ASIA MINERALS CORP.

Copperstone Gold Project

Scoping Study

Volume I

Report and Appendices

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NOTICE

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February 24, 1999

Asia Minerals Corp.
Suite 1480 - 777 Hornby Street
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Attention: Mr. David C. Owens,
President

Dear David:

RE.: Copperstone Gold Project - Scoping Study

We are pleased to submit our report entitled:

Asia Minerals Corp.
Copperstone Gold Scoping Study

We are sending you the balance of the copies of this report.

We thank you for the opportunity to study the Copperstone Gold Project and trust that this report will help you in evaluating the project's development.

Yours truly,
MRDI Canada

DLinden.

David C .P. Lindeman
Study Manager

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Imperial units are used in this report, i.e. feet (ft), tons (t), unless indicated otherwise. Mining and processing rates have been expressed as tons per day (t/d) and tons per hour (t/h). Gold grades have been expressed as troy ounces per ton (opt).

Unless otherwise indicated, all costs and prices in this report are $US, 1st Quarter, 1999.

Common abbreviations used in this report include the following:

Asia Minerals Corp.	AMC
MRDI Canada	MRDI
Golder Associates Inc.	Golder
The Patch Living Trust	PLT
Royal Oak Mines Inc.	RYO
Cyprus Copperstone Gold Corporation	Cyprus
Pincock, Allan & Holt, Inc.	PAH
Environmental Impact Statement	EIS
Aquifer Protection Permit	APP
Load Haul Dump	LHD
Carbon-in-Pulp	CIP
Adsorption, Desorption and Recovery	ADR
General and Administration	G&A
Motor Control Centre	MCC
Gold	Au
Copper	Cu
Standard Reference Material	SRM
End-of-Hole	EOH
Parts per billion	ppb
Quality Assurance/Quality Control	QA/QC
Visual Block Model	VBM
Inverse Distance Weighted to the power 3	IDW3
Inverse Distance Weighted to the power 5	IDW5
Net Present Value (Pre-Tax)	NPV
Discounted Cash Flow Rate of Return (Pre-Tax)	DCFROR

EXECUTIVE SUMMARY

Introduction :

The Copperstone Gold property comprises 284 unpatented mining claims in La Paz County, Arizona, USA. Cyprus Minerals operated a 2,500 tons per day open-pit gold mine on the property between 1987 and 1993. The mine produced 500,000 ounces of gold and was closed at the economic limit of open-pit mining.

Royal Oak Mines Inc., (RYO) explored the property between 1995 and 1997 and discovered a new zone of high grade gold mineralization to the northwest of the open-pit. Asia Minerals Corp., (AMC) purchased a 25% interest in the property from RYO in August 1998 and acquired an option to earn up to an 80% interest. AMC explored the high grade discovery zone with 15 surface drill holes in the 4^{th} quarter of 1998.

MRDI Canada a division of H.A. Simons Ltd., (MRDI) and Golder Associates Inc., (Golder) were retained by AMC in September 1998 to assist in a scoping level evaluation of an underground mine to develop the high grade zones to the northwest of the Cyprus open-pit. The scope of the work provided by MRDI and Golder was to;

- Recommend a QA/QC program for exploration drilling, sampling procedures and laboratory protocols
- Validate the geological and assay database provided by AMC
- Perform a geostatistical analysis of the assay database
- Estimate the geological resource on a MEDSYSTEM™ platform
- Design an underground mine layout and production schedule
- Design a metallurgical process flow sheet
- Identify key environmental issues and mine permit requirements
- Estimate mine capital and operating costs
- Provide a project pre-tax cash flow model and sensitivity analysis
- Outline a project development schedule

Geology :

Gold mineralization occurs principally within the low-angle Copperstone Fault. This is a listric fault related to an underlying mid-Tertiary detachment fault. Gold occurs as native flakes within fault breccia, gouge and shears related to the faulting. Exploration drilling has outlined four zones of mineralization that are down-dip and on-strike of the Cyprus open-pit. These are defined as the A, B, C and D zones according to location, grade and stratigraphy. This scoping study evaluates the underground mine development of the C and D zones only. The database for this study comprises 71 drill holes and 253 sample assays.

Resource Estimate :

The MRDI resource estimate is based on a geological model provided by AMC and an inverse distance weighting to the power 3 block model (IDW3). Gold grades were capped at 2.5 opt Au in the C zone and 4.7 opt Au in the D zone. A 0.00 opt Au block cutoff grade was used for the global geological resource estimate. The results of the MRDI resource estimate are tabulated below;

C and D Zones		**Tons**	**Au opt**	**Au ounces**
Capped		2,085,900	0.340	708,700
Uncapped		2,085,900	0.580	1,209,800
Capped		**Tons**	**Au opt**	**Au ounces**
C Zone	Indicated	478,400	0.194	92,700
	Inferred	696,700	0.323	225,000
	Total	1,175,100	0.270	317,700
D Zone	Indicated	413,800	0.466	193,000
	Inferred	497,000	0.398	198,000
	Total	910,800	0.430	391,000
Total	Indicated	892,200	0.320	285,700
	Inferred	1,193,700	0.354	423,000
	Total	2,085,900	0.340	708,700

There is exploration potential to increase the geological resources in the B, C and D zones; the northern strike extension of the Copperstone Fault; and, in the foot-wall of the Copperstone Fault.

Mining :

Underground access to the C and D zones will be from a portal located in the open-pit and close to the bottom of the existing haul road. The recommended mining method is drift and fill utilizing paste tailings backfill.

The resources available for mining are estimated to be 827,400 tons at a grade of 0.555 opt Au (459,500 ounces Au). These resources are based on the material within a geologic grade envelope of 0.10 opt. Au and greater and having overall diluted grades greater than a mining cutoff grade of 0.25 opt. Au. A mining recovery of 95% and dilution of 10% at a grade of 0.08 opt. Au has been applied.

The recommended mine production capacity for the currently available resource is 520 tons per day or 182,500 tons per year of ore for a mine life of 4.5 years.

Metallurgical Processing :

Gold occurs predominantly in a native form with some quartz and iron-oxide encapsulation. Deleterious metals and carbon minerals are not present. Copper oxides are typically associated with the gold mineralization. No metallurgical test work has been performed for this scoping study.

The process flow sheet is a modified design of the Cyprus recovery system. Run of mine material will be crushed in a two-stage crushing complex and ground in a single stage ball mill circuit to 80% passing 200 mesh. An agitated leach tank system followed by a carbon-in-pulp (CIP) circuit, pressure elution, electrowinning and a smelting circuit will be used to produce gold doré. A gravity concentrator will be used to recover coarse free gold.

A 90% gold recovery has been assumed on the basis of Cyprus historical data. Annual gold production is forecast to be 156,000 ounces in year one and 72,000 in subsequent years. Total mine life gold recovery is estimated to be 413,570 ounces.

Environmental Issues and Mine Permitting :

The underground mine will be developed below the water table and mitigating the impact of mining on the aquifer will be a critical environmental condition. No other critical environmental issues have been identified at this stage. A full base line study, including hydro-geological studies of the aquifer, will be required for environmental and mine operating permits.

A new mine will require approved records of decision for the Environmental Impact Statement (EIS), issued pursuant to the National Environmental Policy Act (NEPA), the Arizona State Aquifer Protection Permit (APP) and an Air Quality Permit. It is estimated that the project could receive all the required permits within two years of submitting the Plan of Operation at an estimated cost of up to US$ 1.00 million. Copperstone Gold is an existing mine site and therefore the permitting costs and time are expected to be less than the typical requirements for a 'greenfield' mine project.

Capital and Operating Costs :

The estimated capital cost is US$ 22.54 million, including direct costs of US$ 14.67 million and indirect costs of US$ 7.87 million. Indirect costs include US$ 1.75 million in owners' costs and a 20% contingency of US$ 3.76 million. These capital costs make no allowance for performing exploratory drilling, geotechnical environmental and metallurgical studies, or performing the feasibility study.

The average mine life operating cost is estimated at US$ 74.52/ton of ore processed. This total includes mining costs of US$ 39.64/ton, processing costs of US$ 25.21/ton and G & A costs of US$ 9.67/ton.

Financial Analysis :

The project base case is a 520 tons per day underground mine and a gold price of US$ 300/troy ounce over a 5 year mine life. The results of a pretax cash flow analysis of the base case are as follows;

• Cumulative Cash Flow	US$ 31.56 million
• Net Present Value (NPV) at a 10% discount	US$ 18.18 million
• Discounted Cash Flow Rate of return (DCFROR)	45.4%
• Capital Payback	1.2 years
• Cash Cost of Production	US$ 149 per ounce

Project economics are most sensitive to changes in mine grade, metallurgical recovery and gold price.

Project Advancement Activities :

The next stage of evaluation is a bankable feasibility study. The work required to complete this study includes surface and underground drilling to establish proven and probable mining reserves, metallurgical testing of a bulk ore sample, environmental and geo-technical studies and detailed estimates of the capital and operating costs. An Environmental Impact Statement and a Plan of Operations will also be required to support mine permit applications.

It is estimated that the above activities can be completed in one year at a cost of US$3.60 million.

1.0 INTRODUCTION AND SCOPE OF WORK

1.1 Introduction

MRDI Canada (MRDI), a division of H. A. Simons Ltd., and Golder Associates Inc.,(Golder) were retained by Asia Minerals Corp., (AMC) to assist in the evaluation of the Copperstone Gold Project, located in La Paz County, Arizona, USA. AMC has a joint venture agreement with Arctic Precious Metals Inc., (APM) a subsidiary of Royal Oak Mines Inc., (RYO) to explore and develop Copperstone.

All pertinent data pertaining to the project are contained at AMC's offices in Vancouver and the Copperstone site. MRDI and Golder personnel visited the Arizona site from October 15 to October 17, 1998, at which time the proposed underground portal locations and potential plant sites were inspected; drilling procedures and results reviewed; and potential environmental and infrastructure-related requirements identified.

1.2 Study Basis

The Tertiary age detachment fault-type gold mineralization present on the property has been actively explored since 1975. The property was initially developed and operated by Cyprus Minerals as a 2,500 tons per day open pit mine in 1987. Exhausted open pit reserves led to the mine closure in 1993. The property was returned to the original holder of the mining claims, the Patch Living Trust (PLT), upon completion of mining and termination of the mining lease. RYO optioned the property in 1995, from the PLT under a 10 year renewable lease contract, and undertook exploration from 1995 to 1997 focusing on the gold mineralization below the floor of the open pit. Diamond drilling of the gold mineralization resulted in the discovery of a new high grade zone in the northwest strike extension of the mineralized deposit. AMC optioned the property in August 1998 from RYO and proceeded with an exploratory drilling program.

By late 1998, AMC had defined the northwest high grade zone, and requested that MRDI carry out an independent review of the available data to assist in their economic assessment of the property. The objective of this study is to provide AMC with a "Scoping Level" study to determine the economic and technical viability of developing a small to medium scale underground gold mine at Copperstone. The major component of this study was a 14 hole, 12,000 ft, surface drilling program that was used to develop a pre-feasibility level engineering and environmental development plan.

1.3 Scope of Work

The scope of work for this study involved preparation of the following deliverables to be completed to a scoping level:

- Quality control review of project exploration drilling programs at sample logging, preparation and assaying stages.
- Review and endorse AMC's resource model's assay and geological database, with subsequent calculation of resource estimate on a MEDSYSTEM™ platform.
- Concept to develop, mine, and process the recoverable resource.
- Ore production forecast to supply feed material for processing.
- Process layout scheme and block diagram.
- Site plan and facility layout showing location of plant site, site roads, surface decline and infrastructure.
- A list of potential environmental liabilities and an evaluation of the associated risk and costs based on regulatory requirements.
- A regulatory review and recommended strategy for the permitting of a new underground operation at the Copperstone Mine.
- Estimates of capital and operating costs.
- A Project pre-tax Cash Flow analysis.

1.4 Limitation and Assumptions

MRDI's assessment was based on a review of existing reports on the Copperstone Property prepared by others, a site visit, and information gained from publications and preliminary inquiries. Not all of the information has been verified; detailed engineering and cost estimates were not made. Therefore attention is drawn to the following exclusions from, and limitations to, MRDI's report.

- Verification of drill hole collars, accuracy of site drill core logs and journals and assay value results, was not completed by MRDI.
- MRDI did not complete a title search nor determine the status of the land or mineral title tenure.
- There are no data on the relationship of the ore grind size to cyanide leach gold extraction. The preliminary process design data was based on Cyprus Copperstone Gold Corporation (Cyprus) engineering records.

Many of the design data were not available and thus engineering work standards and judgements had to be made to complete this scoping study. A list of the major assumptions by study area is as follows;

1.4.1 Geology

- Mineralization is continuous within the outlines of the area drilled to date, and extends up to 140 feet beyond the outermost holes.
- The global geological resource will use an assay cap of 2.5 opt Au for C-Zone and 4.7 opt Au for D-Zone.

1.4.2 Mining

- The mineable reserve was factored from the geological resource. A dilution of 10% and recovery of 95% was applied. A grade of 0.08 opt Au was assumed for diluting material.
- There are no geotechnical concerns affecting the selection of excavation method or waste dump stability.
- Natural water seepage into the underground workings will be controlled.
- The underground mine will be developed and operated by a mining contractor.
- The mine will be accessed by a ramp driven from the bottom of the existing open pit. The portal will be located in the west pit wall.
- Drift and fill mining will be the selected mining method.
- Paste tailings will be used for mine backfill.
- The waste rock will not generate acid rock drainage.

1.4.3 Process

- Gold recovery estimates were based on assumed industry standards for Carbon-in-Pulp (CIP) operations as well as data from a documented project summary dated June 1997, issued by RYO, on the Copperstone property. Gold recovery was estimated to be 90% at a grind size of 80% passing 75 micron (200 mesh). MRDI has not reviewed any metallurgical studies on the deposit.
- The ore is amenable to treatment by conventional crushing, grinding, carbon adsorption and gold elution recovery systems.
- Provision will be made to direct underflow, from a single hydrocyclone unit, to the gravity concentration circuit for free-gold recovery.
- Ground pulp residue can be thickened to 75 percent solids by weight.
- Cyanide in solution will be recycled from the washed thickened tailings, whilst weak acid dissociable cyanide complexes (CN_{WAD}) contained within the thickened tailings underflow will be detoxified with the addition of sodium hypochlorite solution.
- Hydraulic deposition of plant tailings to a new or existing tailings impoundment facility has not been considered. A paste backfill system has been identified as the most economic tailings deposition program by underground stope placement or disposal into the existing open pit.
- Copperstone tailings will contain sufficient ultra-fine material (less than 20 micron) to form a stable paste. No testwork to confirm paste fill cement addition has been performed.

1.4.4 Infrastructure and Environmental

- Land tenure is secured for the area in the site layout identified as process facilities.
- The 1993 closure plan and associated issues for the Copperstone Gold Mine have been approved and implemented by Cyprus.
- Copperstone will be developed as a zero discharge facility whereby all process water shall be contained in a closed loop for reclaim.
- It is proposed to limit the mine disturbance to the open pit area, the mill site and existing mine haul roads.

- Pre-feasibility level reviews of various state and federal permits required for an underground mining plan will be made by Golder, Tucson, Arizona.
- For surface deposition of plant tailings World Bank dischargeable water quality guidelines can be achieved at Copperstone using the sodium hypochlorite detoxification technology.

1.4.5 Capital Cost

- All mining and process equipment will be purchased as new.
- Owner's costs have been included in the cost estimate.
- Estimated costs associated with underground exploration, development drilling, metallurgical testing, geotechnical, hydrology, environmental studies and producing a bankable feasibility document have been excluded from the cost estimate.

1.4.6 Operating Cost

- Salaries for AMC mine employees will be loaded with burdens of 35 percent.
- Pulverized pebble lime and cement will be supplied by bulk tanker truck from within the continental USA.

1.4.7 Financial Model

- Base case gold price will be US$ 300 per troy ounce.

2.0 GENERAL PROPERTY DESCRIPTION

2.1 Location and Access

The Copperstone project is located on public land in La Paz County, Arizona, approximately 9 miles north of Quartzite, see Figure 2.1 and Figure 2.2. The site is accessed by 4 miles of gravel road from Arizona Route 95. The highway and site access road are suitable for the transportation of major project components. Driving time from Phoenix to the mine site is approximately two hours.

The few small buildings remaining on the project site are currently used as offices for the geological exploration crew and a warehouse for mining surface equipment. Three shipping containers are used by AMC as secure storage for exploration reverse-circulation cuttings and diamond drill core.

2.2 Topography and Climate

The Copperstone deposit is favorably located in flat desert terrain. A main line of the Santa Fe railroad passes to the north of the property. Process water, from site surface wells, and overhead commercial power is routed to the project site.

The climate in the area is classified as hot, dry desert . Occasional high wind conditions produce periods of naturally blowing sand. The predominant wind directions are from the north and south. Pertinent climatic statistics for the area are as follows:

Average daily temperature, April to October	-	105°F
Extreme annual temperatures	-	121°F maximum
	-	20°F minimum
Average annual rainfall	-	4 inches

Figure 2.1: Location Map - Copperstone Project

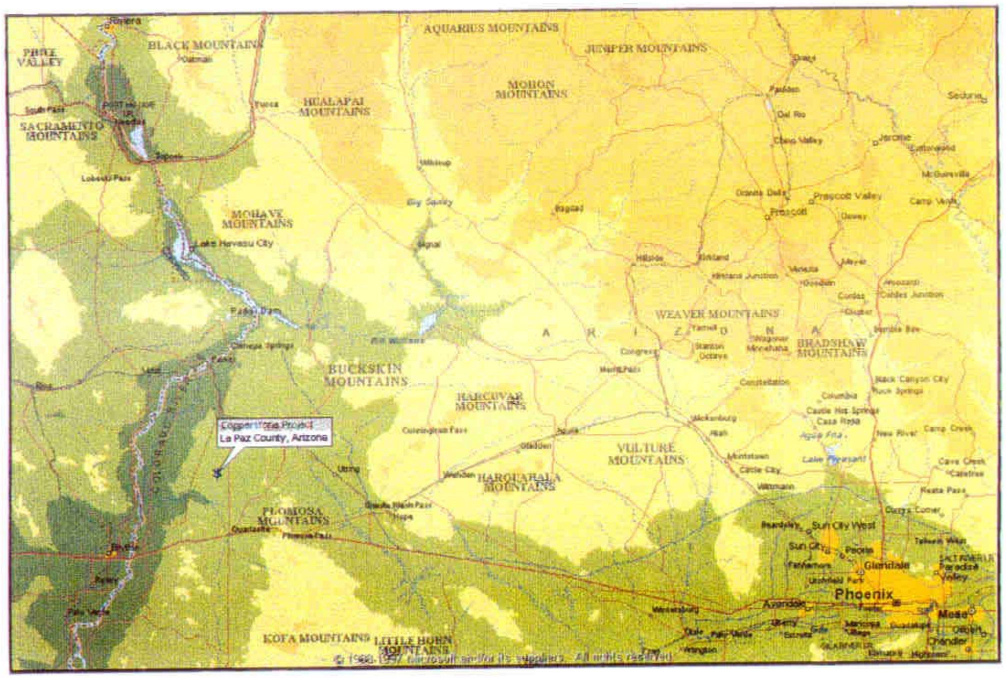


Figure2.2: La Paz County, AZ

Water resources in the region are derived from precipitation or from surface water recharge. The region is the driest area in the United States and large areas are categorized as arid and semi-arid. Annual precipitation generally is related to altitude of the land surface.

Topography in the area is flat to moderate, located on a dry, sandy terrain. Several small knolls and prominent longitudinal northeast trending sand dunes occur in the project area. Surface elevations range from 725 to 900 feet above mean sea level.

2.3 Project Background

The Copperstone property and environs were first identified as a copper prospect in 1968. In 1975, the Newmont Gold Company leased the property, from the original prospectors, and drilled a single hole in an unsuccessful attempt to outline weak porphyry copper mineralization. Dan Patch (PLT) acquired the current claims in March 1980 and leased the claims to the Cyprus Mines Corporation (Cyprus).

Cyprus acquired a 100 percent working interest in the original 65 Copperstone claims, expanded the property to 284 claims and proceeded to outline broad areas of gold mineralization throughout the property. Drilling campaigns from 1980 to 1985, totaled over 400 reverse-circulation and 70 diamond drill holes and resulted in Cyprus conducting baseline economic studies, metallurgical testwork programs and financial analysis, leading to mine construction in 1986.

The proposed open pit area was drilled off in 1986 and a decline was driven from surface to intercept the resource at an elevation approximately equal to the current open pit bottom. In 1987 open pit mine stripping commenced and ore was treated at a rate of 2,500 tons per day through a CIP milling operation.

Open pit mining continued through to late 1992 in accordance with original mining permits which allowed mining to proceed to the groundwater table. In early 1993, open pit blast hole drilling penetrated the water. Mining was halted at the economic limit of open pit mining and all surface ore stocks were processed. Upon completion of the milling operations in 1993 Cyprus removed the majority of the mine and mill equipment, however, much of the mill infrastructure remains operational, including the power, water and communications systems.

The mine was closed with the approvals from the Arizona Department of Environmental Quality Aquifer Protection Permit and the Bureau of Land Management. The mine and waste dump closure plan and reclamation study of surface disturbance associated with the mine was implemented and operated by Cyprus.

Santa Fe Pacific Gold Corp., (Santa Fe) leased the property in 1993 and completed 12,500 ft of reverse-circulation drilling on 7 exploration targets. Optioned in 1995 by RYO, a total of 35 exploration and development drill holes totaling 28,300 ft were completed between 1995 and 1997. The majority of the RYO upper interval drill holes were reverse circulation with coring performed over the last several hundred feet to focus on the deep extension of mineralization in the Copperstone fault, below, and to the north of the existing pit. This drilling outlined an underground geological resource of 2.62 million tons at a grade of 0.23 opt containing 606,000 ounces of gold.

The geological resource included a new high grade zone on the northwest margin of the deposit, containing a diluted resource (10% dilution at 0.0 opt) of 455,000 tons at 0.48 opt gold.

In August 1998 AMC entered into joint venture agreement with RYO to explore and develop the Copperstone property. AMC purchased a 25% interest in the property and acquired an option to earn up to an 80% interest.

3.0 GEOLOGY

This section was prepared by AMC for inclusion in this report. MRDI has reviewed the geological model used as the basis for the resource estimate and found it consistent with AMC's views as presented in this section.

3.1 Regional Geology

The Copperstone gold deposit is located in the Basin and Range province of the southwestern United States. The most significant regional structures associated with gold mineralization in the Copperstone area are Mid-Tertiary age detachment faults. Typically, these faults separate "lower plate" Precambrian to Mesozoic rocks, comprising metamorphic core complexes with Tertiary intrusions, from "upper plate" Triassic to Tertiary volcanics. Upper plate rocks deform by brittle fracture which produces normal and listric fault patterns. The rock fracturing associated with the faulting is a significant control to the mineralization. Lower plate rocks deform in a more ductile fashion and develop mylonitic fabrics. High angle Tertiary Basin and Range type faults cut the detachment faults and are related to the same extensional events.

Table 3-1, Geologic History Of The Copperstone Area

Age	Event
Mid-late Tertiary	basin and range normal extensional faulting
Mid-Tertiary	detachment faulting, mineralization, intrusions metamorphism, formation of core complexes
Late Cretaceous	intrusion of plutons
Triassic-Jurassic	volcanic-plutonic rocks, thick clastic sequences
Paleozoic	carbonate and clastic sedimentation erosion, development of unconformity
Precambrian	Metamorphic rocks, accompanying intrusions

The critical structural control to the mineralization at Copperstone is the Copper Peak Detachment Fault (CPDF). This fault strikes approximately east-west and dips gently NNE at an exposure 1.5 miles south of the Copperstone open-pit. Drilling in the mine area has not penetrated the CPDF.

3.2 Copperstone Geology

3.2.1 Stratigraphy

The oldest rocks observed at Copperstone are Triassic phyllites of probable volcanic origin. Triassic metasediments overly the phyllites and comprise an upward fining sequence of quartzite, chlorite schist and marble. This sequence is the principal host for high grade gold mineralization discovered to the north of the Cyprus open-pit (D zone).

Jurassic quartz latite porphyry (QLP) intrudes the Triassic metasediments. The QLP hosts the gold ore mined in the Cyprus open-pit and is the predominant lithologic unit in the area.(Fig. 3.4). An early Miocene monolithic breccia (MSB) overlys the QLP and is interpreted as a sub-aerial sedimentary unit. Younger Miocene basalt occurs at the southern limit of the open-pit. The property is covered by unconsolidated sand and gravel overburden to a depth ranging from 0 ft. to over 600 ft. The stratigraphy is described in Table 3.2.

Table 3-2, Copperstone Stratigraphy

Age	Name	Description
Early Miocene	Basalt	Basalt to andesite. Cut by mineralized amethyst-quartz-specularite veins to the SW of the pit where economic mineralization developed.
Early Miocene	Monolithic Breccia (MSB)	Monolithic fragments derived from Jurassic QLP. Locally developed above the Copperstone fault. Hematization and quartz - specularite mineralization. Contains economic gold mineralization. A sub-aerial sedimentary unit (chaotic breccia?)
Jurassic	Quartz Latite Porphyry (QLP)	Volcanic flows with well developed metamorphic foliation. The principle ore host in the pit where it occurs in both the hangingwall and footwall of the Copperstone Fault. Where cut by the Copperstone Fault, a brecciated and mineralized interval about 50 ft thick is developed. A minimum 900 ft thickness is indicated by drilling.
Triassic	Meta-sediment unit	A fining upwards sedimentary cycle; quartzite, chlorite schist (siltstone) and marble (limestone). The principle host rocks for D-Zone.
		Marble or limestone (LST) occurs at the top of the meta-sediments. It contains intervals of massive specular hematite +/- manganese oxide and secondary Cu minerals as veins and in nodular replacements. The mineralization and brecciation observed in the unit is related to the Copperstone Fault.
		Chlorite schist or siltstone (SLT). This unit typically occurs at the quartzite-marble transition or interbedded within the marble.
		Quartzite (QTZ) is present in the D-Zone area at the base of the meta-sediment package. Characterized by vein and stockwork stringer mineralization
Triassic	Phyllite (PHY)	The oldest exposed unit in the upper plate, up to 300 ft thick in drillholes. Phyllite only occurs in the footwall of the Copperstone Fault in the north part of the pit and in the D-Zone and C-Zone drillholes.

3.2.2 Structure

The Cyprus open-pit is developed in the Jurassic QLP and Miocene MSB stratigraphy. The rocks strike NW and dip at 25 to 45 to the NE. A fault close to the north wall of the open-pit exposes the older Triassic metasediments which extend north of the open-pit into the D zone discovery area. These rocks also strike to the NW and dip at 25 to 45 to the NE.

3.2.2.1 The Copperstone Fault

The excepted regional model for the Copperstone gold deposit is based on a detachment-fault control to the mineralization. The mineralization is locally controlled by the low angle Copperstone Fault which is interpreted to be a listric splay fault related to the underlying Copper Peak Detachment Fault. High angle NW, NE and WNW normal faults cut the Copperstone Fault and are also important structural controls to the distribution of mineralization. In the open pit, the Copperstone Fault dips NE at 25° to 45° and strikes NNW-SSE. This low angle fault is approximately conformable with the dip and strike of the Triassic-Jurassic stratigraphy. Consequently, the fault occurs predominantly within the Jurrasic QLP in the open-pit and close to the contact of the Triassic metasediments and Jurassic QLP to the north of the open-pit in the D zone. (Fig. 3.4).

3.2.2.2 High Angle NW, WNW and NE Normal Faults

Mapping in the open-pit identified several near vertical and mineralized NW-SE faults. A closely related set of WNW-ESE faults may also control mineralization. Steeply dipping NE-SW faults also appear to influence alteration and gold mineralization in the hanging-wall of the Copperstone Fault. The mineralized Copperstone Fault is offset about 300 ft in a left lateral sense by a NE-SW fault in the south of the open-pit.

3.3 Mineralization

The Copperstone gold-copper mineralization occurs predominantly within the Copperstone Fault. Gold occurs in association with breccia, gouge, shearing and fracturing related to the fault. Specularite, chrysocolla, malachite, siderite, manganese-oxide, adularia and magnetite are commonly associated with the gold mineralization. Massive specular hematite and manganese-oxide replacement mineralization commonly occurs in carbonate rocks within or close to the Copperstone Fault. NW-SE trending gold-amethyst-quartz-specular hematite veins occur in the QLP within the open-pit.

Petrographic studies by Cyprus show that about 80% of the mineralization occurs as native gold flakes ranging in size from 4 to 40 microns. Gold flakes up to 150 microns in size have been reported in thin section studies and in gravity concentrate samples. Gold encapsulated in amethyst, quartz, iron-oxide and calcite has also been identified. Visible gold is commonly observed in D zone drill core intersections. The average grade of the mineralization increases from 0.1 opt gold in the Cyprus open-pit to about 0.5 opt gold in the C and D zones to the north and northeast of the open-pit. The significant drill intersections in C and D zones are included in Appendix B.

Copper mineralization associated with the Copperstone Fault occurs as chrysocolla with minor malachite and azurite. Chrysocolla is interpreted to be a primary hydrothermal mineral. The average copper content of the mineralization ranges from 0.32% Cu in assay composites from the open-pit to 0.96% Cu in assay composites in the C and D zones to the north and northwest of the pit. Silver values in the Copperstone mineralization are typically less than 1.5 ppm. Arsenic, antimony and mercury do not occur above background levels and no carbon mineral impurities have been reported. Gold mineralization is typically iron and manganese rich due the presence of massive iron and manganese oxide in the Copperstone Fault.

The reported paragenetic sequence for the mineralization and hydrothermal is shown in Table 3.3

Table 3-3, Principle Phases of Alteration and Mineralization

Alt/Min Phase	Description
Oxidation	Host rocks are oxidized, often producing earthy red hematite. [Most specularite and chrysocolla are primary and predate oxidation. Sulfides are rarely observed.]
Post mineral veins	Quartz-fluorite-barite-hematite veins
Late stage mineralization	Fine grained quartz and earthy hematite with minor chalcopyrite, chrysocolla, malachite. *Auriferous.*
Early stage mineralization	Amethyst-quartz-chlorite-specularite veins/replacements. *Auriferous.* Pyrolusite is a commonly associated mineral. Well developed in meta-sediments, includes massive Fe-oxide replacement of marble in D-Zone. In volcanic host rocks, characterized by thinner veinlets with open space filling textures.
Propylitic alt'n.	Pre-mineralization phase
Potassic alt'n	Pre-mineralization phase

The 1998 drilling program in C and D zones indicates that gold mineralization is strongly associated with late stage iron-rich hydrothermal chlorite and silica alteration.

3.4 Descriptions of The Resource Zones

Exploration drilling at Copperstone has identified gold mineralization both down-dip and on-strike to the north of the Cyprus open-pit. The grade, style, geometry and host stratigraphy of this mineralization varies according to location and is therefore separated into the A, B, C and D zones.(Fig. 3.4)

The 1998 AMC drilling program explored the C and D zones only. The result of this program is a new and comprehensive geological model which is the basis for the MRDI block model and resource estimate described in Section 4.0. The geological model incorporates all available geological data from Cyprus, RYO and AMC drilling, together with Cyprus pit mapping and blast hole assay data. This detailed information provided geological evidence of structural and stratigraphic sub-zones of mineralization within the C and D zones. These sub-zones are incorporated in the MRDI block model and resource estimate and are described below.

3.4.1 A and B Zones

This scoping study is based on the C and D zones only and does not include the A and B zones. The A zone is the NE down-dip extension of the Cyprus ore body below the southeast corner of the open-pit. The B zone is the equivalent down-dip extension in the center of the open-pit. These two zones are tabular in shape and occur within the Copperstone Fault. The results of a 1998 RYO resource estimate for A and B zones are given in Table 3.4. AMC and MRDI have not validated this estimate.

Table 3-4, RYO Uncapped Resource Estimate : A and B Zones

Zone	Tons	Grade (opt Au)	Ounces Gold
A Zone	222,084	0.149	33,000
B Zone	553,977	0.168	93,000

3.4.2 C-Zone

The C zone is the NE down-dip extension of the Cyprus ore body in the NE lobe of the open-pit. Bench assay plans and geological observations from the bottom of the open-pit indicate that gold grade and continuity of mineralization were above average for the mine in this area. The mineralization occurs predominantly within the Jurassic QLP stratigraphy.

The C zone in the geological and resource model has been subdivided into the C1, C2, and C2A zones. (Fig.3.4) Two high grade intersections on the eastern boundary of the C zone are classified as a distinct hanging wall zone (HW3). A NE-SW cross fault defines the boundary between the C and D zones.

3.4.2.1 C1-Zone

C1 is a tabular zone of mineralization that extends from Section 132 to Section 147, a strike distance of 1,150 ft. The zone dips NE at 20°-35° and has a width of about 360 ft. The average thickness is 15 ft. C1 is separated from the parallel and up-dip C2 zone by a NW-SE fault.

3.4.2.2 C2 and C2A-Zones

C2 is a tabular zone up-dip and separated from C1 by a steep NW-SE fault. The zone extends from Section 132 in the south to Section 146 in the north, a strike distance of about 1,000 ft. C2 is 260 ft. in width and dips 20 ° - 35 ° NE. C2A is separated from C2 by a waste zone with a 15 - 40 ft. thickness. C2 and C2A zones are both well defined in the Cyprus blasthole assay plans for the lowest benches in the northeast lobe of the open-pit.

3.4.3 D-Zone

The D zone is located northwest of the Cyprus open-pit and was discovered by RYO in 1995. High grade gold mineralization in D zone occurs in a tabular wedge within Triassic metasediments below and within the Copperstone Fault. NW-SE and NE-SW normal faults intersect the D zone and separate the mineralization into discrete blocks that have been classified as D1, D1A, D1B, D2, D2A, D3 and D4 in the geological and resource models. (Fig. 3.4)

3.4.3.1 D1-Zone

D1 is a tabular wedge of mineralization extending over a 350 ft. strike interval between Sections 153 and 158. The northern boundary of this zone is not defined by current drilling. D1 dips 25° NE within the Copperstone Fault and has a down plunge extent of 500 ft. The average thickness of D1 is about 15 ft and the maximum observed thickness is 33 ft. in drillhole A98-5. In the down plunge direction, D1 projects towards an area of cataclastic breccia and possible NE faulting near drillholes A98-12 and C97-25.

The up-plunge extent of D1 terminates against a NW-SE fault. D2 continues to the south of this fault. D1A and D1B are minor splay zones in the hanging-wall and foot-wall of D1.

3.4.3.2 D2-Zone

The D2 zone is within a narrow graben-like fault block located between two NW-SE faults. It extends from Section 152 to Section 154.

3.4.3.3 D3-Zone

D3 extends from Section 146 to 152 and occurs within the same fault block as D2. The two zones are separated by a NE-SW cross fault. The faults that outline the boundaries of D2 and D3 are defined by consistent changes in thickness of the Triassic metasediments plus intra-formational stratigraphic changes.

3.4.3.4 D4-Zone

D4 is a west dipping conjugate zone within Triassic carbonate rocks. The zone is located northwest of the fault block that hosts D2 and D3.

3.5 Property Exploration Targets

There is strong exploration potential on the Copperstone property to expand the existing geological resources and to discover new zones of gold mineralization. The major exploration targets are described below;

3.5.1 D - Zone

The high grade D zone is open on-strike to the north. Further drilling is required north of Section 157. In-fill drilling is also required between Sections 148 and 157 to fully define the up and down-dip limits of mineralization.

3.5.2 Copperstone Fault

The Copperstone Fault is a major structural control to mineralization on the property. The projected strike extension of this fault has not been drill tested north of Section 160.

3.5.3 C - Zone

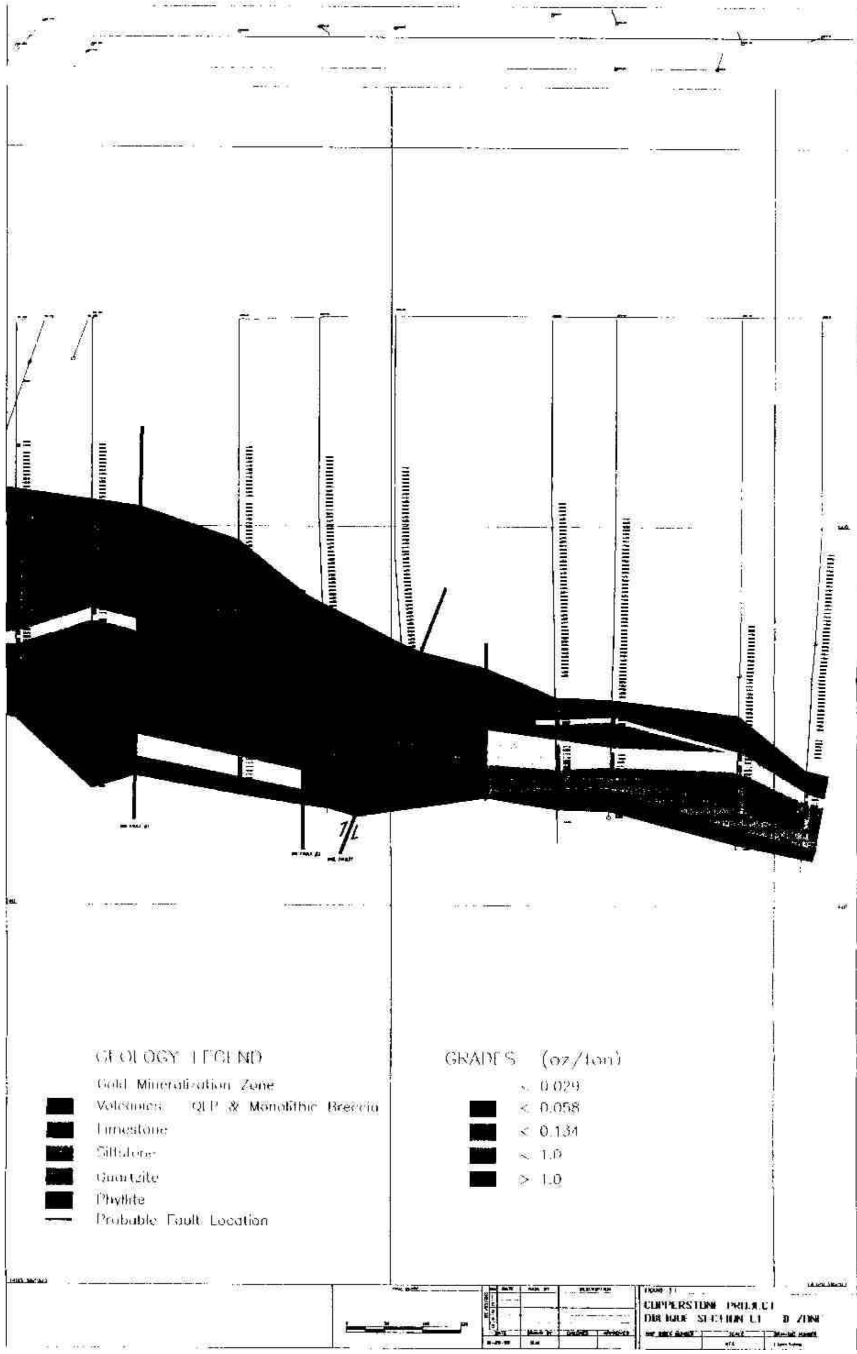
The down-dip extension of high grade gold mineralization in the floor of the Cyprus open-pit in the C- Zone has not been adequately tested by diamond drilling.

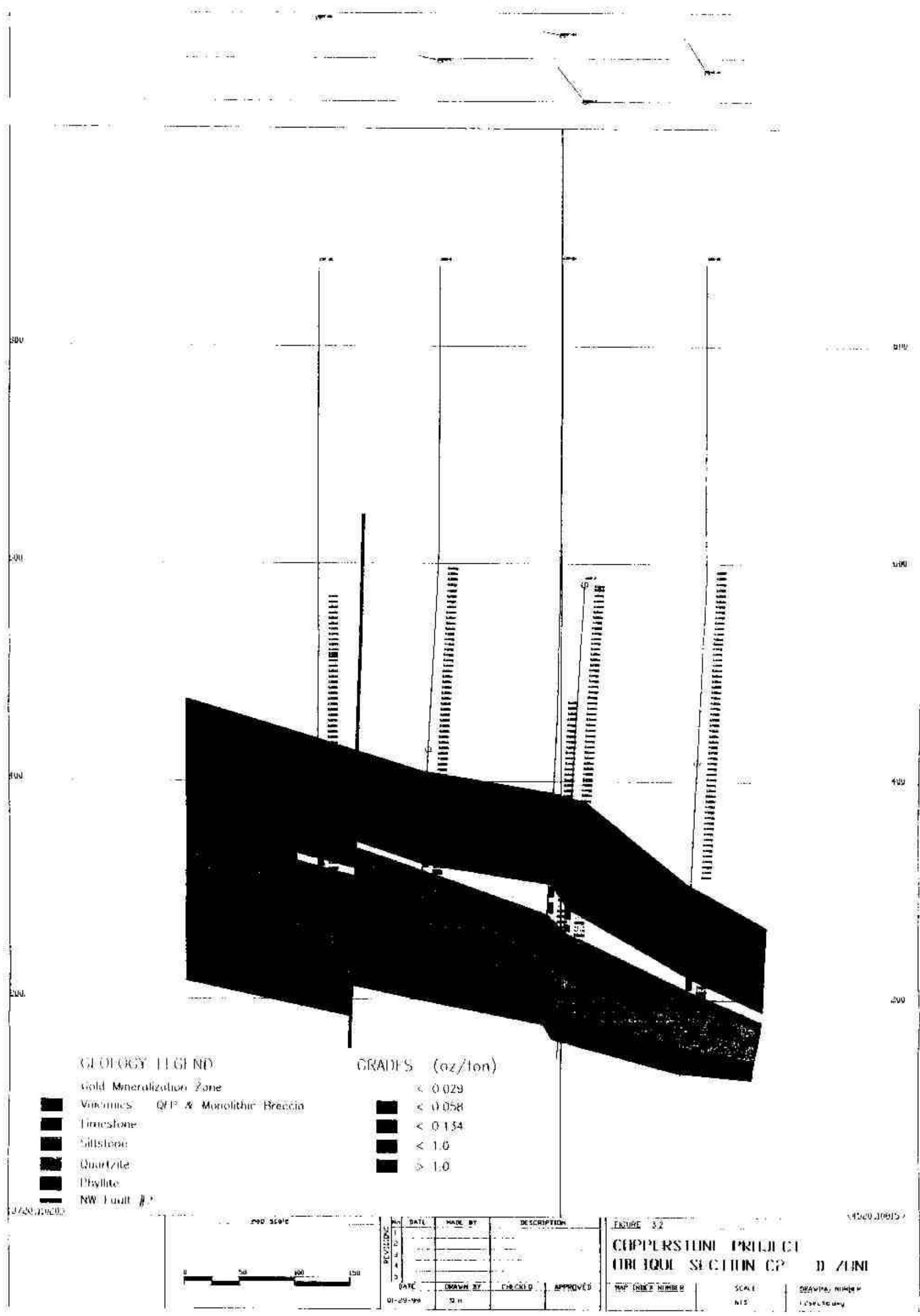
3.5.4 B - Zone

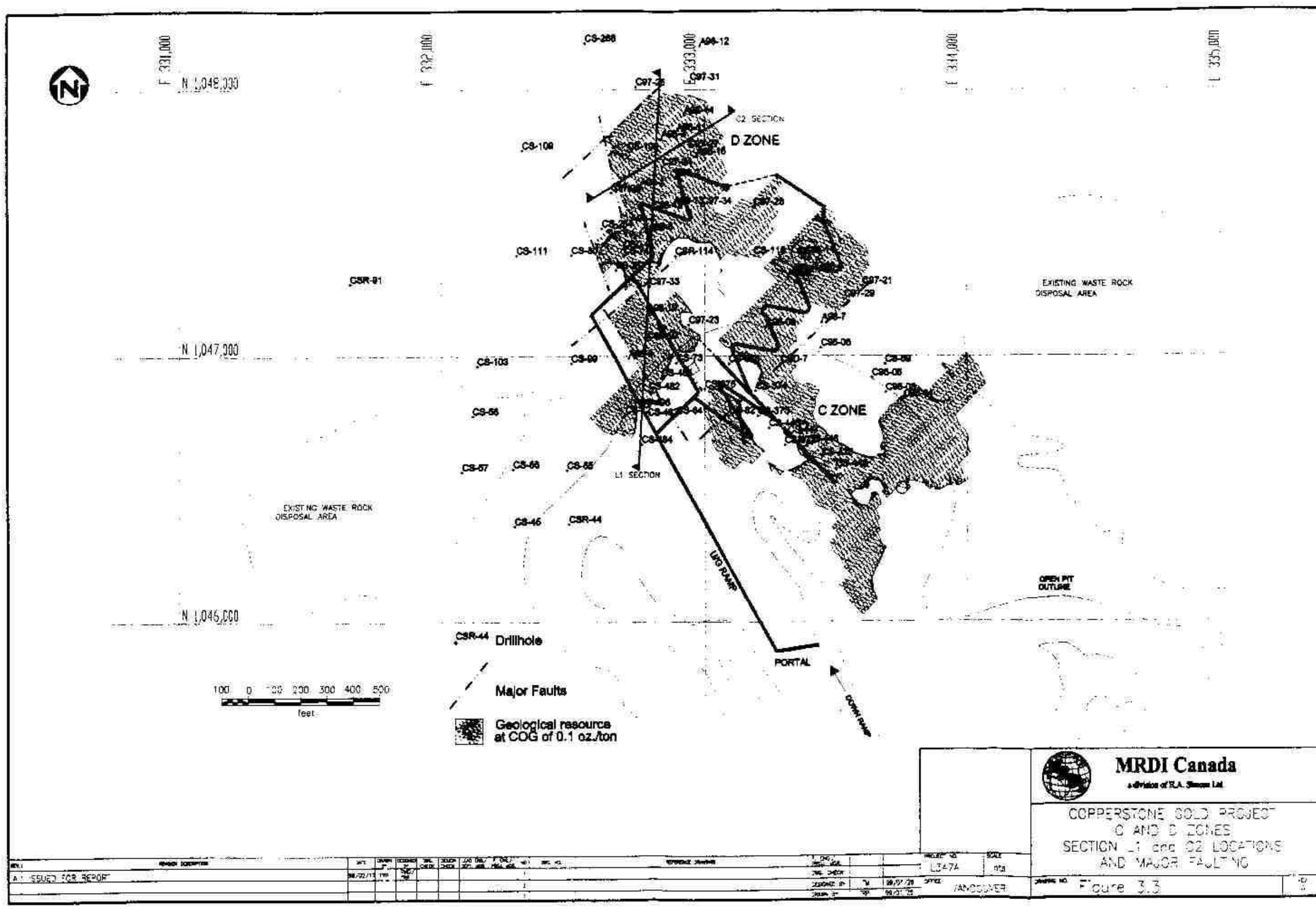
The geology of B- Zone is not well understood. Down-dip and structural off-sets of mineralization have not been fully evaluated or drill tested.

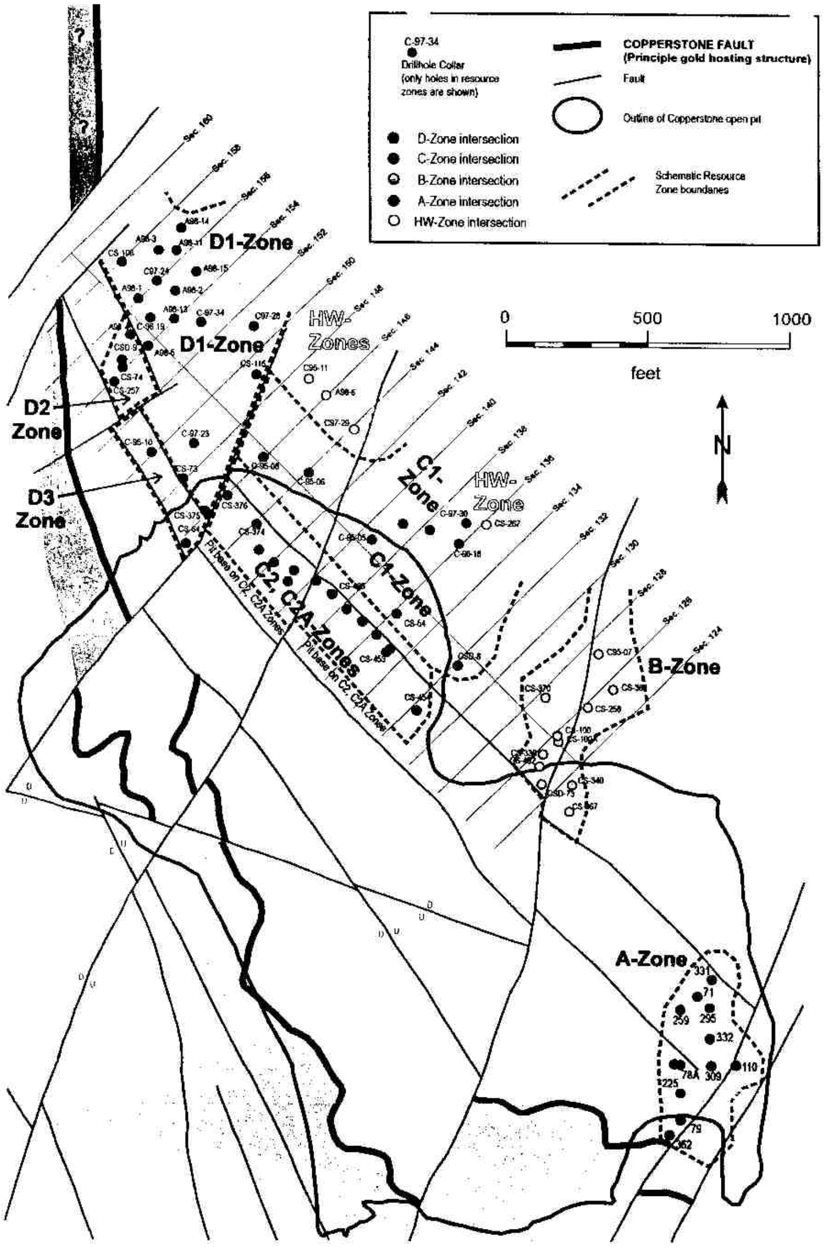
3.5.5 Footwall Structures

A single drill hole, DCU-8, intersected 15 ft. of gold mineralization grading 0.646 opt about 400 ft. below the open-pit in the footwall of the Copperstone Fault. Footwall mineralization also occurs in holes CS-143 (0.268 opt Au over 40 ft.) and CS-67 (0.139 opt Au over 30 ft.) Further drilling is required to test the large scale potential for new zones of mineralization in footwall structures parallel to the Copperstone Fault.


GEOLOGY LEGEND
Gold Mineralization Zone
Volcanics - QLP & Monolithic Breccia
Limestone
Siltstone
Quartzite
Phyllite
Probable Fault Location
GRADES (oz/ton)
< 0.029
< 0.058
< 0.134
< 1.0
> 1.0
COPPERSTONE PROJECT
OBLIQUE SECTION L1 - D ZONE


GEOLOGY LEGEND
Gold Mineralization Zone
Volcanics QFP & Monolithic Breccia
Limestone
Siltstone
Quartzite
Phyllite
NW Fault
GRADES (oz/ton)
< 0.029
< 0.058
< 0.154
< 1.0
> 1.0
0
50
100
150
DATE
DRAWN BY
CHECKED
APPROVED
01-29-98
MADE BY
DESCRIPTION
REVISIONS
FIGURE 3.2
COPPERSTONE PROJECT
OBLIQUE SECTION CP D ZONE
MAP INDEX NUMBER
SCALE
NTS
DRAWING NUMBER


N 1,048,000
N 1,047,000
N 1,045,000
E 331,000
E 332,000
E 333,000
E 334,000
E 335,000
D ZONE
C ZONE
C2 SECTION
L1 SECTION
U/G RAMP
PORTAL
DOWN RAMP
EXISTING WASTE ROCK DISPOSAL AREA
EXISTING WASTE ROCK DISPOSAL AREA
OPEN PIT OUTLINE
CS-288
A96-12
C97-31
CS-109
CS-111
CSR-114
CSR-91
C97-33
C97-23
C97-21
C97-29
A96-7
CS-103
CS-99
CS-56
CS-57
CS-58
CS-55
CS-45
CSR-44
CSR-44 Drillhole
Major Faults
Geological resource at COG of 0.1 oz./ton
100 0 100 200 300 400 500
feet
MRDI Canada
COPPERSTONE GOLD PROJECT
C AND D ZONES
SECTION L1 and C2 LOCATIONS
AND MAJOR FAULTING
Figure 3.3
VANCOUVER
ISSUED FOR REPORT



Tertiary Basalt	**Asia Minerals Corp.**
Jurassic Quartz Latite Porphyry Hanging wall sequence	COPPERSTONE GOLD PROJECT Schematic Plan of the Geology and Resource Zones
Jurassic Quartz Latite Porphyry Foot wall sequence	
Triassic Metasediments	
Triassic Phyllite	Figure 3.4 January 1999 / ptgeo99 detail2.cdr

4.0 GEOLOGIC RESOURCES

4.1 Data Compilation and Analysis

The Copperstone assay database supplied to MRDI contains 30,391 assays from 586 exploration and ore outline drill holes completed during the period 1980 to 1998. A subset of this database containing 71 drill holes with 253 associated assays has been used in this scoping study to develop the geological and resource model of the C and D zones in the Northwest High Grade Zone. Details of the database construction and loading procedures are included in Appendix A.

The assay database was validated through the generation of collar plots, cross-sections and exploratory data analysis. All the drill holes used in the geological model were plotted to:

1. confirm the loading of the drill holes in MEDSYSTEM™
2. provide a check on the drill hole coordinates, downhole survey data and topography, and
3. verify the geological interpretations.

Assay certificates, drill logs and down hole surveys from selected holes (15 holes or 21%) were verified as correct input into the database. Minor errors and omissions observed were detected and corrected in the database.

4.2 Assay Data

AMC completed a 15 hole drill program in November 1998. This program objective was to explore the C and D zones in the Northwest High Grade Zone. Each hole was drilled from surface by reverse circulation to a predetermined depth and then core drilled through the target interval. Boart Longyear was the contractor for the drilling program. All holes were HQ (2 3/4") diameter core drilled through the mineralized zone while the reverse circulation holes were a standard 5 1/2" in diameter.

The primary assay laboratory was Intertek Testing Services (Bondar Clegg). Samples were prepared in Sparks, Nevada and assayed in Vancouver using a 2 assay ton pulp sample. A fire assay protocol with an Atomic Absorption

(AA) finish was used on all samples. A gravimetric finish was used on all samples above the upper limit for AA finish (10,000 pbb). Coarse blanks, reject duplicates, core shed reference samples and CANMET Standard Reference Material (SRM) were inserted blind into the sample stream.

Metallic screen analysis was completed on samples within significant intervals, including low grade samples, to evaluate the coarse gold content at Copperstone. A comparison of metallic screen with fire assays realized an increase up to 9.16% in Au grade. This indicates a potential upside and future assaying protocol should incorporate metallic screen analysis.

Down hole surveys were completed in core holes only. The first reading was taken within 40 feet after the end of the casing, thereafter readings were taken every 200 feet using a Kodak-Eastman single-shot. If more than 3º deviation was noted additional readings were taken.

Core recoveries were reviewed for the holes completed to date (Appendix A—*Site Visit*). The core recovery in the drill holes reviewed within the mineralized gold zone was 90-100%. The mean core recovery for the drilling completed by AMC was 92%. Grade as a function of core recovery was not reviewed to establish whether any significant bias is resulting using samples with low recovery. No data was available to review core recoveries for drill holes completed by RYO or Cyprus.

Data was not available to compare assayed grades in rotary drilling and reverse circulation intersections with core drilling except for a few cases. If future resource estimation is to be based on rotary, reverse circulation and core drilling then this data needs to be reviewed.

The populations for each of the different sources of data were reviewed as shown in Table 4.1.

Table 4.1: Copperstone Assays Summary Statistics

Mineralization Zone	Number of Samples	Mean Au, opt	Coefficient of Variation	Minimum Au, opt	Maximum Au, opt
D1	55	0.7860	2.5810	0.0000	12.0099
D2	11	0.1900	1.1600	0.0029	0.6665
D3	8	1.3820	1.0770	0.0034	3.7434
D4	7	0.1000	0.7690	0.0070	0.2300
D1A	3	0.0900	0.3030	0.0642	0.1180
D1B	4	0.1840	0.8000	0.1820	0.3591
D2A	7	0.0600	0.5320	0.0133	0.0954
C1	23	0.2380	1.5780	0.0020	1.4615
C2	80	0.1820	1.0370	0.0040	1.0550
C3	3	0.1110	0.2930	0.0780	0.1430
C2A	31	0.1260	0.7580	0.0160	0.3110
HW1	11	0.1350	1.9080	0.0015	0.8751
HW2	3	0.0950	0.7930	0.0190	0.1692
HW3	6	7.7550	1.3890	0.0104	23.5189
HW4	1	0.2150	0.0000	0.2150	0.2150
Other	**Number of Assays**	**Mean Interval, ft.**	**Coefficient Of Variation**	**Minimum Interval, ft.**	**Maximum Interval, ft.**
Assay Interval	253	5.7	0.39	1.0	10.0

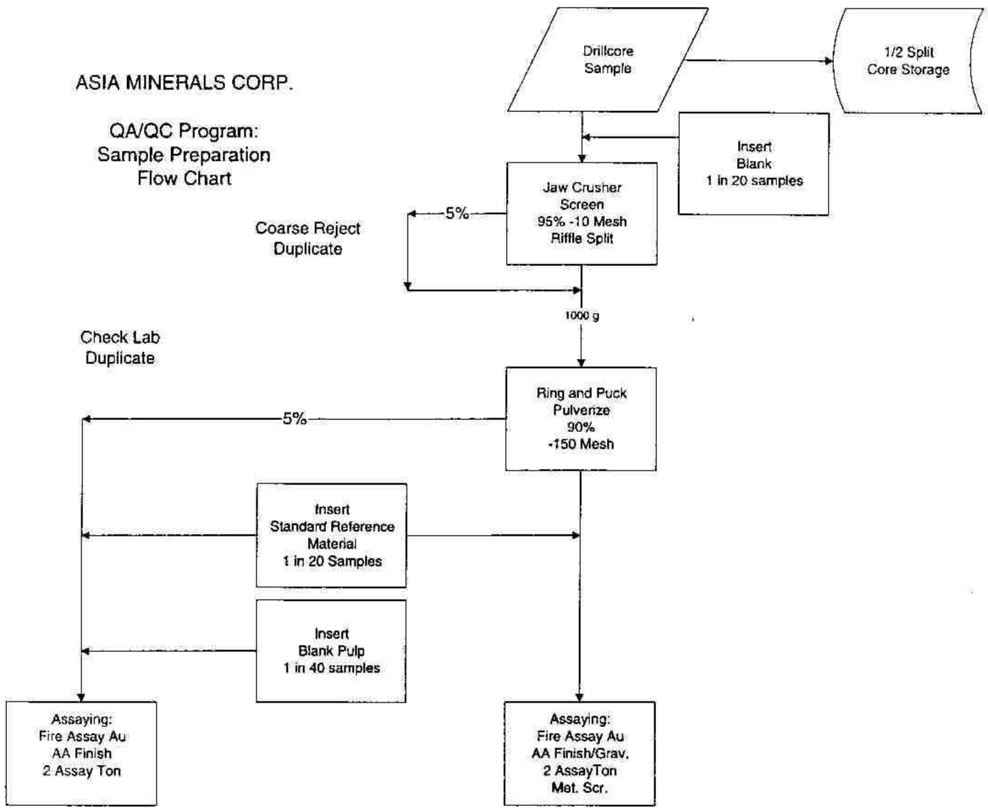


Figure 4-1: QA/QC Program Flow Chart

4.3 Quality Assurance/Quality Control

The Quality Assurance/Quality Control (QA/QC) program implemented by AMC is illustrated in Figure 4-1. Details on the program are included in Appendix A: Copperstone-Standard Procedures.

4.3.1 Standard Reference Material (SRM) and Blank Material

The control on quality is provided by the standards and blanks. These samples indicate whether the lab can consistently produce acceptable results. Standard material including blank material is produced and certified so that intrinsic sampling variation is very low (usually so that the standard deviation is 2% or less of the accepted value). The regular assaying of standards is intended to show that the assay process is in control, i.e., is producing unbiased results with as little variation as possible.

CANMET SRM MA-1b was selected as the Standard Reference Material for the AMC QA/QC program. The mean of the SRM assayed values was 0.508 opt Au versus the certified value of 0.497 ± 0.008 opt Au. The relative difference of +2.11% indicates a slight bias in the analysis. The SRM grades fell within 5% relative difference control lines as shown in the graph included in Appendix A *(MRDI QAQC.xls:SRM 5%).* These are acceptable results.

One blank sample returned a value of in excess of two times the lower detection limit. Bondar Clegg re-analyzed the sample and obtained the same results. This problem was not satisfactorily resolved.

4.3.2 Coarse Reject Duplicates

The assaying of coarse rejects is intended to check the sample preparation protocol. Laboratory to laboratory variation should be removed from this analysis to see as much of the sampling only variation as possible. Therefore, coarse reject duplicates were assayed by the primary lab. These duplicates provide a measure of the variance included in the check laboratory duplicates, plus the variance introduced by the sample preparation procedures, plus the sub-sampling variance of the coarse reject. The coarse reject duplicate analysis did not indicate any problems, 86% of the samples were within 30% relative difference (Appendix A—*MRDI QAQC.xls:Graphs Chart 6).*

4.3.3 Pulp Check Assays

Pulp check assays were processed by the umpire laboratory, American Assays Laboratories Inc., from Reno, Nevada. These provide a measure of the accuracy of the initial determination performed by the primary laboratory and an estimate of the analytical variance plus pulp sub-sampling variance. MRDI's recommended compliance is 90% of the pulps within 10% relative difference; the results however were 80% of the pulps within 10% *(Appendix A—Asia Check Samples.xls:Au>500ppb)*. This indicate that values from American Assays are approximately 5% higher than Intertek Testing Services (Appendix *A—Asia Check Samples.xls:Check Assays*). Most of the differences occurred on grades greater that 1.0 opt Au. On grades less than 1.0 opt Au there was little difference between the two laboratories (Appendix A—*Asia Check Samples.xls:Sheet2 Chart2*) .

As the check assays were submitted at the end of the assay program by AMC they could not be used to monitor quality control . Check assays should be submitted to the umpire lab concurrently with the original samples to the primary laboratory to facilitate monitoring of the QA/QC program.

4.4 Mineralization Envelopes

In order to provide a proper framework and constraint of the geological resource, the mineralization envelopes were developed and validated by AMC. Solid models derived from sectional mineralized zone boundaries were obtained from AMC's geological modeling process. These were used by MRDI with minor adjustments. The mineralization envelopes formed the basis for tagging the drill hole assays with the appropriate zone code and for coding the block model ore codes and percentages.

In some instances, the mineralization envelopes were extended up to constraining drill holes based on AMC's inferred geological continuity of 140 feet. Composites within the constraining drill hole were not coded with the zone of mineralization. These shapes should reflect the constraints imposed by geology, structure and assays.

4.5 Drill Hole Compositing

4.5.1 Methodology

Five foot down hole composites were created from the assay data honoring the mineralization zone codes stored in the assay database. Assays from different zones were not mixed together in a single composite. The average grade of the mineralized zone composites were calculated using a length weighting methodology. Specific gravity weighting was not factored into the compositing method.

The comparison of the composite statistics (Table 4.2) and declustered statistics (Table 4.3) did not identify a bias arising from clustered data.

Table 4.2: Copperstone Composites Summary Statistics

Mineralization Zone	Number of Samples	Mean Au, opt	Coefficient of Variation	Minimum Au, opt	Maximum Au, opt
D1	60	0.7661	2.5882	0.0000	12.0099
D2	17	0.1901	0.9650	0.0314	0.5660
D3	11	1.0102	1.3841	0.0034	3.7434
D4	9	0.1029	0.6756	0.0070	0.2300
D1A	3	0.0902	0.2891	0.0664	0.1181
D1B	5	0.1745	0.5889	0.0182	0.2568
D2A	10	0.0660	0.4213	0.0133	0.0954
C1	27	0.2532	1.5208	0.0097	1.4615
C2	93	0.1686	1.0666	0.0040	1.0550
C3	3	0.1110	0.2929	0.0780	0.1430
C2A	39	0.1286	0.7714	0.0160	0.3110
HW1	11	0.1348	1.9076	0.0015	0.8751
HW2	3	0.0948	0.7926	0.0190	0.1692
HW3	6	7.7547	1.3890	0.0104	23.5189
HW4	2	0.2150	0.0002	0.2150	0.2150
Other	**Number of Assays**	**Mean Interval, ft**	**Coefficient of Variation**	**Minimum Interval, ft**	**Maximum Interval, ft**
Composite Length	299	4.8	0.15	0.2	5.0

Table 4.3: Copperstone Uncapped Composites Declustered Statistics, Au opt

	Number of Samples	Mean	Standard Deviation	C.V.
C & D Zone				
Original	299	0.4670	1.9971	4.276
Declustered	295	0.4716	2.0102	4.263
No. of cells with 1 sample = 291		*No. of cells with 2 samples = 4*		
	Number of Samples	**Mean**	**Standard Deviation**	**C.V.**
D Zone				
Original	115	0.5482	1.5246	2.781
Declustered	112	0.5589	1.5436	2.762
No. of cells with 1 sample = 109		*No. of cells with 2 samples = 3*		
	Number of Samples	**Mean**	**Standard Deviation**	**C.V.**
C Zone				
Original	184	0.4163	2.2450	5.393
Declustered	183	0.4181	2.2510	5.383
No. of cells with 1 sample = 182		*No. of cells with 2 samples = 1*		

4.5.2 Capping Methodology and Implementation

The objective of capping is to limit the risk associated with erratic, extreme values. In order to quantify the risk, a statistical methodology was used to address the uncertainty associated with the high-grade mineralization. For C and D Zones, the assay histograms were reviewed to determine the outlier population. This was resolved to be 2.5 opt Au for C Zone and a 4.7 opt Au for the D Zone.

For the C Zone, a capping grade of 2.5 opt Au removes 458,000 ounces (56%) to address that metal at risk. As a result, a total of 2 (1.1%) out of the 184 composites within the mineralized zone are capped.

For the D Zone, a capping grade of 4.7 opt Au removes 114,000 ounces (21%) to address that metal at risk. As a result, a total of 4 (3.5%) out of the 115 composites within the mineralized zone are capped.

As more holes are drilled the level of confidence in the amount of high-grade metal present will increase. Typically 3 to 8 % of the metal would be removed by capping at the final feasibility study stage of project development.

4.6 Spatial Analysis

2D correlograms were calculated (as discussed in Appendix A) for C Zone and D Zone composites with extreme outliers removed. The modeled correlograms for the mineralized zones using the variable, Au opt, are summarized in Table 4.4.

Table 4.4: Correlogram Summary

Area	Structure	Nugget	Sill	Range, ft (East/North)	Rotation (ºAz)
C Zone	Spherical	0.644	0.201	70/81	335
	Spherical		0.156	71/96	0
D Zone	Spherical	0.656	0.178	70/66	60
	Spherical		0.166	33/70	310

2D variography was used to establish the lateral range for interpolation parameters. The range approximates the drill hole spacing of 70 feet and may be an artifact of the spacing. Short range variability needs to be established with closer spaced drilling.

No interpretable results were available from the 3D variography because the 3D correlograms were erratic as expected in skewed data sets and due to the noise effect created by the high values.

4.7 Geological Resource

4.7.1 Block Model

A three-dimensional block model was developed for the Copperstone project using MEDSYSTEM™ mine modelling software. The block size for each model is summarized in Table 4.5. Further information on the model limits and number of blocks is found in Appendix A.

Table 4.5: Model Block Size

Easting, ft	Northing, ft	Elevation, ft
15	35	5

4.7.2 Tonnage Factor

The results of the bulk density work by Hazen Research on behalf of RYO as part of an Ore Characterization Study (August 3, 1995) were reviewed. Cyprus data was not available. The tonnage factor proposed was 10.7 cu.ft/ton. This tonnage factor was the default used for all zones. AMC and RYO had previously used 11.0 cu.ft./ton for resource estimation.

4.7.3 Interpolation Parameters

The blocks were coded based on the mineralization zones developed by AMC and revised by MRDI. Interpolations matched mineralization codes in composites with mineralization codes in the block being interpolated.

A two pass interpolation methodology was employed using a 3D spherical search. The first pass incorporated an expanded search range and revised interpolation parameters to enable the grade interpolation of all blocks in the geological model. The parameters are summarized in Table 4.6.

Table 4.6: Interpolation Parameters

Interpolation Parameter	First Pass	Second Pass
Minimum # of Composites for Interpolation	3	3
Maximum # of Composites for Interpolation	10	10
Maximum # of Composites per Hole	No Maximum	2
Primary Search Range	1000 ft.	110 ft.
Maximum Distance to Project Single Composite	1000 ft.	110 ft.
Maximum Distance to Closest Point	1000 ft.	110 ft.

Tonnage factors were assigned to interpolated blocks in the mineralized zones.

Mineralization codes and block percentages were assigned based on the interpreted mineralization zones (plan VBMs). A model was created which stored an ore zone and percentage of ore in each block.

The percentage of the block below topography was stored for all blocks.

The Nearest Neighbor (polygonal method) gold grade was interpolated and stored in each block based on the same search parameters as the Inverse Distance Weighted gold grade. Inverse Distance models were built for the powers 3 (IDW3) and 5 (IDW5). Zone codes were matched between the composites and blocks. (See Appendix A: *Geology modelling documentation - Copperstone*)

No minimum length was used to restrict selection of composites for grade interpolation. The population statistics of composites (composite length<3 feet) were reviewed and it was found that a bias could be introduced by restricting selection from this population. Although only 5% of the composites were less than 3 feet, their grade was approximately 50% less (0.2483 opt Au vs. 0.4743 opt Au).

MRDI used search orientations based on AMC's geological understanding of the deposit and the apparent attitude of the mineralization zones. Table 4.7 summarizes the search orientations employed.

Table 4.7: Search Orientations (degrees)

Parameters	C Zone	D Zone
Strike	0	30
Dip	30	25
Plunge	0	0

Tables 4.8 and 4.9 summarize the block model statistics for polygonal and inverse distance weighted models for uncapped and capped gold.

Table 4.8: Copperstone Model Summary Statistics for Uncapped Au (opt)

	Polygonal		IDW3	
	C Zone	D Zone	C Zone	D Zone
Mean	0.5552	0.4328	0.6195	0.4807
Coefficient of Variation	4.5549	3.0336	3.366	1.665
Minimum	0.0015	0.0000	0.0059	0.0000
Maximum	23.5189	12.0099	22.127 3	10.639 9

Table 4.9: Copperstone IDW3 Model Summary Statistics for Capped Au (opt)

	C Zone	D Zone
Capped Grade	2.5000	4.7000
Mean	0.2842	0.3825
Coefficient of Variation	1.244	1.478
Minimum	0.0059	0.0006
Maximum	2.4983	4.6620

4.7.4 Model Validations

Part of the model validation involves a global check on the accuracy of any model by comparing geologically constrained models using kriging, inverse distance weighting and polygonal interpolation. If the volume of blocks used in the polygonal and kriged models are constrained to be the same as those used in the inverse distance power model, the following relationships should hold:

1. The contained metal (and hence grade) at a zero cutoff should be approximately the same for both models.

2. The kriged estimate should give the most tons and the lowest grade at a given cutoff (in most cases).

3. The polygonal method should give the highest grade and the least tons of any of the methods at a given cutoff.

As a suitable variogram model could not be developed with any confidence, a kriged resource model was not developed. Comparisons were made between the polygonal and inverse distance weighted (to the powers 3 and 5) models (Table 4.10).

The contained metal and gold grades are higher in the inverse distance weighted (IDW) models which indicates the impact of the high grade assays on the interpolation. Since the IDW models included up to 10 composites, within a range of 110 feet, their grade was positively impacted.

Table 4.10: Model Validation of Uncapped Au Models

Interpolation Method	Tons	Au Grade opt	Contained Metal oz
Polygonal	2,085,900	0.559	1,166,000
IDW 3	2,085,900	0.580	1,209,800
IDW 5	2,085,900	0.601	1,253,600

The model was also validated by comparing plan and sectional plots of the block grades, polygonal and inverse distance weighted, to the surrounding composite grades. None of the checks appeared to identify any problems with the resource model. When mining there will be unavoidable errors in classification because ore will be selected based on estimates and not on the true selective mining unit (SMU) grades. This is the reason for choosing a model that under-estimates grade and over-estimates tonnage a slight amount. With additional drilling and refining of the geological model, future work may facilitate the generation of a kriged model to aid in this model refinement.

A comparison of the block model using block partials with the solids provided by AMC indicated a slight difference in volume. The block partials model defined a greater volume than the 3D solid. For C Zone the difference in volume of the block partial model relative to the 3D solid model is 2.7%. For D Zone the difference in volume of the block partial model relative to the 3D solid model is 2.0%. MRDI has reported from the 3D solid model for all grades and tonnages in this report.

Based on the review of the grade interpolation and AMC's proposed geological model and inferred continuity, MRDI feels that the IDW3 model best reflects the potential grade of the Copperstone project.

4.7.5 Resource Summary

MRDI has estimated a capped resource in the C and D zones of 2,085,900 tons at a grade of 0.304 opt Au. This capped resource contains 708,700 ounces of gold. Table 4.11 summarizes the capped resource by block cutoff grade.

Table 4.11: Copperstone Geological Resource by Block Cutoff IDW3 Model for Capped Gold Grade

Block Cutoff	Deposit	Tons	Au, opt	Au, ounces
0.000 opt	C Zone	1,175,000	0.270	317,700
	D Zone	910,800	0.430	391,000
	Total	2,085,800	0.340	708,700
0.100 opt	C Zone	769,000	0.382	293,800
	D Zone	601,800	0.622	374,300
	Total	1,370,800	0.487	668,100

In an economic scoping study reported by AMC in September 1998, a resource of 890,000 tons at 0.532 opt Au was defined. That study used an assay cap of 2.5 opt Au and a block cutoff grade of 0.134 opt Au. Based on the additional drill holes by AMC, the revised geological interpretations and new estimation parameters have given a net increase in the contained gold of approximately 41%.

Table 4.12 summarizes the Copperstone geological resource by zone for the IDW3 model using the capped gold grades and no block cutoff.

Table 4.12: Copperstone Geological Resource by Zone IDW3 Model for Capped Gold Grade

	Zone	Tons	Au, opt	Au, ounces
C Zone	C1	435,000	0.295	128,300
	C2	342,400	0.131	44,900
	C3	6,800	0.120	800
	C2A	121,500	0.098	11,900
	HW1	120,500	0.186	22,400
	HW2	41,600	0.106	4,400
	HW3	73,300	1.326	97,200
	HW4	33,900	0.229	7,800
D Zone	D1	533,100	0.409	218,000
	D2	73,000	0.157	11,500
	D3	138,900	0.988	137,200
	D4	62,900	0.104	6,500
	D1A	14,900	0.379	5,600
	D1B	24,600	0.270	6,600
	D2A	63,400	0.089	5,600

4.7.6 Resource Classification

The Copperstone Gold Project Geological Resource, as shown in Table 4.13, has been classified into Measured, Indicated and Inferred Resources based upon the level of confidence according to the proposed TSE guidelines using the drilling grid spacing and continuity of mineralization as determined through the geological and geostatistical review of the data.

Measured - No blocks were classified as "Measured" resource blocks in this model.

Indicated - Blocks having at least three composites within 110 feet from a minimum of two different holes are classified as "Indicated" resource blocks.

Inferred - Blocks with estimates not meeting the criteria above are classified as "Inferred" resource blocks.

Table 4.13: Copperstone Classified Geological Resource IDW3 Model for Capped Gold Grade

	Deposit	Tons	Au, opt	Au, ounces
C Zone	Measured	-	-	-
	Indicated	478,400	0.194	92,700
	Inferred	696,700	0.323	225,000
	Total	1,175,100	0.270	317,700
D Zone	Measured	-	-	-
	Indicated	413,800	0.467	193,000
	Inferred	497,000	0.399	198,000
	Total	910,800	0.430	391,000
All Zones	Measured	-	-	-
	Indicated	892,200	0.320	285,700
	Inferred	1,193,700	0.354	423,000
	Total	2,085,900	0.340	708,700

4.8 Conclusions & Recommendations

Further work is recommended on the definition of the mineralization envelopes and selection of assays for interpolation. Additional work will be required with subsequent drilling to refine these shapes to clearly reflect the constraints imposed by geology, structure and assays.

The QA/QC program for Copperstone should be more rigorously applied in future drill programs. These programs should reflect this in the design and implementation of the assay protocol. The metallic screen analysis has identified an upside potential which could be further evaluated with the existing assay pulps from AMC's drill program.

All previous drilling and mining information should be reviewed with respect to their potential contribution to improving the geological resource estimate for the Copperstone Project. One aspect which must be reviewed is the type of drilling used, i.e. rotary, reverse circulation or core. Also specific gravity information may be gathered from previous mining completed by Cyprus. Geostatistical analysis of the blast hole data within the open pit could better define the spatial variability and give insight into ore distribution within the C Zone.

5.0 MINING

Based on the currently identified resources and the nature of the ore zones it is considered that an achievable and realistic mining rate would be from about 450 to 550 tons per day. A production rate of 520 tons per day has been selected. Assuming that the mine operates for 350 days per year, this rate would provide 182,500 tons of mill feed annually. This study assumes that all mining will be by contractor(s) with personnel working two eight hour shifts per day on a continuous rotating schedule.

5.1 Mine Access and Development

It is assumed that underground access to the deposit will be developed during an exploration program. This will allow core drilling of the mineralized zones to establish proven resources and the mining of a bulk sample of mineralized material for metallurgical testwork.

Underground access will be by a ramp driven from near the bottom of the existing open pit haul ramp Figure 5.1. The portal will be located at about 480 ft elevation in the west pit wall. This location has been selected because it would provide the shortest access development to the ore zones and appears to have a stable pit wall requiring a minimal amount of preparatory work. However, during final mining of the open pit the east wall was mined with few or no safety berms and before final selection of the portal site the stability of the east wall must be confirmed.

Just north of the selected portal location a large pile of rubble blocks access to the bottom of the pit. A location closer to the bottom of the ramp could establish a portal 60 ft lower in elevation and 450 ft closer to the mining areas than the selected site. However, removing the rubble material and conditioning the pit wall at this location is not considered to be cost effective.

The pit wall above the portal will be bolted and screened prior to driving the ramp to prevent loose material from falling in the portal area. An area 50 ft wide will be bolted to a 70 ft vertical height, reaching the first small bench above the portal location. Access will be by crane equipped with a man cage and work platform.

The pit haulage road was ripped as part of the mine closure and will require reconditioning for use as the access and haulage route. The full width of the road is not required as the vehicles which will use the ramp are much smaller than open pit haulage vehicles. A width of 20 feet with some wider passing sections would be sufficient considering the light volume of traffic anticipated.

Locating the portal at the general surface elevation was considered uneconomic due to the increased ramp development (3,600 ft) required to reach the potential mining areas, additional vertical development, increased underground ore and waste haulage distances, increased ventilation requirements, and the increased time required for exploration development.

During exploration about 3,960 ft of ramp, drift, and bulk sample development will be performed. Footwall ramps will be developed to the currently identified lowest elevation of Zone D1 and to about elevation 300 ft in Zone C1.

In order for the ramps to serve as main haulage routes during production mining they will driven at 13% downgrade and located in the footwall at an offset distance of about 70 ft from the footwall contact of the orebodies (see Figures 5.3 to 5.5). All lateral and ramp development will be driven 13 ft wide by 13 ft high to accommodate 22 ton trucks and 5 cu.yd. load-haul-dump (LHD) units.

It is estimated that, following exploration development, a further 5,400 ft of lateral and ramp development and 1,620 ft of raise development will be required over the mine life to access the currently identified lower elevations of C and D zones. Of this total, the preproduction lateral development is 1,320 ft and includes the completion of the ramps and start of diamond drill and access drifts for the D and C Zones. The remaining development will be completed as required during mining operations.

The two ventilation raises to surface will also provide alternate means of egress from the mine. The first of these will be completed and the second started during the preproduction period. The second will be completed early during mining operations. These will be bored from surface with the upper parts of each cased through the overburden to bedrock. In the area selected for these raises (drill hole 98-10) there appears to be about 200 ft. of overlying unconsolidated sands and gravels.

5.2 Stoping Method

Drift and fill stoping has been selected for the following reasons:

- It is flexible and can be responsive to changes in orebody width and orientation and will allow the selective application of a cut-off grade.
- It can achieve relatively high recovery of the orebody with low dilution.
- Because the orebody appears to be narrow and gently dipping the application of lower cost bulk stoping methods would be impractical.

Access to the ore zones will be by crosscuts driven from the footwall ramp (Figure 5.2). Three stope cuts will be served by each crosscut by slashing down the back twice and using the broken rock as the driving surface.

Stope cuts will be 13 ft high and will be driven along the strike of the ore. In wider and flat-lying zones mining will be in multiple parallel passes with the successive cuts retreating from the hangingwall to footwall of the ore. Each cut will be filled before the next is mined and thus some will be mined with backfill forming one wall. Geotechnical analysis will be required to determine the maximum allowable open span during mining.

Mining of each zone will begin at the lowest elevation and advance updip with paste fill used to fill each cut.

It is proposed to concurrently develop and mine two stopes in each of two mining zones. Each stope will provide two working faces as mining advances along strike in opposite directions from the access drift, thus providing a total of eight possible working faces.

It is estimated that the average stope round will contain about 150 tons of ore. Hence the daily ore production target of 520 tons will require between three and four face blasts each day. The required stope face availability of 40 to 50% is considered typical and will provide for face unavailability due to backfilling and other operating conditions and to unplanned delays from adverse ground conditions, low ore grades, or other geological factors.

When ore zones of significant thickness, say greater than 25 ft, are identified, then more productive, lower unit cost stoping methods could be applicable. For example, bench drilling from a top cut established within the ore and on the hangingwall could be used. Mucking would be by remote controlled LHD's within the open stope. The larger stope sizes would allow the use of bulk fill, such as development waste rock, to supplement the paste fill. After filling, mining would advance updip to the next stope block.

Although current drill hole data indicates some relatively thick mineralized intersections, no areas large enough for effectively establishing such longhole stopes are currently apparent and this study only considers cut and fill stoping. Better definition of the ore zones could result in long hole stoping being applicable.

5.3 Resources

The total geologic resource is estimated at 2.086 million tons containing 708,900 ounces of gold and with an average grade of 0.340 opt Au.

Economic mining cutoff grades based on a gold price of $300 per ounce, a milling recovery of 90%, and estimates of operating costs were determined for each zone and varied from about 0.25 to 0.30 opt (Appendix B). Overall processing and general and administration costs per ton of ore were based on the planned daily processing rate of 500 tons. Mining costs assumed the same stoping method in all ore zones and an increase in cost per ton of ore with increasing distance of the ore zone from the mine portal.

There appear to be relatively large variations in the gold grades in the ore zones. However current knowledge of the grade distribution is considered insufficient to allow the assumption of selective mining of only material above the cutoff grades of each zone. In order to identify potential stoping blocks of sufficient dimensions and continuity to permit efficient extraction the resources available for mining are based on the material within a geologic grade envelope of 0.10 opt and greater and having overall diluted grades greater than the calculated cutoff grades.

Within that grade envelope the geologic resource for each zone has been factored for 95% mining recovery and 10% mining dilution. Because there appears to be lower grade material surrounding the envelope a grade of 0.08 opt has been assumed for the diluting material. The resource for each zone, factored for mining dilution and recovery, is shown in Table 5.1.

Table 5.1: Resources

Zone	Resource above 0.10 opt grade				Diluted, Recoverable Resource			
	Resource		Contained Oz Gold	Gold Distr.	Resource		Contained Oz Gold	Gold Distr.
	Tons	Grade Opt			Tons	Grade Opt		
Included in mining plan								
D1	379,700	0.552	209,594	31.3%	396,800	0.509	201,971	31.3%
D3	78,700	1.722	135,521	20.3%	82,200	1.574	129,383	20.0%
D1A	8,900	0.582	5,180	0.8%	9,300	0.537	4,994	0.8%
D1B	24,300	0.272	6,610	1.0%	25,400	0.255	6,477	1.0%
C1	300,200	0.401	120,380	18.0%	313,700	0.372	116,696	18.1%
Total	791,800	0.603	477,285	71.4%	827,400	0.555	459,522	71.1%
Excluded from mining plan – below mining cut off grade								
D2	53,000	0.192	10,176	1.5%	55,400	0.182	10,083	1.6%
D2A	16,900	0.153	2,586	0.4%	17,700	0.146	2,584	0.4%
D4	40,300	0.122	4,917	0.7%	42,100	0.119	5,010	0.8%
C2	180,200	0.199	35,860	5.4%	188,300	0.189	35,589	5.5%
C3	6,800	0.120	816	0.1%	7,100	0.117	831	0.1%
C2A	42,300	0.173	7,318	1.1%	44,200	0.165	7,293	1.1%
HW1	104,200	0.207	21,569	3.2%	108,900	0.196	21,344	3.3%
HW2	30,300	0.111	3,363	0.5%	31,700	0.108	3,424	0.5%
Total	474,000	0.183	86,605	13.0%	495,400	0.174	86,157	13.3%
Excluded from mining plan - limited drill hole data								
HW3	72,200	1.346	97,181	14.5%	75,400	1.232	92,893	14.4%
HW4	32,800	0.232	7,610	1.1%	34,300	0.218	7,477	1.2%
Total	105,000	0.998	104,791	15.7%	109,700	0.915	100,370	15.5%
Total	**1,370,800**	**0.488**	**668,681**	**100.0%**	**1,432,500**	**0.451**	**646,049**	**100.0%**

Mining recovery assumed to be 95% of the resource tonnage and dilution to be 10% by weight at a grade of 0.08 opt

Material excluded from the mining plan includes that with diluted grades less than 0.25 opt, totalling about 474,000 tons at a grade of 0.183 opt, and that in the HW3 and HW4 hangingwall zones which is based on limited drill hole data, and totals about 105,000 tons at a grade of 0.998 opt.

5.4 Ore Definition

During the underground exploration programme diamond drilling will be used to quantify the reserve and define the ore zones. When ramp development has passed the junction of the D and C Zone ramps and advanced beneath those zones then locations for ore outline drilling will become available.

Core holes will be drilled from remuck stations or specific drill cutouts developed as the ramps advance.

This drill programme will be resumed during the mine preproduction period and an allowance for about 5,000 ft of core drilling has been included in the preproduction cost estimates. During mine production drilling will continue at a nominal annual rate of about 26,000 ft.

These allowances are for drilling to better define the currently identified ore zones. There are no allowances for exploration activity outside of the zones considered for mining in this study.

5.5 Mine Equipment

It is assumed that all major mining equipment will be provided by the contractor(s).

All development and stope drilling will be by two boom electric hydraulic jumbos and it is estimated that a maximum of three will be required. Two 22 ton capacity diesel trucks will be used to haul ore and waste from the mine.

Most mucking of development waste rock and stope ore will be by 5 cu.yd. LHD's and it is estimated that three will be required. One will be equipped for remote control operation. Smaller LHD's, typically 3 cu.yd. capacity, will be used for mining in narrow areas and for general service and construction.

Rock bolting of development and stope backs will be by hand held stoper drills from the decks of scissors lift vehicles. It is estimated that three scissors lift units will be required.

Table 5.2 shows a list of the major equipment assumed to be used by the contractor(s) and Table 5.3 the equipment and facilities to be provided by the owner.

Table 5.2: Major Mining Equipment (Contractor)

Type	Number
2 boom drill jumbo	3
3 cu.yd. LHD	1
5 cu.yd. LHD	3
22 ton haul truck	2
Explosives truck	1
Scissors lift	4
Lube vehicle	1
Flat deck with hydraulic crane	1
Utility truck for mechanics	1
Mancarrier – 8 man capacity	1

Table 5.3: Major Mining Equipment (Owner)

Type	Number
Utility truck - for supervisors	2
Utility truck - for surveyors	1
Main ventilation fan – 48" axial, 300 hp	1
Secondary fan – 42", 75 hp	2
Secondary fan – 30", 50 hp	10
Ventilation bulkhead	2
Main pumps – high head	4
Secondary pumps – low head	8
Compressor	1
Portable emergency hoist	1
Portable latrine	2
Portable refuge station	2

5.6 Mine Manpower

It is assumed that the contractors' personnel will work two eight hour shifts per day on a continuous rotating schedule. The estimated average personnel required, including those on days off, is shown in Table 5.4. It is assumed that the personnel required for technical control of the operations will be owner's employees. Six personnel will be required as listed in Table 5.5.

Table 5.4: Mine Operating Personnel (Contractor)

Position	Preprod.	Year 1	Year 2	Year 3	Year 4	Year 5
Lead miner	4	4	4	4	4	4
Development miner	8	-	3	-	-	-
Truck drivers	2	3	3	3	2	2
Miner	-	16	17	17	8	8
Diamond driller	2	2	2	2	-	-
Mechanic	6	6	6	6	6	6
Electrician	1	1	1	1	1	1
Labourer/fill crew	2	4	4	4	4	4
Total	25	36	40	37	25	25

Table 5.5: Mine Technical Personnel (Owner)

Position	Number
Engineer	1
Engineering technician / surveyor	2
Geologist	1
Geological technician	1
Sampler	1
Total	**6**

5.7 Backfill System

Paste backfill was selected as the mine fill material to minimize the surface impact of the operation by maximizing the tailings disposed underground.

Paste will be prepared at the mill (see Section 6.3) and placed underground at a nominal rate of 30 t/hr. The elevation difference between the fill preparation plant and the stoping areas will be inadequate for gravity flow and a positive displacement pump will be required at the plant.

Preparations for filling will include the placement of waste rock berms at the stope entrance Figure 5.2. The fill distribution pipeline will be placed to the ends of the stope and filling will retreat back to the stope entrance. Final fill placement, tight to the stope back, will be accomplished by decreasing the slump of the fill material.

The mining method does not require a strong fill. Accordingly, it has been assumed that the cement addition will be limited to 2% by weight, the minimum considered necessary to completely absorb the fill water during hydration and to ensure the paste flow characteristics.

The paste will be delivered to the mine via a 4 inch diameter pipeline laid on surface. The pipeline will pass through the portal and down the main ramp to the stoping areas. Schedule 80 steel pipe will be used and will be suspended from the backs of the ramps and drifts. This line will be under high pressure. The final run into the stopes will use HDPE pipe to reduce costs. This pipe will either be suspended from J-hooks and chain, or simply run along the floor of the stope. Both the HDPE and schedule 80 pipe will be salvaged for re-use as mining areas are completed.

5.8 Ventilation

Based on the mobile mining equipment described in Section 5.5, the current MSHA regulations for diesel equipment, and assumed leakage of 10% it is estimated that about 180,000 cfm of ventilating air will be required.

The ventilation system will be modified as the mine is developed.

- The initial ramp development will be ventilated by an intake fan at the portal with fresh air ducting to the ramp face and exhaust flow back to the portal.
- After Ventilation Raise #1 has been developed and equipped at the collar with a 42", 75 hp fan it will be used as a temporary intake system.
- After Ventilation Raise # 2 has been developed it will be equipped at the collar with a 48", 300 hp fan and be the permanent intake system.

The C Zones will be ventilated via a lateral drift and a raise connecting the bottom of the C Zone ramp to the bottom of Ventilation Raise #2. An airflow regulator will be installed in the drift to control flow rates according to the relative levels of mining activity in the D and C zones.

The ramp and Ventilation Raise #1 will be used as the main exhaust circuit with flow regulated by two bulkheads in the drift either side of Raise #1.

5.9 Ore and Waste Handling System

5.9.1 Ore

Ore will be loaded by LHD's into haulage trucks located in the ramp. A backslash will be taken at each stope access intersection with the ramp to provide sufficient head room for truck loading. The remuck stations driven during development of the ramp will be used to store ore for truck loading.

The underground trucks will haul the ore on surface up the existing pit haul road directly to the mill where it will either be dumped to a surface stockpile or fed directly into a hopper ahead of the primary crusher.

The pit haul road has been ripped and will have to be graded prior to use. The full 50 to 60 ft width of the road will not be required, a 20 ft wide surface with occasional wider passing areas will be adequate.

5.9.2 Waste Rock

Development waste will be dumped into the pit bottom close to the portal. There appears to be ample room to store all waste rock broken during the project. Some will be end-dumped from the edge of the pit haul road and some can be placed on the pit floor in the area of the current rubble pile. Some leveling of the waste by bulldozer may be required.

Some development waste rock may be disposed in completed stopes before placing paste fill. However, as surface disposal of waste rock appears to be very convenient it may be more advantageous to maximize the disposal of tailings as paste fill and reduce the surface storage requirements for tailings.

The tonnage of waste rock forecast to be produced during development is shown in Table 5.6.

5.10 Mine Services

5.10.1 Dewatering

The potential for groundwater inflow into the mine workings and possible rates of flow are unknown. Drainage water during mine operations will be limited to that used for drilling and for dust control. Water within paste fill is used during the hydration of the cement and does not result in drainage water.

Assuming that there will be some groundwater flow an allowance for four main dewatering pumps has been included. These will be portable, high head, submersible pumps. Sumps will be developed at the lowest elevations of the D and C Zone ramps and each equipped with two pumps with one operating and one as a spare on standby.

Drainage water from the C Zone sumps will be pumped to the D Zone sump and all water then pumped to surface through a pipeline installed in Ventilation Raise #2.

Local drainage from development headings and stoping areas will be pumped by portable, low head, submersible pumps to either of the two main sumps. An allowance for eight of these secondary pumps has been included.

5.10.2 Compressed Air

A portable diesel compressor will be installed on surface near the portal. It will be located at the large switchback northwest of the portal and at an elevation about 240 ft higher. The compressed air pipeline will be located along the pit wall to the portal then down the main ramp to the mine workings.

5.10.3 Water

Water for drilling and dust control will be conveyed through a surface pipeline from the process plant area to the portal and down the main ramp.

5.10.4 Mine Facilities

Two underground refuge stations and latrines will be provided. These will be portable units, one for each of D and C Zones.

Small powder and cap magazines will be installed in the ramps convenient to the mining zones.

A fuel and materials storage area will be placed in the strike drift near the base of Ventilation Raise #1, the exhaust raise.

5.10.5 Emergency Escapeways

Ventilation Raise #2 will be equipped with an emergency conveyance to provide a second means of egress from the mine for D- Zone. The hoist could be either a fixed installation or truck mounted. An advantage of a truck mounted hoist is that it could also service Ventilation Raise #1 if needed.

The diamond drill drift for C Zone definition will have a 110 ft conventionally driven raise connecting the zone to the base of Ventilation Raise #1. A manway will be installed in this raise to provide a second means of egress.

A 600 ft long drift will connect the bottom of the C Zone ramp to the base of Ventilation Raise #2 and a conventionally driven raise developed between C and D Zones will provide a second means of egress for C Zone.

5.11 Mine Schedules

5.11.1 Development Forecast

The forecast preproduction development is shown in Table 5.6. An average daily advance of 22 ft has been forecast from one crew on each of two shifts per day. This is equivalent to blasting two 12 ft rounds per day and breaking 11 ft per round. The location numbers noted in the table correspond to those noted on Figure 5.1.

Three months of preproduction development are forecast. This completes the D Zone ramp to elevation 170 ft and the C Zone ramp to elevation 220 ft. Development of the C Zone ore definition diamond drill drift and of the two ventilation raises is then started. Ventilation Raise #1 will be equipped with a surface fan and will also serve as the emergency egress from the mine.

The development of ore access drifts and ramps continues during mine production from year 1 to year 4 at the rates required to serve production. This includes development of the ramp to the bottom of Zone C1 and a drift and raise connection to the base of Ventilation Raise #2 which will serve as a second means of egress for C Zone.

Table 5.6: Development Forecast (feet)

	Preproduction (months)			Operating (years)				
	1	2	3	1	2	3	4	5
Preproduction – lateral								
D1 ramp - leg 4 (8 to 9)		160						
C1 ramp - leg 1 (10 to 11)		200						
C1 ramp - leg 2 (11 to 12)		220	40					
C1 ramp - leg 3 (12 to 13)			260					
Diamond Drill Drift			360					
D1 access drifts		50						
D1B access drifts		30						
Preproduction – raise								
Vent Rse #1 to Surface (8 ft dia)	350	225						
Vent Rse #2 to Surface (8ft dia)		125	350					
Preproduction total	350	1,010	1,010	-	-	-	-	-
Lateral	-	660	660	-	-	-	-	-
Raise	350	350	350	-	-	-	-	-
Operating – lateral								
D1 access drifts				130	130	130	130	40
D3 access drifts				100				
Diamond Drill Drift				390				
D1A access drifts						30		
D1B access drifts				40	80			
C1 access ramp				1,620				
C1 escape drift to VR#1				110				
C1 escape drift to VR#2				660				
C1access drifts					150	150	150	40
Operating – raise								
Vent Rse #2 to Surface (8ft dia)				230				
Escapeway to VR#1, conv				110				
Escapeway to VR#2, conv				230				
Operating total	-	-	-	3,620	360	310	280	80
Lateral	-	-	-	3,050	360	310	280	80
Raise	-	-	-	570	-	-	-	-
Development waste tonnage	1,613	11,837	11,837	50,308	5,577	4,802	4,338	1,239

5.11.2 Production Forecast

Ore production is forecast to start at 250 tpd in the first month, to increase to 375 tpd in the second month, and attain the full planned rate of about 520 tpd in the third month.

This production rate is sustained for four years and then decreases in the fifth year. The total forecast production over the five year period is 827,400 tons. The production grade averages 1.016 opt for the first year and then the average annual grade varies from 0.419 to 0.440 opt for the remainder of the forecast period. The overall grade is 0.555 opt and contained gold about 459,500 oz.

Mining is forecast to start in Zones D1 and D3. Zone D3 is depleted at the end of year 1 and replaced by Zone C1 to maintain two active mining areas for the duration of the mine life. The smaller zones D1A and D1 B are mined concurrently with D1 since they share a common access from the ramp.

Table 5.7: Production Forecast

Zone	Year 1					Year				Total
	Mth 1	Mth 2	Mth 3	Mth 4-12	Total	2	3	4	5	
Zone D1	3,800	5,700	7,600	68,400	85,500	91,200	91,200	91,200	37,700	396,800
Zone D3	3,800	5,700	7,600	65,100	82,200	-	-	-	-	82,200
Zone D1B	-	-	-	3,350	3,350	22,050	-	-	-	25,400
Zone D1A	-	-	-	-	-	9,300	-	-	-	9,300
Zone C1	-	-	-	-	-	59,950	91,300	91,300	71,150	313,700
Tons	7,600	11,400	15,200	136,850	171,050	182,500	182,500	182,500	108,850	827,400
Grade Opt	1.042	1.042	1.042	1.009	1.016	0.435	0.440	0.440	0.419	0.555

5.12 Mine Closure and Reclamation

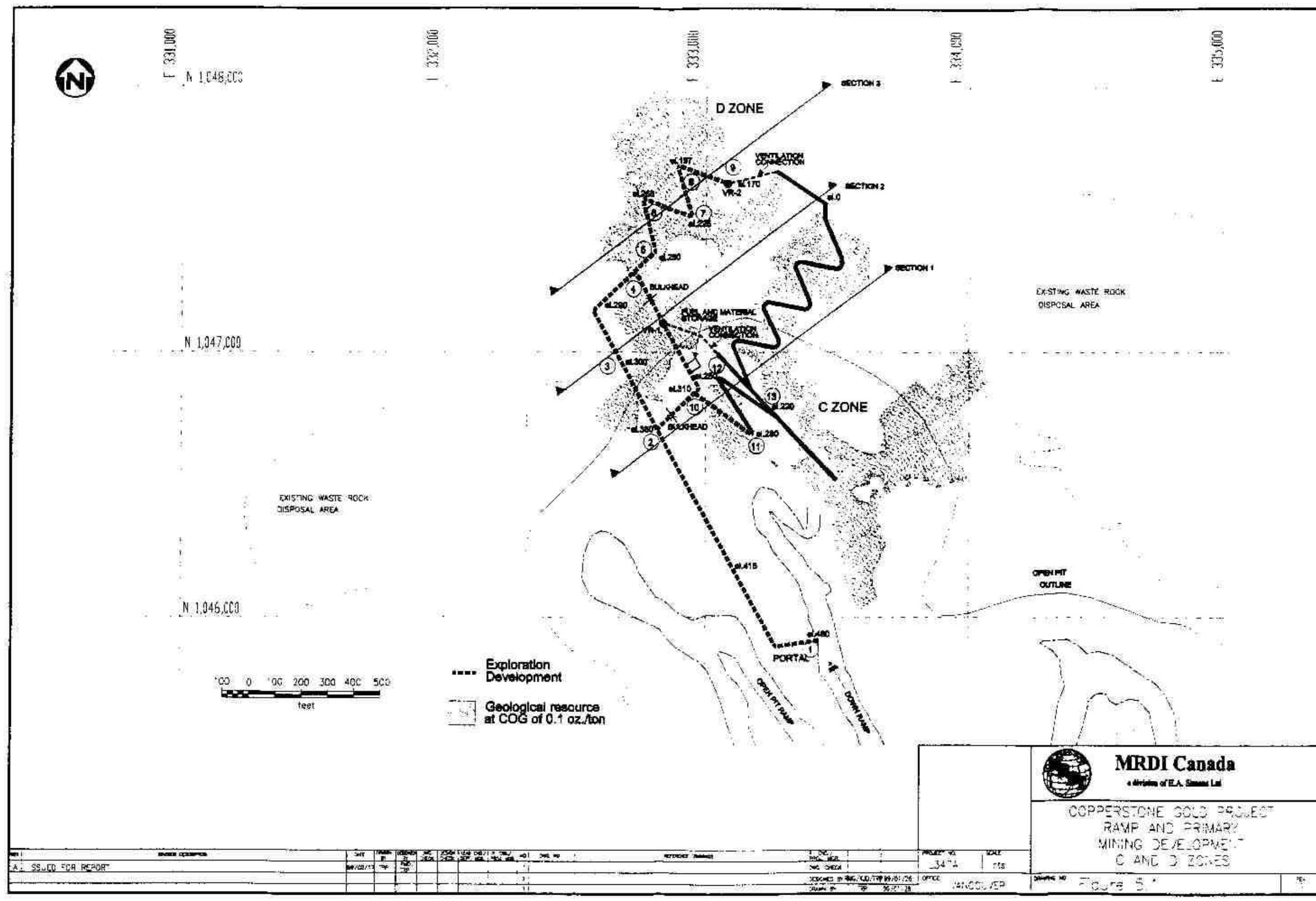
On completion of the mining and milling activities, the plant facilities will be removed and the site regraded to conditions similar to that which existed prior to the underground mine development. The objective of the decommissioning program will be to remove obvious man-made disturbances and to remove potential long-term sources of pollution or environmental upset.

Mobile and process equipment would be offered for resale. That which is not saleable will be drained of fuel and lubricants, cleaned, and buried in a landfill.

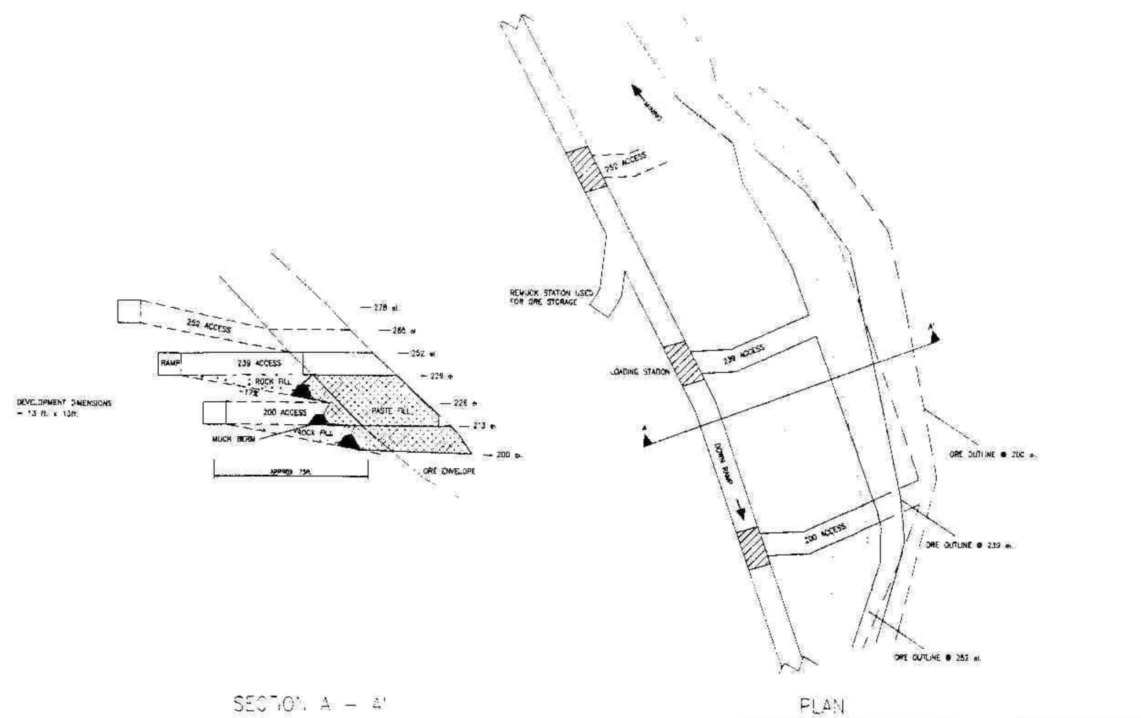
All pre-engineered buildings will be dismantled and sold to local dealers. Other buildings will be demolished and buried in the open pit.

After removal of the buildings, the plant site will be re-graded. Ditches would be filled to blend into the contours of the re-graded site. The mine haul road from the open pit, and all surfaces of the plant site and roads would be scarified to enhance natural vegetation. The portal and all raises to surface would be permanently sealed to prevent access into the underground mine workings.

In the event of a temporary closure, the environmental monitoring program and treatment of effluents, if any, would continue in accordance with permit requirements. All pipelines and the plant facilities would be drained. The portal and raise collars would be bulkheaded and locked to prevent access. Watchmen would remain on site to provide security and maintenance.



E 331,000
E 332,000
E 333,000
E 334,000
E 335,000
N 1,048,000
N 1,047,000
N 1,046,000
D ZONE
C ZONE
SECTION 3
SECTION 2
SECTION 1
VENTILATION CONNECTION
VR-2
BULKHEAD
FUEL AND MATERIAL STORAGE
PORTAL
OPEN PIT RAMP
DOWN RAMP
OPEN PIT OUTLINE
EXISTING WASTE ROCK DISPOSAL AREA
EXISTING WASTE ROCK DISPOSAL AREA
100 0 100 200 300 400 500
feet
Exploration Development
Geological resource at COG of 0.1 oz./ton
MRDI Canada
a division of H.A. Simons Ltd
COPPERSTONE GOLD PROJECT
RAMP AND PRIMARY
MINING DEVELOPMENT
C AND D ZONES
VANCOUVER
Figure 5.1
A ISSUED FOR REPORT



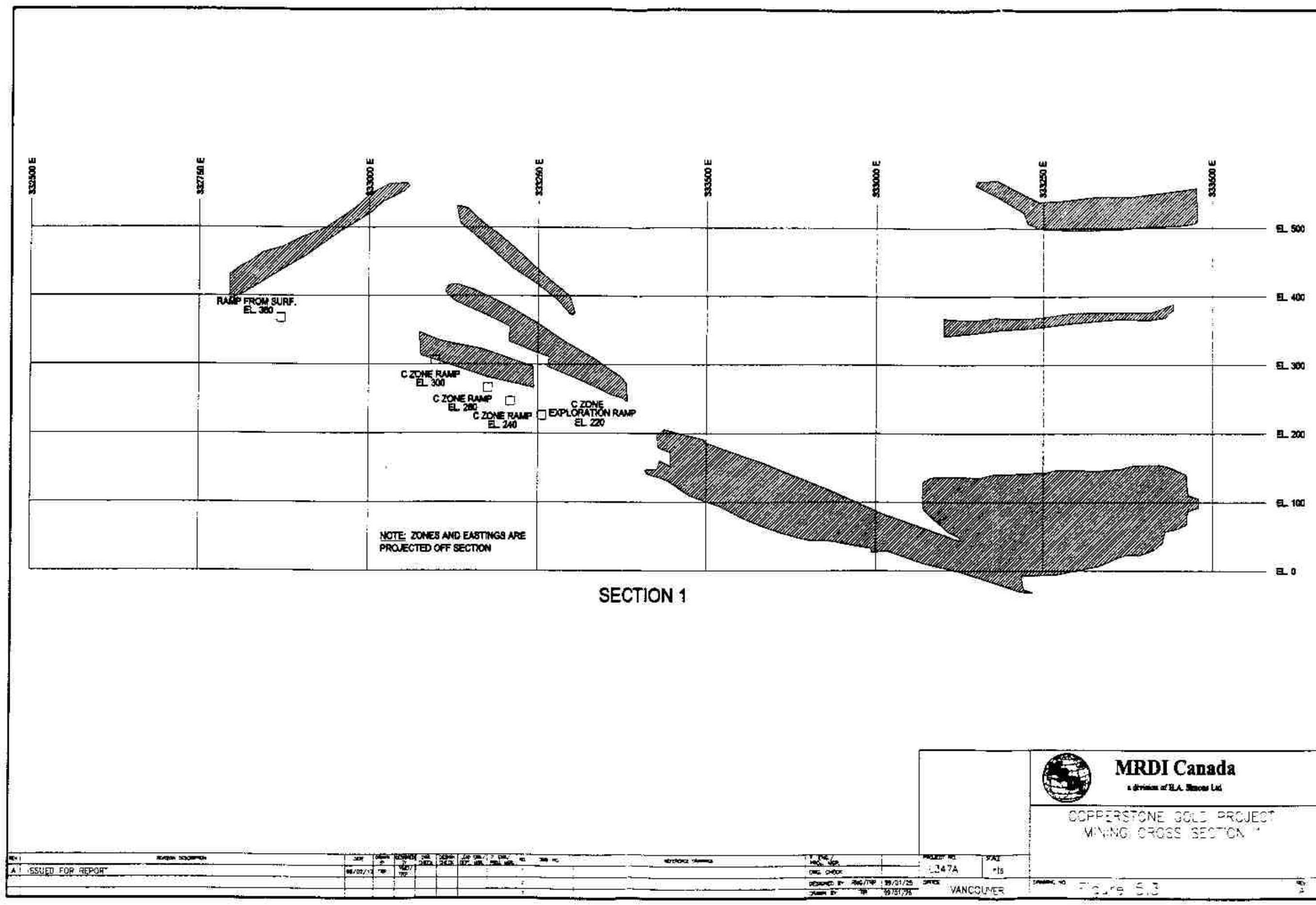
MRDI Canada
a division of H.A. Simons Ltd

COPPERSTONE GOLD PROJECT
SCHEMATIC DEPICTING
DRIFT AND FILL
MINING METHOD

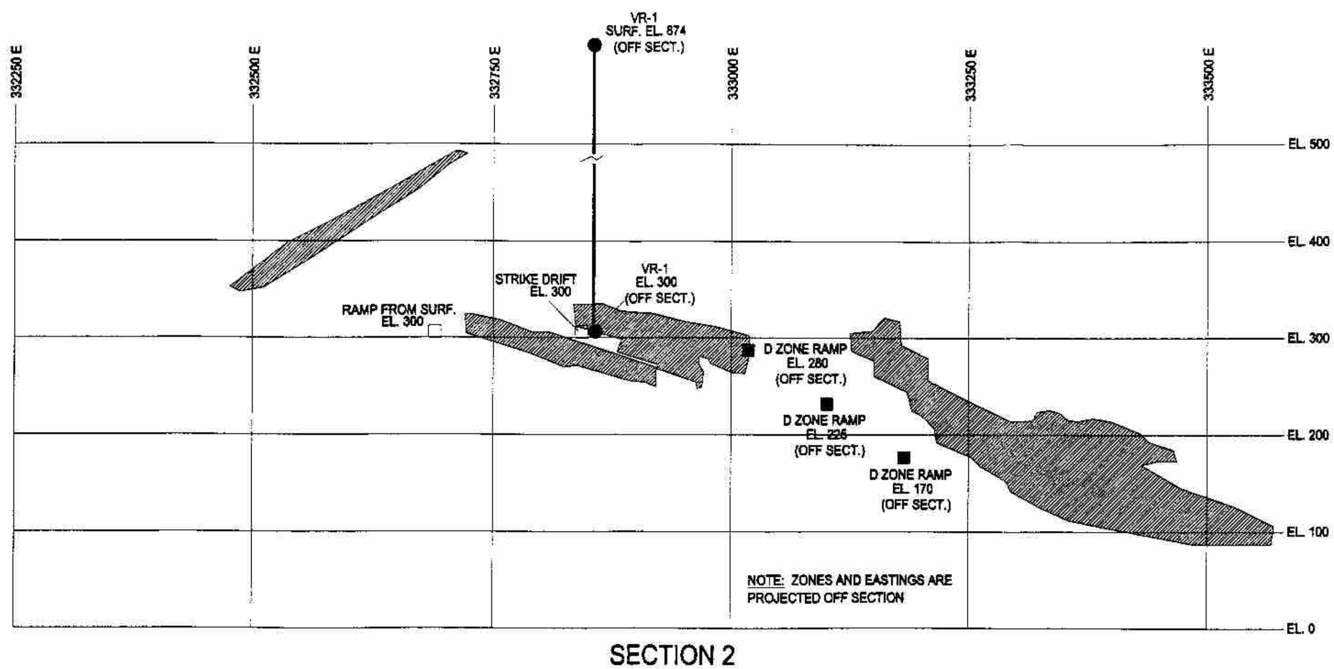
Figure 5.2

VANCOUVER

ISSUED FOR REPORT



EL 500
EL 400
EL 300
EL 200
EL 100
EL 0
RAMP FROM SURF.
EL 360
C ZONE RAMP
EL 300
C ZONE RAMP
EL 280
C ZONE RAMP
EL 240
C ZONE
EXPLORATION RAMP
EL 220
NOTE: ZONES AND EASTINGS ARE
PROJECTED OFF SECTION
SECTION 1
MRDI Canada
COPPERSTONE GOLD PROJECT
MINING CROSS SECTION 1
ISSUED FOR REPORT
VANCOUVER



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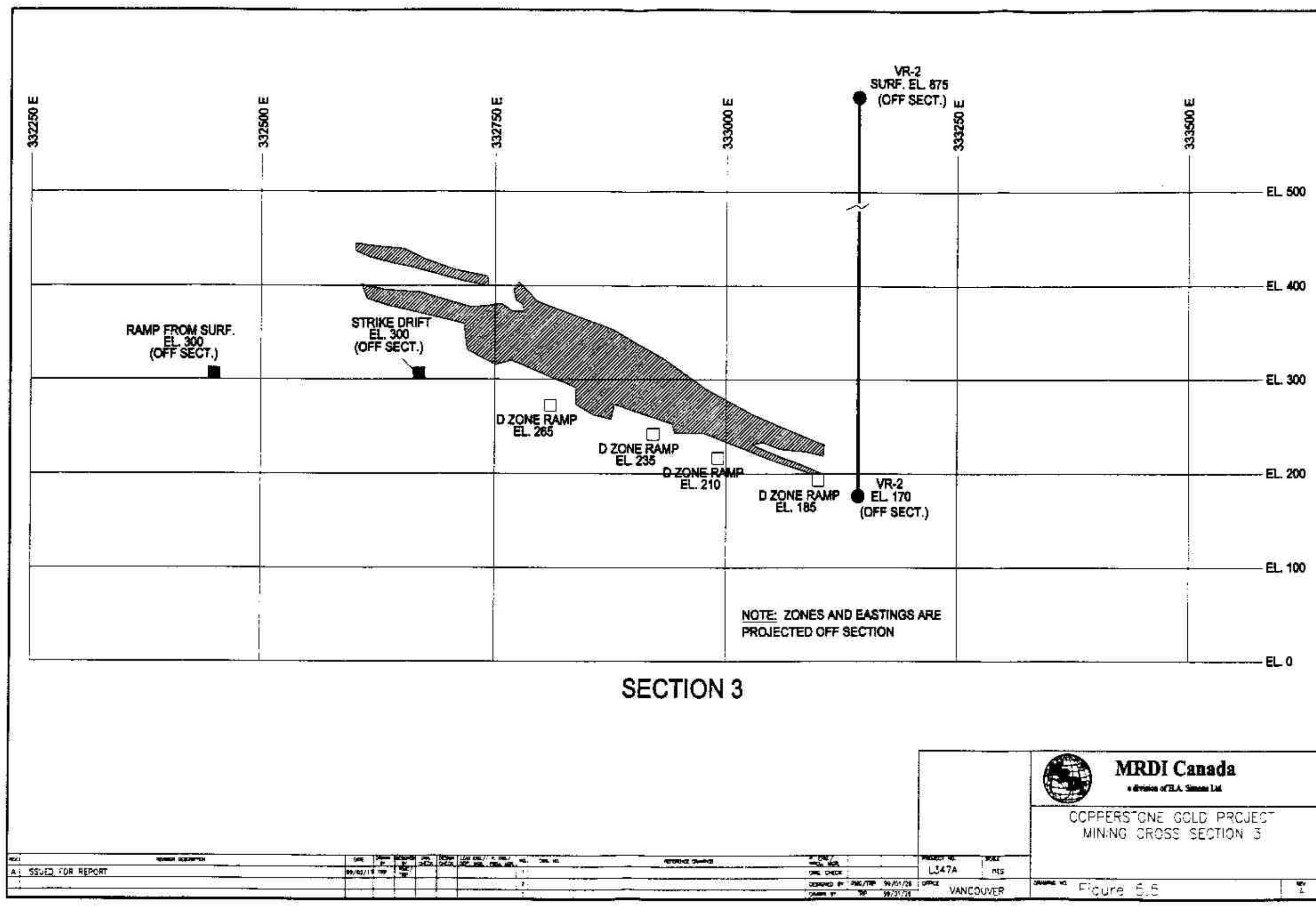
COPPERSTONE GOLD PROJECT
MINING CROSS SECTION 2

A ISSUED FOR REPORT

VANCOUVER

Figure 5.4


332250 E
332500 E
332750 E
333000 E
333250 E
333500 E
EL. 500
EL. 400
EL. 300
EL. 200
EL. 100
EL. 0
VR-2
SURF. EL. 875
(OFF SECT.)
RAMP FROM SURF.
EL. 300
(OFF SECT.)
STRIKE DRIFT
EL. 300
(OFF SECT.)
D ZONE RAMP
EL. 265
D ZONE RAMP
EL. 235
D ZONE RAMP
EL. 210
D ZONE RAMP
EL. 185
VR-2
EL. 170
(OFF SECT.)
NOTE: ZONES AND EASTINGS ARE
PROJECTED OFF SECTION
SECTION 3
MRDI Canada
COPPERSTONE GOLD PROJECT
MINING CROSS SECTION 3
A ISSUED FOR REPORT
L347A
VANCOUVER
Figure 5.5

6.0 PROCESSING

6.1 Metallurgical Testwork

Although the proposed processing scheme has not been tested at laboratory scale, the operating records from Cyprus indicate that the proposed scheme is feasible. Confirmation of the metallurgical performance of the proposed processing scheme will be required during the feasibility study stage of the project development.

The principal objectives for the feasibility testing program will be to assemble information to develop preliminary design criteria, flowsheet and equipment lists for processing the Copperstone ore. This may be achieved by testing under the following conditions:

1. Drill core composites and/or bulk ore samples should be used throughout the various phases of the testing program. A minimum of 200 lbs. of each variability parameter composite will be required to complete a scope of work. Usually four (4) composite types are prepared according to rock type lithology, oxidation state, precious metal:base metal ratios and alteration/mineralization types.

2. Head Assay Determination.

   - Triplicate fire assay for gold on each composite;
   - Single 'acid-base accounting' analyses on each composite; and,
   - Crush each composite to the coarsest feed size (¼" or 6 mesh), assay size analyses on all composites. Each size fraction will be assayed for gold, silver and for total sulphur.

3. Bottle roll tests on a 2 pound charge of 6 mesh crushed material form each composite. Bottle rolls should be for 48 hours at 40% solids, pH 10.5 to 11.0, 2lbs. NaCN/tonne solution, five (5) interim pregnant solution samples (at 2, 6, 8, 24, 36 and 48 hours) to establish recovery rates. Stage grind evaluation at P_{80} 65, 100, 150 and 200 mesh sizes.

4. A Bond Abrasion and Mill Work Index determination conducted on a sample of each composite.

5. Conventional thickening and filtration tests to assess the settling and filtration characteristics of leached residues from all composites.

6. Perform cement paste confirmatory tests on each leached residue composite material. Factors affecting the paste stability include non-segregation of fine particles (i.e. less than 20 micron), binder (cement, sand, aggregate) addition for strength and viscosity.

6.2 Process Development

The process scheme developed by MRDI for this study was based on a review of topographic electronic files on the Copperstone property prepared by others, and information gained from publications and preliminary inquiries. Based on limited mill operating data, a conceptual flowsheet for the Copperstone project was developed for a 500 ton per day milling, CIP gold recovery circuit. Figure 6.1 is a block-flow diagram showing the selected processing scheme. Conceptual process design criteria are tabulated in Table 6.1.

The following assumptions were made during this study:

1. There is no metallurgical data available to determine gold, copper and silver responses to traditional gold recovery processes. The overall gold recovery was assumed to be 90 percent.

2. MRDI assumed a crush, grind, whole ore leach, CIP and carbon adsorption, desorption and gold recovery (ADR) operation.

3. Coarse gold is amenable to recovery by gravity concentration methods.

4. Paste technology will be used for both underground backfill applications and surface tailings deposition.

5. No mineralogical data exists to determine the nature and size of the metals contained in the Copperstone deposit.

6. There is no data on the relationship of differing ore types to precious metal recoveries.

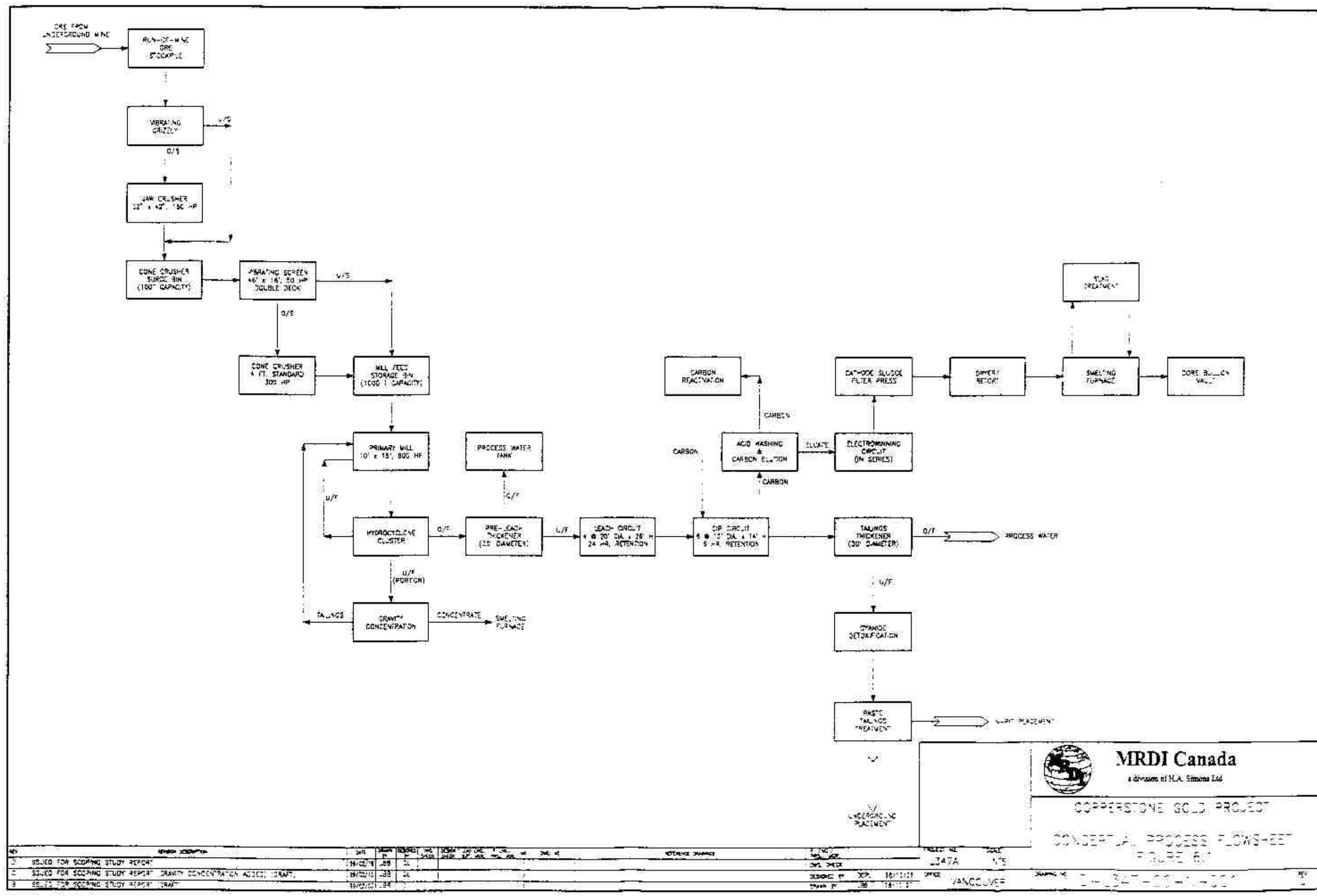

ORE FROM UNDERGROUND MINE
RUN-OF-MINE ORE STOCKPILE
VIBRATING GRIZZLY
O/S
JAW CRUSHER
CONE CRUSHER SURGE BIN (100T CAPACITY)
VIBRATING SCREEN
DOUBLE DECK
U/S
O/S
CONE CRUSHER
MILL FEED STORAGE BIN
PRIMARY MILL 10' x 15', 800 HP
U/F
HYDROCYCLONE CLUSTER
U/F (PORTION)
TAILINGS
GRAVITY CONCENTRATION
CONCENTRATE
SMELTING FURNACE
O/F
PRE-LEACH THICKENER (30' DIAMETER)
PROCESS WATER TANK
O/F
U/F
LEACH CIRCUIT
24 HR. RETENTION
CARBON
CIP CIRCUIT
RETENTION
CARBON REACTIVATION
CARBON
ACID WASHING & CARBON ELUTION
CARBON
ELUATE
ELECTROWINNING CIRCUIT (IN SERIES)
CATHODE SLUDGE FILTER PRESS
RETORT
SMELTING FURNACE
SLAG TREATMENT
DORE BULLION VAULT
TAILINGS THICKENER (30' DIAMETER)
O/F
PROCESS WATER
U/F
CYANIDE DETOXIFICATION
PASTE TAILINGS TREATMENT
UNDERGROUND PLACEMENT
MRDI Canada
a division of H.A. Simons Ltd.
COPPERSTONE GOLD PROJECT
CONCEPTUAL PROCESS FLOWSHEET
FIGURE 6.1
ISSUED FOR SCOPING STUDY REPORT
ISSUED FOR SCOPING STUDY REPORT GRAVITY CONCENTRATION ADDED (DRAFT)
ISSUED FOR SCOPING STUDY REPORT (DRAFT)
VANCOUVER

Table 6.1: Conceptual Process Design and Production Data

Ore Type	
Nominal Mill Feed Rate dry t/a	182,500
Crushing	
Shifts/Day	1
Hours/Shift	12
Plant Availability During Operating Shifts, %	80
Required Availability, %	75
Operating Hours per day	9.6
Operating Hours per year	3505
Grinding/Leaching/Gold Recovery	
Shifts per day	2
Hours per Day	24
Plant utilization, %	95
Effective Hours per year	8320
Crushing	
Design Crushing Throughput, t/h	125
Number of Crushing Stages	2
Primary Crusher Type	Jaw
Primary Crusher Size	32" by 42"
Primary Crusher Product Size, P80, inch	4-1/2
Secondary Crusher Type	Cone
Secondary Crusher Size	4 ft. standard
Secondary Crusher Product Size, P80, inch	1/2
Secondary Screen Aperture Size, inch	1/2
Stockpile and Reclaim	
Storage Type	Steel Silo
Live Storage Capacity, ton	1,000
Stockpile Reclaim Type	Mechanical Feeder
Number of Feeders	2
Milling, Classification and Gravity Conc.	
Operating hours, per annum	8320
Availability, %	95
Design Throughput, t/hr solids	30
Feed Size, F80, micron	12,500
Product Size P80, micron	75
Number of Mills	1
Work index (assumed) kWhr/t	17.0
Estimated Installed Power, HP	800
Classifier Type	Hydrocyclone
Feed Density, % solids	53.5

Ore Type	
Overflow Density, % solids	35
Volume Flowrate, Feed USgpm	215
Mill Circulating Load, %	300
Portion of classifier underflow fed to Gravity, %	12.5
Gravity Concentrator type	Centrifugal
Number of Gravity Concentrators	1
Type of Gravity Cleaner unit	Inclined, single deck shaking table
Leach/CIP	
Feed Rate, t/h solids	22
Feed Volume Flow Rate, USgpm	132
Number of Leach Tanks	4
Nominal Capacity, US gallons per tank (live)	51,700
Total Residence Time, h	26
Agitation	Mechanical
NaCN Addition, lb./ton	1.20
Lime Addition, lb./ton	3.28
Number of CIP Tanks	6
Nominal Capacity, US Gallons per tank, (live)	8,460
Agitation	Mechanical c/w interstage carbon screens
Tailings Thickening/ Paste System	
Type of Dewatering Unit	High Density Thickener
Flocculant Addition	Anionic-type
CN- ppm (tailings solution)	150 (typical)
Tailings Flowrate, t/h solids	22
Tailings Flowrate, USgpm slurry	75
Binder Type	Cement, 2%
Paste Pipeline Size, inch	4.0
Elution	
Batch Size, ton	2
Strips per week	3 (minimum)
Elution Process	Pressure Zadra
Elution Heater Fuel	Electric
Electrowinning	
Number of E/W Cells	2
Nominal Capacity, cubic feet	75
Carbon Regeneration	
Kiln Type	horizontal, rotary
Regeneration Temperature, °F	1400
Refinery	
Smelting Furnace	Crucible type

The basic unit operations incorporated into the processing scheme are:

- primary and secondary crushing;
- crushed ore stockpiling and reclaim;
- single stage grinding incorporating a ball mill;
- two-stage coarse gold recovery circuit;
- pre-leach thickening for leach feed;
- four-stage mechanically agitated leach circuit;
- carbon-in-pulp gold adsorption;
- desorption and reactivation of granular carbon;
- electrowinning and smelting to produce doré;
- tails thickening for solution cyanide recovery; and,
- paste tailings production for underground placement and in-pit disposal.

6.2.1 Paste For Tailings Disposal

Some advantages of the proposed processing scheme are that if a final tail can be produced with good settling characteristics and contain a sufficient degree of fine material, a cement paste tailings system can be adopted. To avoid additional permitting issues with land disposal and the high maintenance costs and poor availability associated with a hydraulic fill tailings system, paste thickening for both in-pit disposal and backfill to all underground stopes, is the proposed option.

The preliminary flowsheet selected for Copperstone will consist of dewatering the total plant tailings, mixing the detoxified thickener underflow with metered additions of cement and pumping the paste to either the underground stopes or impoundment areas within the existing open pit.

In addition, the recovered solution from the tailings thickener will allow a high proportion of the soluble cyanide to be retained within the process plant water balance, thereby reducing the amount of fresh cyanide solution make-up.

Paste backfill has been adopted by many underground mines in North America and a growing number elsewhere. Paste offers a number of advantages for underground backfill applications and tailings disposal including improved overall stability, potentially easier permitting, a smaller water inventory and earlier reclamation.

A. Stability

Elimination of impoundment dikes is a major cost saving feature of using paste for tailings disposal. Because of the increase in shear strength of cemented paste, low moisture tailings deposits are stable and do not need impoundment structures.

B. Acid Generation

Low permeability and capillary action will limit the acid generation of sulphidic tailings deposited using paste technology. Oxygen availability will be reduced due to limited infiltration of surface water.

C. Water Balance/Inventory

The inventory of process water will be greatly reduced compared to slurry tailing disposal because there is no inventory of process water in the impoundment. This will inherently decrease the risk from sudden impoundment failures and reduce the likelihood of wildlife mortality.

D. Reduce Land Usage

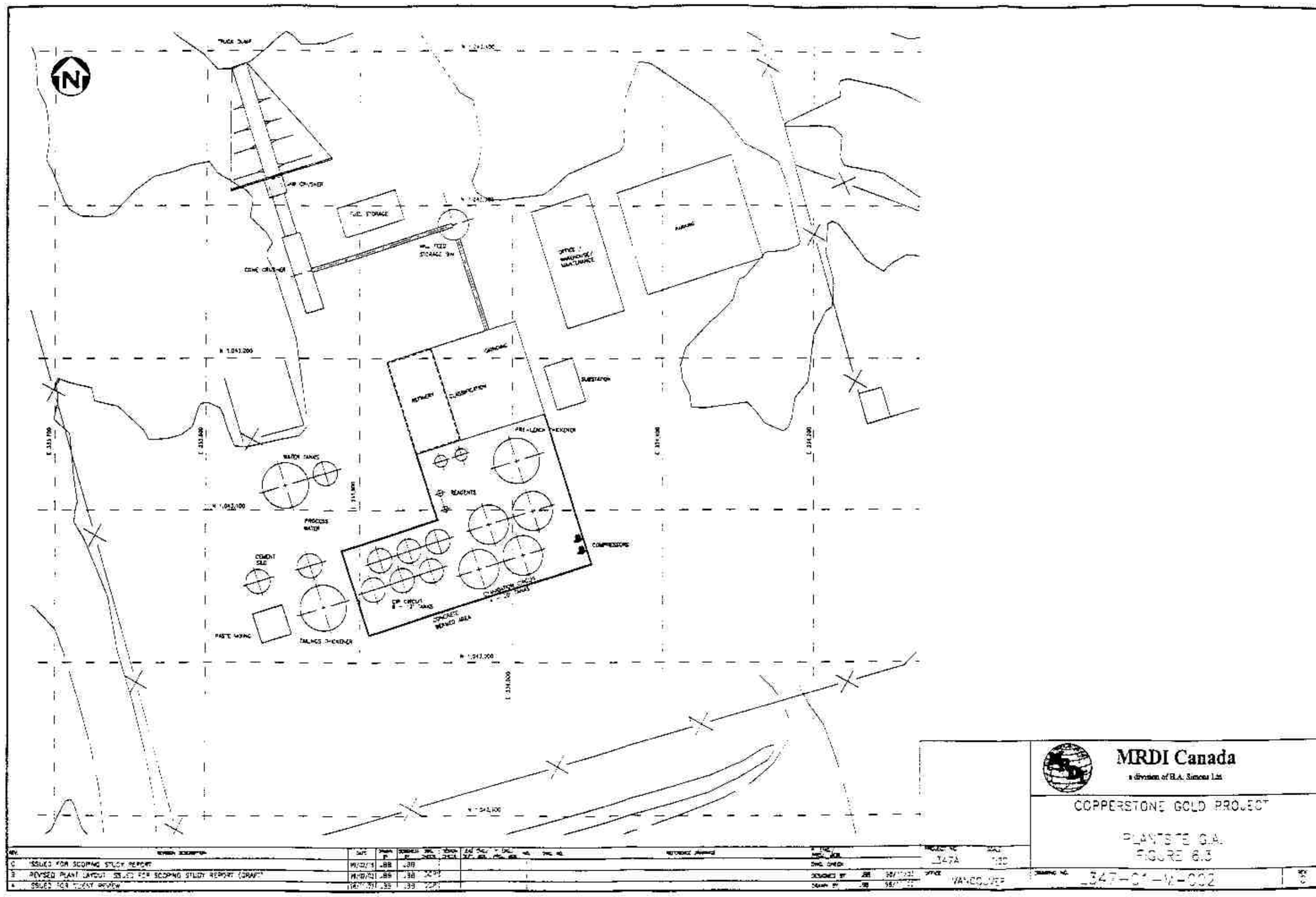
Land usage requirements will be reduced, which in turn will likely reduce permitting requirements and decrease the potential for conflicts with other land uses.

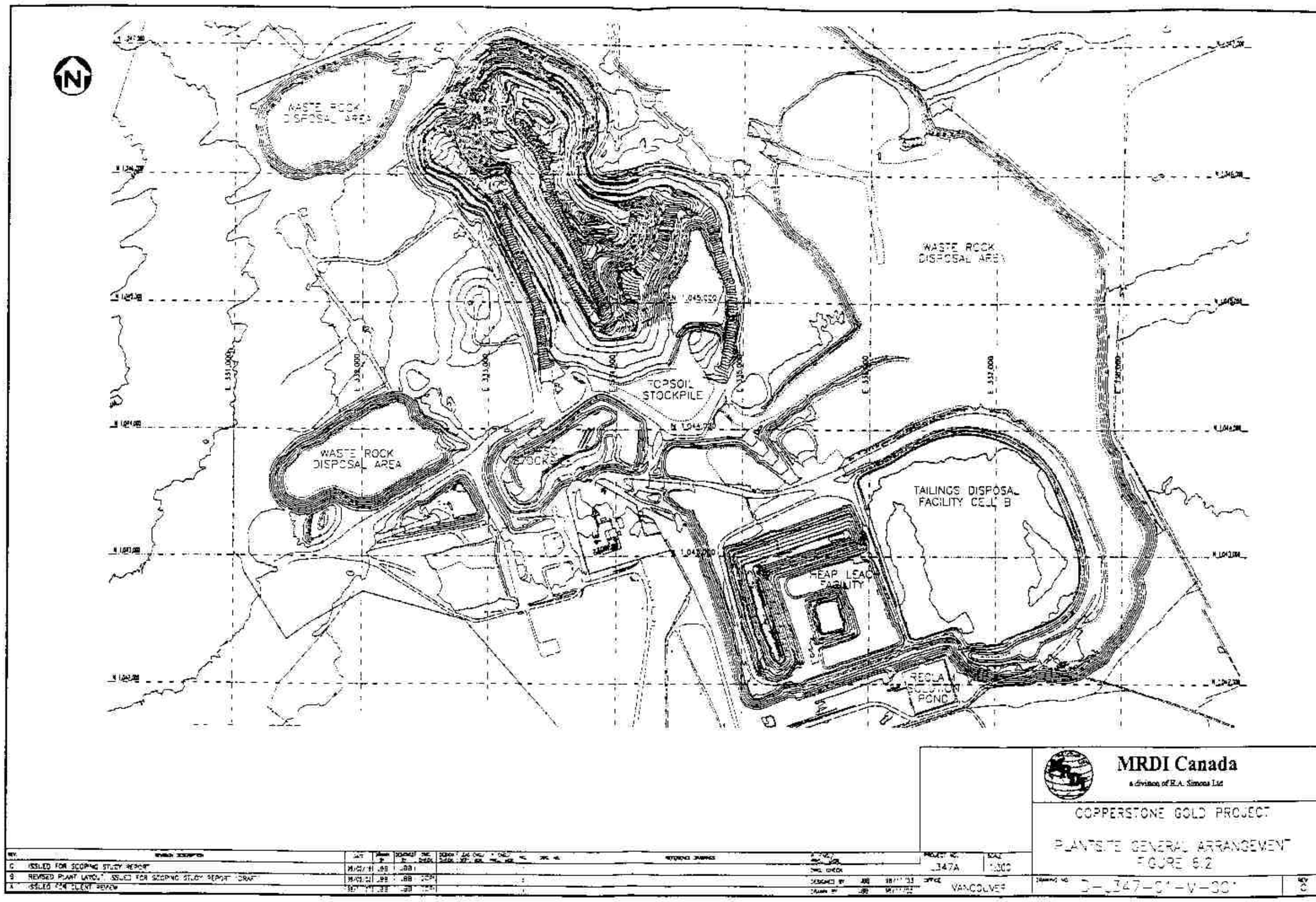
A number of technical papers detailing the use of paste for mill tailings handling are contained in Appendix F.

6.3 Process Facilities Description

The following sections present a brief description of the process facilities addressed in the conceptual flowsheet and the capital and operating cost estimates.

The location and layout of facilities are shown in the drawings Figures 6.2 and 6.3.


TRUCK DUMP
JAW CRUSHER
CONE CRUSHER
FUEL STORAGE
MILL FEED STORAGE BIN
OFFICE / WAREHOUSE / MAINTENANCE
PARKING
GRINDING
REFINERY
CLASSIFICATION
SUBSTATION
PRE-LEACH THICKENER
WATER TANKS
PROCESS WATER
REAGENTS
COMPRESSORS
CEMENT SILO
CIP CIRCUIT
CONCRETE BERMED AREA
TAILINGS THICKENER
PASTE MIXING
MRDI Canada
a division of H.A. Simons Ltd.
COPPERSTONE GOLD PROJECT
PLANTSITE G.A.
FIGURE 6.3
ISSUED FOR SCOPING STUDY REPORT
REVISED PLANT LAYOUT ISSUED FOR SCOPING STUDY REPORT (DRAFT)
ISSUED FOR CLIENT REVIEW
VANCOUVER
1347A
347-01-M-002



WASTE ROCK
DISPOSAL AREA
WASTE ROCK
DISPOSAL AREA
TOPSOIL
STOCKPILE
WASTE ROCK
DISPOSAL AREA
TAILINGS DISPOSAL
FACILITY CELL B
HEAP LEACH
FACILITY
MRDI Canada
a division of H.A. Simons Ltd
COPPERSTONE GOLD PROJECT
PLANTSITE GENERAL ARRANGEMENT
FIGURE 6.2
VANCOUVER
ISSUED FOR SCOPING STUDY REPORT
REVISED PLANT LAYOUT, ISSUED FOR SCOPING STUDY REPORT DRAFT
ISSUED FOR CLIENT REVIEW

dissociable (WAD) cyanide concentration. Treated tailings will report to the cement paste circuit.

6.3.4 Carbon Elution

Loaded carbon is washed with a weak acid solution prior to stripping the gold and silver from the carbon with a caustic-cyanide solution at a temperature of 275° F and 80 pounds per square inch (psi). Barren strip solution is heated using a in-line solution heater and heat exchangers. Cooled pregnant solution flows through electrowinning cells where the gold is deposited onto stainless steel wool cathodes. Solution exiting the cells is recirculated through the stripping cycle until complete. When complete, the stripped carbon is transferred to carbon regeneration, prior to returning the to the CIP circuit.

6.3.5 Refining

Cathodes are washed and the resultant gold sludge is filtered in a plate and frame filter press. The filter cake is dried, weighed and added along with flux reagents to the smelting furnace. Gravity concentrates will be fluxed and direct smelting in the smelting furnace. Doré will be poured and shipped off-site to a refiner for further processing (by others).

6.3.6 Tailings and Paste Handling

Thickened tailings at a high slump paste of 70 to 75% solids is achievable in the high density tailings thickener. Conventional thickening followed by vacuum filtration is considered uneconomic at present. The thickened paste will be pumped to a continuous paste mixer, where it will mix with Portland cement at 2 % (by weight) addition rates metered out of a 50 ton silo. Water is added for flushing and viscosity control. The mixer will discharge into a paste hopper feeding a concrete pump. The concrete pump will pump the paste through an overland, 4 inch diameter pipeline into the open pit. Most of the time, paste will be routed down shaft pipeline system to the underground stopes areas.

6.3.7 Reagents

Reagent mixing is provided in a separate dedicated area and provides mixing, storage and pumping facilities for cyanide, lime, sodium hypochlorite, caustic soda, hydrochloric acid and flocculant. A lime silo and slaking mill is provided to produce a lime slurry for pH control.

6.3.8 Assaying

In order to accurately control the mining and metallurgical aspects of the operation it will be necessary to have an on-site assay laboratory capable of carrying out atomic absorption (AA), fire assaying methods and environmental programs. In addition, a small metallurgical laboratory within the milling building will be required to make regular shift analysis of the process solutions.

6.3.9 Air Services

Service air has been provided for both high and low pressure applications. High pressure air is required for service points and instrument air. Low pressure air blowers will supply compressed air to the cyanide leach tanks.

6.3.10 Water

Water supply includes the provision of the following services:

- process water
- fire water
- potable water
- fresh raw water

6.3.11 Spill Contingency

Spill-prevention measures and contingency plans for containing accidental spills and for preventing uncontrolled discharges to the environment will be developed for the project as required by laws and regulations for the State of Arizona. Protection measures will be provided in the processing facility area to ensure that any spills of solutions and/or chemicals are contained, collected, and introduced back into the process stream or safely disposed, as appropriate.

6.3.12 Buildings and Protective Shelters

The following buildings have been included in the scope of the cost and operating estimate:

- Mill Feed Bin Silo
- Milling, CIP and ADR circuit covered structure
- Service and Reagents covered building
- Refinery Building
- Crushing area MCC
- Process Plant area MCC

6.4 Process Equipment List

The main process equipment (and motor horsepowers) required for crushing, grinding, leaching and gold recovery are listed in Table 6.2.

6.5 Process Operations

Ore crushing operations will be carried out in a single 12 hr shift per day. The crushing operating schedule has been based on 350 planned work days a year.

Process operations will be carried out on two (2) 12 hr shifts a day. The operating schedule has been based on 365 planned work days a year.

Arizona has key management and mine operators with a qualified local labour base for all disciplines; and major projects, including mining, have been constructed in the State. Based on previous operations experience in Arizona, it is considered unnecessary to use more than the normal quota of supervisors and operators to provide on-the-job training and supervision.

Table 6.2: Process Equipment List

Qty	Description	Size	HP Total
	Primary & Secondary Crushing		
1	Vibrating grizzly c/w hopper		40
1	Primary Jaw Crusher	32.0" x 42"	150
1	Primary crusher collection conveyor	36" wide	25
1	Tramp iron magnet		
1	Surge Bin (Secondary crusher feed)	100 tonne (nominal)	
1	Secondary Crusher feed conveyor	36"	15
1	Double Deck secondary screen	6' wide x 16' long	50
1	Secondary Hydrocone crusher	4' standard	300
1	Crusher Product discharge conveyor	36" wide	10
1	Mill feed storage conveyor	36" wide	10
1	Weightometer		2
	Dust Collector, ancillary power and lighting		50
	Grinding, Gravity & Pre-Leach Thickening		
1	Mill feed storage bin	1000 tonne capacity	
2	Mill reclaim belt feeder		30
1	Mill feed conveyor	18" wide	10
1	Primary Grinding mill	10' dia. X 15' egl	800
1	Mill Liner Handler		20
1	Mill Clutch Compressor		10
1	Primary Mill discharge pumpbox		
2	Cyclone feed pump	6" x 4"	100
1	Hydrocyclone Cluster		
1	Gravity concentrator	20" diameter	7.5
2	Gravity tailings pump		5
1	Shaking gold table		7.5
1	Pre-leach Thickener	30' diameter	15
2	Thickener overflow pump	3" x 3"	20
2	Leach Feed pump	4" x 3"	50
1	Grinding area sump pump		25
1	Process Water tank		
2	Process Water distribution pump	3" x 2"	30
	Overhead Crane, ancillary lighting		50
	Leach & CIP		
1	Leach feed distribution box		
4	Leach Tanks	20' dia. x 26' high	
4	Leach Tank agitators		80
1	Leach Area sump pump		20
2	Leach Air blowers		150

Qty	Description	Size	HP Total
6	CIP Tanks	12' dia. x 14' high	
6	CIP Tank agitators		90
6	CIP Intertank screens		45
6	Carbon forwarding pumps	3" vertical	30
1	CIP Area sump pump	4" vertical	10
1	Carbon Safety screen	2' wide by 4' long	15
1	CIP tailing pumpbox		
2	CIP tailing thickener feed pumps	4" x 3"	40
	Ancillary power and lighting		30
	Carbon Recovery		
1	Loaded Carbon Screen	2' wide by 4' long	5
1	Acid Wash Column	2 ton capacity	
1	Acid Wash Discharge pump		3
1	Acid make-up tank and circulating pump		3
1	Carbon Strip column	2 ton capacity	
2	Heat Exchanger		
1	In-line solution heater	$1x10^6$ kJ	150
1	Strip solution tank		
2	Strip solution pumps	2" x 1"	20
1	Pregnant Solution tank		
2	Pregnant Solution pumps	3" x 2"	10
2	Electrowinning Cells c/w rectifiers, fans	50 cu. Feet	30
1	Barren Solution tank		
1	Barren Solution pump	3" x 2"	10
1	Carbon Strip Area sump pump	2' vertical	5
1	Carbon Reactivation dewatering screen	2' wide x 4' long	5
1	Carbon Reactivation Kiln c/w exhaust	250 lbs./h	250
1	Carbon Quench tank c/w agitator		5
1	Reactivated carbon pump	2" x 1"	3
	Carbon Recovery miscellaneous		20
	Gold Recovery		
1	Sludge Filter feed pump	3" x 2"	10
1	Sludge Filter	Plate and Frame	
1	Drying Oven		2
1	Smelting Furnace c/w combustion blower		15
1	Exhaust Scrubber		2
1	Gold Room vault		
	Refinery miscellaneous, lighting		20
	General Site Services		
2	Plant air compressor		50
1	Instrument Air Dryer		5

Qty	Description	Size	HP Total
1	Instrument Air Receiver		
1	Fire Water Pump		20
1	Fire/Fresh Water Tank (existing)	350,000 Gallons	
3	On-site Water Well Pump (existing)		100
	Service Miscellaneous		10
	Reagents		
1	Lime Storage Bin c/w filter, feeder	35 ton capacity	10
1	Lime Mix tank		
1	Lime Transfer pump	2" x 2"	5
1	Lime Holding Tank c/w agitator		7.5
2	Lime Distribution Pumps	3" x 2"	15
1	Lime Area sump pump	2" vertical	5
1	Cyanide Mix tank		
1	Cyanide Transfer pump	1" x 1"	2
1	Cyanide area eyewash and alarm features		2
1	Cyanide Holding tank		
2	Cyanide Transfer pumps	2" x 1"	10
1	Cyanide Area sump pump	2" vertical	5
1	Caustic Storage/Mix tank		
1	Caustic dosing pump	1' x 1'	1.5
1	Dry Flocculant mixing system		5
1	Flocculant Mixing tank c/w agitator		2
1	Flocculant transfer pump	1" x 1"	0.5
1	Flocculant Holding tank c/w agitator		2
2	Flocculant transfer pumps	1" x 1"	2
	Reagent Area miscellaneous, lighting		10
	Paste Tailings		
1	Tailings Thickener (high density)	30' diameter	15
2	Thickener overflow pump (mill solution)	3" x 2"	20
2	Paste Circuit feed pump (peristaltic)		50
1	Cement Silo	50 ton capacity	
1	Cement screw conveyor		5
1	Weightometer (cement feed		1.5
1	Rapid pan mixer		5
1	Paste hopper		
1	Concrete pump		210

7.0 PLANT INFRASTRUCTURE AND ANCILLARY FACILITIES

7.1 Road Access

The road from Arizona Route 95 to the project site will be upgraded, with local materials, along the approximate 4 miles length of plant access corridor. No drainages will be crossed. Access to the site administration area will be made through a man-operated security gate, through which access will be gained to the shop/warehouse/office facilities and process plant facilities.

7.2 Site Preparation

Geotechnical investigations at the site were not completed for this study, however it is anticipated that the surface conditions will consist of good quality, load-bearing, bedrock. Site grading will be kept to a minimum as it has been assumed that little disturbance of the plant site area has occurred since the Cyprus mine closure in 1993.

A ramp to the primary crusher will be required and will be constructed from material available from the mine pre-production activities. The open-pit haul road will require some grading and fill material to upgrade for passage of the mine haul fleet.

The Cyprus tailing impoundment is lined with geomembrane and still discharges small quantities of poor quality seepage to a seepage collection pond located at the toe of the facility. The waste rock dumps for the most part are at angle of repose slopes and are not vegetated. Topsoil and seeding has been performed on a portion of the mine waste rock piles. It is intended that future development of Copperstone will avoid disturbing the existing mine waste disposal facilities.

7.3 General Site Arrangement

The plant site facilities will include the following installations:

- Small Vehicle repair shop, sub-station, warehouse and reagent storage
- Office and change room modules
- Fire/Raw water storage tank

- Sewage system
- Crusher and ROM open stockpile area
- Covered, open-structure grinding, CIP area building
- Covered, block-wall refinery and gold room

In addition, there is a need to install enclosure covers, where applicable, on open tanks, electric motors and other equipment for protection from seasonal weather patterns.

7.4 Power Supply & Distribution

Power to the new mine and process facilities will be achieved through a 'tie-in' to the commercial 69KV overhead power transmission line that runs parallel to the state highway and feeds the power transformer presently on site. High voltage power is supplied to the main transformer sub-station and MCC area for distribution around the mine and plant. It was decided to install emergency diesel generated power (one 250 HP generator) to allow for continuous operation of essential mechanical equipment following a power interruption.

7.5 Water Supply and Distribution

The process plant will require considerable raw water at the start-up of operations. It is assumed that the three (3) water wells drilled on the Copperstone site will supply the raw, fire and process water needs. Existing vertical well pumps are maintained by AMC and supply required water volumes through fully instrumented water lines to an existing 375,000 US gallon raw water tank. It is intended that this storage capacity be used to supply the required regulatory volumes for fire water service as well as supply a 4,000 gallon road water tank. Potable water will be trucked to the site to meet human consumption needs.

7.6 Mill Structures

It is envisaged that the mill building will be a pre-engineered steel structure, with galvanized roof and partial siding. The shelter will effectively house the grinding mill, CIP tanks and carbon desorption circuit. The gold refinery area, including the gravity concentration circuit, will be a block-walled fabricated building with secured expanded metal up to the mill roof elevation.

All other plant facilities will be uncovered in a contained area. The mill building will include vehicular doors for access to the grinding area and gold refinery. Foundations will consist of concrete column bases and grade beams, with concrete floor slabs over prepared fill.

Process plant consumables (other than lime and cement) will be stored in a fenced area c/w concrete pad and suspended roof.

7.7 Administration and Plant Wash-up Facilities

Offices for general site and plant administration will be designed to suit a pre-fabricated trailer construction complete with partitions and all fittings, except the moveable office furniture. A dry will be located adjacent to the offices and maintenance shop and is sized for 40 men and 5 women. It will serve the mine, plant and maintenance personnel. The dry will be equipped with one clean locker and handling baskets for each user. Showers and washrooms will be included within the dry.

7.8 Warehouses and Plant Shops

A secure warehouse building will be built from connected trailers ('used' shipping containers) for plant equipment spares. A mobile plant equipment maintenance shop, adjacent to the administration offices, will be provided. Foundations will consist of concrete sills and floor slabs over fill. Floor loadings will be considered light.

A mine maintenance shop with an outside wash/lube bay and tire bay will be built for the maintenance of the contractors mine vehicle fleet.

7.9 Laboratory Facilities

Standard gold sample preparation and solution fire assay equipment will be located in a ventilated trailer housing complete with installed benches, cupboards, shelving, sinks, dust and fume hoods, etc. A preparation room and analysis equipment are set aside for cyanide and environmental assay work. Facilities are also supplied in the mill building to conduct bottle roll testwork and daily plant metallurgical control tests.

7.10 Utilities and Services

7.10.1 Lighting

All process area lighting fixtures will be attached to the outside of buildings and pole lines for area floodlighting. Fluorescent lighting will be used in offices, electrical rooms, control rooms and other ancillary buildings.

7.10.2 Fire Protection

Fire protection system will be provided for the administration, maintenance and warehouse facility. Hose stations throughout the facilities will be provided with strategically placed hand-held fire extinguishers.

7.10.3 Security

A guard house is located at the entrance to the plant administration complex complete with area fencing and gate. The gold refinery room is fenced and access protected per industry standards. A vault is provided to hold bullion bars prior to shipment.

7.10.4 Sewage

At the mining operation, contractor supplied portable chemical toilets will be used. For crushing and milling operations a septic system will be installed for sewage disposal in the mill facility.

7.10.5 Surface Mobile Equipment

Mobile equipment needed for surface transportation and yard maintenance are:

Grader (rental)	1
Pick-ups (4 wheel drive all-terrain)	3
Cat dozer c/w backhoe	1

Forklift	2
Plant Area Crane (rental)	1

8.0 ENVIRONMENTAL ASSESSMENT AND PERMITTING

8.1 General

This pre-feasibility regulatory review has been prepared to provide a background and recommended strategy for the permitting of a new underground operation at the Copperstone Mine in La Paz County, Arizona.

Copperstone, as an existing property, represents a different situation than a 'greenfield' new mine development project for both permitting duration and costs. The significant permits to be obtained for the startup of the Copperstone mine include approval of a record of decision for the Environmental Impact Statement (EIS), the Aquifer Protection Permit (APP), and an Air Quality Permit.

New proposed mines in Arizona are currently requiring 5 to 8 years for permitting and legal challenges. However, Copperstone, as an existing property, represents a different situation than a 'greenfield' new development project for both permitting duration and costs. There are no recent comparable properties where a closed mine has resumed operations with a new operator. It is therefore anticipated that the permitting time line for Copperstone would be considerably shorter than that for a 'greenfield' new mine development, i.e. on the order of 2 to 4 years.

The proposed strategy for the in-pit placement of paste tailings should reduce the potential environmental impacts associated with the project. Based on the conceptual mine plan detailed in this study including underground mining from within the pit, and disposal of both waste rock and tailings within the pit, it is strongly believed that the project could receive all the required permits within 2 years of submittal of the Plan of Operations.

Permitting costs for the EIS, APP, and Air Quality permit including legal review and support would be in the range of US$ 800,000 to US$ 1,000,000. Opposition groups to the mine can cause significant delays and result in increased legal fees with little costs to themselves which provides a measure of uncertainty which should be addressed early in project feasibility.

Cyprus operated the mine from 1987 to 1993, comprising of an open pit, an exploration decline, a mill and ore processing facility, waste rock dumps, heap leach pad, and two tailings impoundments. The mine was closed under the approvals from the Arizona Department of Environmental Quality Aquifer Protection Permit and the Bureau of Land Management. While the existing mine closure and reclamation plan were approved by the proper regulatory authorities it is likely that more stringent standards would be required in 1999.

Consequently, future development by AMC should avoid disturbance of the existing mine waste disposal facilities to the degree possible. This would provide a clear demarcation between the mining conducted by the former operator and AMC.

Furthermore, it is proposed that AMC keep the proposed mine disturbance limited to the open pit area, the mill site, existing haul roads, and if necessary design and construct a new tailings impoundment outside the existing Copperstone waste piles. The proposed strategy for paste tailings disposal within the open pit is preferred from permitting and environmental standpoints over construction of a new tailings impoundment.

Recent permitting of new mines in Arizona has resulted in considerable delays due to changing regulations and opposition by environmental groups (e.g. Yarnell and Carlotta) Given these potential hurdles, it is recommended that significant effort be placed up front in the integration of mine planning and environmental protection.

The proposed underground mining operation at Copperstone offers a number of opportunities to minimize new impacts and potentially improve the existing environmental conditions. A potential strategy for using paste tailings disposal and in-pit placement would minimize water retention in the tailings; reduce the required area of disposal, probably eliminate the need for lining (evaluation of cyanide detoxification would be required), and greatly simplify closure requirements. Construction of a new tailings impoundment may allow lower capital and operating costs but result in greater permitting and closure costs.

8.2 Environmental Regulations in Arizona

Golder has documented procedures, timelines and potential costs for acquiring State and Federal permits for mineral exploration and mining operations. The principal objectives for acquiring the major permits are discussed in Appendix F.

The three major permits are regulated by the following federal laws:

1. National Environmental Policy Act (NEPA)

- Copperstone has existing mining related impacts and surface disturbance and will not have the same complexities as permitting a new mining project. However, the NEPA process has frequently be used to tie pre-existing environmental liabilities to the new operator's responsibility. To minimize this potential the proposed mining plan for Copperstone gold would prevent disturbance outside of the existing surface disturbance boundaries and not utilize the pre-existing tailings or heap leach facilities. The proposed strategy for minimizing the NEPA process includes the following:
- Design mine access and waste rock disposal facilities to lie within the limits of the existing open pit disturbance.
- Design tailings disposal facility to lie within the limits of the existing open pit disturbance.
- Use paste tailings disposal method to maximize water recovery and minimize water consumption.
- Locate mill and associated plant facilities in area of previous facilities to minimize disturbance.
- The NEPA process for an Environmental Impact Statement (EIS) ideally should require 18 to 24 months, given baseline contracting, public scoping, establishment of a memorandum of understanding with the lead agencies, one year of baseline data collection, analysis, and document preparation. However in practice, the EIS process for new 'greenfield' mines in Arizona is presently requiring 5 to 8 years. Given that there is some existing environmental documentation and that the project could be designed to be constructed within the existing surface disturbances, the EIS process could likely fall between the 24 to 36 month timeframe.

- EIS costs are driven by the number, and complexity of the baseline studies required, and by the duration of the analysis and reporting. Typical EIS costs would fall within a $500,000 to $1,000,000 range, which includes paying agency personnel for review time and assuming a third-party subcontractor. If the AMC proposal minimizes new surface disturbance, utilizes the pit for waste disposal and minimizes water consumption, it is our judgement that the NEPA process should be completed in 24 months with total costs less than $500,000.

2. US Army Corp of Engineers (ACOE)

- The U.S. Army Corps of Engineers (ACOE) regulates the discharge of dredged or fill material and adverse modification of waters of the United States. Statutory authority of the ACOE is Section 404 of the Clean Water Act.
- The determination of whether a 404-Permit will be required for Copperstone will depend upon the mine plan, e.g., the degree to which facilities or roads impact drainages, and the determination by the ACOE of impacted watersheds. There is a potential that a 404-Permit will not be required, and if required, would likely only require a nationwide permit. However, the presence of sensitive issues, e.g., threatened and endangered (T&E) species could lead to an individual permit. Therefore a T&E species clearance survey should be conducted early in the permitting process. Other potential resource concerns (e.g., transportation, visual, air quality, socioeconomic, land use, recreation, noise, wildlife, vegetation, soils, etc.) do not appear to be significant in relation to the proposed project.
- The State of Arizona also requires a 401-Permit, which is similar to the ACOE 404-Permit. The purpose of the 401-Permit is to ensure that federal activities do not violate state water quality standards when a facility may result in discharge of waters of the state.
- It is recommended that a 404-Permit review and delineation be performed early in the feasibility process to allow the mine plan to be designed to eliminate the need for a formal 404-Permit. This preliminary review and waterway delineation could be accomplished in two to three months and should be completed by a recognized expert in close consultation with the appropriate regulatory authorities.

- It is our judgement that given appropriate information concerning the drainages and waterways that the scoping level mine plan can be designed to prevent the 404-Permit requirement at Copperstone.

3. National Pollutant Discharge Elimination System (NPDES)

- The NPDES program utilizes numeric water quality limits for the discharge of effluent to public and federal lands. In the general provisions that address these discharges an exemption from the standards is permissible if the discharging facility is designed, constructed and maintained so as to contain the maximum volume of wastewater that would be generated during a 24 hour period, plus the runoff contribution resulting from a 10 year, 24 hour precipitation event. The facility must take all reasonable steps to maintain treatment of the wastewater and minimize the amount of overflow.
- It is recommended that the mine design be developed as a zero discharge facility, in that, all process water (i.e. tailings water) would be contained in a closed loop for reclaim. Any mine water encountered in the underground workings would be utilized in the process circuit or dust control. Under these conditions a NPDES permit is not required for the Copperstone project.
- The time frame to process a new permit is typically six to eight months, with the permit being valid for a period of 5 years. The project water balance and water consumption rates need to be designed such that all mine water, reclaimed process water, and storm water from the pit are contained within the zero-discharge loop. No costs are anticipated for a NPDES permit. A storm water permit and pollution prevention plan will however be required. Costs associated with this effort should be less than $100,000.

8.3 State of Arizona Permits

Through the passage of the Environmental Quality Act in 1986, the State of Arizona has established the general requirements of the Aquifer Protection Permit (APP). The APP is a groundwater protection program designed to maintain the quality of Arizona's groundwater resources. The ground water underlying the Copperstone site is found at a depth of 550 to 685 feet, corresponding to an elevation of 213 to 330 feet above sea level. No permanent streams or springs exist and ephemeral drainages are poorly defined. Surface water resulting from precipitation is minimal and of short duration.

An APP applicant with a potentially discharging facility (e.g., waste rock, non-stormwater ponds, tailings facility, leach operations, etc.) capable of discharging pollutants to the groundwater is obligated to make two demonstrations:

- That the facility will be designed, constructed and operated in a manner that ensures the greatest degree of discharge reduction achievable through the application of the 'Best Available Demonstrated Technology Manual (BADCT), processes, operating methods or other alternatives.
- That the discharge will not cause or contribute to a violation of an aquifer water quality standard at the applicable point of compliance, nor cause further degradation of a previously impacted aquifer.

Demonstrations also must be made of the applicant's technical and financial capability and compliance with local zoning ordinances and regulations.

In brief, the various application requirements deal with the characterization of the hydrogeologic setting, evaluation of the physical and chemical characteristics of any mining wastes, project design, and a BADCT analysis. The proposed process facilities at Copperstone will be designed, constructed, operated, and maintained so as not to discharge to the environment. Under the state general requirements these facilities are exempt from APP permit requirements. A listing of the various hydrogeologic and analytical requirements of the permit program are detailed in Appendix F.

The APP process needs to be conducted on a parallel path with the design of regulated facilities, e.g., waste dumps, process impoundments, non-stormwater ponds, tailings facilities, etc., as the design engineer (engineer of record) for these facilities needs to be intimately involved in the APP process. For planning purposes, the APP/design process generally requires on the order of 2 years.

As with the NEPA process, the complexity of obtaining the APP depends on the mine plan and associated potential impacts. Design of paste tailings disposal within the existing open pit would require a BADCT demonstration. A conventional lined tailings impoundment, with slurry tailings deposition, would also meet BADCT. It is however, recommended that due to lower water consumption, less disturbed area associated with waste disposal facilities, and significantly reduced closure effort, that in-pit paste tailings disposal be proposed for this project. Potential combined disposal of waste rock and tailings may also reduce potential impacts to groundwater quality.

The APP, given the proposed approach, could likely be obtained within 12 months. Much of the information for the APP would be developed during the Feasibility Study and EIS.

Costs associated with the development of the APP including legal support would likely be in the $300,000 to $500,000 range.

There are a number of other permits that the State of Arizona requires to ensure safety and construction of water retention structures. Construction standards and procedures need to be met, however for the proposed approach of using in-pit tailings disposal, precludes the necessity for a dam safety permit. Regulatory requirements for monitoring well construction should not cause any significant project delays of added costs.

The Air Quality Permit required for Copperstone would be administered under the Arizona Revised Statutes (ARS). The permit usually requires a minimum of four months to obtain depending on the size and complexity of the facility under control. The purpose of the Air Quality Permit is to protect public, animal and plant life from the adverse effects of pollution discharge into the atmosphere. Once approved the permit is valid for a period of five years.

Essentially the same as a closure and post-closure plan documented in the APP application report, the Arizona State Mine Inspector requires the submission of a separate Reclamation Plan. As most closure and post closure plans for new projects are essentially conceptual in nature, the primary purpose of the Reclamation Plan is to demonstrate that the mine plan is compatible with acceptable closure criteria.

Other miscellaneous permits, notifications, or certificates that may or may not be required, depending on the mine plan adopted for the next level of study, include:

- Notice of Intent to Clear Land (if there are listed T&E plants);
- Cultural Resource Compliance Review (archeological survey prior to sale of land); Wastewater Reuse Permit;
- Hazardous Waste Treatment, Storage or Disposal Permit; Solid Waste Disposal;
- Mine Start-up Notice (Registration/Safety);
- Notification of Underground Storage Tanks; and,
- Notification of Mining Industry Off-Road Motor Vehicle Waste Tires.

Local zoning requirements applicable to La Paz county will also need to be complied with, but these requirements generally do not have onerous cost of time implications.

9.0 CAPITAL COST ESTIMATE

9.1 Summary

The capital cost estimates for the proposed facilities are based on the scope of work described below and should be considered order of magnitude estimates with a probable range of accuracy of +/- 25%

The estimated capital cost to carry out the design, supply, construction and commissioning for the stated scope is $ 22.5 million 1st quarter, 1999 U.S. dollars with no allowance for escalation beyond this point.

The estimated direct and indicated capital costs are summarized below:

Table 9.1: Project Capital Costs (US$ 1,000's)

Category	Total Cost
Mining and Site Preparation	3,307
Crushing, Process Plant	8,066
Infrastructure and Ancillary Buildings	2,058
Plant Site Services	1,241
Total Direct Costs	14,672
Project Indirects	2,366
Owners Cost	1,748
Contingency (20%)	3,757
Total Indirect Costs	7,871
Total Project Costs	**22,543**

9.2 Scope of Work

Costs included in this estimate allow for the complete design, construction and commissioning of the equipment, materials and structures necessary to complete the facilities shown on Figures 6.1, 6.2 and 6.3. Costs are also included for underground mining pre-production development.

Costs associated with surface and underground ore definition drilling; fees payable to consulting companies to complete geotechnical and hydrology studies; environmental data collection; and, the preparation of bankable feasibility documents are not included.

For the purpose of producing the capital cost estimates, the projects have been divided into areas of direct and indirect costs.

9.3 Direct Costs

For the purpose of this study, all direct costs have been estimated using the following methods:

9.3.1 Equipment Costs

All equipment was priced based on in-house historical data.

9.3.2 Other Costs

Structures were priced on a plan area basis including foundations, structural and platform steel, architectural finishes and mechanical and electrical services as required. In plant piping, electrical and instrumentation were priced as a factor based on the purchase cost of process equipment. Allowances were made for site preparation and fencing and site wide electrical and water distribution systems.

9.4 Mining

The capital costs for mine development are estimated to be about US$ 3.31 million and include:

Contractor's mobilization	$ 42,000
Ore definition diamond drilling	$ 100,000
Owner's mining equipment	$ 1,748,500
Preproduction development	$ 1,416,840
Total	$ 3,307,340

The details of the owner's mining equipment costs are shown in Table 9.2. As a contractor will be used for the duration of the mine life, no major pieces of mobile equipment are purchased. The rental for these units is included in the contractor unit rates applied to each activity. Three utility vehicles are purchased by the owner to be used by the technical staff.

The fixed plant equipment includes ventilation fans, bulkheads, pumps, a portable compressor, paste backfill line, an emergency hoisting system, and an allowance for underground electrical equipment.

Installation costs for the ventilation fans, the compressor, and bulkheads have been estimated at 30% of the equipment purchase price. New equipment is assumed for all items.

Table 9.2: Owner's Mining Equipment Costs (US$)

Equipment		Number	Unit Cost	Cost
Utility truck	Supervision	2	40,000	80,000
Utility truck	Surveyors	1	45,000	45,000
Main ventilation fan	48"	1	350,000	350,000
	Installation			105,000
Secondary fan	42"	2	11,000	22,000
	30"	10	7,000	70,000
Ventilation bulkhead	Double door	2	30,000	60,000
	Installation			18,000
Main pumps	High head	4	30,000	120,000
Secondary pumps	Low head	8	6,000	48,000
Compressor	Allowance	1	100,000	100,000
	Installation			30,000
Emergency hoist	Allowance	1	300,000	300,000
Latrine	Portable	2	1,000	2,000
Refuge station	Portable	2	15,000	30,000
Powder/cap magazines	Allowance	2	20,000	40,000
Storage area	Fuel/supplies	1	50,000	50,000
Paste fill pipeline	Sched 80	3,600 ft.	35	126,000
Paste fill pipeline	HDPE	500 ft.	5	2,500
Electrical systems	Allowance	1	150,000	150,000
			Total	**1,748,500**

Preproduction capital development costs are estimated at US$ 1.42 million, including $0.60 million for lateral development and US$ 0.82 million for ventilation raises and are summarized in Table 9.3.

The unit costs are based on a current mining contractor quotation and include all men, material, and equipment costs associated with development and waste haulage and disposal near the portal.

An allowance of US$ 150,000 is included for each bored ventilation raise for surface casing through about 200 ft of overburden.

Table 9.3: Preproduction Capital Development Costs (US$)

	Unit Cost	Total Cost
Preproduction – lateral		
D1 ramp - leg 4 (8 to 9)	525	84,000
C1 ramp - leg 1 (10 to 11)	455	91,000
C1 ramp - leg 2 (11 to 12)	455	118,300
C1 ramp - leg 3 (12 to 13)	455	118,300
Diamond Drill Drift	425	153,000
D1 access drifts	425	21,250
D1B access drifts	365	10,950
Preproduction - raise		
Vent Rse #1 to Surface (8 ft dia)	495	284,784
Vent Rse #2 to Surface (8 ft dia)	495	235,256
Borehole Casing (allowance)	150,000	300,000
	Total	**1,416,840**
	Lateral	596,800
	Raise	820,040

9.5 Process Plant

The direct capital cost for installed process equipment and facilities is US$ 8.07 million, tabulated by plant area, in Table 9.4, below:

Table 9.4: Process Equipment costs (US$ 1,000's)

Process Area	Total Cost
Crushing and Conveying	1,193
Ore Storage, Grinding and Gravity Concentration	1,905
Thickening and Leaching	929
CIP, Carbon Treatment and Gold Recovery	2,256
Process Reagents	288
Paste Tailings System	1,495
Total	**8,066**

A mining equipment supplier was consulted on the cost of the major crushing and grinding equipment units. Estimated installed costs for the leach, CIP, gold recovery plant and associated reagent handling facilities were collected from MRDI records and scaled to suite the area and plant conditions of the Copperstone project.

Golder were consulted on the equipment and chemical needs of a small-scale cement paste tailings. Installed costs for paste tailings circuits were obtained from documented operational data available to MRDI.

9.6 Plant Infrastructure

The direct capital cost for plant site infrastructure and service requirements is US$ 3.30 million.

Power distribution and emergency power costs at site were estimated from other MRDI projects. A budget cost allowance was supplied for site water supply and distribution.

Buildings, other than the mill and refinery structure, were limited to modular or trailer structures to minimize costs. Rehabilitation costs of structures presently on-site were accounted for. Walls were omitted where appropriate (reagent storage, small vehicle shop) due to the amenable climate, whilst wind and heat covers on selective equipment were costed.

The costs for mobile plant equipment were based on historical data available to MRDI. MRDI supplied a budget allowance of US$ 68,000 for sewage disposal and fuel storage.

9.7 Indirects

The total for project Indirect Costs including Owner's costs and Contingency is US$ 7.87 million. Costs are included for engineering, procurement and construction management; capital spares; freight and taxes; and, start-up and commissioning.

9.7.1 E.P.C.M. Costs

Costs are included for the engineering design, procurement and construction management for the project. Costs have been estimated as a percentage of the total direct costs based on similar projects.

9.7.2 Contractor Indirects

Contractor indirects have been included in the direct costs.

9.7.3 Spare Parts

An allowance has been included for spare parts.

9.7.4 Freight and Taxes

An allowance of 3% of equipment and material costs has been included for freight. No sales taxes are included.

9.7.5 Commissioning and Start Up

An allowance has been included for engineering and contractor personnel for start up and commissioning.

9.7.6 Contingency

A "contingency" allowance is intended to cover the cost of items typically missed at the current level of project definition. For a scoping study, the contingency allowance is typically in the order of 25% of total direct costs plus project indirects.

In this study, the assumed contingency is 20%, since a number of cost elements had higher than normal levels of confidence, i.e.:

- Mine mobile fleet and major process equipment costs based on vendor "budget' quotations.
- Costs for process sections were based on data from a recently completed project of similar scope.

9.8 Owners Cost

Owners costs have been included in the project cash flow analysis. These costs cover Owner's project staff personnel, time/travel, recruitment, relocation and training of operating personnel, insurance, and expenses related to project implementation. These cost are paid by the Owner once the corporate decision to proceed with project production has been made.

An allowance of US$ 1.75 million has been included for Owner's Cost.

9.9 Assumptions

The following assumptions have been made in the preparation of this estimate:

- All equipment supply and installation subcontracts will be competitively tendered on a lump sum basis.
- All equipment will be new.
- Site work will be continuous and will not be constrained by the Owner.
- The work week for the construction phase of the project will be 50 hours per week.
- Skilled trades persons, supervision and contractors will be readily available.

- Construction workers will commute daily.
- Construction will be on an open shop basis.
- Temporary water and power for construction will be provided by the Owner.
- Any material or equipment which is to be removed or relocated requires no special handling or protective equipment and can be disposed of on site.

9.10 Exclusions

The following costs have not been included in this estimate:

- Building, environmental or other permits.
- The costs of rights, royalties and licensing fees.
- Ongoing capital costs.
- Land acquisition costs.
- Legal costs.
- Financing costs including interest during construction.
- Escalation beyond first quarter, 1999.

10.0 OPERATING COST ESTIMATE

10.1 Summary

Operating cost estimates were calculated for the mining, processing and ancillary facilities described in corresponding sections of this report. All operating costs are expressed in first quarter 1999 US dollars with no allowance for escalation.

All operating and maintenance costs have been summarized into the major categories of Mine, Process Plant Operations and General and Administration (G&A) costs.

The estimated annual operating costs for the Project are shown in Tables 10.1 and 10.2.

Table 10.1: Total Annual Operating Costs (US$ 1,000's)

	Operating Year					Total
	1	2	3	4	5	
Mining	7,609	6,983	7,215	7,079	3,914	32,800
Processing	4,311	4,600	4,600	4,600	2,744	20,855
G and A	1,654	1,765	1,765	1,765	1,053	8,002
Total	13,574	13,348	13,580	13,444	7,711	61,657

Table 10.2: Unit Operating Costs (US$ / ton processed)

	Operating Year					Average	Distr.
	1	2	3	4	5		%
Mining	44.48	38.26	39.53	38.79	35.96	39.64	53
Processing	25.21	25.21	25.21	25.21	25.21	25.21	34
G and A	9.67	9.67	9.67	9.67	9.67	9.67	13
Total $/ton processed	79.36	73.14	74.41	73.67	70.84	74.52	100
Total $/oz recovered	86.80	186.93	187.71	185.83	187.64	149.08	
Processed (tons ore)	171,050	182,500	182,500	182,500	108,850	827,400	
Head grade (opt)	1.016	0.435	0.440	0.440	0.419	0.555	
Gold recovered (oz)	156,381	71,405	72,346	72,346	41,091	413,570	

10.2 Mining Operating Costs

Mining costs are based on operating two eight hour shifts per day for 350 days per year. Unit rates for development, stoping, and other mining activities are taken from recent budgetary quotations from a USA based mining contractor for a similar project and are considered suitable for this study. The annual operating costs for underground mining are summarized in Table 10.3.

Table 10.3: Mine Operating Costs (US$ 1,000's)

	Operating Year					Total
	1	2	3	4	5	
Mining	4,789	5,350	5,475	5,475	3,266	24,355
Development	1,395	148	130	119	34	1,826
Ore Haulage (surface)	319	340	340	340	202	1,541
Diamond Drilling	525	525	525	525	-	2,100
GME	581	620	620	620	370	2,811
Equipment Replacement	-	-	125	-	-	125
Demobilize Equipment	-	-	-	-	42	42
Total	7,609	6,983	7,215	7,079	3,914	32,800

10.3 Processing Operating Costs

Process plant annual operating costs are based on a treatment rate of 182,500 tons per year (nominal 500 tpd, 365 days per year). The annual process operating cost estimates are summarized in Table 10.4.

Table 10.4: Process Operating Costs (US$ 1,000's)

	Operating Year					Total
	1	2	3	4	5	
Operating labour	1,347	1,439	1,439	1,439	856	6520
Maintenance labour	695	741	741	741	443	3361
Maintenance supplies	316	338	338	338	201	1532
Consumables	989	1,055	1,055	1,055	630	4784
Electric power	795	848	848	848	506	3845
Mobile equipment	169	180	180	180	107	816
Total	4,311	4,601	4,601	4,601	2,743	20,857

Labour cost estimates are based on a project work schedule of two 12 hour shifts per day, 365 days per year for process plant operations and of one 12 hour shift per day, 350 days per year for crusher operations. Salaries, hourly wage rates and fringe benefits in this study are based on a recent survey of North American mining industry wage rates.

Power costs are based upon a unit-rate of $0.05 per kWh applied to the average power draw.

Cost of consumables and supplies are based on current North American pricing obtained from vendors, supplemented as necessary with information from MRDI's files on similar operations. The annual cost for equipment spares and routine maintenance supplies is based on 3% of new equipment cost. The operating cost for process mobile equipment is derived from an allowance of US$180,000 per year.

Details of the various processing cost categories are given in Appendix C.

10.4 General and Administration Operating Costs

General and administration (G&A) costs are based on a treatment rate of 182,500 tons per year. The estimated annual G&A costs for a typical operating year are summarized in Table 10.5.

Table 10.5: General and Administration Operating Costs (US$ 1,000's)

	Typical Operating Year
Administration personnel	697
Administration expenses	916
Doré transportation and treatment charges	152
Total	1,765

Annual salaries for plant staff are based on MRDI's database of costs.

General and administration expenses include items such as property and equipment taxes and insurance, office and communication expenses, safety and office supplies, legal and accounting supplies, etc, and are shown in detail in Appendix D.

11.0 FINANCIAL ANALYSIS

The financial model for the Coppertone Project was built to determine the financial returns for the project and to test the sensitivity to changes of various parameters on a pre-tax project basis.

The inputs to the model were generated to a scoping study level of accuracy as stated previously. US Federal and Arizona State tax regulations were not included in generating the pre-tax cash flow model.

11.1 Basis of Financial Analysis

The model uses pre-tax Discounted Cash Flow (DCF) techniques to determine the pre-tax Net Present Value (NPV) and the pre-tax Discounted Cash Flow Rate of Return (DCFROR) of the project base case. The project base case is:

- a 500 tpd mining operation;
- operating 365 days per year;
- a mining resource of 827,400 tons, at a grade of 0.56 opt;
- based on the above resource estimate and mining rate, a mine life of 4.5 years results;
- reflected in underground mining using cut and fill methods and ramp access;
- processing using CIP producing a doré product, with a mill recovery of 90%;
- paste tailings disposal in the open pit, and underground as backfill; and,
- the price of gold at US$ 300 per ounce.

Some simplifying assumptions used in the financial model were:

- Refining charges reflect 95% paid gold, US$ 1.50 per oz treatment charge and US$ 0.40 per oz charge for transportation and insurance;
- Working capital is calculated on an annual basis, based on an initial fill of 40% of the first year operating costs prior to year one, and thereafter

based on annual changes to inventory, accounts receivable and accounts payable;

- Gold inventory days were assumed to be 10, accounts receivable were 30 days and accounts payable were 20 days;
- recovery of working capital is completed during the final year of mining;
- a reclamation costs of US$ 250,000 were assumed for the year following the completion of mining;
- No salvage value for equipment was applied:
- No inflation was incorporated in the model;
- Analysis was done on a 100% equity basis; and,
- Patch Living Trust royalties of 1% of Gross Proceeds.

Capital and operating cost inputs were based on those derived in Sections 9 and 10 respectively. In addition to the direct mining costs, refining charges were added to arrive at the final revenues.

11.2 Results of Financial Analysis

Under the present assumptions and design parameters, indications are that the project has fairly robust economics. At a 10% discount rate, the project pre-tax NPV is US$ 18.18 million. The project pre-tax DCFROR of the base case is 45.4%.

The project is most sensitive to Gold Price showing an 12.9% return before tax if the gold price drops to levels near US$ 240, and is nearly as sensitive to changes in mill recovery and ore grade. The project is least sensitive to changes in capital costs showing a pre-tax DCFROR of 35.4% if the costs were to increase by 20%, while operating costs show a reduced pre-tax DCFROR of 28.1% for the same increase.

On a pretax basis, the project payback is approximately 1.2 years.

Based on this analysis, this project would benefit from an increase in tonnage delineated through further exploration. This would improve the potential for securing debt financing for a portion of the capital costs and provide financial leverage thus increasing returns.

Sensitivities for the various financial parameters are summarized in Table 11.1 and 11.2 and are shown graphically in Figures 11.1 and 11.2.

Table 11.1 - Project Cash Flow - Pretax

Asia Minerals Corp.
Copperstone Joint Venture
Cash Flow Model (US$) - Project Basis Pretax
February-99

Year	0	1	2	3	4	5	Total
Inflation Factors							
Price	1.00	1.00	1.00	1.00	1.00	1.00	
Costs	1.00	1.00	1.00	1.00	1.00	1.00	
Ore Processed (tons)	-	171,050	182,500	182,500	182,500	108,850	827,400
Grade (oz/ton)	-	1.02	0.43	0.44	0.44	0.42	
Bullion Sales - gold (oz)	-	152,722	72,995	72,225	72,270	43,125	413,337
Gold Price (US$ /oz)	-	300.00	300.00	300.00	300.00	300.00	
CASH FLOW (US$)							
+ Revenue	-	43,296,652	20,693,948	20,475,788	20,488,545	12,226,023	117,180,954
- Operating Costs	-	13,574,393	13,347,760	13,579,910	13,443,960	7,710,387	61,656,410
= Gross Margin	-	29,722,259	7,346,188	6,895,878	7,044,585	4,515,636	55,524,544
- Third Party Royalty	-	432,967	206,939	204,758	204,885	122,260	1,171,810
= Net Operating Profit	-	29,289,292	7,139,248	6,691,120	6,839,700	4,393,375	54,352,735
- Capital Costs	22,542,890	-	-	-	-	250,000	22,792,890
- Working Capital Additions (Reductions)	5,429,757	3,186,728	- 1,862,939	- 13,147	- 5,424,643	- 1,315,755	- 0
= Cash Flow Before Taxes	- 27,972,647	26,102,564	9,002,187	6,704,267	12,264,343	5,459,130	31,559,845
Cumulative Cash Flow	- 27,972,647	- 1,870,083	7,132,105	13,836,372	26,100,715	31,559,845	
Payback Factor	-	-	1.2	-	-	-	1.2

Asia Minerals Corp.
Copperstone Joint Venture
Cash Flow Model (US$) - Financial Summary
February-99

Table 11.2 - Project Cash Flow Pretax

Sensitivities to Changes in Major Parameters

Discount Rate	0%	5%	10%	15%	20%
NPV ($'000)	31,560	24,010	18,182	13,623	10,016

Gold Price	250.00	275.00	300.00	325.00	350.00
NPV@10% ($'000)	4,191	11,186	18,182	25,178	32,173
DCFROR	18.7%	32.3%	45.4%	58.1%	70.6%

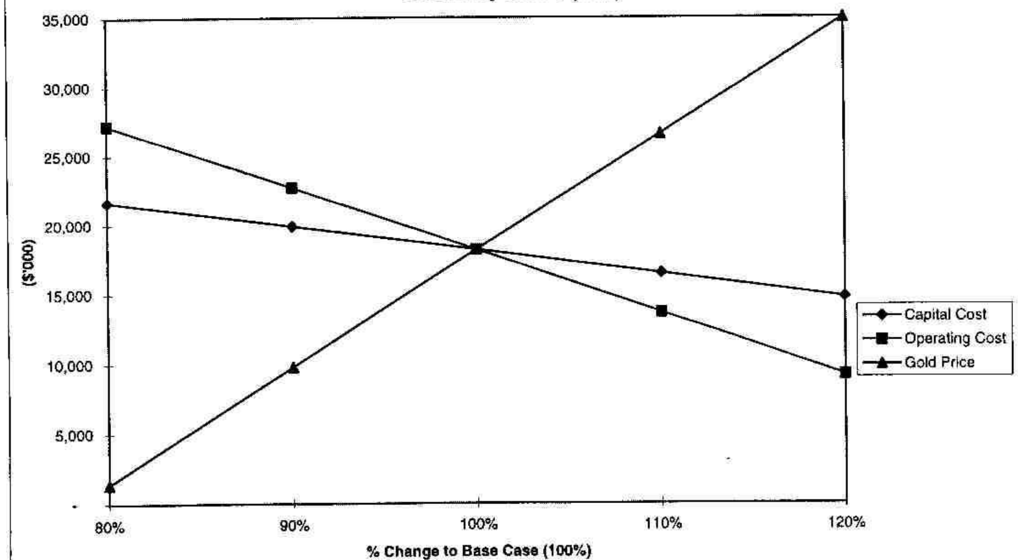
Mill Recovery	89%	90%	91%	92%	93%
NPV@10% ($'000)	17,254	18,182	19,110	20,038	20,966
DCFROR	43.7%	45.4%	47.1%	48.8%	50.5%

Parameter	NPV@10% ($'000)				
Change from base:	80%	90%	100%	110%	120%
Capital Cost	21,626	19,904	18,182	16,460	14,738
Operating Cost	27,208	22,695	18,182	13,669	9,156
Gold Grade	1,481	9,831	18,182	26,533	34,883
Mill Recovery	1,481	9,831	18,182	26,533	34,883
Gold Price	1,392	9,787	18,182	26,577	34,972
Parameter	DCFROR				
Change from base:	80%	90%	100%	110%	120%
Capital Cost	58.6%	51.5%	45.4%	40.1%	35.4%
Operating Cost	62.8%	54.0%	45.4%	36.8%	28.1%
Gold Grade	13.1%	29.7%	45.4%	60.5%	75.3%
Mill Recovery	13.1%	29.7%	45.4%	60.5%	75.3%
Gold Price	12.9%	29.7%	45.4%	60.6%	75.5%

Payback 1.2 Years

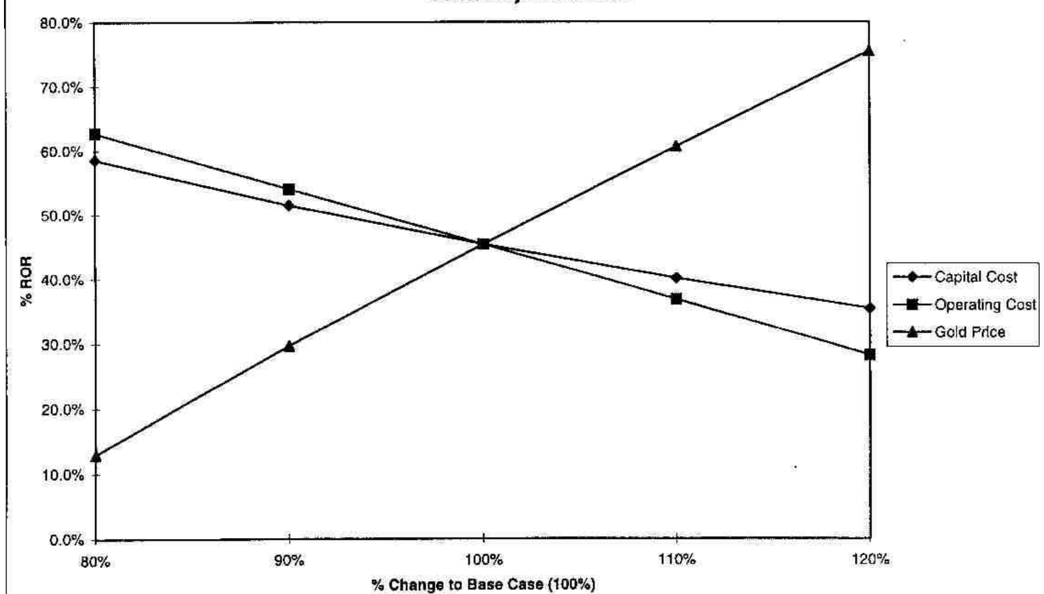
Asia Minerals Corp.
Copperstone Joint Venture
Cash Flow Model (US$) - Financial Summary
February-99

Figure 11.1 - Project Basis Pretax
Sensitivity of NPV (10%)



Asia Minerals Corp.
Copperstone Joint Venture
Cash Flow Model (US$) - Financial Summary
February-99

Figure 11.2- Project Basis Pretax
Sensitivity of DCFROR



12.0 PROJECT SCHEDULE

The next stage of project evaluation would be to bring the project to a level of full bankable feasibility study. The following recommended development programs and studies listed below must be completed to achieve this. Upon approval to proceed past the feasibility stage, the Project will enter the design engineering phase which will be followed by the 6 month construction phase. Start of plant construction is scheduled for the third quarter of 2000, with the first gold pour scheduled for the second quarter of 2001.

12.1 Geology and Resource Model

- Establish proven and probable reserves capable of servicing the loan for capital cost expenditure, or expand the mineable reserves for at least five years production.
- At least 40% of the resource (mining resource) at Copperstone should be proven.
- The resource estimate should incorporate structural, lithological and other ore controls with the objective of providing a geological-metallurgical model.

12.2 Underground Mining

- A surface "in-fill" drilling and underground core drilling program to increase confidence in the resource estimate.
- The size of the deposit, number of zones, mineable widths, etc., are needed to determine a suitable mine production rate. There are significant advantages from increasing the production rate above the 520 tpd assumed in this study.
- Study and select appropriate stoping method(s) based on geological and geotechnical information. Detailed mine development and stoping layouts are required to prepare feasibility level schedules, cost estimates and production forecasts.

- Need to acquire geotechnical data during diamond drilling. This is required to determine maximum stope excavation sizes and required ground support systems.

- A hydrology study to assess underground water seepage and determine need for drainage and pumping capabilities.

12.3 Metallurgical Testing and Process Design

- A metallurgical core drilling program to obtain representative rock type samples for investigation of the metallurgical response(s) of the Copperstone ores by evaluating specific metallurgical similarities or differences by oxidation state, deposit area, deposit depth and ore grade. A typical feasibility level testing program is itemized in Section 6.1.

- Interpretation of the test program results to develop process flowsheets, design criteria, major equipment lists, general arrangement drawings and process description narrative to support the capital and operating cost estimates of the most economically viable plant for Copperstone.

12.4 Permitting

- A preliminary study by local professional to review all socio-economic issues related to the project, i.e. manpower availability/housing in local townsites, unit wage rates, personnel transport, relocation of public roads, etc.

- Develop a project development plan (plan of operations) to support permit application submission to federal and state regulatory agencies.

12.5 Capital Costs, Operating Costs and Project Economics

- Obtain site specific quotations for mining services and major equipment from potential mining contractors and suppliers.

- Determine more detailed scope of facilities, services, personnel, etc., that contribute to the owner's costs and then prepare accurate cost estimates.

- The developed detailed capital and operating cost estimates will be combined with the mine plans and expected processing results to create a feasibility cash flow model for the project.

12.6 Project Management and Coordination

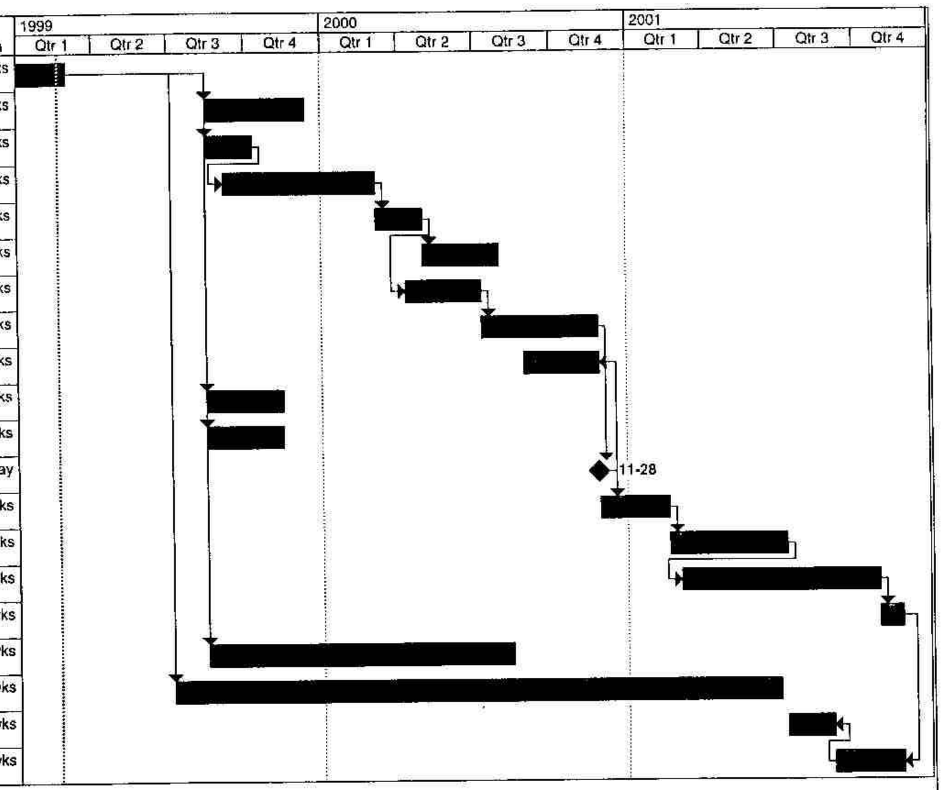
- The provision of coordination services throughout the exploration program and feasibility study should be adopted. This will ensure that all stages and aspects of the project are fully implemented in a controlled manner and in close cooperation with AMC's technical manager and staff.

12.7 Work Plan

Based on the review of the exploration mining plan and previous proposals for a similar project scope, the estimated costs to advance the Copperstone project to start of mine production are:

Description	Cost
Underground Exploration Development Program, Including Definition Drilling and Bulk Sample Collection (8month Period)	US $2,000,000
Metallurgical Testwork Program	US $50,000
Bankable Feasibility Study	US $380,000
Geotechnical and Hydrology Data Collection and Analysis	US $100,000
Environmental Data Collection, EIS, APP Permit Application	US $500,000
Contingency (at 20%)	US $600,000
Estimated Total Cost	**US $3,630,000**

ID	Task Name	Duration
1	Scoping Study	26 wks
2	Permitting and Surface In-fill Drilling	17 wks
3	Permit U/G Drilling Contract/Mobilize	8 wks
4	Exploration Development	26 wks
5	Definition Drilling, Bulk Sample	8 wks
6	Metallurical & Paste Testwork	13 wks
7	Resource Calculation	13 wks
8	Feasibility Study	20 wks
9	Arrange Financing	13 wks
10	Geotechnical Analysis	13 wks
11	Groundwater Analysis	13 wks
12	Production Decision	1 day
13	Basic Engineering	12 wks
14	Detailed Engineering	20 wks
15	Construction	34 wks
16	Commissioning	4 wks
17	Environmental Data (Plan of Operation)	52 wks
18	Permitting (EIS,APP,Air quality)	104 wks
19	Drilling Program & Predevelopment	8 wks
20	Preproduction Development	12 wks



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Date: Fri 99-02-19

Task | Split | Progress | Milestone | Summary | Rolled Up Task | Rolled Up Split | Rolled Up Milestone | Rolled Up Progress | External Tasks | Project Summary

Figure 12.1 - Preliminary Copperstone Project Schedule