

Corriente Resources Inc.

Mirador Copper Project

Feasibility Study Report



May 2005
142314

Knight Piésold LLC



11 May 2005

Corriente Resources Inc.
Suite 520 – 800 West Pender Street
Vancouver, BC
V6C 2V6

Attention: Ken Shannon, P.Geo.,
President

Dear Mr. Shannon:

**Reference: Mirador Copper Project
Feasibility Report**

I am pleased to present to Corriente Resources Inc, the Mirador Copper Project Feasibility Report.

I would like to acknowledge the diligent work and cooperation of your consultants Merit Consultants International Inc. and Knight Piésold Consulting during the course of the study and preparation of this Report.

I trust this report meets your needs and we wish you success with your ongoing development of this promising project.

Yours truly,

AMEC AMERICAS LIMITED

A handwritten signature in black ink, appearing to read "Tom Nikiforuk", written over a horizontal line.

Tom Nikiforuk, P.Eng.,
Study Manager

IMPORTANT NOTICE

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MIRADOR COPPER PROJECT FEASIBILITY STUDY REPORT

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GLOSSARY

Ampere.....	A
Annum (year).....	a
Centimetre.....	cm
Cubic centimetre	cm ³
Cubic metre	m ³
Day	d
Days per week.....	d/wk
Days per year (annum).....	d/a
Dead weight tonnes.....	DWT
Degree	°
Diameter.....	Ø
Dry metric ton	dmt
Foot.....	ft
Gallon.....	gal
Gram	g
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m ²)	ha
Horsepower	hp
Hour	h
Hours per day.....	h/d
Hours per week	h/wk
Hours per year.....	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre.....	kg/m ³
Kilograms per hour	kg/h
Kilograms per square metre	kg/m ²
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilovolt.....	kV
Kilovolt-ampere	kVA
Kilovolts.....	kV
Kilowatt.....	kW
Kilowatt hour.....	kWh
Kilowatt hours per tonne (metric ton).....	kWh/t
Kilowatt hours per year.....	kWh/a
Less than.....	<
Litre	L
Litres per minute.....	L/m
Megapascal	MPa



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Megavolt-ampere	MVA
Megawatt.....	MW
Metre	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second.....	m/s
Micrometre (micron)	µm
Milliamperes	mA
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre.....	mm
Million	M
Million tonnes	Mt
Minute (plane angle).....	'
Minute (time)	min
Month	mo
Ounce.....	oz
Parts per billion.....	ppb
Parts per million.....	ppm
Pascal (newtons per square metre).....	Pa
Pascals per second	Pa/s
Percent.....	%
Pound(s).....	lb
Pounds per square inch	psi
Second (plane angle)	"
Second (time).....	s
Specific gravity	SG
Square centimetre	cm ²
Square foot.....	ft ²
Square inch	in ²
Square kilometre	km ²
Square metre.....	m ²
Thousand tonnes.....	kt
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Volt.....	V
Week	wk
Weight/weight.....	w/w
Wet metric ton	wmt
Year (annum)	a



1.0 EXECUTIVE SUMMARY

1.1 Summary

In November 2003, Corriente Resources (Corriente) commissioned AMEC Americas Limited (AMEC) to be the primary consultant for the preparation of a bankable feasibility study for the Mirador Copper Project. The deposit is located along the valley of the Rio Zamora in the Zamora Chinchipe Province of southeast Ecuador, adjacent to the border with Peru. The Mirador site is approximately 340 km south of Ecuador's capital city of Quito, 70 km east-southeast of the city of Cuenca, and about 418 km from the Pacific Ocean port of Machala. The site coordinates are 03°34'N latitude and 78°26'W longitude and the elevation ranges from 800 to 1,400 masl).

The site is accessible year round by road. The access from Machala to the city of Loja is on 259 km of paved road and then an additional 170 km on paved and gravel roads to site. Tundayme is the closest village to the deposit and is also accessed by about 170 km of paved and gravel roads from Cuenca or Loja. There is national highway road access from Cuenca to the capital city Quito 270 km to the north. Scheduled air service is available from Quito to Cuenca or Loja. Only small, chartered aircraft can fly to Gualaquiza, the closest asphalt airfield, about 40 km to the northwest of the deposit.

The region is characterized by a wet equatorial climate with a reported rainfall of 2,700 mm/a. Individual rainfall events can be over 60 mm in a 24 hour period. The site is bisected at its base by the valley of the Zamora River that lies at an elevation of about 700 masl and is flanked by highland areas that rise to the east and west. Drainage is into the Zamora River via the Wawayme River. Variations in the local topography have influenced the location of the main plant facilities.

Regional exploration was initiated in southeastern Ecuador by BHP Billiton S.A. in 1994. They identified possible porphyry copper systems in the region, including Mirador. Corriente has carried out exploration on the Mirador site since April 2000. The work completed to date has included 24,350 m of core drilling in 91 diamond drill holes arranged in a grid on approximately 75 m to 100 m centres.

The feasibility study is based on conventional open pit mining of the porphyry copper deposit. The deposit is covered by an average 22 m depth of overburden and a leached cap, a portion of which must be pre-stripped to access the orebody. The sulphide ore body is relatively homogenous, consisting dominantly of primary copper sulphides and is open at depth. Secondary enrichment is thinly developed in places over the primary sulphide mineralization. Overall, the metallurgy is regarded as relatively simple.



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The estimated mineral resources included in the mine plan total approximately 111 Mt grading 0.67% Cu and 0.22 g/t Au. Approximately 91 Mt of waste rock will be removed over the mine life, resulting in an average strip ratio of about 0.8:1.

The mine plan is based on a contract mining company providing ore to a conventional copper concentrator to support an average milling rate of 25,000 t/d (9.125 Mt/a). All facilities are designed for this throughput and operate on a continuous basis, 24 h/d, 365 d/a. Average production is estimated to be about 174,000 t/a of copper concentrate over the mine life of 12 plus years.

Run-of-mine ore will be crushed in a gyratory crusher. The mill flow sheet selected for Mirador will be a conventional copper-gold porphyry flowsheet, with relatively coarse primary semi-autogenous and ball mill grinding to about 150 µm followed by copper rougher flotation, concentrate regrind to 30 µm, and cleaner flotation and dewatering. Concentrates produced are predicted to average 29.8% copper at a recovery of 91%. Gold recovery is expected to average 47%. A laboratory analysis of concentrates indicated that no significant deleterious penalty element impurities are present. The concentrate will be trucked via the existing road network in the area to a port facility in Machala for shipment to smelters. Tailings from the process will be impounded in a zero discharge tailings pond; water will be reclaimed from the tailings pond and reused in the process.

The major infrastructure required to develop the property includes road access upgrades, a run-of-river hydroelectric scheme, and power line. The hydroelectric development is not part of the scope of this study and the supply of power will be through an independent build-own-operate arrangement. A 2.7 km access road is needed to connect the plant and administration areas to the existing highway that passes by the property. Access between the plant area and the mine will be via upgraded existing roads including a new 5 km section to the pit area. A 10 km overland conveyor will connect the process plant and the crusher. In addition, a 100 km power line will have to be constructed, as part of the Sabanilla Hydroelectric Project, to connect the site to the hydroelectric power station and the existing power grid. The power demand of the project is about 28.3 MW. Seasonal variations in the output of the hydroelectric scheme result in an average output of about 23.5 MW and, on average, 4.8 MW of supplementary power will be imported from the existing power grid.

Merit Consultants International Inc. (Merit) estimated the capital cost to build the facilities as described in this report to be US\$202 million.



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Mining costs were based on contract mining budget quotations. Process operating costs and G&A costs were estimated by AMEC with input from Corriente. The estimated life of mine operating costs is US\$3.23/t mined or US\$5.41/t milled.

At a copper price of US\$1.00/lb and a gold price of US\$400/oz, the project has a pre-tax IRR of 14.8%, an NPV of US\$71.2 million at 8% and a five-year payback.

The financial analysis indicates the project is most sensitive to changes in revenue, either through increased copper prices or increased tonnage. It is AMEC's recommendation that Corriente investigate increasing the plant throughput.

1.2 Project Background

1.2.1 Terms of Reference

Corriente is evaluating the development of the Mirador Copper Project in southeastern Ecuador. Corriente commissioned AMEC to be the prime consultant for the feasibility study and to compile this feasibility study report, with input from Corriente's various consultants as listed below. The feasibility study addressed the resource, mine plan, processing and support facilities, site access, transportation of concentrate, port requirements, concentrate marketing and project execution.

The following consultants were responsible for input to the feasibility study:

AMEC

- overall study compilation
- review of project geology
- review of sample QA/QC program
- estimate of mineral resources
- mine planning
- specification and management of metallurgical testwork program
- flowsheet development
- design of the process facilities and site support facilities
- design of mine roads and concentrator access road
- input to capital cost estimating:
 - equipment pricing for major process equipment
 - quantity estimation for major civil and structural components
- operating cost estimates for mining, process and administration



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- pre-tax financial analysis
- geotechnical engineering to support pit design
- site water management
- mine waste management
- ARD analysis
- specification and management of environmental and socio-economic baseline studies, including impact analysis.

Caminosca Caminos y Canales Cia. Ltda. (Caminosca)

- design of the site access roads and the mine haul road
- surveying.

Knight Piésold Ltd (KP)

- design of Tailings Management Facility
- geotechnical investigations to support tailings impoundment design
- investigations for freshwater wells
- design of tailings distribution pipework for TMF
- design of reclaim barge and pipeline for TMF
- TMF capital and operating cost estimates
- TMF closure plan.

Merit Consultants International Inc. (Merit)

- study coordination
- development of project implementation plan and schedule
- assessment of off-site infrastructure
- capital cost estimating.

Terrambiente

- environmental base line monitoring
- environmental impact assessment report
- socio economic studies
- environmental management plan.



Corriente

- geologic solids for dikes and supergene material
- geotechnical investigation of plant site area
- overland conveying system
- port operations and costing
- concentrate marketing
- concentrate trucking and shipping
- power supply.

1.2.2 Property Description and Location

The Mirador Project is located approximately 340 km south of Ecuador's capital City of Quito, along the valley of the Zamora River in the Zamora Chinchipe Province of southeast Ecuador, adjacent to the border with Peru at approximately 9 604 200 N, 785°000 E; UTM Zone 17S coordinates. The nearest village to the Mirador deposit is Tundayme, which is accessed by 170 km of paved highway and gravel roads from either Cuenca (about five to six hours travel time) or Loja (four hours travel time).

The area has a wet equatorial climate with a reported rainfall of 2,700 mm/a. Individual rainfall events can be over 60 mm in a 24-hour period. Due to variations in local topography there is a wide range in rainfall levels, and the area could be characterized as having numerous microclimates.

The valley of the Zamora bisects the project area at an elevation of about 700 masl. The flanking highland areas of the Paramos de Matanga on the west, and the Cordillera de Condor on the east, rise to maximum elevations of 4,200 and 3,500 masl, respectively. The elevation of the Mirador project ranges from about 800 masl to 1,400 masl. Drainage is into the Zamora via Wawayme River.

A site location map is provided in Figure 1-1 followed by photographs of the site.



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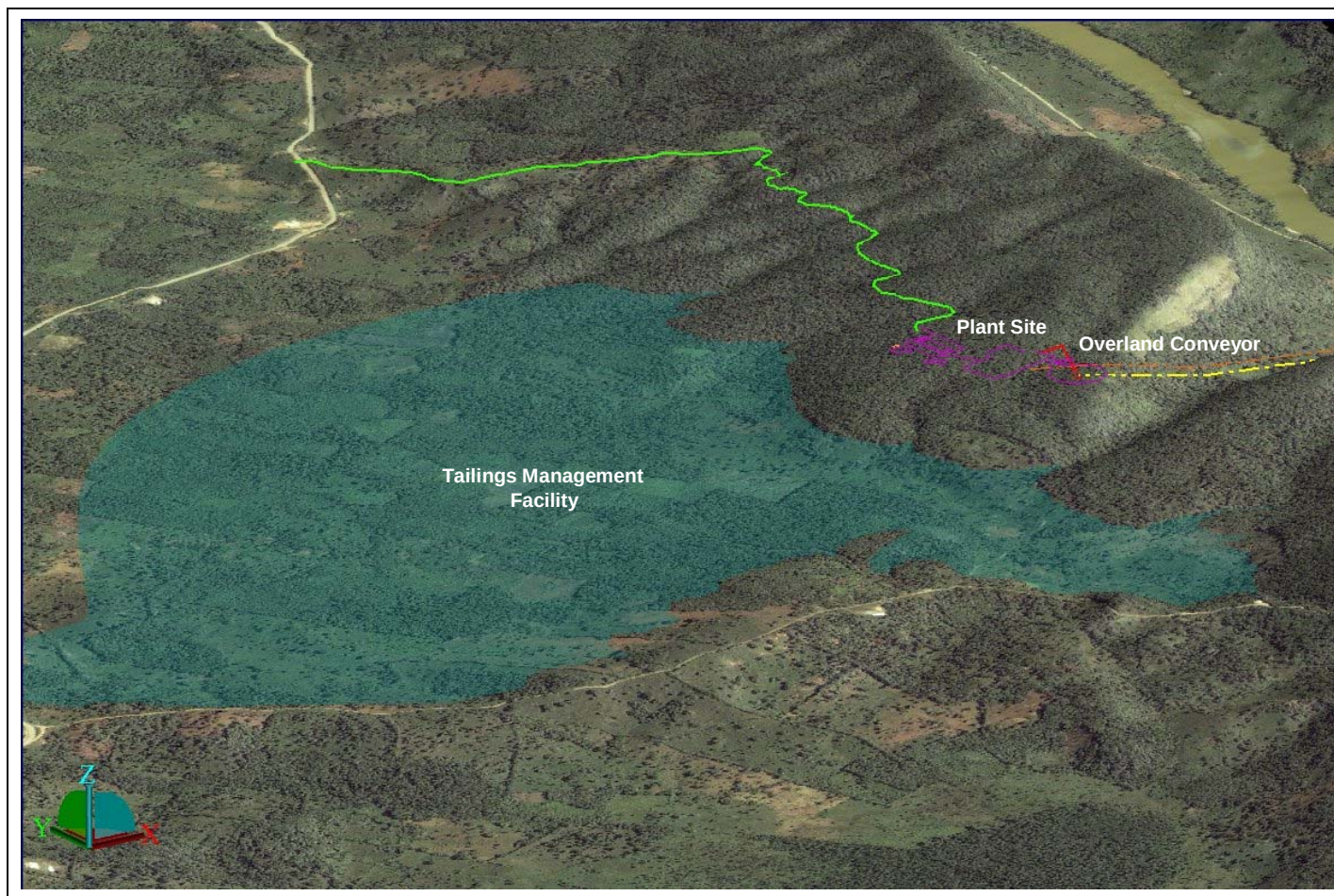
Figure 1-1: Location Map





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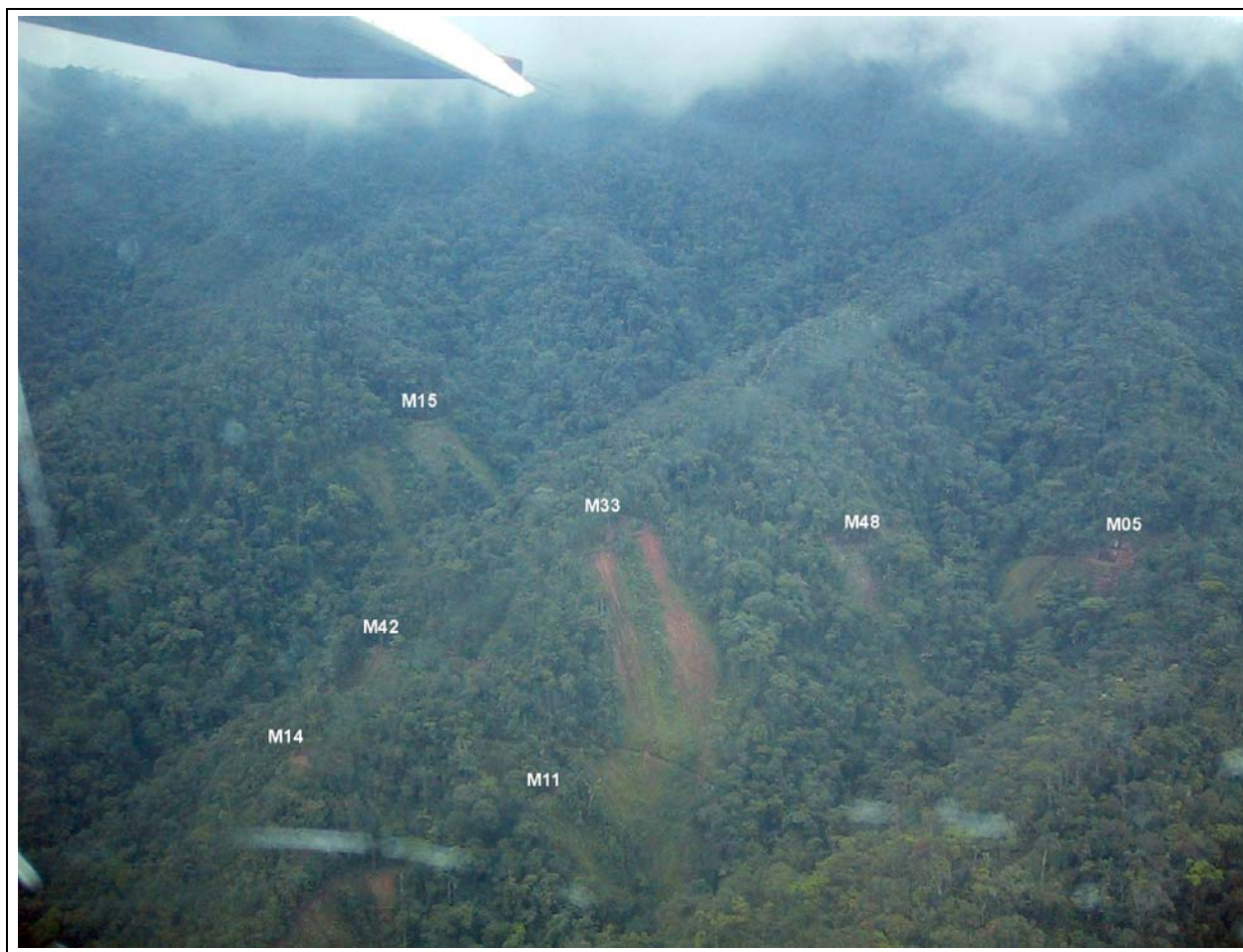
Figure 1-2: Plant Site Area





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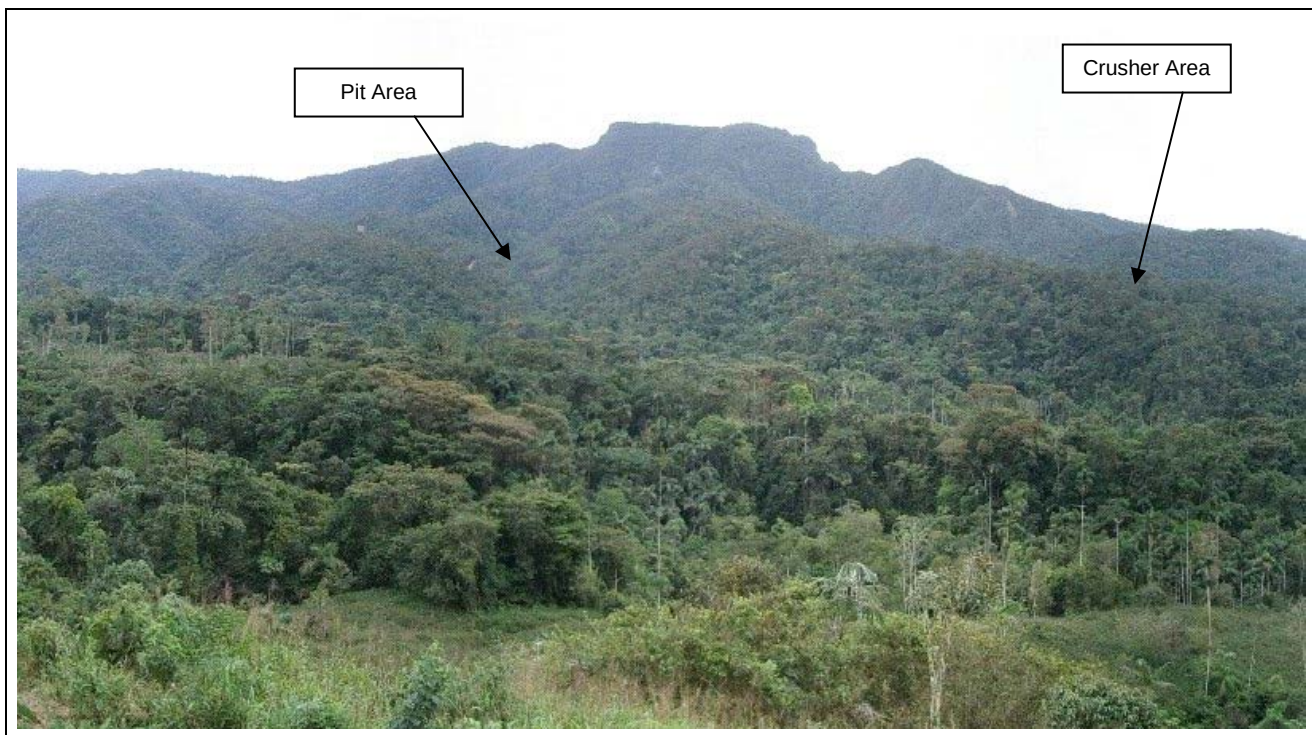
Figure 1-3: Pit Area Looking South





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Figure 1-4: View of the Crusher and Pit Areas





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Figure 1-5: Tailings Area Looking North





1.3 Geology and Resource Modelling

Corriente engaged AMEC to provide a mineral resource estimate and Qualified Person's review and Technical Report for the Mirador Project. The work entailed estimating mineral resources in conformance with the CIM Mineral Resource and Mineral Reserve definitions referred to in National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects. It also involved the preparation of a Technical Report as defined in NI 43-101 and in compliance with the format set out in 43-101F1. Susan Lomas, P.Geo., an employee of AMEC, served as the Qualified Person responsible for preparing the resource estimation.

1.3.1 History

Regional exploration was initiated in southeastern Ecuador by BHP Billiton S.A. in 1994. BHP Billiton used silt sampling to identify favourable areas of base metal anomalies. With further mapping they identified possible porphyry copper systems associated with these anomalies. At least eight separate porphyry copper systems have been identified in the region.

The area of Mirador was recognized as an area of interest during the original reconnaissance geological and geochemical surveys completed in December 1994. These surveys, which included 315 pan concentrates of stream sediments, defined an area roughly 50 km² that contained anomalous values for Cu, Mo, Au, Zn, and Ag. However, BHP Billiton was forced to focus its efforts in the north part of the belt since the present property was off limits between 1995 and 1999 due to the border conflict in the area between Ecuador and Peru.

After a peace treaty was signed with Peru in July 1999, BHP Billiton completed detailed follow-up surveys to better define the anomaly areas at the Mirador property. They collected 746 soil samples along ridges and 219 rock chips from outcrops found in drainages traversing the anomalous zones. This work along with geological and alteration mapping defined the Mirador (previously known as Wawayme) zone and the Chanco and Chanco Norte Zones to the southwest.

In October 1999, BHP Billiton entered into an option agreement with Corriente covering 880 km² in the northern part of the porphyry belt. Subsequently, BHP Billiton and Corriente entered into a further agreement in April 2000 covering 230 km² in the southern part of the porphyry belt (the Mirador – Chanco property).

In 2002, after completion of 52 diamond drill holes, results of a polygonal resource estimate were released (12 February 2002). The estimated tonnage and grade,



calculated at a 0.65% Cu cutoff grade, were 218 Mt grading 0.73% Cu, all in the Inferred Mineral Resource category.

In 2003, Corriente released a second mineral resource for the Mirador Project in a NI 43-101 Technical Report (February 2003). The updated resource estimate was based on 10 additional holes and estimated using polygonal methods. This resource estimate, calculated at a 0.65 Cu % cutoff grade, yielded 182 Mt grading 0.76% Cu and 0.221 g/t Au. The estimate was classified as Inferred Mineral Resources.

1.3.2 Deposit Type and Mineralization

The copper and gold mineralization encountered at the Mirador deposit display many of the characteristics of calc-alkaline type porphyry copper systems. Supergene zones of enrichment are weakly developed beneath a superficial leach cap that averages less than 22 m thick.

Mineralization at Mirador can be divided into early molybdenum, early copper $\pm$ gold, and late copper-gold stages. Molybdenum is associated with early quartz veining. Both copper-gold events are sulphide dominated. Chalcopyrite occurs as mainly irregular masses between clasts in the early breccia (with some disseminated) and as fine disseminations in a halo zone around the breccia within the Zamora granite. The syn- to post-mineral porphyry dykes are variably and weakly mineralized by the late copper-gold disseminations and sparse sulphide veins.

A late stage series of 1 cm to 2 cm quartz sulphide stingers transect the project. They are subvertically emplaced but their strike is unknown, as they have only been seen in drill core. These veinlets are sulphide rich and have elevated gold and zinc values. They were intersected in the vertical drill holes and were found running down the core over metres of core length. AMEC developed a capping strategy to limit the influence of these high gold values during gold grade interpolation.

The Mirador porphyry system exhibits well-defined alteration zoning, with a large (approximately 4 km²), quartz-sericite alteration zone covering a large part of the wall rock and quartz diorite porphyry. This alteration has overprinted a potassic alteration whose remnants are small secondary biotite nuclei mainly in the quartz monzonite.

1.3.3 Resource Classification and Summary

The mineral resources of the Mirador project were classified using logic consistent with the CIM definitions referred to in NI 43-101. The project mineral resources were classified into one of two categories: Indicated or Inferred Mineral Resources. Review



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of the model relative to the drill hole data, in sections and in plans, showed good geologic and grade continuity in areas where sample spacing is around 100 m. The current drill hole spacing is nominally 100 m with some areas approaching 75 m. Combined with spatial statistical work and investigation of confidence limits in predicting planned annual production work, AMEC decided that blocks covered by this data spacing may be classified as Indicated Mineral Resources. In the hypogene mineralization two drill holes needed to be within 100 m of the block centre or a single drill hole within 35 m to be classified as Indicated Mineral Resources. Supergene mineralization needed to have two drill holes within 75 m of a block centre or a single drill hole within 35 m. Locally a second pass utilizing a one-hole rule was necessary to prevent a “striping” effect of isolated Inferred Mineral Resource blocks amongst mostly Indicated mineral resource blocks.

All interpolated blocks that did not meet the criteria for Indicated mineral resources were assigned as Inferred mineral resources if they fell within 200 m of a drill hole composite in the hypogene mineralization or 150 m if in the supergene zone.

Table 1-1 contains the results of the resource estimation for the Mirador Deposit as of 23 September 2004. The estimate is made by mineralization type. The resource estimate result for Mirador is being declared using the 0.40% Cu cutoff. For comparative purposes, additional copper grade cutoffs equal to 0.50. 0.55. 0.60. 0.65 and 0.70 % were also tabulated and they are included in Table 1-2. The resources are reported to a depth of 850 m elevation, which is approximately 500 m below the surface.

Table 1-1: Mirador Deposit Mineral Resource Summary, 23 September 2004

Zone		Tonnes	Au (g/t)	Cu (%)
0.4 Cu % Cutoff				
<i>Indicated Mineral Resource</i>				
	Mixed	1,300,000	0.23	0.57
	Enriched	6,700,000	0.24	0.99
	MNZD (mineralized)	301,700,000	0.20	0.65
Total	Indicated hypogene	301,700,000	0.20	0.65
	Indicated supergene	8,000,000	0.24	0.92
<i>Inferred Mineral Resource</i>				
	Mixed	-	-	-
	Enriched	1,200,000	0.25	0.83
	MNZD	313,900,000	0.17	0.56
Total	Inferred hypogene	313,900,000	0.17	0.56
	Inferred supergene	1,200,000	0.25	0.83



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Table 1-2: Mirador Deposit Mineral Resource Summary – Comparative Cutoffs, 23 September 2004

Zone		Tonnes	Au (g/t)	Cu (%)
0.5 Cu % Cutoff (Cu ID)				
<i>Indicated Mineral Resource</i>				
	Mixed	800,000	0.25	0.65
	Enriched	6,600,000	0.24	1.00
	MNZD	254,800,000	0.21	0.68
Total	Indicated hypogene	254,800,000	0.21	0.68
	Indicated supergene	7,400,000	0.24	0.96
<i>Inferred Mineral Resource</i>				
	Mixed	-	-	-
	Enriched	1,200,000	0.25	0.83
	MNZD	215,300,000	0.18	0.62
Total	Inferred hypogene	215,300,000	0.18	0.62
	Inferred supergene	1,200,000	0.25	0.83
0.6 Cu % Cutoff (Cu ID)				
<i>Indicated Mineral Resource</i>				
	Mixed	500,000	0.28	0.76
	Enriched	6,200,000	0.24	1.03
	MNZD	182,400,000	0.22	0.73
Total	Indicated hypogene	182,400,000	0.22	0.73
	Indicated supergene	6,700,000	0.24	1.01
<i>Inferred Mineral Resource</i>				
	Mixed	-	-	-
	Enriched	900,000	0.22	0.89
	MNZD	93,300,000	0.20	0.71
Total	Inferred hypogene	93,300,000	0.20	0.71
	Inferred supergene	900,000	0.22	0.89
0.65 Cu % Cutoff (Cu ID)				
<i>Indicated Mineral Resource</i>				
	Mixed	400,000	0.25	0.75
	Enriched	6,000,000	0.25	1.05
	MNZD	140,600,000	0.23	0.77
Total	Indicated hypogene	140,600,000	0.23	0.77
	Indicated supergene	6,400,000	0.25	1.03
<i>Inferred Mineral Resource</i>				
	Mixed	-	-	-
	Enriched	800,000	0.25	1.00
	MNZD	56,900,000	0.22	0.76
Total	Inferred hypogene	56,900,000	0.22	0.76
	Inferred supergene	800,000	0.25	1.00
0.7 Cu % Cutoff (Cu ID)				
<i>Indicated Mineral Resource</i>				
	Mixed	200,000	0.35	1.05
	Enriched	5,800,000	0.24	1.05
	MNZD	101,600,000	0.24	0.80
Total	Indicated hypogene	101,600,000	0.24	0.80
	Indicated supergene	6,000,000	0.25	1.05
<i>Inferred Mineral Resource</i>				
	Mixed	-	-	-
	Enriched	800,000	0.25	1.00
	MNZD	40,400,000	0.23	0.79
Total	Inferred hypogene	40,400,000	0.23	0.79
	Inferred supergene	-	-	-



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1.4 Mine Plan

1.4.1 Introduction

The resource model was divided into four copper/gold-bearing zones: leached, mixed, enriched, and mineralized. After resource modelling was completed, these four zones were combined into a single zone per block to reflect the mineability of each block, on a diluted basis. The mine plan treats leached material as saprolitic and all other material as sulphide.

Mining at Mirador will be by conventional open pit truck and shovel methods. Budget pricing was obtained for contract mining. The mining fleet is estimated to consist of two 19.9 m³ front-end loaders and eight 140 t trucks. Support equipment will include bulldozers, graders, and excavators to maintain the surfaces of the roads, dumps and operating benches and the water collection system at the pit rim and in-pit.

Mining occurs in three phases: a starter pit which is mined for eight years, an intermediate pit which is mined over a 9 year time frame and the ultimate pit which is mined over 11 years. Overlaps of the pits occur to balance waste stripping and ore feed grade. The annual mine production and mill feed forecast for the three mining phases is summarized in Table 1-3.

Table 1-3: Mine Production Forecast

Year	Total	Grade			Total	S.R.
	Mill Feed (t x 000)	NSR (\$/t)	Cu (%)	Au (g/t)	Waste (t x 000)	
-1*	2,100	8.570	0.661	0.223	14,175	6.75
1	7,025	9.622	0.743	0.227	14,344	2.04
2	9,125	9.880	0.762	0.228	7,481	0.82
3	9,125	8.846	0.683	0.237	7,132	0.78
4	9,125	9.066	0.700	0.241	5,974	0.65
5	9,125	8.850	0.683	0.234	5,903	0.65
6	9,125	8.736	0.674	0.230	5,907	0.65
7	9,125	8.453	0.653	0.238	5,908	0.65
8	9,125	7.962	0.615	0.198	5,911	0.65
9	9,125	8.383	0.647	0.200	5,877	0.64
10	9,125	8.242	0.636	0.196	5,898	0.65
11	9,125	7.880	0.608	0.184	3,811	0.42
12	9,125	8.178	0.631	0.187	2,628	0.29
13	1,127	9.387	0.724	0.203	223	0.20
Total	110,627	8.662	0.668	0.216	91,172	0.82

* Includes stockpiled ore for plant feed in Year 1



1.4.2 Open Pit Optimization

To run the pit optimization, an NSR model was created within Gemcom®. The calculated NSR values are based on copper values only, and do not include gold. Only model blocks carrying ore grades within the indicated category were assigned a revenue value. Blocks carrying grades in the inferred category were treated as waste, and referenced as Waste-Inferred in this report. This model was then imported into Whittle 4X® mine optimization software package. A series of pit shells were created at various net smelter return (NSR) values utilizing a revenue factor, these pit shells were then analyzed to select the best one for pit design. The best pit is the maximum NPV at the stated criteria. The Whittle® Milawa algorithm was used to generate production forecasts at varying throughputs, from 20,000 t/d through 35,000 t/d, assuming three pit phases. The shells were loaded back into the mine planning package and evaluated for accessibility and overall pit footprint. For the purposes of this study, Corriente chose 25,000 t/d, based on company objectives at this time, and design work, which had completed to date. The discount rate used in all Whittle work was 10%.

The parameters used to create the NSR and optimization models are summarized in Table 1-4.

The Whittle forecast output for the series of generated shells is shown in Figure 1-6. The Worst Case mining forecast simulated mining the shell in a top-to-bottom manner, a scenario that is unlikely to occur in even the most challenging environment. The Best Case mining forecast mines in a shell-by-shell manner, which is far too optimistic to occur and due to mining width, is not feasible. The Specified Case allows mining each specified shell. In this case, three shells were chosen to represent the starter, intermediate and ultimate pit shells. These shells could then be forecast using the Milawa algorithm, which allows for simultaneous mining of multiple phases in an optimized manner. In summary, the ultimate pit was selected based on maximizing the deposits NPV. It is expected that the Specified Case will be achieved with proper scheduling, and therefore significantly greater gross value than the Worst Case indicates.



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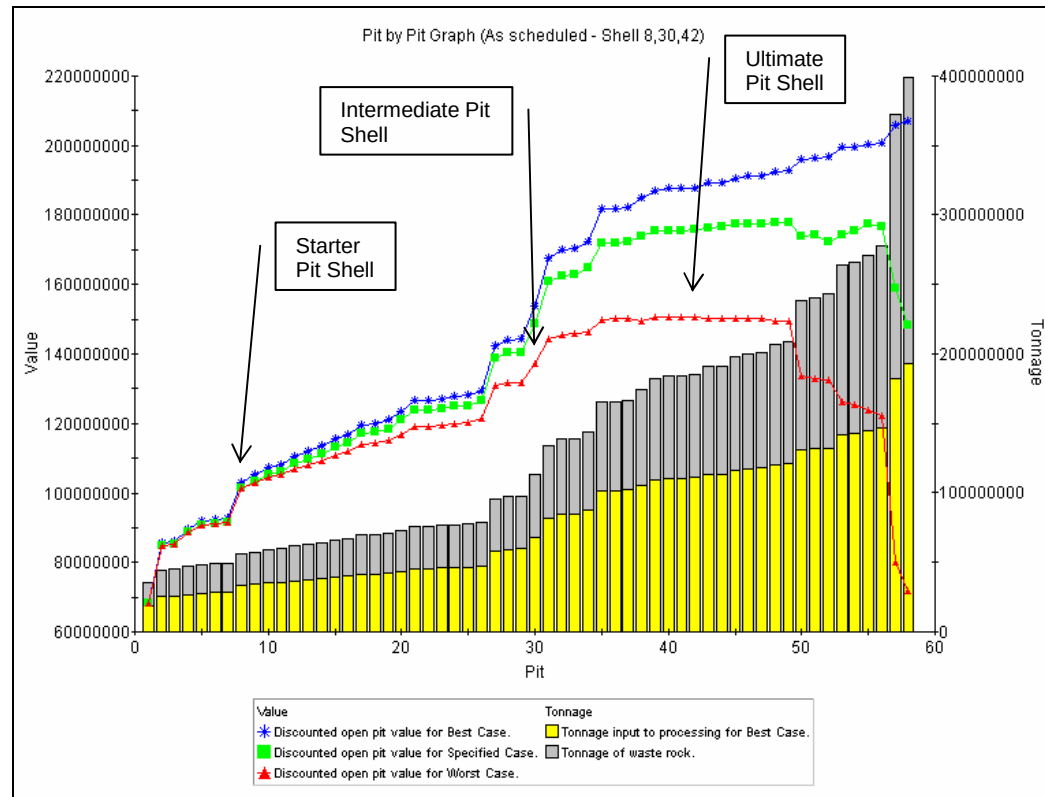
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Table 1-4: Pit Optimization and NSR Parameters

Parameter	Unit	Base Case
<i>Block Size</i>		
X	(m)	15
Y	(m)	15
Z – oxide	(m)	12
<i>Bulk Density</i>		
Ore & Waste	(t/m ³)	Variable from model
<i>Metal Price</i>		
Copper	(\$/lb)	1.00
<i>Primary Process Recovery</i>		
	(%)	Cu 91.4
<i>By-product Credits</i>		
		N/A
<i>Concentrate Grade</i>		
Copper	(%)	29.8
<i>Smelter Terms</i>		
		Cu
Treatment Charge	(\$/dmt)	75
Amount Payable	(%)	96.5
Refining	(\$/lb)	0.075
<i>Transportation (all products)</i>		
Road freight & storage	(\$/wmt)	81.62
Ocean Freight	(\$/wmt)	Incl. above
Moisture	(%)	8.00
Losses	(%)	0.25
<i>Mining Recovery</i>		
	(%)	100
<i>Preliminary Costs</i>		
Process and G&A	(\$/t milled)	3.90
Ore mining @ 25,000tpd	(\$/t mined)	0.89
Waste mining	(\$/t mined)	0.89
<i>Pit Slope Angles</i>		
	(°)	35 to 42
<i>Maximum Slope Height</i>		
	m	500



Figure 1-6: Whittle Nested Shell Output



1.4.3 Pit Design

The parameters used in the detailed pit design, including the geotechnical data described above, are as follows:

- bench height, single benching 12 m
- berm width 8 m
- haul roads and pit ramps
 - total width allowance 22 m
 - running surface 18 m
 - berms and ditches 4 m
 - berm access from ramp bottom only
 - maximum grade 10%

The three pit phases share a final wall on the southwest of the pit. Stripping will start on the south highwall and progress towards the north. Waste from the pit will be



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hauled to the waste dump or used for haul road construction and maintenance. Waste dump construction will isolate saprolitic (leached) waste so that it is contained within a sulphide waste berm.

The primary road access will be from the pit entrance through to the crushing facilities. All haul roads, both within and outside the pit, are designed to a minimum width of 22 m, except the lowest segments of the pit, which are 18 m (single-lane access).

1.4.4 Mining Estimate

The resources within the Mirador pits are summarized in Table 1-5, only the Indicated resources are considered mill feed; the Inferred resource is treated as waste.

Table 1-5: Resources by Material Type

	Tonnes (000)	Cutoff NSR (US\$/t)	Cutoff Cu (%)	NSR (US\$/t)	Cu (%)	Au (g/t)
<i>Indicated</i>						
Phase One	38,294	6.67	0.44	9.74	0.751	0.241
Phase Two	29,807	5.84	0.41	8.23	0.635	0.211
Phase Three	42,526	5.46	0.35	8.00	0.617	0.198
Total	110,627	-	-	8.66	0.668	0.216
<i>Inferred</i>						
Phase One	2,407	6.67	0.44	8.86	0.683	0.228
Phase Two	2,330	5.84	0.41	6.91	0.533	0.191
Phase Three	5,057	5.46	0.35	5.94	0.458	0.166
Total	9,794	-	-	6.89	0.531	0.187

Note: Copper cutoff shown as a NSR equivalent; actual NSR cutoff was completed in Whittle.

Waste material was categorized into the three types listed in Table 1-6. Material that was within the leached area was considered saprolite. Inferred waste was material above copper cutoff grade, but within the inferred resource category. All other material is been tabulated as sulphide waste.

Table 1-6: Waste Quantities and Type

Waste Type	Phase One (t x 000)	Phase Two (t x 000)	Phase Three (t x 000)	Total (t x 000)
Sulphide Waste	12,300	9,549	23,726	45,575
Inferred Waste	2,407	2,330	5,057	9,794
Saprolitic (Leached) Waste	21,129	6,143	8,531	35,803
Total	35,836	18,022	37,314	91,172



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1.4.5 Mining Equipment

A budget quotation for contract mining was received from Mamut Andino C.A., a large Ecuadorian civil construction company. The contract mining was quotation was based on AMEC's mine schedule, pit development plans, and layout drawings for the pit crusher and waste dump.

Loading and hauling estimated were carried out using Caterpillar software Fleet Production and Cost. Major input values are shown in Table 1-7. Blasting was estimated at 50% ANFO and 50% emulsion.

Table 1-7: Estimate Input Values

Schedule	
Calendar days	365 d/a
Shifts per day	2
Shift utilization	83%
FEL filling factor	90%
FEL & Truck availability	92%
FEL Passes / truck	5
Net Load / Truck	138 t

The estimated fleet requirements are listed in Table 1-8:

Table 1-8: Mine Equipment Fleet Requirements

Years	Quantity
203 mm blasthole drill	1
19.9 m ³ front-end loader	2
140 t haulage truck	8
"D10" track dozer	2
"D9" track dozer	1
Grader	2
"966" loader	2
"330" excavator	2
20 t dump trucks	2
Emulsion mix truck	1
Water truck	1
Lube truck	1
Drum vibrator	1



1.5 Metallurgy and Process Design

1.5.1 Metallurgical Testwork

A significant amount of metallurgical testwork has been undertaken on mineralized samples from the Mirador deposit since 2001. SGS Lakefield Research (Lakefield), in Lakefield, Ontario, Canada, carried out the main program of feasibility testing between December 2003 and September 2004. The groups responsible historically for the metallurgical testing aspects of the project are summarized below:

Geomet S.A., Santiago, Chile (May 2001)

- scoping batch rougher tests.

Resource Development Inc. (RDI), Co, USA (May 2002)

- scoping batch rougher tests.

Sumitomo Metal Mining Co, Limited, Japan (July 2003)

- scoping batch rougher and cleaner tests
- concentrate chemical and mineralogical analysis
- composite bond ball work index (BWI).

SGS Lakefield Research Limited, Lakefield, Ontario, Canada (Dec 2003 – Sept 2004)

- feasibility bench-scale flotation test program (batch and locked cycle flowsheet development and locked cycle variability testing)
- comminution testing (BWI, RWI, CWI, Ai and JK and SMC drop-weight testing) and modelling (JK SimMet)
- QEM*SEM characterization of composite samples for variability testing
- concentrate characterization and dewatering.

G&T Metallurgical Services Limited, Kamloops, BC, Canada (April – Sept 2004)

- supporting bench-scale flotation testwork
- mineralogical modal analysis.



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MinnovEX Technologies Inc, Toronto, Ontario, Canada (July 2004)

- comminution testing (SPI) and modelling (CEET).

In 2001 Geomet S.A. conducted a scoping rougher flotation test, on behalf of Billiton Chile, on an unidentified sample (Muestra 2) from Mirador.

In April 2002, Resource Development Inc. (RDI) conducted three batch rougher flotation tests also on unidentified samples.

In June 2003, Sumitomo Metal Mining Co., Ltd. (Sumitomo) conducted an independent scoping level metallurgical program on five selected drill core samples. This testing included batch rougher and cleaner flotation, mineralogical and chemical analysis of concentrates, and a bond work index determination. The liberation characteristics of the ore were also investigated. AMEC reviewed this work and found it to be done to industry standard.

Overall, the results of the Geomet, RDI and Sumitomo testwork showed the samples tested had relatively simple metallurgy and favourable commercial concentration potential. The subsequent follow-up feasibility work by Lakefield on 3,000 kg of split diamond drill core from 18 drill holes across the ore body, and at various depths, has confirmed their conclusions.

The feasibility metallurgical testwork carried out by Lakefield was done under the direction of AMEC. Lakefield also provided samples to MinnovEX and G&T Metallurgical Services (G&T) to conduct SPI grindability testwork, and mineralogical and flotation quality control testwork respectively. Lakefield and MinnovEX also conducted grinding circuit evaluations using their JKSimMet and Comminution Economic Evaluation Tool (CEET) simulation models respectively.

Lakefield's testing was conducted in two phases:

- Flowsheet Development December 2003 – March 2004
- Mapping and Recovery Variability April – September 2004.

Four master composites were produced from the core samples for an initial flowsheet and design criteria development program. This indicated the mill flow sheet for Mirador will be a conventional copper-gold porphyry flowsheet, with relatively coarse primary SAG and ball mill grinding to about 150 µm followed by copper rougher flotation, concentrate regrind to 30 µm, and cleaner flotation and dewatering. Metallurgical testing and mineralogical quantitative modal liberation analysis,



conducted by G&T, supported the selection of the primary grind and regrind parameters.

A recovery and mineralogical variability mapping program completed during the third quarter of 2004 subsequently confirmed that the metallurgy and mineralogy of the ore body is quite simple and homogenous, and the samples tested responded consistently well to the conventional flowsheet and reagent scheme selected. Over 44 variability sub-composite samples were produced from 17 drillholes and tested by hole and depth. Each sample was subjected to chemical and QemSCAN (Quantitative Evaluation of Mineralogy by Scanning Electron Microscopy) mineralogical analysis, grindability testing, and locked cycle flotation. Locked cycle concentrates were subjected to mineralogical, chemical, pyroforicity, and dewatering testing.

Chemical analysis of the head samples indicated a range of copper grades from 0.20% to 1.07%, with average overall grade of 0.67%. Gold grades ranged from 0.05 g/t to 0.43 g/t with an average value of 0.22 g/t.

Concentrates produced are predicted to average 29.8% Cu at a recovery of 91%. The average gold grade and recovery was 5.2 g/t and 47.2%, respectively. A gold behaviour model developed from the flotation test data suggests gold tracks chalcopyrite, pyrite, and gangue, with near equal weighting throughout the process. There is good reconciliation between the test gold recovery data and that predicted by quantitative mineralogy.

A laboratory analysis of the individual locked cycle concentrate products indicated that no significant deleterious penalty element impurities were present and this is in good agreement with mineralogical mapping. Concentrate thickening and filtration testwork was conducted. The concentrates settled rapidly and no dewatering problems were identified. Pyroforicity results indicated the concentrate is not expected to be self-heating.

Grindability tests were conducted on the sub-composite intervals of core from individual drill holes. Two dedicated whole core geotechnical and comminution holes were also drilled and used for additional grinding testwork, including Bond Work and Abrasion Indices, JK drop-weight and MinnovEX SPI testing.

Most of the ore in the pit falls geologically in an alteration zone of intense gypsum depletion. This is indicated by low RQD data and poor rock quality observed in drill core boxes. Comparative Bond low energy impact (CWI) and drop-weight test data also indicates the +150 mm ore lumps will break relatively easily at low-energy, but that the resulting reduction may be small. On this basis it is reasonable to assume the



SAG mill feed granulometry will be relatively finer than the copper porphyry industry average.

JK and SPI testing data ranked the samples in the medium range of resistance to impact breakage for SAG milling. The ore exhibits low to moderate abrasivity. The average Bond ball mill work index is about 14.5 kWh/t and ranks the ore in hardness to ball milling as moderately soft relative to other copper porphyry ores in Lakefield's industry database and with relatively low variability. The JKTech drop weight and SPI test SAG mill parameters, and ball mill work indices, were used in JK SimMet and CEET simulation software models to confirm the grinding circuit design basis, and there was good agreement between both approaches.

1.5.2 Flowsheet

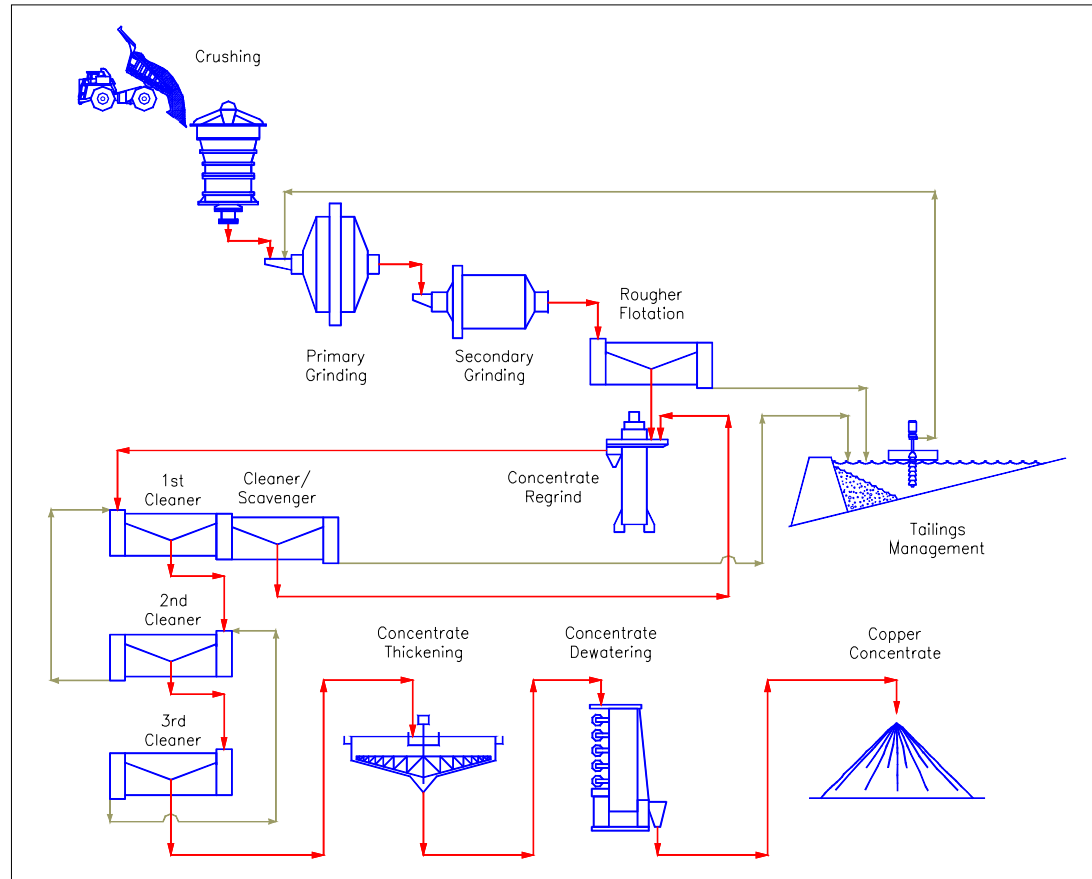
The estimated mineral resources included in the mine plan total approximately 111 Mt grading 0.67% Cu and 0.22 g/t Au. Silver and molybdenum are present but the grades are relatively low. Approximately 91 Mt of overlying waste rock will be removed over the mine life, resulting in an average strip ratio of 0.8:1. The process will be designed to treat 25,000 t/d of material.

A simplified schematic drawing of the proposed flowsheet is provided in Figure 1-7. Run-of-mine open pit ore will be crushed in a gyratory crusher. The crushed ore will be processed by means of semi-autogenous and ball mill grinding followed by rougher flotation, regrind, cleaner flotation, and dewatering to produce copper concentrate. The concentrate will be trucked via the existing road network in the area to a port facility in Machala for shipment to smelters. Tailings from the process will be impounded in a tailings pond; water will be reclaimed from the tailings pond and reused in the process.



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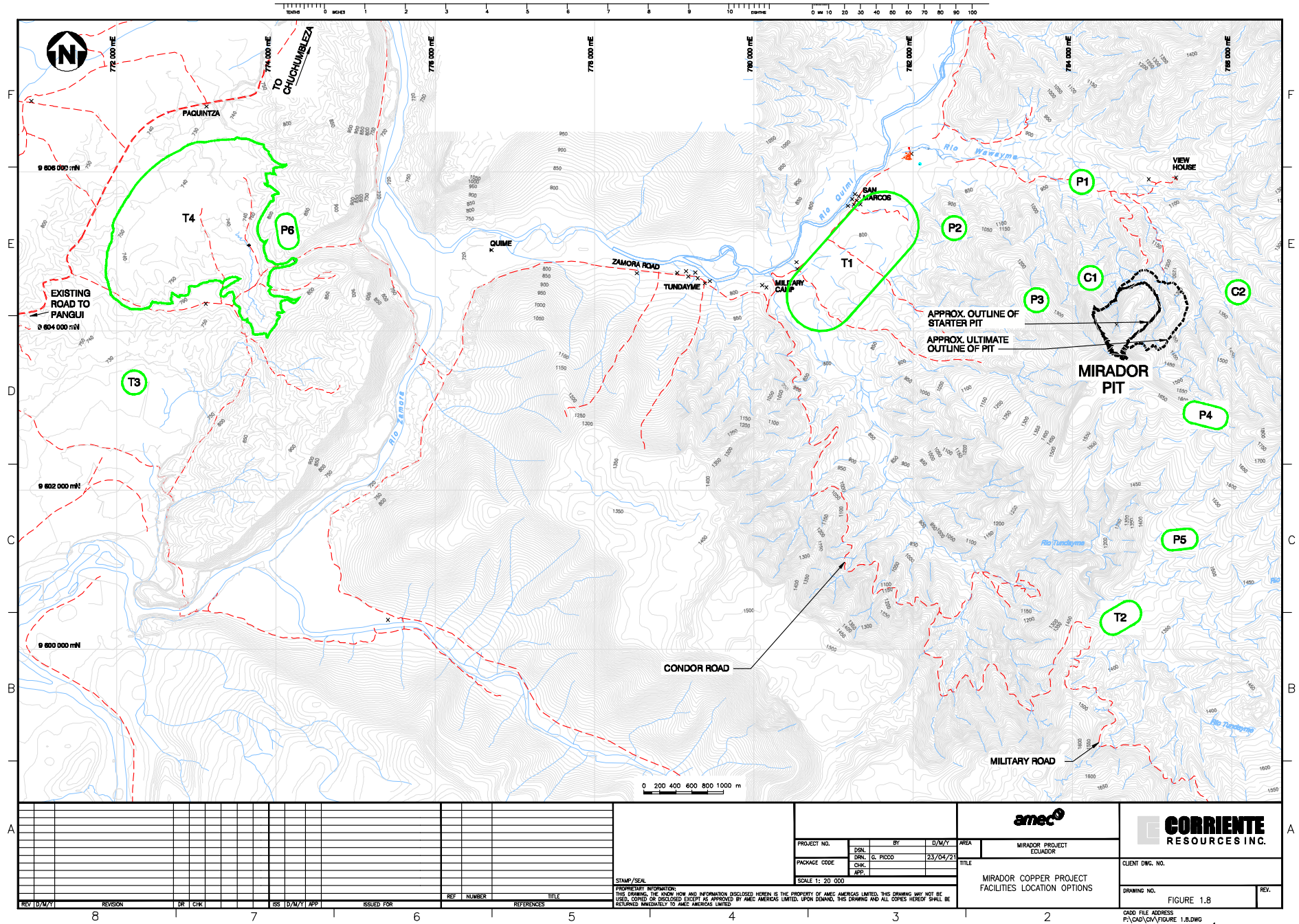
Figure 1-7: Simplified Mirador Flowsheet



1.6 Mine Site Infrastructure

The Mirador project site is located in rugged terrain, characterized by low mountains with steep slopes, and narrow ravines associated with drainage systems. The area is heavily forested.

Six locations were examined as possible sites for process facilities, two sites were examined as possibilities for the crusher, and seven locations were examined as alternatives for the tailings facilities, see Figure 1-8. The process, administration, concentrate filtration and loadout facilities area located approximately 11 km west of the deposit in the Pangui area, close to the towns of Santa Cruz and Paquintza and adjacent to the proposed tailings impoundment facility. These locations were examined by AMEC, Corriente, KP, and Merit personnel during field Investigations.





Tailings site T3 was preferred because it was located away from populated areas and would be a zero discharge impoundment facility. With T3 being the tailings impoundment location, plant locations P1 to P5 were rejected, as it would require pumping of tails several kilometres. Therefore, location P6 was examined. This location provided suitable ground conditions for foundation design, easy access to the existing highway, gravity flow to the tailings impoundment and a short distance for reclaim water pumping. The process plant will be connected to the existing highway and road system via a new 2.7 km access road. An overland conveyor will connect the crusher and the process plant. Upgraded local roads and a new 4 km road section will provide access to the mine and crusher area. The overall site layout is shown in Figure 1-9.

The main entrance to the site will be at the administration area. A restricted road, designed for highway vehicles, will connect the administration area to the crusher area. A four-wheel drive road running parallel to the overland conveyor will provide maintenance access to the conveyor system.

The administration offices, shops, warehouses, process plant, concentrate filtration and loadout facility and the water treatment plant will be located at the administration area. At the crusher area will be the mine maintenance facilities and mine offices as well as the primary crusher. Work crews will report to the administration area before being bussed to the crusher area.

Freshwater, firewater and sanitary facilities will be provided at all areas.

Power will be brought to the site via a new 138 kV power line. The main substation will be at the concentrator area, as this is the location of the largest power demand. Power to the crusher area will be fed from the main substation via a 13.8 kV power line.

1.7 Tailings Impoundment

The tailings impoundment has been designed to accommodate the estimated 106 Mt of solids produced over the 12-year life of the project. The principal design consideration is ensuring that the proposed facility meets the “zero-discharge” criteria, meaning that no water will be released from the facility throughout the life of the project.



A starter embankment will be built to an elevation of 748 m prior to the beginning of operations; this will provide about two years of storage at the proposed 25,000 t/d production rate. This elevation has been selected to provide ample time for commissioning of a cyclone system for coarse sand production and the procedures for placing and compacting the sand fraction. The main embankment will be raised to its ultimate elevation of 776 m using cyclone sand raises. It is estimated that the embankment will rise 3 m/a during year 3 to 6 and after that 2 m/a until closure. In addition to the starter embankment, a small dam will be built at the southeast corner of the facility to an elevation of 760 m to provide a pond in which the cleaner tailings may be stored under a constant water cover. This small dam will eventually be submerged as the pond surface rises to its ultimate elevation.

The close proximity of the tailings facility to the Mill reduces the length of the tailings distribution pipework, resulting in decreased capital expenditures as well as reduced power requirements. Rougher tailings will flow by gravity north, along the eastern bank of the impoundment, where the coarse fraction will be used as embankment material with the cyclone overflow material being deposited behind the embankment. Cleaner tailings (high sulphide cleaner tailings) will flow south through a separate pipe and be deposited sub-aqueously, thus preventing oxidation.

The embankment will be raised using the centreline construction method, with a starter earth embankment and ongoing raises using compacted cycloned tailings.

Embankment foundation and toe drains will be installed and will consist of a drain pipe, drain material, sand and filter fabric. The sand will have a filter relationship with the tailings that will prevent tailings from entering the drain. The seepage collected in the drains will be delivered to the seepage collection sump and pumped back into the impoundment area.

At start-up, a decant channel will be excavated at the east end of the impoundment, in close proximity to the plant site. The reclaim barge will be placed in this channel. Water will be continuously reclaimed from this supernatant pond, brought back to the plant site, and re-introduced into the process.

1.8 Off Site Infrastructure

1.8.1 Roads

The Mirador site is accessed from the Amazon Highway in southeastern Ecuador, about 25 km north of the town of El Pangui. The Amazon Highway is a good, all weather, two-lane road, paved in some sections and gravel in others. From the village



of Paquintza, it is about 3 km to the administration and process plant area and an additional 20 km to the mine site along a single lane gravel road, which will be upgraded as part of the project development. There is one river crossing, across the Zamora River, which is accomplished via a small barge. The overland conveyor will cross the Zamora on a dedicated bridge.

1.8.2 Concentrate Port

Copper concentrate will be trucked from the mine to Puerto Bolivar in Machala. The route is about 418 km and passes through the towns of Zamora, Loja and Santa Rosa, see Figure 1-10.

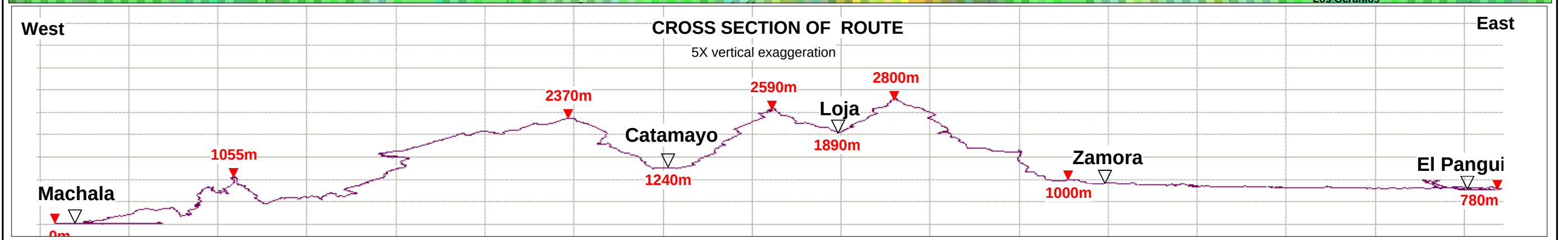
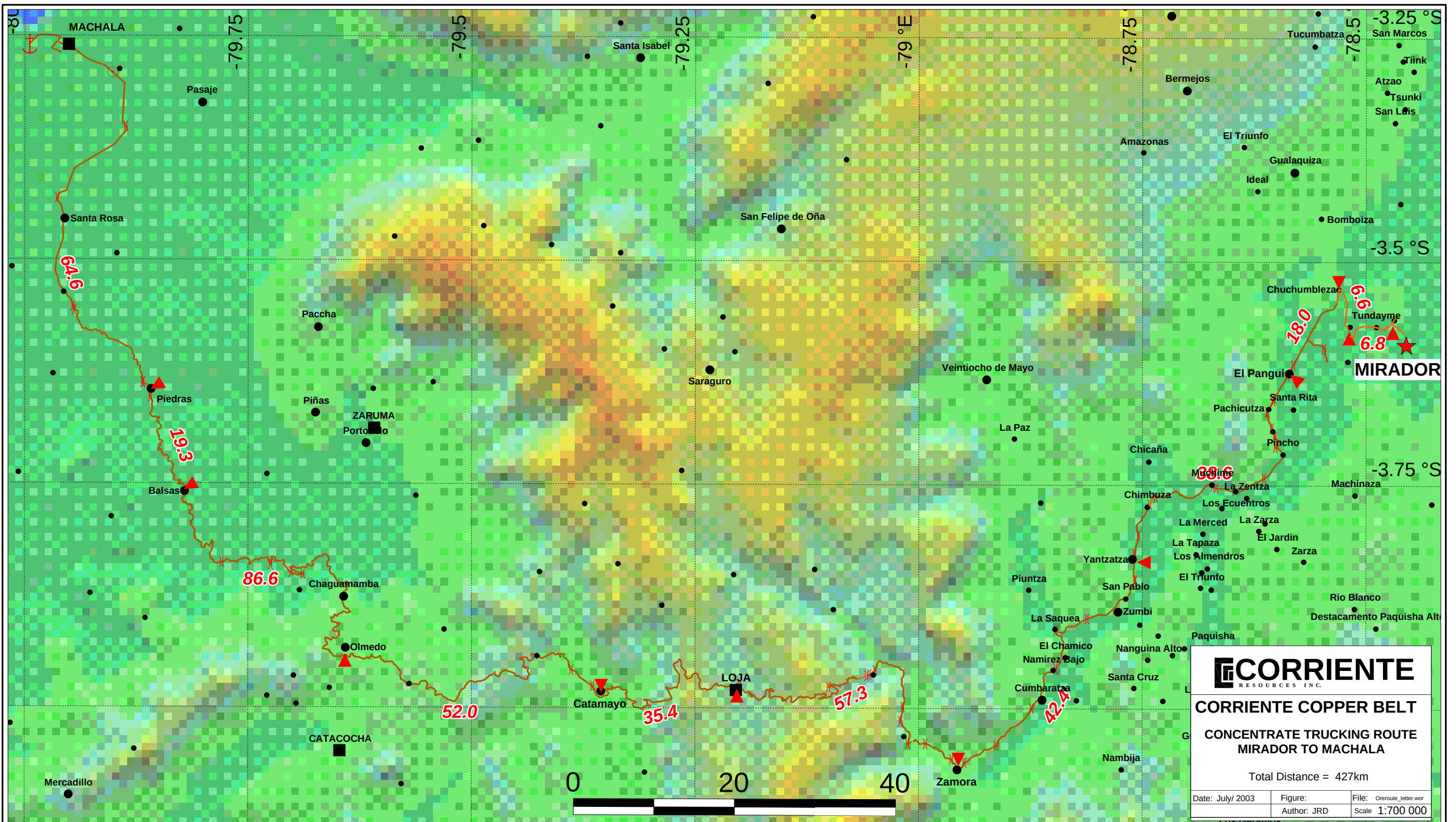
It is estimated there will be 18 to 20 trucks making the trip each day. The route to the coast is along good all-weather two-lane roads, the first hundred kilometres being gravel and the final distance being paved. Two of the bridges on route have been committed to be replaced by the government, the rest of the bridges and culverts along the route are in good condition and are part of the main east-west arterial route for heavy traffic crossing southern Ecuador. Bridges are generally rated for 40 t to 50 t capacity, of concrete construction, about 6 m wide on the gravel section and double laned on the asphalt section. It is not anticipated that any improvements to the existing roads, culverts, or bridges need to be performed before the start of operations.

At this time, it will take a truck about 15 hours to make the trip to the coast through Zamora, Loja, and Santa Rosa.

Machala, the closest port city to the Mirador Project, is located in southwest Ecuador at similar latitude to Mirador. The terminal, Puerto Bolivar, is located some 7 km south of Machala. Machala's industry is based on exportation of bananas and shrimps. The bulk of Ecuador's banana exportations go through Puerto Bolivar.

Puerto Bolivar is run by the Autoridad Portuaria de Puerto Bolivar (APPB), which manages the port concession on behalf of the government. APPB has six docks of which two are not utilized and are available to Corriente to use as a concentrate load out facility.

The concentrate load out facility will consist of a 15,000 t covered storage shed, load out conveyor and ship loading system. The facility will be on APPB property with security and access controlled by APPB.





Based on existing trade patterns for the west coast of South America, vessels are expected to range in size from small Handysize bulk carriers in the 12,000 DWT to 20,000 DWT capacity class, to a maximum Handymax vessel of 47,000 DWT. A detailed analysis of the marine market identified that empty 33,000 DWT bulk carriers currently travel from Guayaquil to Peru. These 33,000 DWT vessels are Corriente's best opportunity to optimize freight rates and therefore the ship loading system is designed for 33,000 DWT vessels.

Corriente's concentrate marketing study showed that copper concentrates will likely be sold in 10,000 wmt lots. An analysis by AMEC of the draft requirements for various size ships, confirmed that a 33,000 DWT vessel could load 10,000 t of product and not be affected by tide or draft restrictions. Depending on timing, some delays from three to six hours might be encountered waiting for tides to access the channel.

1.8.3 Power

At present, the Ecuadorian power grid is not able to supply sufficient power to meet the 28.3 MW needs of the Mirador project. As such, Corriente has purchased all the rights to the Sabanilla Hydroelectric Project in Ecuador, including all permits, studies and designs completed to date. The Sabanilla Hydroelectric Project is located in southern Ecuador, in the Province of Zamora Chinchipe, 40 km east of the City of Loja. The project will be able to supply an average of approximately 23.5 MW of power to the mine, with the remainder coming from the Ecuadorian power grid. A new line will be built, as part of the Sabanilla Hydroelectric Project, from the Sabanilla substation to bring the power to the site.

1.9 Environmental and Permitting

The Mirador project is located within the Cordillera del Condor. This area is considered ecologically important because of its high biological diversity and presence of endemic species.

Ecuador's environmental legislation is extensive and their requirements for early stage operations i.e., exploration, are well defined. Ecuador is one of the few Latin American countries that have adopted an EIA process for exploration activities. Argentina, Chile, and Peru have adopted a similar process to conduct environmental assessments for early stage exploration.

Under Ecuadorian Mining Law, the Ministry of Energy and Mines handles the environmental approval system for new mining projects. Mining concession holders are required to complete environmental impact studies and environmental



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management plans to prevent, mitigate, rehabilitate, and compensate for environmental and social impacts as a result of their activities. These studies are approved by the Ministry of Energy and Mines Sub secretary of the Environment.

Terrambiente, a Quito-based environmental firm, has completed an environmental baseline assessment for the Mirador Project. Baseline data collection commenced in March 2004 and has been ongoing through the study period.

The environmental approval process is summarized as follows:

- Proponent files a Project Description and Terms of Reference (ToRs) regarding how the EIA will be developed with respect to the conditions within the project area as well as the project description with the Ministry of Energy and Mines (MEM) and Ministry of the Environment (MoE). The ToR for Mirador has been reviewed by MEM and MoE.
- These ToRs need to be approved by MEM and published.
- Environmental baseline studies and environmental impact assessment are completed by proponent in accordance with the ToR.
- The EIA is presented to the local affected communities and input to the EIA is requested. Corriente will have community meetings in Valle del Quimi, San Marcos, Tundayme, El Pangui, and at the Ministries of Energy and of the Environment. The EIA is updated to acknowledge community input.
- The EIA is submitted to MEM who reviews within a 45-day period after which the ministry will request Corriente to respond to any comments and questions regarding the EIA.
- Corriente will have a 30-day period to submit responses to all comments and questions.
- The MEM will then take another 30-day period to revise the documentation and pronounce its satisfaction with all information, obtaining in this way the Approval for the EIA.
- Once the EIA is approved, proceedings towards granting of the Environmental License starts. It is estimated that another 30-day period is needed to prepare and grant the Environmental License.
- Submission of EIA to the Ministry of the Environment will take place at the same time as with the Ministry of Energy and Mines. Approval times are expected to be less than MEM.



1.10 Socio-economic

The Project area is located within the province of Zamora Chinchipe near the southern border of the Province of Morona Santiago. There are three parishes in the area, the El Guisme and Tundayme parishes in El Pangui County (Zamora Province) and the Bomboiza parish in Gualaquiza County (Morona Santiago Province).

The population in the Project area is composed primarily of immigrants from other parts of Ecuador. The density of the population in the Project area is low and poverty indexes are high. There is a scarcity of infrastructure and public services.

In general, there is a lack of employment and personal income is low. Productivity from farming is marginal, and primarily for local consumption. Approximately 40% of the households earn less than \$100 per month, while the remainder of the population earns between \$100 and \$300 per month. Responses to interviews with community members clearly indicate, from both high percentages of the responses and constant repetition, that the lack of employment is the most pressing problem in the communities.

Construction, while providing opportunities for local unskilled labour, is temporary. The area may experience an influx of persons from other parts of Ecuador who are looking for employment opportunities. This may create tension between the local area residents and the newcomers. Expectations in the local communities are high. These include improvements to the local road network to the provision of sanitary and potable water treatment systems.

1.11 Closure

The closure and reclamation plan is considered preliminary since some elements of the plan, such as post-closure monitoring requirements and associated cost estimates, may require revision upon project approval. Further, the specific details of the Mine Closure and Reclamation Plan will evolve as mining progresses, and so the plan will be updated periodically during the mine life. The final plan will be generated several years before mine closure.

The closure plan will include both progressive and final reclamation measures. Progressive reclamation will include reclamation following the construction phase and during operation. All borrow pits, quarries, equipment, and storage areas utilized during construction, but not required during mine operations, will be closed out and reclaimed at the end of the construction phase of the project. Prior to disturbing areas, all practically available soil will be salvaged and stockpiled for redistribution during ongoing reclamation and final closure of the mine.



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As far as practical, disturbed areas will be revegetated incrementally during operation to reduce erosion and improve aesthetics. Areas will be regraded, revegetated, and stabilized, as soon as possible.

The open pit mine will be allowed to fill with water when mining operations cease. There will be a height of exposed pit wall remaining above the final flooded surface. Some of the exposed wall materials could have the potential for acid generation. A pit lake study will be completed at Year 5 of the operation, when the ultimate mine life and pit configuration are better known and after more extensive geochemical characterization of the pit wall rocks has been carried out.

Waste dumps will have a cap of impervious material applied as soon as final surfaces are created, at which time they will also be vegetated. The waste dump will be constructed to its final slope angles where possible. Soil salvage will be incremental and concurrent with each phase of waste dump construction, pit development, and waste rock dump development. Acid rock drainage (ARD) will be minimized by the placement of the impervious cap and by maintaining runoff diversions around the waste dumps. The successful implementation of this strategy will minimize the volume of water that will require ongoing collection and treatment. Collection and treatment of acidic drainage from the waste dumps will continue for as many years as required, until the levels of acidity and metals abate to such that they will be acceptable for release or that can be adequately treated by passive systems.

The tailings impoundment facility will be designed such that upon completion of the mining operations, the tailings surface will be made trafficable and the potential for wind and water erosion minimized. During the final years of operation, the extent of the supernatant pond will be minimized and will be removed during the first year following closure. Water will be removed from the surface pond, if required, treated to an acceptable quality, and released to surface waters. The downstream embankment face will be revegetated and the tailings covered with soils from nearby borrow sources.

Closure activities after the end of mine life will also include removal of all facilities and infrastructure that is not planned to be left for other uses or is needed for post-closure maintenance, and reclamation of all disturbed areas.

Post operational reclamation will return the disturbed areas to the pre-mine conditions and habitat. Certain mine features may be left in place as permanent control measures to prevent environmental pollution, for long-term community use, or as a post-mining enhancement.



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Environmental impacts remaining post-closure are expected to be moderate. Key expected impacts would be:

- Potential for acid drainage from exposed wall rocks above the outlet elevation of the open pit. It is expected that, with the significant streamflows in the region, the rate of oxidation of pit wall rocks will be such that water quality in the pit water discharge could be mitigated by dilution. In the worst case the pit lake discharge could necessitate collection and treatment. Even in this worst case, there would be no downstream impact to the receiving waters of the Quimi River as the discharge would be collected and treated if necessary
- The waste dumps are expected to become acid generating and there will be discharges of acidic waters, containing dissolved metals, from the dump. The dump discharges will be monitored and collected and treated as necessary.
- The waste dump will permanently alter the topography. The location of the dump in a side valley and the anticipated vegetation growth will effectively mask waste dump visibility.
- The plant site and all other mine infrastructure will be essentially returned to pre-mining conditions and revegetated. This there will be little long-term impact of the plant site and infrastructure.
- The tailings dam will remain visible after closure, as the downstream face of the dam will be about 40 m high.
- A main impact of the Mirador project will be the loss of forest habitat in the open pit and waste dump area (188 ha).
- Typically, mine closure will impact the economic condition of the closest community(ies). Corriente will work with the communities in the area to enable all stakeholders to have their interests considered during the mine closure process. A consultative process involving local communities, regulators and other interested stakeholders will be established during the operation to determine the long-term use for the mine and all associated lands.

Corriente has developed a water quality monitoring plan that is already in place on the Mirador site. This existing water quality monitoring plan will form the basis of an environmental management and environmental effects monitoring plan during the life of the operation. That plan will be continued as required post-closure.

Maintenance of critical components will also continue post-closure, including inspection and repair of diversion ditches, repairs of any eroded reclaimed areas and repairs to any erosion noted on the tailings pond and waste dump covers, and operation of the water treatment plant and/or passive water treatment systems.



Final closure plans for Mirador will be implemented as soon as the mine reaches the end of its economically viable life.

1.12 Project Implementation

A project group will be established by Corriente to undertake the execution of the Mirador Copper Project. It will be comprised of Corriente's team working with outside specialist consultants to undertake the engineering, procurement and construction management, as well as much of the pre-production work. The proposed organization chart is shown as Figure 1-11.

The overall project capital program will take about 21 months from the scheduled project approval date to engineer and construct the facilities to Mechanical Completion when pre-operations can start.

After acceptance of the feasibility study and the associated trade-offs, and the issuing of permits, Corriente will select the engineering and procurement teams to undertake the design of the material handling, process plant facilities, tailings facility, mining, and infrastructure work.

The Project will generally be engineered to Canadian standards and regulations and will incorporate preferred South American codes when applicable. Measurement will be made in metric units for world-wide procurement.

The engineering contractor's purchasing group will provide procurement of capital equipment, and inspection of all equipment if needed. The construction management contractor's group will organize purchases of bulk materials such as pipe, cable, and cable tray and will provide expediting services of equipment and materials. Orders will be placed with qualified onshore and offshore suppliers on the basis of quality, price and delivery.

Most equipment and materials will be procured from sources within North and South America. All efforts will be made to purchase locally where possible and practical to do so. Corriente will purchase as much of the capital equipment and materials as possible rather than leave it to the contractors. Engineering and construction management expeditors will maintain contact with all suppliers to maximize their performance. The use of third party inspection and expediting services will be used where appropriate to minimise the cost of expediting and inspection.

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Project Manager

Corriente Representative

Infrastructure Engineering

- Power
- Roads
- Overland Conveyor
- Port
- Tailings

Process Plant Engineering

Engineering Disciplines

- Mine Planning
- Metallurgical
- Concrete
- Structural
- Mechanical
- Piping
- Structural
- Electrical
- Instrumentation
- Material Handling

Capital Equipment Procurement

- Prepare and Issue RFQ
- RFQ quote analysis
- Purchase Orders

Geotechnical Consultant

Construction Management

- Trade-off Study Input
- Constructability Reviews
- Bid Document Preparation
- Project Schedule
- Project Cost System
- Construction Management
- Field Engineering
- Field Procurement
- Expediting

Mine Planning

Environmental

- Water
- Waste
- Reclamation
- Construction Monitoring

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Project Organization Chart

Figure 1-11



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Incoming freight during construction and operations will come through the port at Guayaquil to the north of Machala. The port at Guayaquil has the appropriate unloading facilities on the dock to accommodate the heavy loads needed to be transported up to the mine construction site.

Dedicated customs brokers will operate from an office at the port to ensure that the paperwork has been received for inbound freight and the necessary clearances are obtained without delays at the port.

The construction management group will be responsible for the management of all field operations. The Construction Manager will be responsible to Corriente to effectively plan, organise, and manage the construction to meet quality, safety, budget, and schedule objectives.

It is planned to have the contractors originating from Ecuador and investigations indicate that there is a great interest and ability of contractors. Their experience has been generally associated with the oil industry and manufacturing plants and ports, but their labour pool is conversant with the trades needed to build the Mirador facilities.

A top-down commitment to health, safety and the environment (HSE) will be made and communicated to everyone associated with the project. Corriente will develop and issue a project safety program and an environmental control program based on its own policies and standards, internationally recognised requirements, and those required by North American and Ecuadorian law.

Construction contracts will require contractors to submit written health, safety and environmental action plans that conform to the project HSE programs or to provide a written commitment to adhere to Corriente's HSE programs. Each contractor will be responsible for administration of its own HSE programs, and will be required to designate a management-level person who will be personally responsible to carry out the safety program and the environmental control program. Key elements of the program include new employee orientation and weekly HSE meetings.

Workers will be provided a safe place to work. Proper inspection, guidance, and strict adherence to safety policy will ensure that safe working conditions are maintained.

Construction activities will be carried out with an acute awareness of the environment and the need to comply with the policies and program of the project. Regular environmental audits will ensure compliance to the procedures.

The project schedule has an estimated overall project duration of 21 months from the start of the detailed engineering to the completion processing plant ready for the dry



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runs of ore. The Sabanilla Hydroelectric Project will need to initiate the main civil works contracting process some 5 months ahead of the start of the Mirador process plant design in order for it to be supplying power for mill run-in.

For the Mirador Project, the critical path runs through the project financing, approval to proceed, mine access road construction and crusher installation, as well as pre-development work in the open pit.

Engineering for the plant can proceed immediately upon the completion of project financing/approval to proceed. Procurement of major capital equipment will be done at this time also.

The key activities to commence plant operation are the procurement and delivery of major capital components, such as the crusher, mills, and electrical equipment. The milestones may be summarized as shown in Table 1-9.

Table 1-9: Key Milestones for Mirador

Description	Month No.
Project approval	0
Tailings facility engineering complete	5
Concentrator access road complete	6
Overland conveyor design complete	6
Mine access road complete	6
Earthworks at concentrator & filter areas complete	12
Earthworks for overland conveyor complete	12
Tailings facility complete	15
Mill building structural completion	18
Port facilities complete	18
Thickeners complete	19
Flotation mechanical / piping complete	19
Grinding mechanical / piping complete	20
Crusher facilities complete	20
Overland conveyor complete	20
138 kV power on-site	20
Mechanical completion	21

1.13 Capital Costs

Merit prepared a bankable feasibility study level capital cost estimate that is considered by Merit to be within a 10% accuracy of actual costs.



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The total estimated cost to design and build the facilities described in this report is US\$202,306,000, see Table 1-11. All costs are expressed in first quarter 2005 US dollars, with no allowance for escalation, interest during construction or taxes.

The estimate covers the direct field costs of executing this project, plus the indirect costs associated with design, procurement, and construction efforts.

Table 1-10: Summary of Capital Costs by Area, US\$

Area	Cost (\$ x 000)
<i>Direct Costs</i>	
Mine	20,108
Process Plant	87,922
Utilities	6,920
Site Preparation and Roads	13,561
Tailings	8,474
Ancillary Facilities	3,847
Total Direct Costs	140,832
<i>Indirect Costs</i>	
Project Indirects	39,362
Owner Indirects	4,376
Total Indirect Costs	43,738
Subtotal	184,570
Contingency	17,736
Total Project	202,306

1.14 Operating Costs

Mine operating costs are based on a budget quotation from an Ecuadorian transportation and mining contractor. Process operating costs and G&A costs have been estimated by AMEC, with input from Corriente and impoundment operating costs have been estimated by KP. Operating costs have been prepared in fourth quarter 2004 US dollars, and exclude:

- contingency
- allowance for escalation
- value-added (IGV) taxes
- import duties.

The operating cost estimates have been assembled by area and component, based on estimated staffing levels, consumables and expenditures according to the mine plan



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and process design. Concentrate shipping costs are detailed in the financial analysis. Life-of-mine operating costs are shown in Table 1-11, and annual operating costs in Table 1-12.

Table 1-11: Operating Cost Life-of-Mine, US\$

	Labour	Expenses	Total Cost	\$/t Mined	\$/t Milled
Mine Operations	11,966,000	198,565,000	210,531,000	1.14	1.90
Processing Operations	28,179,000	289,784,000	317,963,000	1.71	2.87
Administration	36,093,000	34,302,000	70,395,000	0.38	0.64
Total	76,238,000	522,651,000	598,889,000	3.23	5.41

Table 1-12: Annual Operating Cost, US\$

Year	Mining	Processing	Admin	Total	\$/t Mined	\$/t Milled	\$/lb Cu
1	20,598,000	26,723,000	5,806,000	53,127,000	2.49	5.82	0.41
2	17,890,000	26,202,000	5,806,000	49,898,000	3.00	5.47	0.36
3	16,961,000	26,178,000	5,806,000	48,945,000	3.01	5.36	0.40
4	16,181,000	26,183,000	5,806,000	48,170,000	3.19	5.28	0.38
5	16,473,000	26,183,000	5,806,000	48,462,000	3.22	5.31	0.38
6	16,811,000	26,185,000	5,806,000	48,802,000	3.25	5.35	0.39
7	17,019,000	26,178,000	5,806,000	49,003,000	3.26	5.37	0.41
8	17,729,000	26,167,000	5,806,000	49,702,000	3.31	5.45	0.44
9	17,661,000	26,179,000	5,806,000	49,646,000	3.31	5.44	0.41
10	18,252,000	26,176,000	5,806,000	50,234,000	3.34	5.51	0.42
11	16,930,000	26,160,000	5,806,000	48,896,000	3.78	5.36	0.43
12	15,921,000	26,167,000	5,806,000	47,894,000	4.07	5.25	0.41
13	2,105,000	3,282,000	723,000	6,110,000	4.53	5.43	0.37
Total	210,531,000	317,963,000	70,395,000	598,889,000	3.23	5.41	0.40

1.15 Economic Analysis

The Mirador project was analyzed using a discounted cash flow approach assuming 100% equity in first quarter 2005 US dollars. Projections for annual revenues and costs are based on data developed for the mine, process plant, ancillary facilities, infrastructure, capital expenditures, and operating costs. Estimated project cash flows were used to determine the pre-tax net present value (NPV) and internal rate of return (IRR) for the base case.

Results of the base case analysis indicate that the project has a potential pre-tax internal rate of return of 14.8% and a pre-tax net present value \$71.2 million at a



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discount rate of 8%. Effects of changes to commodity prices, capital cost, and operating cost were examined in a sensitivity analysis. This analysis indicated that the project is more sensitive to changes in copper price than either capital or operating costs and is slightly more sensitive to operating costs than capital costs. The base case scenario has a projected payback period of approximately 5 years.

The variation in NPV with discount rate is shown in Table 1-13.

Table 1-13: Variation in Pre-Tax NPV with Discount Rate, (US\$ M)

	0%	5%	8%	10%
Net Present Value	254.6	122.2	71.2	45.1

Base case sensitivity analysis is summarized in Table 1-14, Figure 1-12, and Figure 1-13. For the range of changes examined, from 80% to 120% of the base case, the project is most sensitive to copper price.

Results of the NPV (8%) and internal rate of return for metal prices, capital cost and operating cost are shown in Figures 1-12 and 1-13.

Table 1-14: Base Case Pre-Tax Sensitivity Analysis

Parameter	Variation (%)	Value	IRR (%)	NPV at 8% (\$ M)
<i>Average Copper Price</i> (\$/lb)	-20	\$0.80	-	-86.2
	-10	\$0.90	7.2	-7.5
	0	\$1.00	14.8	71.2
	10	\$1.10	21.3	149.9
	20	\$1.20	27.2	228.6
<i>Average Gold Price</i> (\$/oz)	-20	\$320	13.5	55.9
	-10	\$360	14.2	63.6
	0	\$400	14.8	71.2
	10	\$440	15.5	78.9
	20	\$480	16.2	86.5
<i>Operating Cost</i> (\$/t ore)	-20	\$4.33	20.2	135.4
	-10	\$4.87	17.6	103.3
	0	\$5.41	14.8	71.2
	10	\$5.95	11.9	39.1
	20	\$6.49	8.7	7.1
<i>Capital Cost</i> (\$ M)	-20	\$161.8	20.1	107.1
	-10	\$182.1	17.2	89.2
	0	\$202.3	14.8	71.2
	10	\$222.5	12.7	53.3
	20	\$242.8	10.9	35.3



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Figure 1-12: Base Case Pre-Tax NPV Sensitivity @ 8%

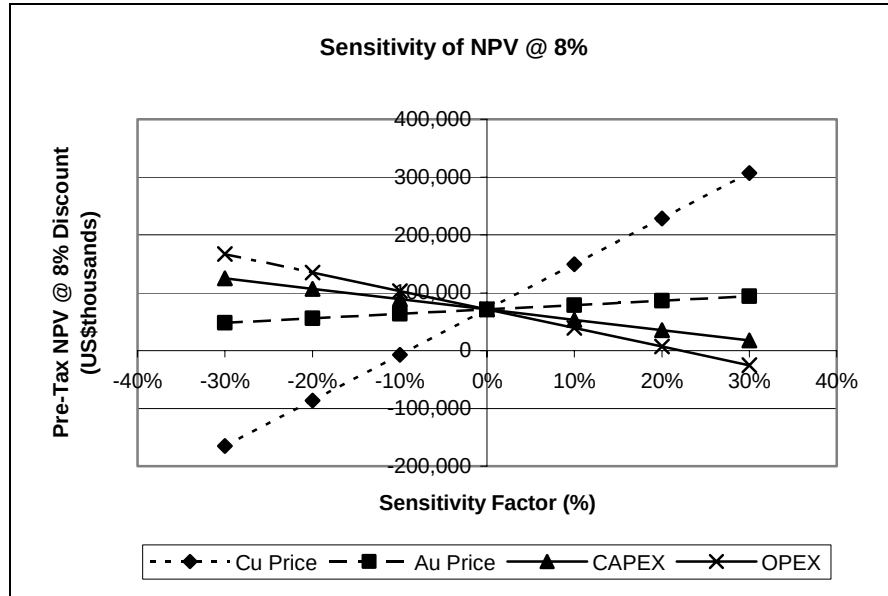
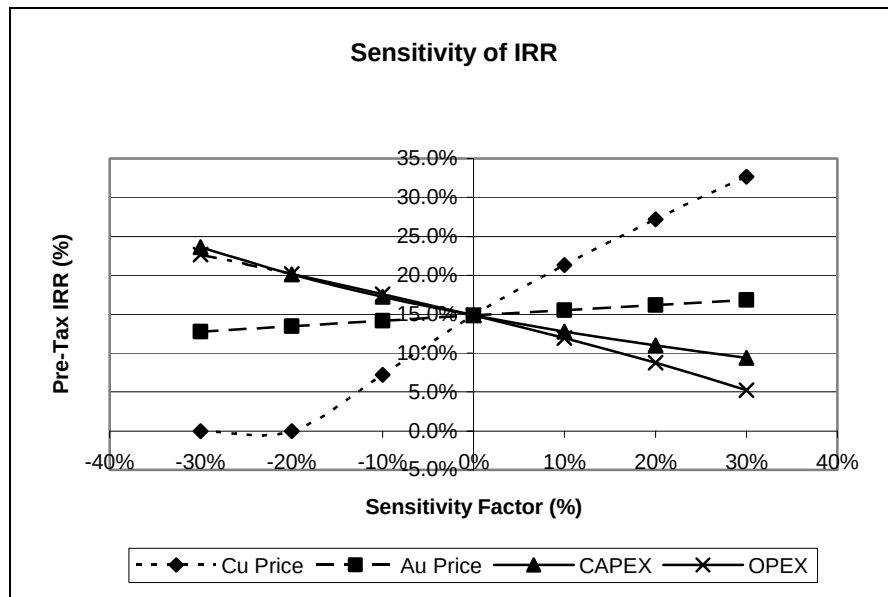


Figure 1-13: Pre-Tax IRR Sensitivity





1.16 Opportunities

Opportunities and recommendations for additional work to improve the project economics are discussed in the different sections of the report. This section summarizes these items.

1.16.1 General

- The plant throughput should be reviewed. Preliminary modeling showed the project IRR and NPV improved with increasing throughput. A mine Whittle shell based on a 75,000 t/d plant throughput using factored operating costs, showed a Measured and Indicated pit shell containing 223.4 Mt of ore at an average grade of 0.606% Cu and 0.194 g/t Au and a stripping ration of 0.65:1. The same model run showed a Measured, Indicated and Inferred shell containing 310.5 Mt at an average grade of 0.576% Cu and 0.185 g/t Au, with a strip ratio of 0.48. Additional studies are required to define the infrastructure requirements and costs to exploit this scenario.
- Silver should be included in the resource model. Silver is observed in the concentrate, but is not assayed in the drill core samples. Running the financial model with the silver grade observed in the concentrate, improves the pre-tax IRR by approximately 1%.

1.16.2 Resource Geology

- More angled drill holes should be drilled due to the presence of various geologic features having sub-vertical contacts (e.g., post-mineral dykes, late quartz-sulphide stringers). This would better define the locations and contacts of the vertical post-mineral dykes and help reduce dilution during the mining process.
- Downhole survey data should be captured for all drill holes in the future to accurately locate the drill holes and grade items during the resource estimation process.

1.16.3 Mine

- The final schedule includes 9,794,000 t of Inferred resources in the waste. This material represents potential mill feed if the classification can be successfully upgraded from Inferred to Indicated with further exploration drilling.
- A better estimate of overburden or saprolite should be established and a solid produced for use in further pit design work. The saprolite will require a flatter slope



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and wider berms, and will add waste tonnes to the current design. The new overburden solid would allow this flatter development to only occur in the poor saprolite.

- There is potential for increasing the open pit slope angles and improving open pit economics, based on additional investigation and analyses, as follows:
 - Additional geotechnical logging of any further exploration drillholes to increase the geotechnical database on the characteristics of the rock mass.
 - A detailed geologic/lithologic model incorporating such items as rock lithology, contact boundaries, faults, and dyking. This will allow for projection of lithologic information onto and behind proposed pit walls for stability assessment.
 - Additional geotechnical drilling to assess the hydrogeology of the pit area and to assess the rock mass strength at depth in the pit walls.
 - A hydrogeologic model to develop an understanding of the groundwater hydrology and the impact the pit will provide on the groundwater flow regime as well as what impact the drainage facilities (horizontal drains and/or drainage galleries) will have on optimizing the pit walls.
 - Additional rock strength testing on the various lithologic rock types within the pit to increase the confidence level in the rock strengths used for stability analysis.
 - Additional investigations into the characteristics and particularly persistence of major and intermediate scale structures.
 - A review of stability of the overall and bench face slope angles with the benefit of improved lithologic and hydrogeologic interpretations. The slope angles recommended are relatively low in comparison to many open pits, but are considered realistic for the rock quality indicated.

1.16.4 Process

- Pilot plant testing should be conducted to evaluate the flotation response of the different ore zones under continuous operation.
- Additional surface and shallow composites should be prepared so that further batch flotation testwork can be conducted to better define the relationship between rougher pH and final concentrate grade.
- Additional surface and shallow samples should be submitted for bulk modal analysis in order to clarify the effect of secondary copper on flotation response.
- The SAG mill size (motor) was fixed early in the feasibility study based on project power supply constraints that no longer apply. Comminution test data indicates a relatively low probability the ore will generate SAG resilient pebbles therefore, a SAB circuit was originally specified. However, a pebble crusher was added later to



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the SAG circuit, which was tonnage limiting, to increase throughput. Based on comminution modelling the revised SABC circuit throughput was increased from 22.5 kt/d to 25 kt/d. It is recommended additional modelling optimization is conducted to consider the economics of eliminating the pebble crusher, and using either a larger SAG mill, or ball mill with a coarser transfer size out of the SAG circuit. This could simplify the circuit and result in incremental capital cost savings.

- Grinding mill availability was originally fixed at 94% based on the use of synchronous drive with LCI speed control. Based on capital cost savings a WRIM drive with slip energy recovery was subsequently selected. It is recommended the mill availability of 94% is reviewed for consistency with this drive selection and a more detailed technical and economic analysis of each option is done during engineering.
- The cyclone configuration criteria selected reflects the lowest cost capital bid received. However, this is based on the use of multiple cyclones relative to other bids. The use of a greater number of smaller cyclones could result in higher ongoing maintenance costs and this should be reviewed during engineering to identify the best overall life-cycle cost cyclone configuration option.
- Secondary and final flotation cleaning is currently based on conventional cells, versus columns, based on Corriente's preference. AMEC believe columns will perform satisfactorily in this application and could provide layout and capital cost advantages. However, it is recommended this is left as an opportunity to be confirmed by future pilot testing.

1.16.5 Power Supply

- The choice of distribution voltage and mill motors should be reviewed. This entails basic design and detail pricing of the system at 13.8 kV as compared to 4.16 kV.

1.16.6 Tailings Management Facility

- A more detailed seismicity study be conducted for future TMF design work, to confirm the magnitude and associated maximum acceleration for the Maximum Design Earthquake. This would require more detailed probabilistic and/or deterministic seismic hazard analyses than have been completed to date.
- Hydrological and climate data gathering should be continued to provide ongoing site specific data. This will allow for further refinement of the water management measures during the detailed design.



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1.16.7 Water Management

- Site hydrologic data analysis to provide more site-specific design flows for sediment control systems, and diversions.
- Soil/waste samples for forested watersheds, the mine pit, and the waste rock dump to determine erodibility factors (K) and eroded particle size distributions for sediment control design.



2.0 PROJECT BACKGROUND

2.1 Terms of Reference

Corriente is evaluating the development of the Mirador copper project in southeastern Ecuador. Corriente commissioned AMEC to be the prime consultant for the feasibility study and to compile this feasibility study report, with input from Corriente's various consultants as listed below. The feasibility study addressed the mineral resource, mine plan, processing and support facilities, site access, transportation of concentrate, port requirements, concentrate marketing and project execution.

The following consultants were responsible for input to the feasibility study.

AMEC

- overall study compilation
- review of project geology
- review of sample QA/QC program
- estimate of mineral resources
- mine planning
- specification and management of metallurgical testwork program
- flowsheet development
- design of the process facilities and site support facilities
- design of mine roads and concentrator access road
- input to capital cost estimating:
 - equipment pricing for major process equipment
 - quantity estimation for major civil and structural components
- operating cost estimates for mining, process and administration
- pre-tax financial analysis
- geotechnical engineering to support pit design
- site water management
- mine waste management
- ARD analysis
- specification and management of environmental and socio-economic baseline studies, including impact analysis.



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Caminosca Caminos y Canales Cia. Ltda. (Caminosca)

- design of the crusher and mine site access road
- surveying.

Knight Piésold Ltd (KP)

- design of Tailings Management Facility (TMF)
- geotechnical investigations to support tailings impoundment design
- investigation for freshwater wells
- design of tailings distribution pipework for TMF
- design of reclaim barge and pipeline for TMF
- TMF capital and operating cost estimates
- TMF closure plan.

Merit Consultants International Inc. (Merit)

- study coordination
- development of project implementation plan and schedule
- assessment of off-site infrastructure
- capital cost estimating.

Terrambiente

- environmental base line monitoring
- environmental impact assessment report
- socio economic studies
- Environmental Management Plan.

Corriente

- geologic solids for dikes and supergene material
- geotechnical investigation of plant site area
- overland conveying system
- port operations and costing
- concentrate marketing



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- concentrate trucking and shipping
- power supply.

2.2 Property Description and Location

The Mirador Project is located along the valley of the Zamora River in the Zamora Chinchipe Province of southeast Ecuador, adjacent to the border with Peru (Figure 2-1, overleaf). The concessions are approximately 340 km south of Ecuador's capital City of Quito and 70 km east-southeast of the City of Cuenca.

The Mirador Concession Group centre is found at 9 604 200 N, 785 000 E; UTM Zone 17S coordinates and is located in an area of steep topography at 1,350 m elevation. The group of six concessions are contiguous and encompass 13,640 ha.

The location of the individual concessions that are collectively known as the Mirador Project are tabulated in Table 2-1, and shown in Figure 2-2.

Table 2-1: Mirador Concession Locations and Areas

Concessions	Area (ha)	Date of Application	Date of Grant or Concession	Region	Number
Caya 36	4,740	27-Nov-98	23-Aug-01	Zamora	500200
Curigem 18 East	800	16-Jan-03	07-Feb-03	Zamora	500806
Curigem 18	1,600	16-Jan-03	07-Feb-03	Zamora	4768
Curigem 19	2,900	16-Jan-03	07-Feb-03	Zamora	4769
Mirador1	2,400	16-Jan-03	07-Feb-03	Zamora	500807
Mirador2	1,200	16-Jan-03	07-Feb-03	Zamora	500805
Mirador 3	1,020	23-Feb-05	In progress	Zamora	500976
Total	14,660	-	-	-	-



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Figure 2-1: Location Map

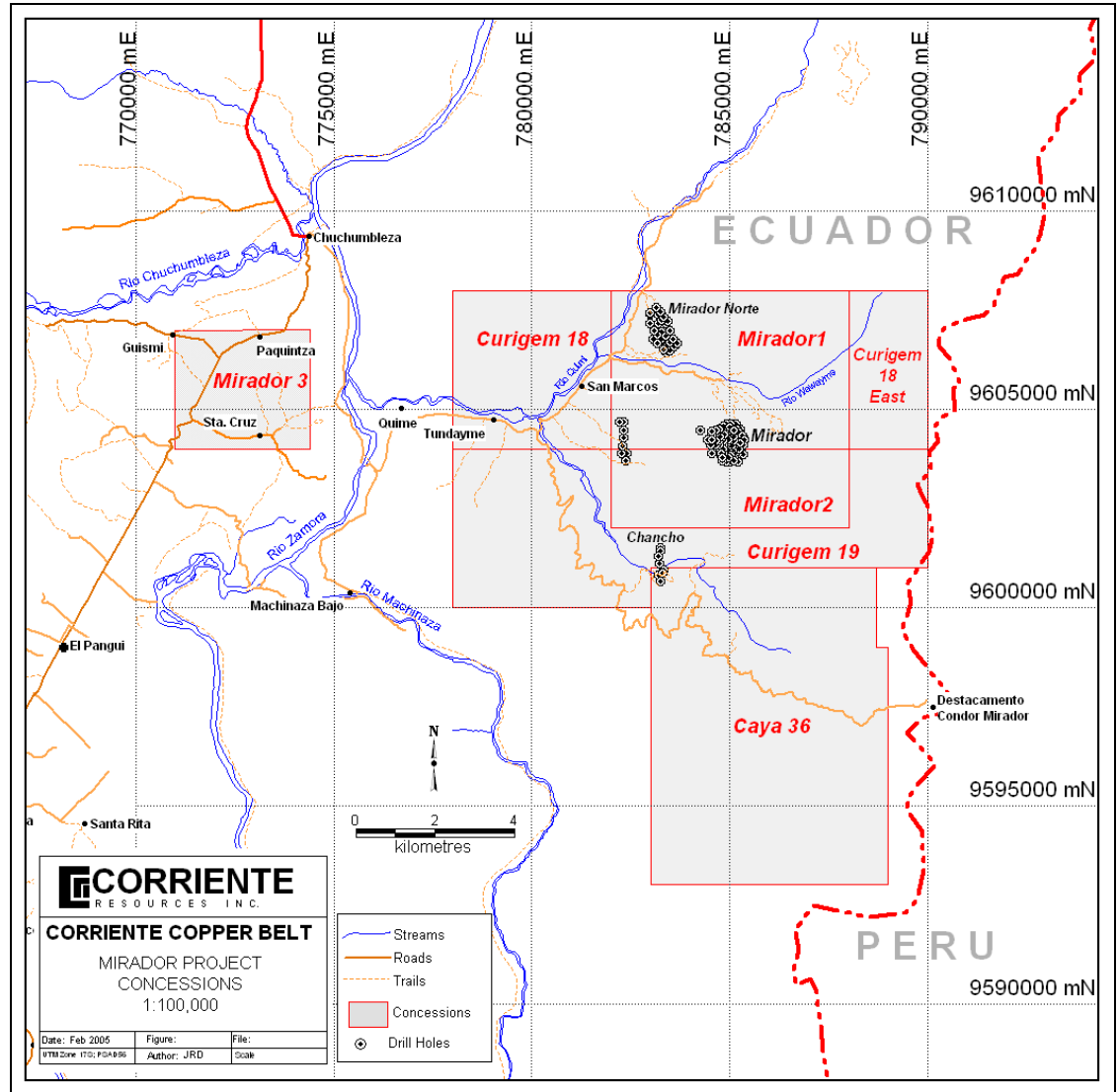




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Figure 2-2: Detailed Map of the Mirador Concessions





2.3 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

2.3.1 Accessibility

The nearest village to the Mirador deposit is Tundayme. Scheduled air service is available from Quito to Cuenca and Loja (one hour flight). Small chartered civilian and military aircraft currently fly to Gualaquiza, the closest airfield to the deposit.

The village of Tundayme is accessed by a system of paved and gravel roads from Cuenca or Loja. From Cuenca, the road distance is approximately 170 km or about five to six hours travel time.

There is highway access from Quito to Cuenca for the transport of samples or heavy equipment. There is also highway access to the Pacific Ocean port of Machala.

Road access to the mine site is under construction with completion scheduled for May 2005. Access to the tailings area is via a gravel road from the Pangui-Chucgunbkeza road. The mill site is access via a one kilometre trail from the end of the Santa Cruz village access road.

2.3.2 Climate

The area has a wet equatorial climate with a reported rainfall of 2,700 mm/a. Individual rainfall events can be over 60 mm in a 24-hour period. Due to variations in local topography there is a wide range in rainfall levels, and the area could be characterized as having numerous microclimates. Fieldwork is possible all year round. The optimum period for airborne surveys or road and trail construction is from October to December because of the drier weather conditions at this time.

2.3.3 Physiography

The valley of the Zamora River bisects the project area at an elevation of about 700 masl. The flanking highland areas of the Paramos de Matanga on the west, and the Cordillera de Condor on the east, rise to maximum elevations of 4,200 masl and 3500 masl, respectively. The elevation of the Mirador prospect ranges from about 800 masl to 1,400 masl. Drainage is into the Zamora River via the Wawayme and Quime rivers.



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2.3.4 Infrastructure

The closest city serviced with daily jet flights is Cuenca, with a population of approximately 350,000 inhabitants. The closest existing airstrip is at Gualaquiza about 40 km to the northwest of the deposit. It has an asphalt runway approximately 2,075 m (6,800 ft) long, suitable for use by aircraft the size of a Hercules L100.

There are a number of small towns in the area of the project such as Gualaquiza and El Pangui. The small village of Tundayme is the closest to the deposit.

There is currently a 13.8 kV power line that approaches the project from Loja in the south and ends at a substation in Cumbaratza. The line continues to the project area as a 69 kV power line and carries on to the north to reconnect to the grid. The stability of these power lines is unknown.

2.4 Supporting Documents

Table 2-2 presents a list of supporting documents that formed part of the conceptual design and investigative process and are referenced for the conclusions presented in this feasibility study.

Table 2-2: Supporting Document List

Document Title	Owner	Document Number
<i>CORRIENTE RESOURCES INC.</i>		
Concentrate Study		
<i>AMEC EARTH & ENVIRONMENTAL, a division of AMEC AMERICAS LIMITED</i>		
Mirador Project, Conceptual Foundation Design for Mine Structures		31 January 2005, AMEC File: VM00343-700
Preliminary Mine Closure and Reclamation Plan, Mirador Project, Ecuador		24 November 2004, AMEC File: VM00343
Water Management Plan, Mirador Copper Project, Ecuador		24 November 2004, AMEC File: VM00343
Mirador Waste Dump Design		09 November 2004, AMEC file: VM00343-600
Mirador Project, Tailings Management Facility, Preliminary Design Report		23 November 2004, AMEC File: VM00343
<i>CAMINOSCA</i>		
Mirador Mining Development, Camp-Crusher-Pit Access Road, Preliminary Studies		18 November 2004, In 270VIAIfi-A-cb1-e0



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Document Title	Owner	Document Number
<i>MERIT</i>		
Mirador Project, Bridge Review from Site to Puerto Bolivar, Ecuador		
<i>KNIGHT PIÉSOLD LTD</i>		
Corriente Resources Inc., Corriente Copper Belt, Mirador Deposit, Report on Initial Tailing Storage Site Alternatives Study		17 February 2003, VA201-00078/1-1
Corriente Resources Inc., Mirador Mine, Assessment of Alternatives for a TMF in the Pangui Area, Letter Report		17 December 2004, V4-1219
Corriente Resources Inc., Mirador Mine, Assessment of Alternatives for a TMF in the Pangui Area, Letter Report		02 March 2005, V5-0209
Corriente Resources Inc., Mirador Mine, Pangui Tailings Management Facility, Feasibility Study for 25,000 t/d TMF		18 March 2005, VA201-78/4-1
<i>TERRAMBIENTE</i>		
Mirador Project, Terms of Reference for EIA		November 2004
<i>WESTMAR CONSULTANTS INC.</i>		
Mirador Conveyor Cost Estimate and Conceptual Investigation		80263, March 2005
<i>OTHERS</i>		
Corriente Copper Belt Project, Southeast Ecuador, Order-of-Magnitude Study (Preliminary Assessment Technical Report)	David K. Makepeace, P.Eng.	22 June 2001
Mirador Project, Corriente Copper Belt, Southeastern Ecuador, Preliminary Assessment Technical Report	David K. Makepeace, P.Eng.	12 February 2002
Corriente Copper Belt Project, Southeast Ecuador, Order-of-Magnitude Study Part 1, Technical Report	David K. Makepeace, P.Eng., James Dawson, P.Eng.	February 2003
Corriente Copper Belt. Mirador Deposit, Report on Initial Tailings Storage Site Alternatives Study	Knight Piésold Ltd	VA201-00078/1-1, 17 February 2003
Copper Flotation Program, Preliminary Results (Ecuador)	Geomet, SA	May 2001
Status of Metallurgical Testing of Samples from Corriente Copper Belt, Ecuador, Progress Report No. 2	RDI Resource Development Inc.	May 2002
Metallurgical Testwork on Mirador Ore	Sumitomo Metal Mining Co. Ltd.	July 2003
Metallurgical Testing of Mirador Ores, Progress Report No. 1	SGS Lakefield Research	LR 10703-001, September 2004



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Document Title	Owner	Document Number
Metallurgical Testing of Mirador Ores, Progress Report No. 2	SGS Lakefield Research	LR 10044-094, October 2004
Mirador Deposit Phase II, Environmental Characterization of Locked Cycle Flotation Products	SGS Lakefield Research	LR 10703-002, February 2005
Corriente Resources Mirador Project Grinding Circuit Evaluation	MinnovEX Technologies Inc.	MinnovEX Project #21 2610, July 2004
Optimum Treatment Conditions for Ores from the Mirador Project	G&T Metallurgical Services	KM 1482, April 2004
A Preliminary Assessment of Metallurgical Response	G&T Metallurgical Services	KM 1540, September 2004
A Preliminary Assessment of Metallurgical Response – Two Samples of Near Surface Areas	G&T Metallurgical Services	KM 1540, September 2004



3.0 GEOLOGY AND RESOURCE MODELLING

Corriente engaged AMEC to provide a mineral resource estimate and Qualified Person's review and Technical Report for the Mirador Project. The work entailed estimating mineral resources in conformance with the CIM Mineral Resource and Mineral Reserve definitions referred to in National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects. It also involved the preparation of a Technical Report as defined in NI 43-101 and in compliance with the format set out in 43-101F1. Susan Lomas, P.Geo., an employee of AMEC, served as the Qualified Person responsible for preparing the resource estimation.

3.1 History

Regional exploration was initiated in southeastern Ecuador by BHP Billiton S.A. in 1994. BHP Billiton used silt sampling to identify favourable areas of base metal anomalies. With further mapping they identified possible porphyry copper systems associated with these anomalies. At least eight separate porphyry copper systems have been identified in the region.

The area of Mirador was recognized as an area of interest during the original reconnaissance geological and geochemical surveys completed in December 1994. These surveys, which included 315 pan concentrates of stream sediments, defined an area roughly 50 km² that contained anomalous values for Cu, Mo, Au, Zn, and Ag. However, BHP Billiton was forced to focus its efforts in the north part of the belt since the present property was off limits between 1995 and 1999 due to the border conflict in the area between Ecuador and Peru.

After a peace treaty was signed with Peru in July 1999, BHP Billiton completed detailed follow-up surveys to better define the anomaly areas at the Mirador property. They collected 746 soil samples along ridges and 219 rock chips from outcrops found in drainages traversing the anomalous zones. This work along with geological and alteration mapping defined the Mirador (previously known as Wawayme) zone and the Chanco and Chanco Norte Zones to the southwest.

In October 1999, BHP Billiton entered into an option agreement with Corriente covering 880 km² in the northern part of the porphyry belt. Subsequently, BHP Billiton and Corriente entered into a further agreement in April 2000 covering 230 km² in the southern part of the porphyry belt (the Mirador – Chanco property).

In 2002, after completion of 52 diamond drill holes, results of a polygonal resource estimate were released (12 February 2002). The estimated tonnage and grade,



calculated at a 0.65% Cu cutoff grade, were 218 Mt grading 0.73% Cu, all in the Inferred Mineral Resource category.

In 2003, Corriente released a second mineral resource for the Mirador Project in a NI 43-101 Technical Report (February 2003). The updated resource estimate was based on 10 additional holes and estimated using polygonal methods. This resource estimate, calculated at a 0.65% Cu cutoff grade, yielded 182 Mt grading 0.76% Cu and 0.221 g/t Au. The estimate was classified as Inferred mineral resources.

3.2 Regional Geology

The porphyry copper mineralization of the region is associated with late Jurassic porphyry intrusive phases of the Jurassic batholiths of the Abitigua subdivision in the Cordillera Real and sub-Andean regions of Ecuador. The Zamora batholith, which hosts the known porphyry mineralization, is part of the Abitigua subdivision.

3.3 Local and Property Geology

The Zamora batholith forms the wall rocks of the Mirador porphyry system. Within the mineralized zone, the intrusion comprises mainly granodiorite, with some leucogranite dikes along the west and southwest margins. An early, hornblende-feldspar porphyry intrudes the Zamora rocks as north-south discontinuous dikes.

The core of the deposit is characterized by a circular area of breccia composed of clasts of the host porphyry and granite, and early mineral quartz-stockwork fragments. This breccia obscures the contact relationships between the granite and porphyry units. The porphyry and the granite were impregnated with anhydrite stringers during the chalcopryite-mineralizing phase. Meteoric fluids percolated down from the surface and hydrated the anhydrite. The hydration of anhydrite created gypsum and a corresponding 50% volume increase that shattered the rock (Figure 3-1 and 3-2). Gypsum is soluble and leaches from the rock leaving weakly cemented to open fractures. The poor rock quality migrated downward in the deposit area as a hydration front. Away from the hydration front, the porphyry and the granite are less fractured and the anhydrite fracture filling is unaltered.



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Figure 3-1: Drill Core Photo of M64 from 140.78 to 146.14 m showing Low RQD prior to Core Cutting



Figure 3-2: Drill Core with Unaltered Anhydrite (Below the Gypsum Front)





A series of mostly northeast trending, steeply northwesterly dipping, hornblende-feldspar porphyry dykes cut the breccia and the country rocks. These dykes, ranging in age from syn- to post-copper mineralization, are more numerous and thickest along the southeast and northwest margins of the mineralization. A quartz-bearing variety appears to be the youngest in the series.

Post-mineralization, hydrothermal breccia dikes, with a polymictic clast assemblage of mineralized and unmineralized rock are associated with the porphyry dikes and are also most common along the southeast margin. The breccia also occurs as irregular plugs around the north and northeast margins of mineralization. Copper grades within the breccia range from very weak to slightly below average for the deposit, depending on the amount of mineralized rock incorporated.

Fifty- to eighty-metre high cliffs of Cretaceous Hollin Formation quartz sandstone occur to the south, above Mirador. These resistant rocks cover the southern extent of the Mirador alteration/mineralization complex.

3.4 Deposit Type and Mineralization

The copper and gold mineralization encountered at the Mirador deposit display many of the characteristics of calc-alkaline type porphyry copper system. Supergene zones of enrichment are weakly developed beneath a superficial leach cap that averages less than 22 m thick.

Mineralization at Mirador can be divided into early molybdenum, early copper $\pm$ gold, and late copper-gold stages. Molybdenum is associated with early quartz veining. Both copper-gold events are sulphide dominated. Chalcopyrite occurs as mainly irregular masses between clasts in the early breccia (with some disseminated) and as fine disseminations in a halo zone around the breccia within the Zamora granite. The syn- to post-mineral porphyry dykes are variably and weakly mineralized by the late copper-gold disseminations and sparse sulphide veins.

A late stage series of 1 cm to 2 cm quartz sulphide stringers transect the project. They are subvertically emplaced but their strike is unknown, as they have only been seen in drill core. These veinlets are sulphide rich and have elevated gold and zinc values. They were intersected in the vertical drill holes and were found running down the core over metres of core length. The impact of these stringers is discussed in more detail in Sections 3.6 and 3.10 below.

The Mirador porphyry system exhibits well-defined alteration zoning, with a large (approximately 4 km²), quartz-sericite alteration zone covering a large part of the wall



rock and quartz diorite porphyry. This alteration has overprinted a potassic alteration whose remnants are small secondary biotite nuclei mainly in the quartz monzonite.

3.5 Exploration

Since April 2000, Corriente has carried out exploration on the Mirador Project. The work completed has included geological mapping, pan concentrate sampling of drainages, geochemical soil sampling, rock chip sampling and the completion of 24,349 m of core drilling in 91 diamond drill holes. (The pan concentrate work was carried out by BHP Billiton previously to Corriente's participation, but Corriente conducted all subsequent sampling and project work.)

- **2000**, in May the first phase drilling on Mirador began and 30 holes were completed resulting in 5,383 m of drill core under the supervision of J. David Lowell, who was under contract by Corriente. Corriente assumed total management of the project in October 2002.
- **2001**, between January and May of 2001, the second phase of drilling was completed at Mirador and 22 holes were drilled for 8,136 m of core.
- **2002**, between February and April of 2002, the third phase of drilling was completed where 10 holes were drilled for 2,739 m.
- **2003**, in February a Preliminary Assessment Report of the Mirador deposit was completed and was filed as a Technical Report entitled "Mirador Project, Corriente Copper Belt, Southeast Ecuador, Order of Magnitude Study, Part 1, Technical Report", under the supervision of Qualified Person David K. Makepeace, P.Eng., and James M. Dawson, P.Eng.. A polygonal resource estimation was completed for this study based on 62 drill holes completed on the Mirador Project.

In July, Sumitomo of Japan completed independent metallurgy tests showing favourable concentrate potential. AMEC reviewed this work and found it to be done to industry standard and subsequent follow-up work has confirmed their conclusions. Metallurgy is discussed in detail in Section 16 of this report.

- **2004**, in April the fourth phase drilling was completed and 29 drill holes were drilled for 8,091 m of core.

3.6 Drilling

The Mirador deposit has been intersected by 91 diamond drill holes totalling 24,350 m, arranged in a rough grid on approximately 75 m to 100 m centres. The drill hole



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information has been tabulated in Table 3-1 and the collar locations are illustrated in Figure 3-3.

Table 3-1: Summary Table of Drill Holes

Hole ID	X	Y	Z	Depth	Downhole Survey	Dip	Azimuth	Phase
M01	784429.06	9604229.02	1335.91	104.56	N	-70	180	1
M02	784548.07	9604336.79	1230.56	198.16	N	-90	360	1
M03	785225.46	9604083.82	1389.82	368.81	N	-90	360	2
M04	784533.41	9604095.86	1345.16	92.35	N	-90	360	1
M05	784728.65	9604200.73	1363.85	226.42	N	-90	360	1
M06	784734.12	9604104.12	1369.27	199.64	N	-90	360	1
M07	784731.22	9603996.78	1418.72	104.39	N	-90	360	1
M08	784846.97	9604315.52	1296.32	158.19	N	-90	360	1
M09	784837.34	9604099.41	1408.63	150.27	N	-90	360	1
M10	784821.49	9603899.27	1408.04	163.98	N	-90	360	1
M11	784954.04	9604295.75	1312.64	150.88	N	-90	360	1
M12	784923.83	9604105.48	1391.69	211.23	N	-90	360	1
M13	784912.06	9603788.57	1459.88	153.01	N	-90	360	1
M14	785031.76	9604294.29	1327.44	150.88	N	-90	360	1
M15	785030.21	9603974.82	1385.97	176.17	N	-90	360	1
M16	785022.78	9603887.73	1385.58	213.97	N	-90	360	1
M17	785158.22	9604410.25	1304.30	131.06	N	-90	360	1
M18	785175.45	9604299.31	1298.23	86.26	N	-90	360	1
M19	785168.60	9604192.59	1318.96	106.06	N	-90	360	1
M20	785134.51	9604084.62	1356.43	244.75	N	-90	360	1
M21	785254.51	9604493.84	1243.87	54.86	N	-90	360	1
M22	784551.19	9603892.78	1507.67	128.02	N	-90	360	2
M23	785342.00	9604280.11	1339.23	400.81	N	-90	360	1
M24	784514.99	9604112.03	1345.40	69.65	N	-60	260	1
M25	784429.06	9604229.02	1335.91	223.42	N	-60	255	1
M26	785106.59	9603790.80	1464.02	99.20	N	-90	360	1
M27	785332.78	9604204.01	1370.16	438.91	Y	-70	270	2
M28	785333.85	9604093.70	1325.75	258.47	Y	-70	135	2
M29	785209.72	9603890.66	1385.05	457.20	Y	-90	360	2
M30	785124.34	9603976.42	1419.83	175.26	N	-90	360	1
M31	784934.00	9603982.00	1360.65	279.50	N	-90	360	1
M32	785219.85	9603983.51	1361.20	295.35	N	-90	360	1
M33	784932.02	9604200.78	1378.04	285.60	N	-90	360	1
M34	785109.58	9603791.65	1464.04	429.77	Y	-70	90	2
M35	784920.79	9603890.85	1414.43	301.75	N	-90	360	1
M36	785121.42	9603884.58	1451.54	175.26	N	-90	360	1
M37	785335.76	9603867.16	1354.07	400.00	Y	-90	360	2
M38	785002.46	9603680.09	1498.12	486.16	Y	-90	360	2
M39	785329.16	9603682.64	1395.37	272.49	N	-70	135	2
M40	785325.20	9603684.65	1394.52	298.70	N	-90	360	2
M41	785040.51	9604081.10	1311.76	249.94	N	-90	360	2
M42	785025.27	9604190.25	1330.42	294.13	N	-90	360	2
M43	784903.62	9603679.88	1539.60	399.29	N	-90	360	2
M44	784928.05	9603935.74	1391.86	419.10	Y	-90	360	2
M45	784927.04	9604034.14	1368.02	303.28	N	-90	360	2
M46	784822.81	9604005.57	1424.18	408.13	N	-90	360	2



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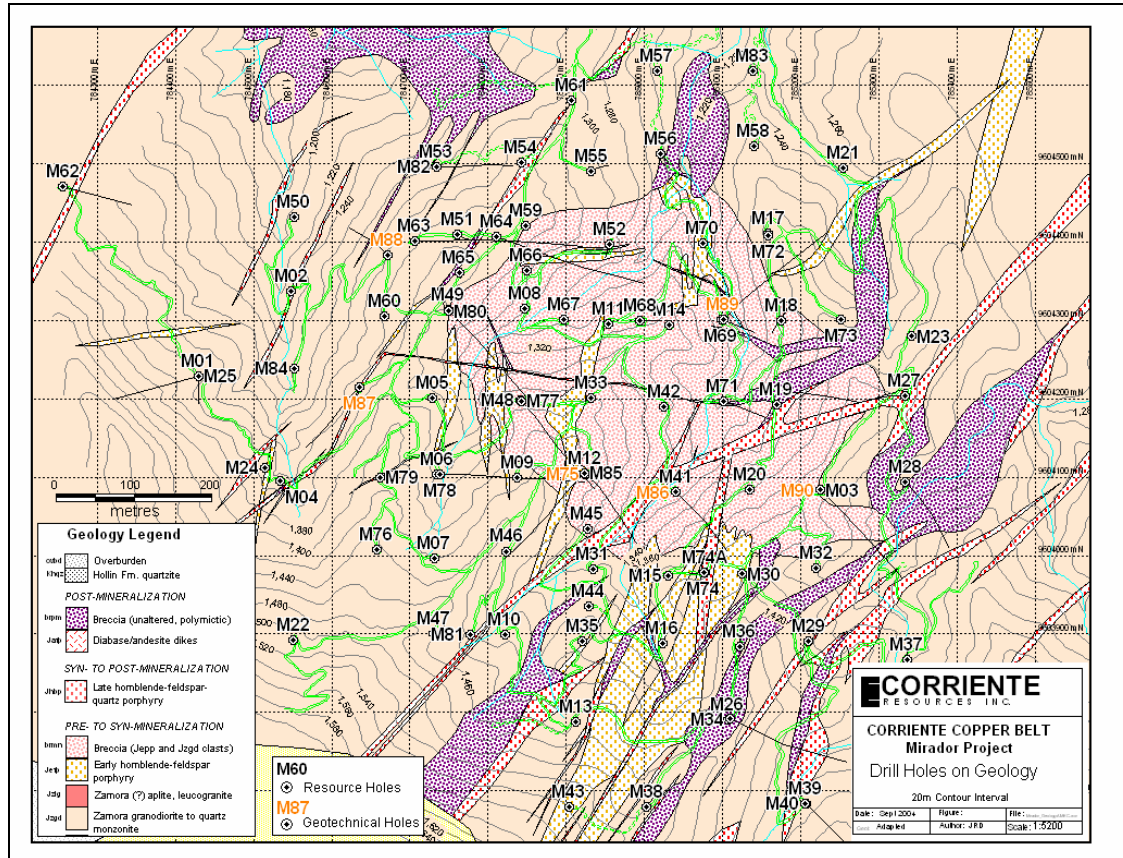
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Hole ID	X	Y	Z	Depth	Downhole Survey	Dip	Azimuth	Phase
M47	784730.04	9603900.14	1464.08	349.00	N	-90	360	2
M48	784840.47	9604198.33	1372.03	448.06	N	-90	360	2
M49	784748.93	9604313.58	1358.08	492.25	Y	-90	360	2
M50	784551.33	9604431.37	1208.15	341.06	N	-90	360	2
M51	784760.68	9604409.17	1357.24	449.58	Y	-90	360	2
M52	784955.70	9604396.72	1294.20	443.48	Y	-90	360	2
M53	784733.46	9604496.33	1291.29	198.42	N	-90	360	3
M54	784842.37	9604500.75	1335.86	188.67	N	-90	360	3
M55	784932.04	9604490.70	1316.40	363.02	Y	-90	360	3
M56	785020.91	9604512.41	1243.26	399.90	Y	-90	360	3
M57	785015.86	9604618.65	1239.11	250.85	Y	-90	360	3
M58	785139.87	9604521.79	1267.86	180.75	Y	-90	360	3
M59	784848.69	9604420.90	1352.61	391.67	Y	-90	360	3
M60	784667.52	9604304.61	1341.69	410.26	Y	-90	360	3
M61	784907.07	9604580.28	1307.20	171.91	Y	-90	360	3
M62	784256.02	9604470.38	1309.96	183.30	N	-60	105	3
M63	784706.16	9604401.16	1345.00	400.51	Y	-60	90	4
M64	784810.73	9604406.45	1352.00	400.81	Y	-60	110	4
M65	784763.08	9604360.13	1363.00	300.23	Y	-90	0	4
M66	784849.12	9604364.10	1322.00	300.23	Y	-90	0	4
M67	784896.78	9604300.56	1295.00	300.23	N	-90	0	4
M68	784994.73	9604299.24	1315.00	315.47	N	-85	90	4
M69	785098.00	9604300.56	1270.00	269.75	N	-80	110	4
M70	785074.16	9604398.51	1240.00	249.94	N	-90	0	4
M71	785100.63	9604197.31	1278.00	300.23	Y	-80	110	4
M72	785158.22	9604408.00	1305.00	286.51	Y	-90	0	4
M73	785251.53	9604300.56	1338.00	188.98	Y	-90	0	4
M74A	785077.89	9603978.05	1400.00	73.15	N	-90	0	4
M74	785076.00	9603978.05	1400.00	288.04	Y	-85	270	4
M75	784925.00	9604104.00	1391.69	300.25	Y	-60	135	4
M76	784657.38	9604007.78	1445.00	254.51	Y	-90	0	4
M77	784842.00	9604197.00	1372.03	300.25	Y	-60	135	4
M78	784738.37	9604103.27	1370.00	300.25	Y	-55	90	4
M79	784661.60	9604099.98	1410.00	269.75	Y	-90	0	4
M80	784750.00	9604312.00	1358.08	300.23	Y	-60	135	4
M81	784776.95	9603899.64	1430.00	256.03	Y	-90	0	4
M82	784734.50	9604496.35	1280.00	225.55	Y	-65	90	4
M83	785138.24	9604618.10	1225.00	150.88	Y	-90	0	4
M84	784552.00	9604239.00	1260.00	249.94	Y	-90	0	4
M85	784923.83	9604105.48	1392.00	300.23	Y	-60	225	4
M86	785040.51	9604081.10	1312.00	202.69	Y	-60	135	4
M87	784635.92	9604215.19	1342.00	300.23	Y	-70	225	4
M88	784671.06	9604383.53	1310.00	349.91	Y	-80	315	4
M89	785100.63	9604300.56	1270.00	358.14	Y	-60	45	4
M90	785225.46	9604083.82	1390.00	298.70	Y	-60	135	4



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Figure 3-3: Mirador Drill Hole Location Plan



Kluane International Drilling Inc. of Canada was contracted to complete the diamond drilling programs. They used a man-portable wire-line drill and all platforms were accessed via hand-cut mule trails. There is no road access to the drill pads; access is by a walking trail over the final 1,500 m from the road.

Drill core was recovered in standard core tubes at sizes of NTW (5.2 cm diameter) and BTW (4.8 cm diameter). The smaller size core was recovered in the lower parts of the deeper drill holes. Standard HQ size core was taken from a few of the geotechnical holes.

Most drill holes at Mirador were drilled vertically. AMEC believes more angled drill holes should have been drilled due to the presence of various geologic features having sub-vertical contacts (e.g., post-mineral dykes, late quartz-sulphide stringers). The copper grades do not appear to be influenced by the angle of the holes. However, because the quartz-sulphide stringers contain elevated gold values, samples hosting them were treated conservatively during gold grade interpolation.



The drill hole collars were surveyed using differential GPS equipment to a reported instrument accuracy of ± 1 m (X-Y) and ± 2 m (Z) Segundo Toledo Peláez of Topcon Survey S.A. of Ecuador. Some of these surveys were only done on the excavated pads before drilling. The accuracy here will be somewhat less than the accuracy of the equipment, in the order of ± 3 m.

Table 3-1 includes a column to indicate which drill holes were down hole surveyed. The surveys were completed using a Tropari instrument for the 20 holes in the second and third phases of drilling. A Sperry-sun instrument was used during the fourth phase of drilling in 2004. Of the 91 drill holes completed, 50 have no down hole survey data. Five of the unsurveyed drill holes were drilled at an angle between 60° and 70° ; M01, M25, M25, M39, and M62. These holes were drilled on the periphery of the deposit and have no impact on the resource estimation. Drilling deviation on vertical holes can be very small but it is best practice to complete down hole surveys on all drill holes.

Core recovery is good and is $>90\%$ in the zones of primary mineralization. The RQD values measured from the core indicate that rock quality is very low through large parts of the deposit. The average RQD value from the drilling over the whole of the deposit is 38%. The poor RQD at Mirador is the result of the shattering process due to the hydration of anhydrite mineralization.

Bulk density was determined on pieces of core that ranged in size from 10 g to 30 g each. The samples were collected at 40 m intervals. For phases 1 to 3 the bulk density was determined by weighing the sample dry, then immersing the sample in water and measuring the weight of the water displaced. This was not a very efficient method as it was slow and water could soak into the samples. The procedure was changed for Phase 4 where the samples were suspended with thin nylon monofilament and weighed dry, then immersed in water and weighed wet. There are 665 values in the database for the hypogene samples, 21 for the Leached, 22 for the Mixed and 46 for the Enriched samples for a total of 754 samples tested.

3.7 Sampling Method and Approach

The entire length of all the drill holes was sampled and assayed. When Corriente receives the core at the camp, the core technicians mark up the core into 1 m samples and assign each of these intervals a sample number. The samples are combined into composite samples according to the rock type. The Leached rock type was sampled in 5 m lengths. Within the zone of supergene-copper enrichment the sample length was 2 m. The Mixed lithology was identified at the end of the 4th phase of drilling therefore these samples can be either 5 m or 2 m in length. There was poor identification of these units during the primary logging so the samples typically cross



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boundaries between these three rock types (Leached, Mixed and Enriched). Sample lengths through the zone of primary mineralization are 3 m. The sample interval was frequently extended to five metres within barren post-mineral dikes.

Table 3-2 contains a list of the proposed and the resulting sample lengths by rock type. The enriched samples were often found to be longer than the proposed upper limit of 2 m where 57 samples or 22% of the samples are over 3 m.

The sampling program was planned and executed in a satisfactory manner and the resulting assays are acceptable for resource estimation purposes.

Table 3-2: Sample Lengths by Lithology Unit Mineral Zone

Lithology Name	Lithology Code	Sample Length (m)	Number of Samples	Mean Length	Min Length	Max Length
Leached	90	5	440	4.80	2	14
Mixed	93	5 or 2	169	2.77	2	10
Enriched	95	2	261	2.25	1	7
Mineralized	100 to 150	3	5743	2.98	0.75	6
Post mineral dykes	10 and 50	5	966	3.79	1.6	13

The geologist marks up a sample sheet where the samples are composited by rock type and receive a sample number. The samples are split and placed in plastic sample bags. The core is split using a Boart rock saw. When the core is strongly broken, fragments from one half of the core are picked out of the box. The sampler is instructed to collect the pieces randomly. No line is marked on the core. The sample bags are doubled and the sample numbers are well secured to the bags. The sample bags are then weighed and the weight is recorded on the sample sheet.

The current sample number system uses a 10-digit number, e.g., 2722190680. The number breaks down into the following components:

2_72_219_068_0

- 2: The Project Number (Mirador is 2)
- 72: The drill hole number for M72
- 219: The “to” interval the sample interval. If the sample is an SRM then this number is used to indicate which SRM was inserted into the sample stream. The code used and the corresponding SRM samples are listed in the following table:



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Code	SRM
500	GEM1
600	GEM2
700	GEM3
800	MS1
900	MS2

- 068: The sample number for the drill hole
- 0: This number is used as a QA/QC marker
 - 0 = core sample
 - 7 = ½" crush duplicate
 - 8 = coarse reject duplicate
 - 9 = pulp duplicate

AMEC recommends that Corriente change their sample numbering protocols and use assay laboratory sample tag books. Sample numbers should never identify the hole number, the sample interval, or put emphasis on QA/QC samples by giving them unique identifiers. Going to standardized sample books will minimize risk of recording errors and be consistent with industry best practices.

The sample bags are put into rice bags for shipping. Each bag is marked with the project number, the drill hole number, and then another number for the shipping bag. AMEC advised Corriente that this was not an accepted practice and the bags should only be marked with the bag number, not the drill hole number. The rice bags are secured with tape and rope. The samples are sent to Quito by a contracted driver and are not zip secured or in a locked box.

The remaining half of the sampled core was then logged in detail by Corriente geologists.

3.8 Sample Preparation, Analyses, and Security

Sample pulps were prepared at Chemex Laboratories in Quito by laboratory personnel. Earlier samples were processed and analysed by Bondar Clegg prior to the merging of the two companies. The preparation procedure for the samples and duplicates and standard insertion changed over the four different phases of drill programs.



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For all drill phases the following procedures remained consistent. The whole sample was crushed to 75% passing –10 mesh, then 1 kg of sample was pulverized to 95% passing –150 mesh and a 100 g subsample was split from the mass and shipped to Chemex in Vancouver for 30 g fire assay analysis for gold and copper was analyzed by acid digestion/AAS methods.

No QA/QC samples were inserted in the sample shipments for the Phase 1 drilling campaigns. Five percent of the samples from these drill programs were reanalysed with the benefit of SRM samples.

For Phase 2 drilling, the only QA/QC sample inserted into the sample shipments was a 100 g Lab Check Duplicate. This sample was taken on a 1 in 10 basis and was shipped to Chemex in Vancouver for analysis. The check lab was changed to Acme Labs in Vancouver after ALS Chemex purchased Bondar Clegg. No SRM samples were included in the shipment of any samples for these drill programs. The Lab Check Duplicate sample results were insufficient to certify the analysis results so a reassay program was set up by AMEC. Five percent of the samples from these drill programs were reanalysed with the benefit of SRM samples.

For the Phase 1 and 2 drilling programs the samples were analysed for gold using fire assay analysis and for copper, silver, lead, molybdenum, and zinc using four acid digestion/AAS methods.

The sampling procedures were modified in February 2001 to introduce some QA/QC samples into the sample shipments. The new protocol included inserting one pulp duplicate and one of three standards, GEM1, GEM2, and GEM3 into each batch of 20 samples sent to the labs. These standards were insufficiently homogenous to be used for a QA/QC program. This situation does not impact the quality of the grade values since 5% of the samples were reanalysed with the benefit of SRMs.

In late 2002, for the third phase of drilling, Corriente changed the insertion rate of the pulp duplicates and SRM samples to 1 in 40 samples. MS1 was the SRM sample being included in the lab shipments for this drilling campaign. The check samples were now being shipped to Acme labs in Vancouver but no SRM samples were being included in the shipments of the Lab Check Duplicate.

Samples for the third drilling phase on the project were analysed for gold using fire assay analysis and for copper, molybdenum, and zinc using four acid digestion/AAS methods.

In late 2003, the sampling protocols were modified to include more QA/QC samples. During the 4th phase drilling program, the insertion rate of all QA/QC samples was



increased back to a 1 in 20 insertion rate. The two SRMs included in the sample shipments were MS1 and MS2. A coarse reject duplicate was also included in the sample shipments. (A coarse split at -2 mesh was done only for samples destined for metallurgy test work and were only applied to drill holes M65 to M70.)

Samples for the fourth drilling phase on the project were analysed for gold using fire assay analysis and for copper and molybdenum using four acid digestion/AAS methods.

No blank samples were inserted into any of the sample shipments. Blank samples are used to certify sample preparation procedures by guarding against inter-sample contamination. They also are useful to identify sample mix-ups. It is recommended that Corriente include blank sample insertion in any future sampling programs.

With the inclusion of the reassay program and Corriente's current QA/QC protocols, the sample results are of sufficient quality to support a resource estimate.

3.8.1 ICP Sample Analysis

Three of the angled holes through the deposit, M75, M77, and M80, were selected to check for any significant levels of potentially deleterious elements. 281 samples were analyzed for the following elements by ICP analysis; Au, Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sr, Ti, V, W, Zn, and Hg. No elements reported ranges that would cause potential problems for the concentrate quality. Results from these analyses are presented in Appendix A.

3.8.2 Reassay of Phases 1 and 2 Drilling Samples

The drill core samples from drill Phases 1, 2, and 3 were initially assayed without the benefit of adequate SRM samples. Phase 1 samples were sent to the assay lab with no SRM samples and Phases 2 and 3 samples were sent with an SRM that was insufficiently homogeneous. The pulps for 5% of the samples from these phases, 523 samples, were sent to ALS-Chemex under a blind sample-numbering scheme. Standards (MS1 and MS2) and pulp duplicates were inserted into batches of 20 samples.

The reassay data returned values for copper and gold well within the accepted range to indicate that the original assays are of reasonable quality for resource estimation purposes. AMEC considers $\pm 10\%$ variation for 90% of the pulp duplicate analyses to demonstrate good precision. For the copper grades, 90% of the population has an absolute relative difference of 11% whereas for the gold grades it was 18% (Figure 3-4



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to 3-7). All SRMs submitted with these sample shipments returned values within the SRM accepted ranges. The gold grade differences are a little on the high side, probably due to the small subsample size of 100 g and the small aliquot size of 30 g.

Figure 3-4: QA/QC Reassay of Historical Data, Cu (ppm)

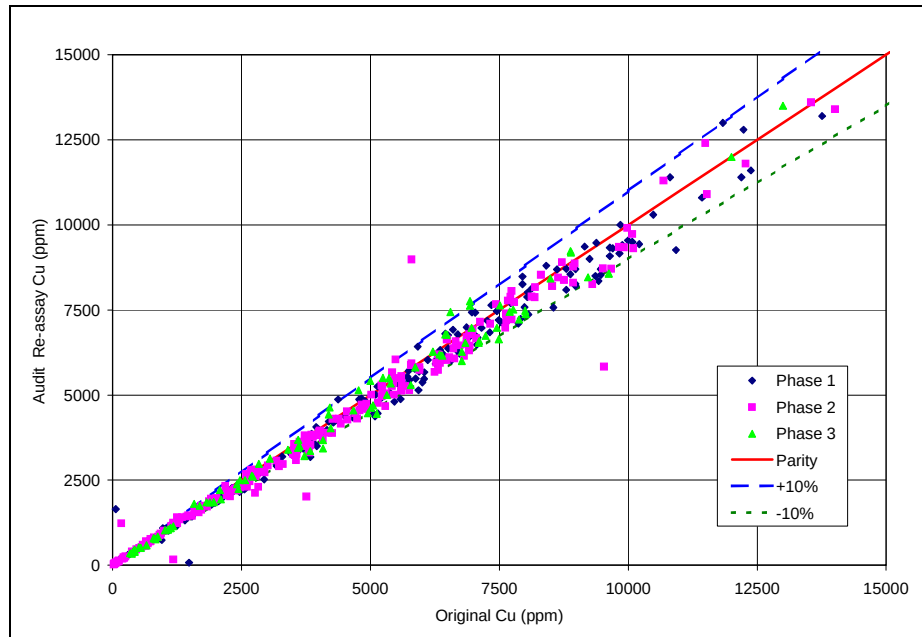
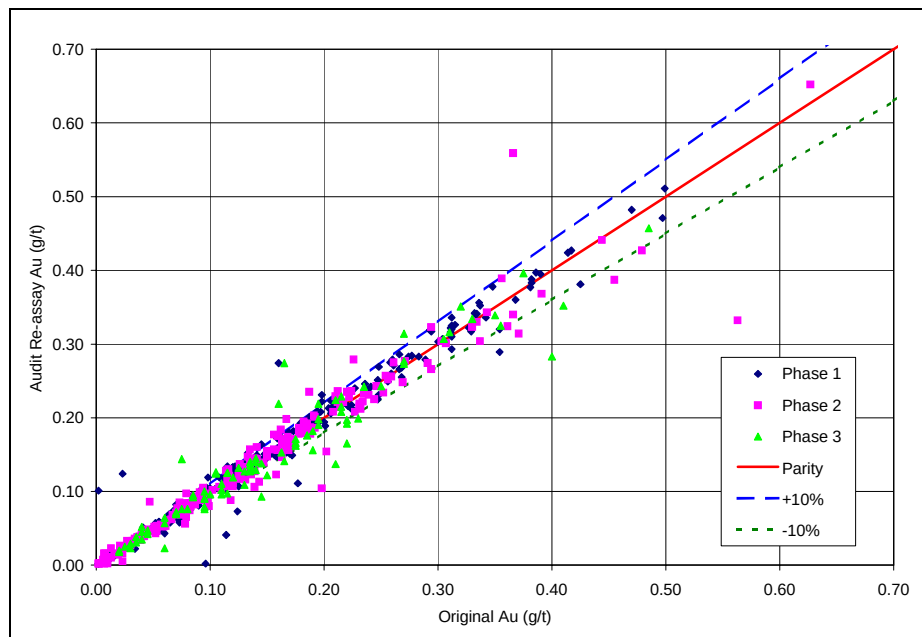


Figure 3-5: Reassay of Historical Data, Au g/t





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Figure 3-6: Percentile Rank Chart, Reassay Results, Cu (ppm), 501 Samples

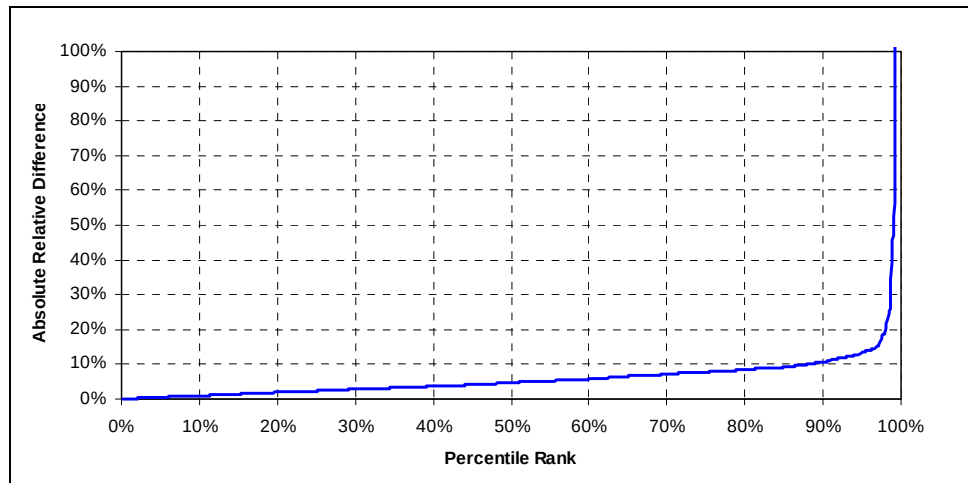
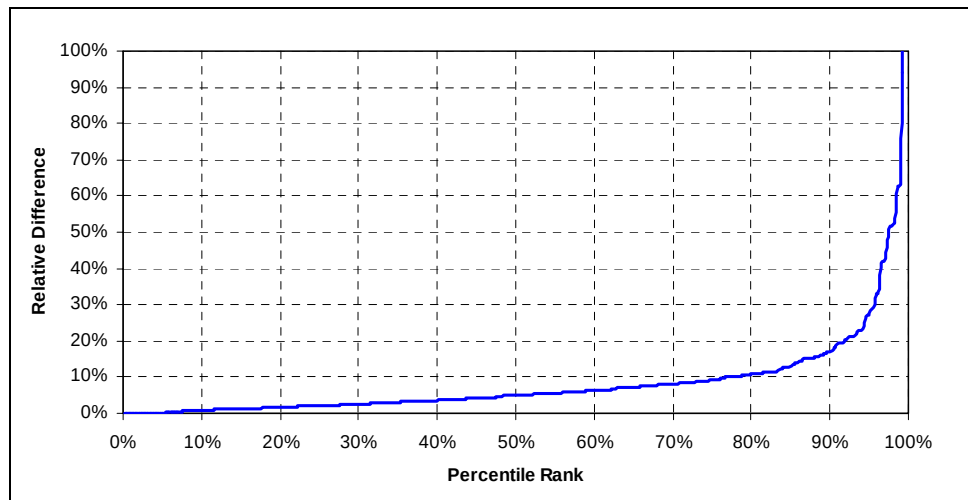


Figure 3-7: Percentile Rank Chart, Reassay Results, Au g/t, 442 Samples



3.8.3 Standard Reference Material

In 2003, Corriente had the ALS prep lab in Quito prepare two new SRMs for them: MS1 and MS2. The procedures outlined below were used for making these SRMs.

The SRMs were prepared by extracting 100 kg of material from a homogenized 500 kg bulk sample of drill core rejects (-10 mesh). The material was pulverized in a standard laboratory rod mill by grinding the 100 kg of drill core rejects to -150 mesh and then blanket rolling them (four corners technique) to homogenize. The pulp was



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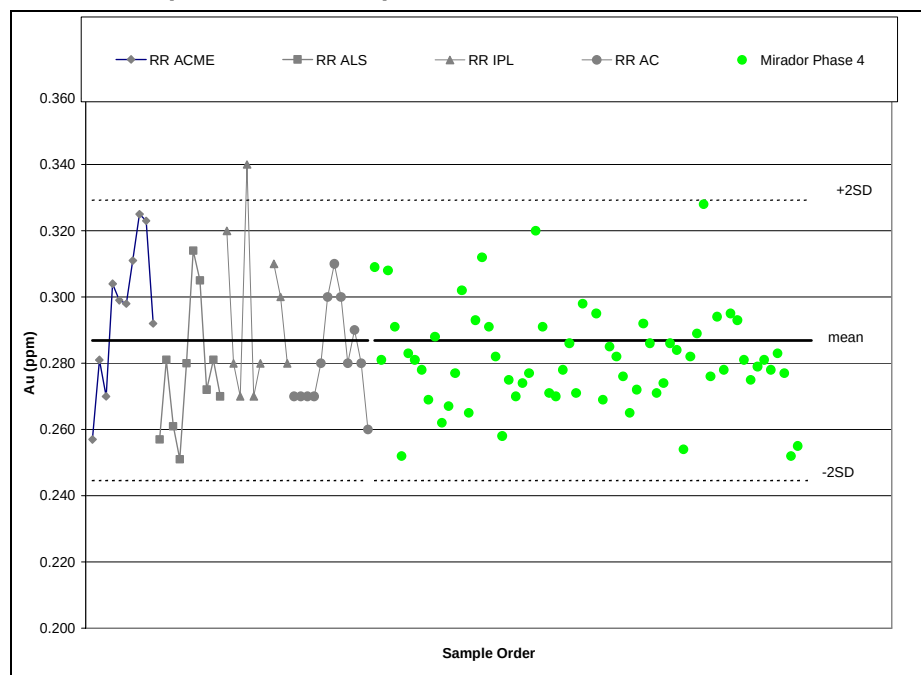
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immediately stored in paper envelopes (100 g) ready for submission for round robin analysis.

MS1 represents the middle grade range for copper and gold at the Mirador deposit (0.80 Cu % and 0.287 Au g/t) and MS2 is a low-grade copper-gold standard (0.37 Cu % and 0.16 Au g/t). Both SRMs were submitted for round-robin analysis at four laboratories, Analytical Laboratories Ltd, ALS Chemex, International Plasma Laboratories (IPL) and Assayers Canada. Each laboratory received 10 samples. The round robin analysis results were of sufficiently good quality to establish the acceptance limits for the SRMs (see Figures 3-8 to 3-11). They were inserted into the sample shipments on a 2 in 40 basis.

If a SRM result was found to exceed twice the standard deviation value from the round robin study then the batch of 20 samples was reanalysed until the SRM returned an acceptable value. Some samples on the SRM charts are outside of the acceptable limits. These were not reassayed as they were in areas outside of the mineralized envelope or in areas of below background grade range.

Figure 3-8: MS1 Round Robin and Sampling Results, Au g/t, (DDH M63 to M90)





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Figure 3-9: MS1 Round Robin and Sampling Results, Cu %, (DDH M63 to M90)

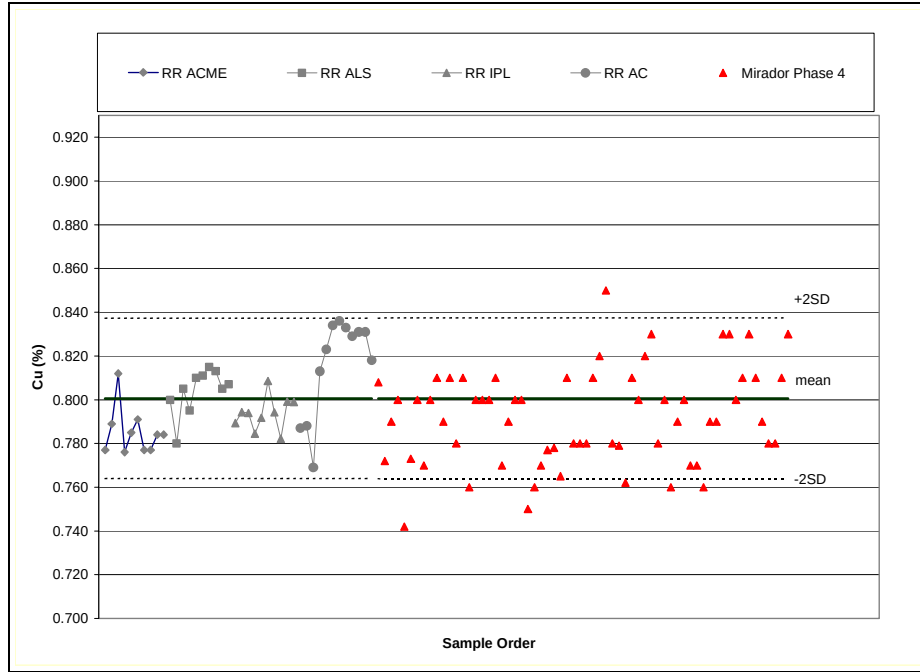
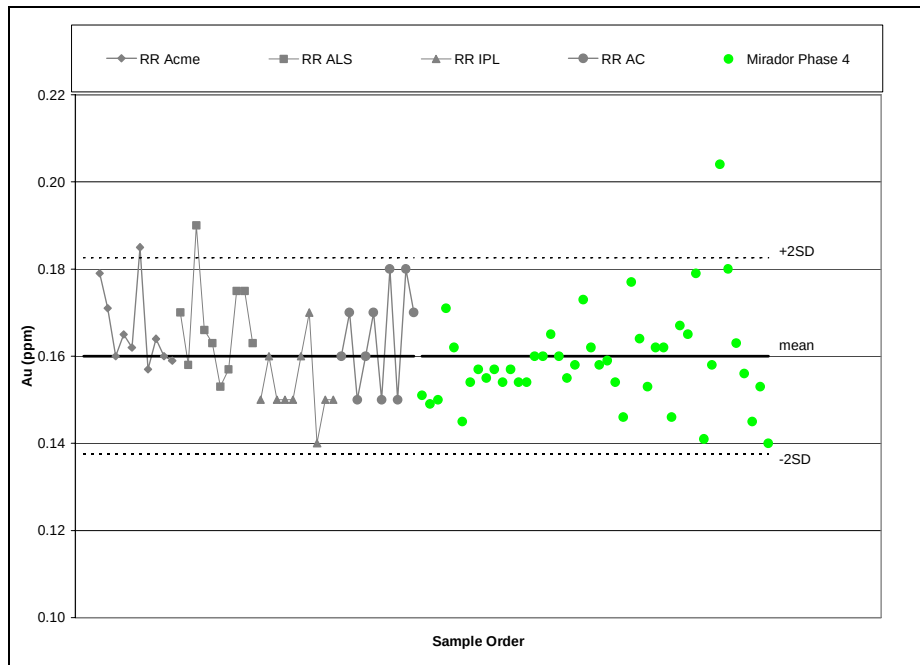


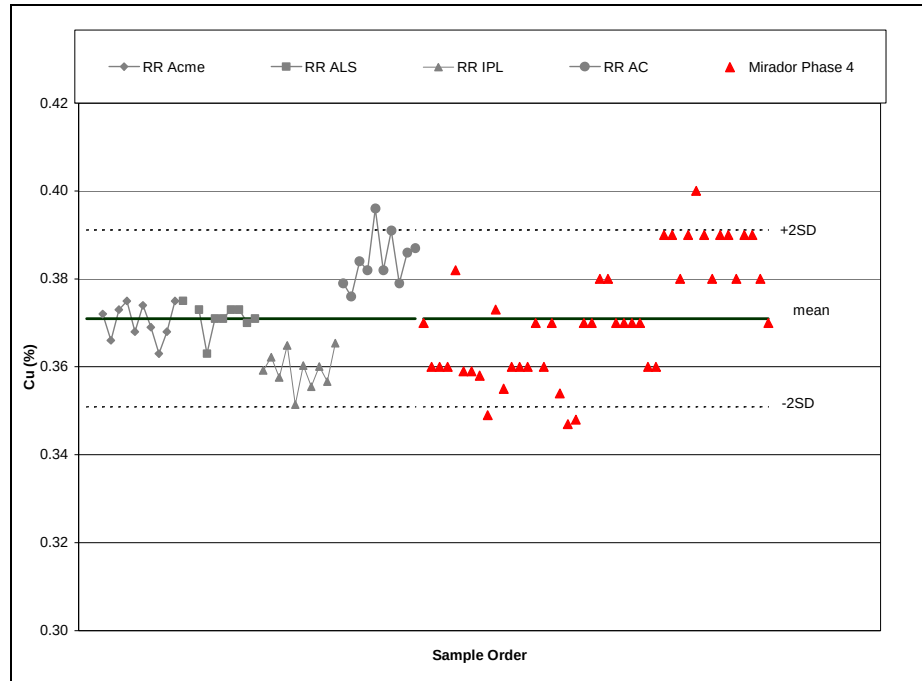
Figure 3-10: MS2 Round Robin and Sampling Results, Au g/t, (DDH M72 to M90)





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Figure 3-11: MS2 Round Robin and Sampling Results, Cu %, (DDH M72 to M90)



3.8.4 Duplicate Samples

Four different types of duplicate samples were variably used at Mirador.

- Phase 1: Only Lab Check Duplicates were taken during this phase of the drilling campaign on a 1 in 10 basis. AMEC has not reviewed the results from these samples. They have been superseded by the reassay results.
- Phase 2: Pulp duplicates were included in the sample shipments on a 1 in 20 basis.
- Phase 3: Pulp duplicates were included in the sample shipments on a 1 in 40 basis.
- Phase 4: Pulp duplicates were included in the sample shipments on a 1 in 20 basis. Coarse Reject duplicates were introduced into the QA/QC program and were taken on a 1 in 20 basis. A small program was begun where for metallurgical holes, the samples were being crushed to ½" and a 1 to 2 kg sample was taken for metallurgical test work. For these holes, a duplicate was taken every 20 samples to monitor the performance of the assay results and study the impact of removing the subsample.



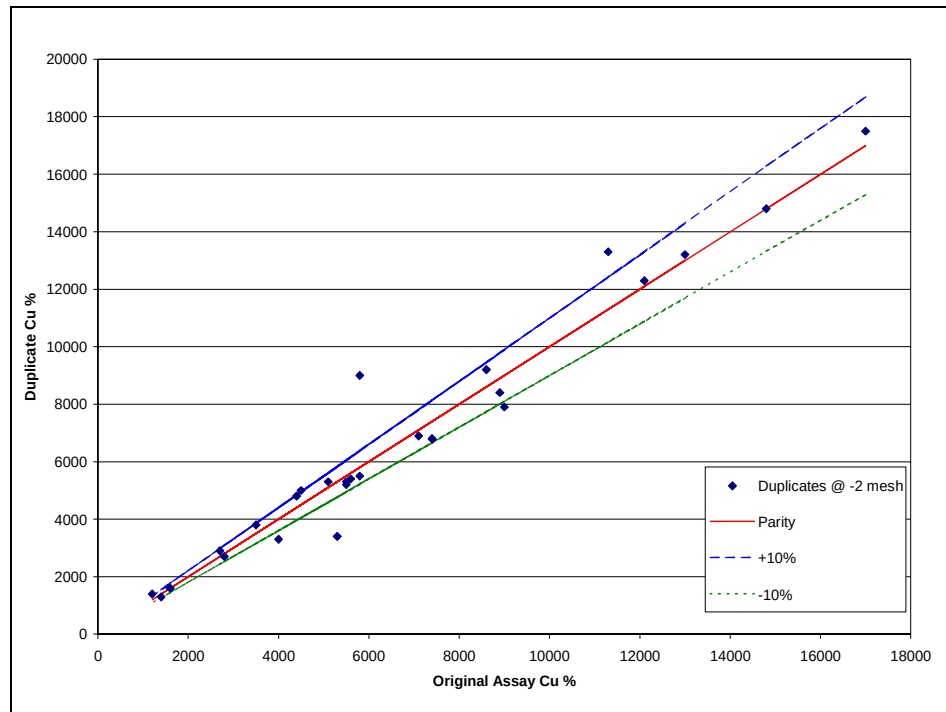
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Coarse Crush Duplicates, ½" Crush Material

These duplicates were taken from drill holes M65 to M70 resulting in only 27 duplicate pairs. This is too small of a sample population for a detailed analysis but the few results that are present indicate no bias to the sample analysis results for copper or gold (Figures 3-12 and 3-13).

Figure 3-12: Coarse Crush Duplicate Results, Cu %, 27 Data Pairs

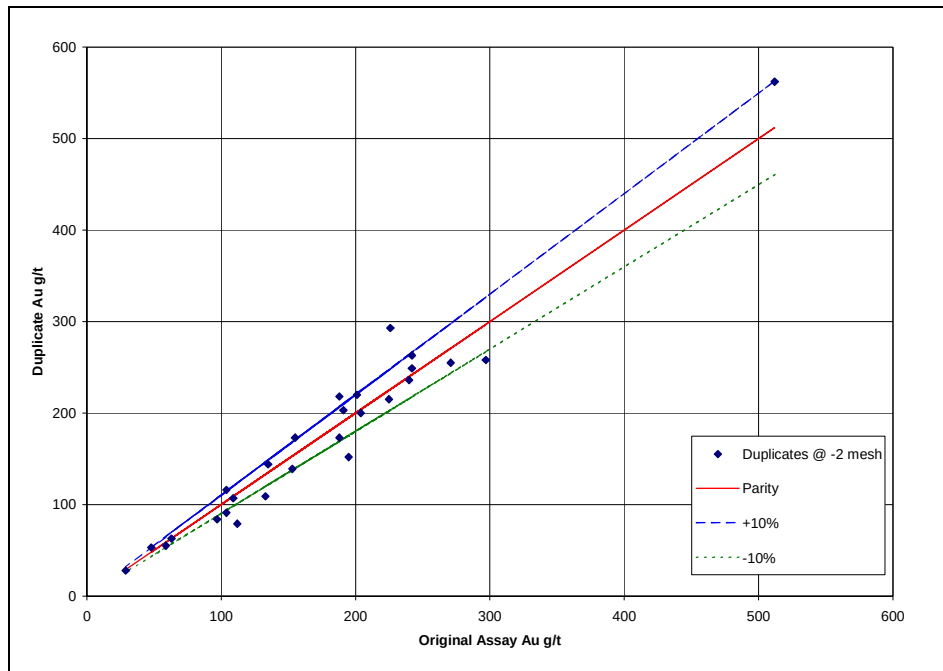




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Figure 3-13: Coarse Crush Duplicate Results, Au g/t, 27 Data Pairs



Coarse Reject Duplicates

This duplicate type was only introduced during the 4th drilling campaign in 2003 to 2004. Corriente currently has 85 coarse reject duplicates in their database, representing coverage of 3.4% of total assayed samples.

The results of the coarse reject duplicate assays are shown in Figures 3-14 and 3-15. The performance of the copper results is of excellent quality and reproduces well.

AMEC considers $\pm 20\%$ variation for 90% of the coarse reject duplicate analyses to demonstrate good precision. As seen in Figures 3-18 and 3-19, the variance for both the copper and gold results are well below the accepted upper limit of 20%.



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Figure 3-14: Coarse Reject Duplicate Results, Cu %, Scatter Plot

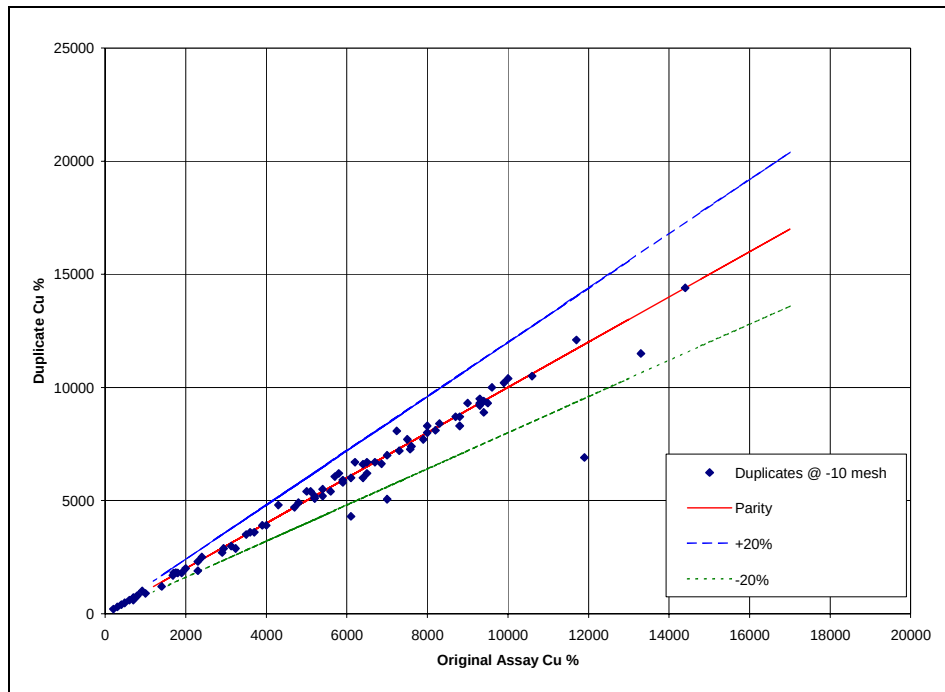
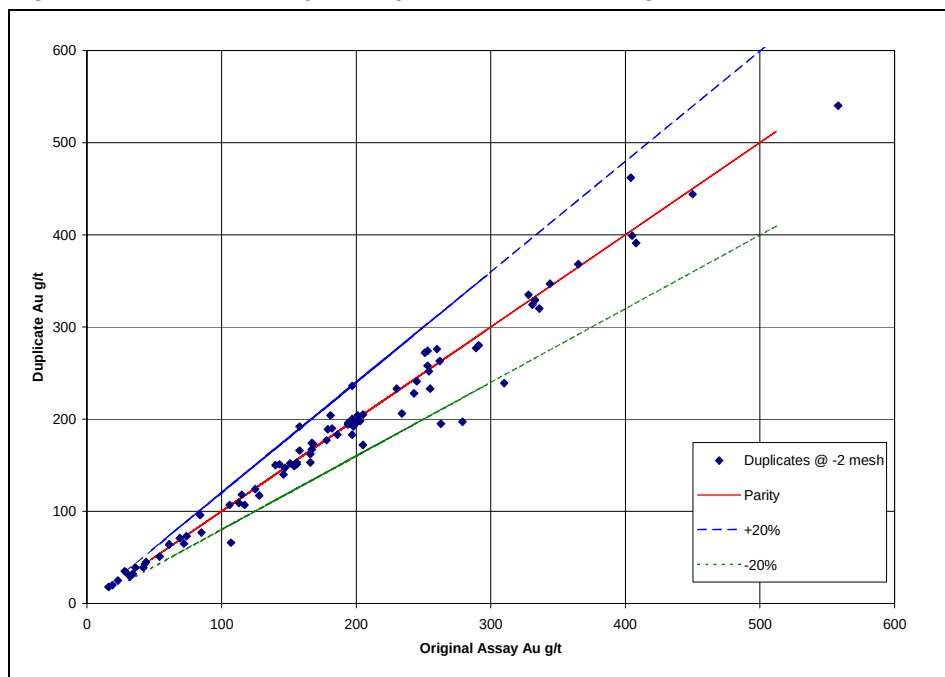


Figure 3-15: Coarse Reject Duplicate Results, Au g/t, Scatter Plot





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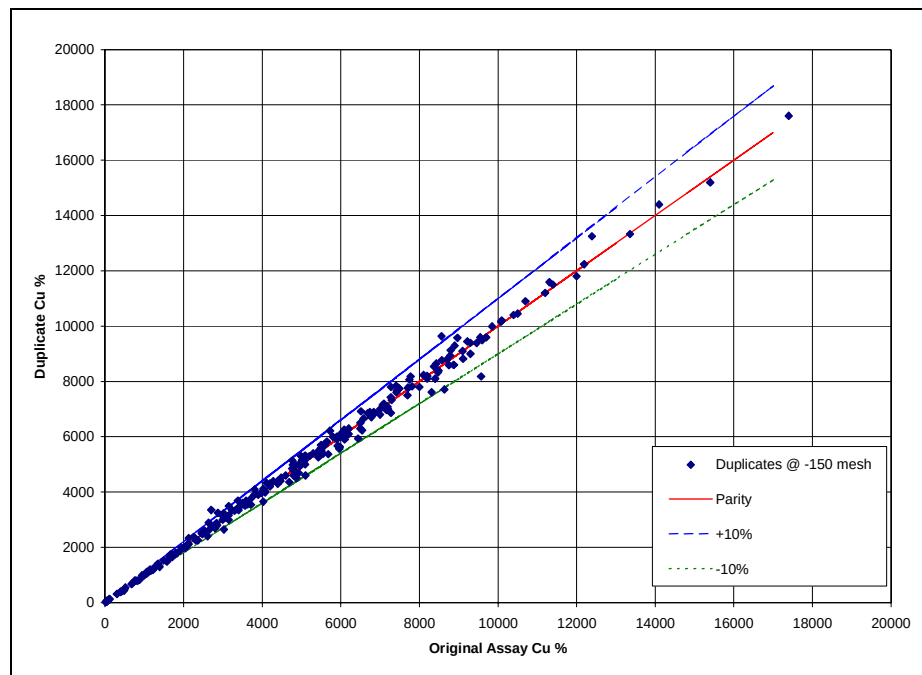
Pulp Duplicates

There are 248 pulp duplicates in the Mirador database representing 4.3% of the assay database.

The results of the pulp duplicate assays are shown in Figures 3-16 and 3-17. Performance for the copper values is excellent. The performance for the gold values is of acceptable quality considering its minor economic role in the deposit.

AMEC considers $\pm 10\%$ variation for 90% of the pulp duplicate analyses to demonstrate good precision. The copper values are plotted in Figure 3-18 on a percentile rank chart. This chart shows that the copper sample pairs are reporting 90% of the samples have a less than a 10% relative difference. The gold values however are reporting that 90% of the sample population has a relative difference of 15% (Figure 3-19); probably due to the small subsample size of 100 g and the small aliquot size of 30 g.

Figure 3-16: Pulp Duplicate Results, Cu %, Scatter Plot





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Figure 3-17: Duplicate Results, Cu %, Percentile Rank Chart

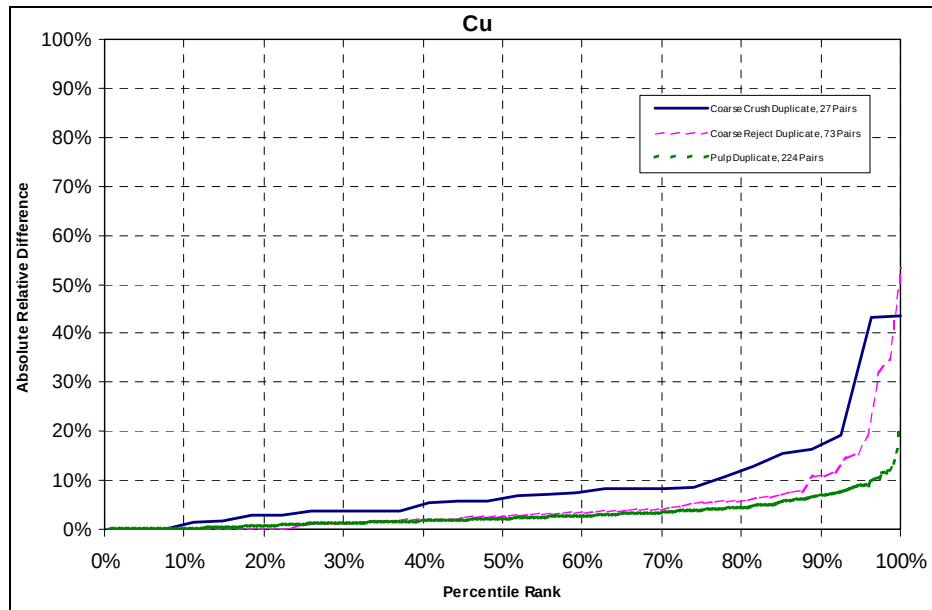


Figure 3-18: Pulp Duplicate Results, Au g/t, Scatter Plot

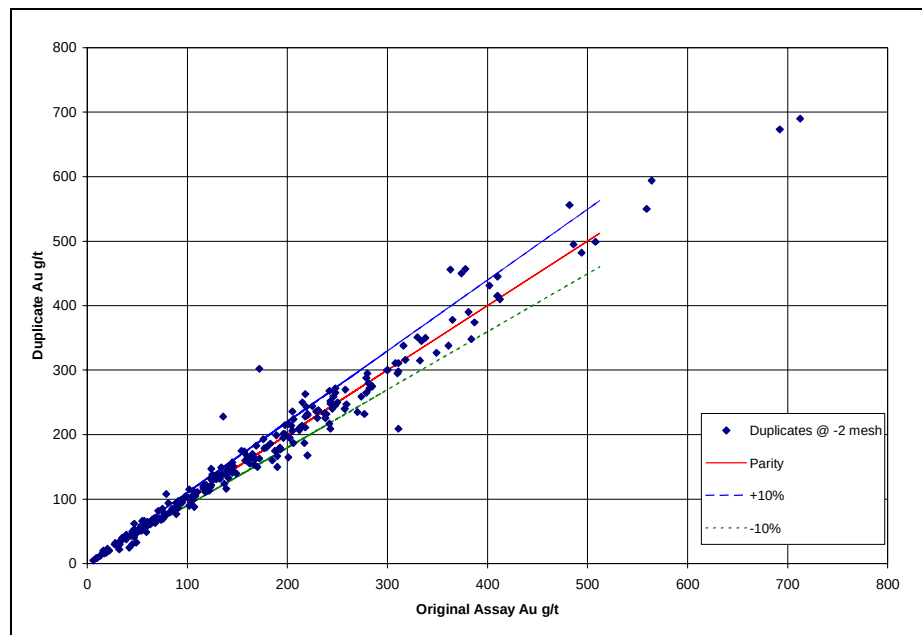
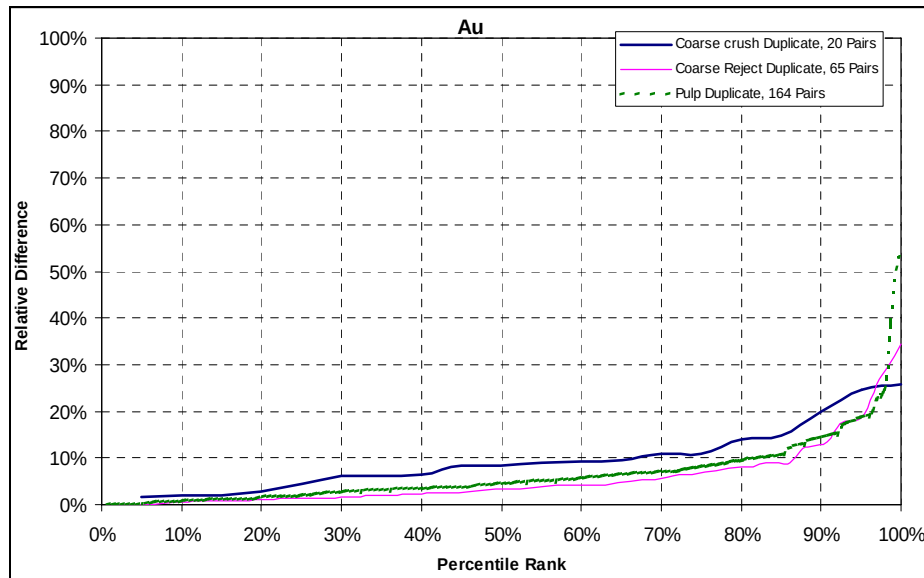




Figure 3-19: Duplicate Results, Au g/t, Percentile Rank Chart



3.9 Data Verification

AMEC completed a data quality check on 5% of the sample database used for the resource estimation. AMEC reviewed the collar surveys, downhole surveys, assay results and bulk density data and used original data sources to confirm the values in the database. The drill holes selected were M02, M12, M18, M33, M46, M54, M67, M77 and M87.

All the data was found to be of excellent quality and adequate for resource estimation purposes.

3.9.1 Assay Results and Lithology Codes

The assay results for 5% of the copper and gold were reconciled back to the original assay certificates issued by the assay laboratory. The sample intervals and sample number were confirmed by reconciliation to the drill logs for each of the drill holes. The lithology codes were confirmed using the drill logs. No errors were found during this review.

3.9.2 Downhole Surveys

AMEC checked all the survey disks for the downhole surveys during the site visit to the project. It was found that the geologists were not reading the down-hole tests



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correctly. On vertical holes, an incorrect azimuth was read on all holes. Corriente did not have a disk reader at the time but have since acquired one. All tests were reread using the proper instrument prior to completing the resource estimation.

3.9.3 Bulk Density

Twenty samples were sent to Teck-Cominco Labs in Vancouver for auditing; results agreed to within 3% for the first method and 0.6% for the second method (see Table 3-3). During the AMEC site visit, Corriente was advised that 5% of these samples should be validated at an independent laboratory.

Overall, the averaged values differ very little between the original and the check result. The bulk density values are of sufficient quality for resource estimation purposes.

Table 3-3: Bulk Density Independent Check Results

Hole ID	Date	Depth	Unit	Method	Corriente	Teck-Cominco	Relative % Difference
M16	2003	160	jefp	1	2.67	2.56	-4.2
M44	2003	300	jzgd	1	2.72	2.66	-2.2
M46	2003	280	jzgd	1	2.78	2.54	-9.0
M48	2003	350	brmn	1	2.65	2.55	-3.8
M49	2003	400	jzgd	1	2.75	2.69	-2.2
M51	2003	200	jzgd	1	2.64	2.53	-4.3
M52	2003	220	jzgd	1	2.62	2.59	-1.2
M55	2003	300	jzgd	1	2.60	2.53	-2.7
M56	2003	340	jzgd	1	2.63	2.66	1.1
				Avg:	2.67	2.59	-3.2
M63	2004	40.0	jzgd	2	2.24	2.41	7.3
M63	2004	110.0	jzgd	2	2.65	2.62	-1.1
M63	2004	170.0	jzgd	2	2.85	2.83	-0.8
M64	2004	67.0	brmn	2	2.56	2.50	-2.3
M64	2004	290.0	brmn	2	2.72	2.69	-0.9
M64	2004	380.0	brtc	2	2.61	2.56	-1.9
M65	2004	140.0	jzgd	2	2.53	2.48	-2.1
M65	2004	260.0	jzgd	2	2.56	2.56	0.0
M66	2004	110.0	jzgd	2	2.68	2.64	-1.6
M66	2004	140.0	brmn	2	2.57	2.52	-2.1
M66	2004	230.0	brtc	2	2.58	2.55	-1.3
				Avg:	2.60	2.58	-0.6

Methods: 1 = sample weighed dry, then immersed in water, and water displaced weighed; arm balance. 2 = suspended sample weighed in air, then immersed in water; electronic balance



3.10 Mineral Resource Estimates

The mineral resource for the Mirador deposit was estimated under the direction of Susan Lomas, P.Geo. of AMEC. The estimate was made from a 3-dimensional block model utilizing commercial mine planning software (Gemcom®).

3.10.1 Geology and Grade Domains

Key geologic units at Mirador deposit comprise intrusive phases of the host porphyry and post mineral dykes. These comprise:

- Zamora Granodiorite to Quartz Monzonite (JZGD)
- Breccia with JZGD and JEFP Fragments (BRMN)
- Early Hornblende-Feldspar Porphyry (JEFP)
- Late Hornblende-Feldspar-Quartz Porphyry (JHBP)
- Late Breccia, Unaltered, Polymictic (BRPM)

Also investigated were the various supergene-altered phases: Leach, Enriched and Mixed. Geologic models were created of the intrusive and altered units. AMEC checked the shapes for interpretational consistency on section and plan, and found them to have been properly constructed.

To constrain grade interpolation in each of the zones, AMEC created 3-dimensional mineralized envelopes based on copper and gold grades. These were derived by a method of Probability Assisted Constrained Kriging (PACK) to initially outline a general shape. Threshold grades were 0.2 g/t for Au and 0.5% for Cu. The probability that gold or copper in any block in the model exceeded the threshold was estimated using indicator variograms and ordinary kriging. Contoured probability values (in increments of 0.05) defined potential volumes used to delineate mineralized and non-mineralized populations. The boundary formed by each contour line was called an estimation envelope or "shell." The grade outline selection was done by inspecting the contoured probability values (in increments of 0.05) in Gemcom®. Final probability values were 0.40 for Au and 0.48 for Cu. Additionally, a copper shell at around the 0.30 probability value was made to help constrain grade interpolation in the background (or Outside Shell) domain. These shapes were then edited on plan and section views to be consistent with the structural and lithologic models and the drill assay data so that the boundaries did not violate data and current geologic understanding of mineralization controls.



The solids were used to code the drill hole data and block model cells. A set of cross sections and plans with drill holes colour-coded by domain and blocks similarly coloured were plotted and inspected to determine the proper assignment of domain.

3.10.2 Data Analysis

The data analyses were conducted on original and composited assay data. Assays were composited into 6 m down-hole composites. The compositing for the data analysis honoured the main lithology categories according to logged data and then were segregated by zone and grade shell or background codes. AMEC reviewed the compositing process and found it to have been performed correctly.

Descriptive statistics (boxplot analyses), contact plots, and histograms have been completed for copper and gold. Results obtained were used to guide the construction of the block model and the development of estimation plans.

Grade distributions of the assays by rock and alteration type, and grade probability shell are shown as box plots in Figures 3-20 and 3-21 (assays) and Figures 3-22 and 3-23 (6 m composites). Box-plots show the frequency distribution of the assay values by way of a graphical summary. The vertical axis of the plot gives the range of values for the particular assay (total copper). The box shows the portion of the sample data that falls between the 25th and the 75th percentiles. In other words, the box captures the half of the data, which falls in the middle of the distribution. The horizontal line that appears in the box represents the median of the data, or that value where half of the assays are greater and the remaining half are less than this median value. The mean or average of the data is shown with a dot. The vertical lines that extend away from the box reach to the minimum toward the bottom of the plot and to the maximum value toward the top.

Key observations from these analyses are:

- The domaining and tagging of the assay and the composite data functioned well. The domains used for grade interpolation (Inside Shell and BRMN (20)) and waste (BRPM (10) and JHBP (50)) were well selected as indicated by the grade populations inside each domain.
- The mean values per domain agree within accepted limits between the assay results and the subsequent composited data.



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Figure 3-20: Box Plot of Cu Assay Results by Litho Code

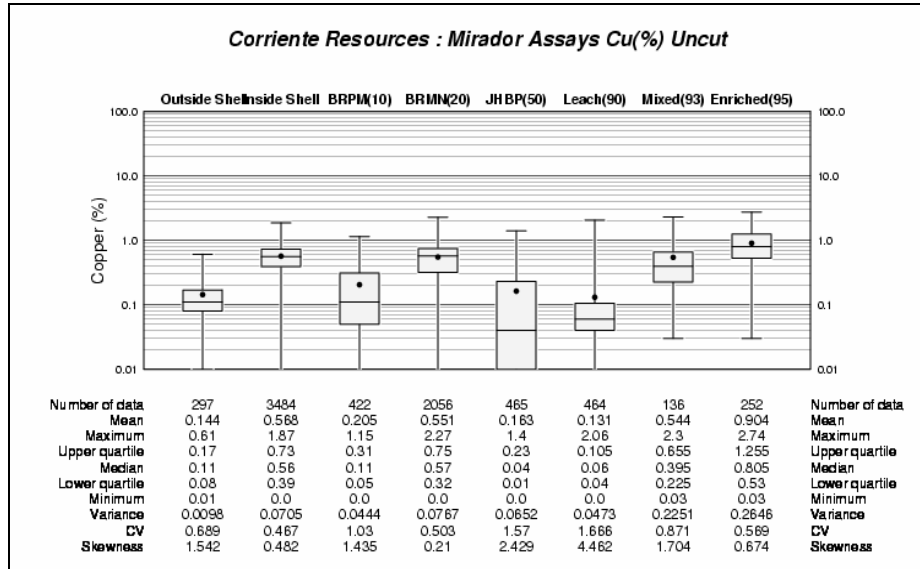
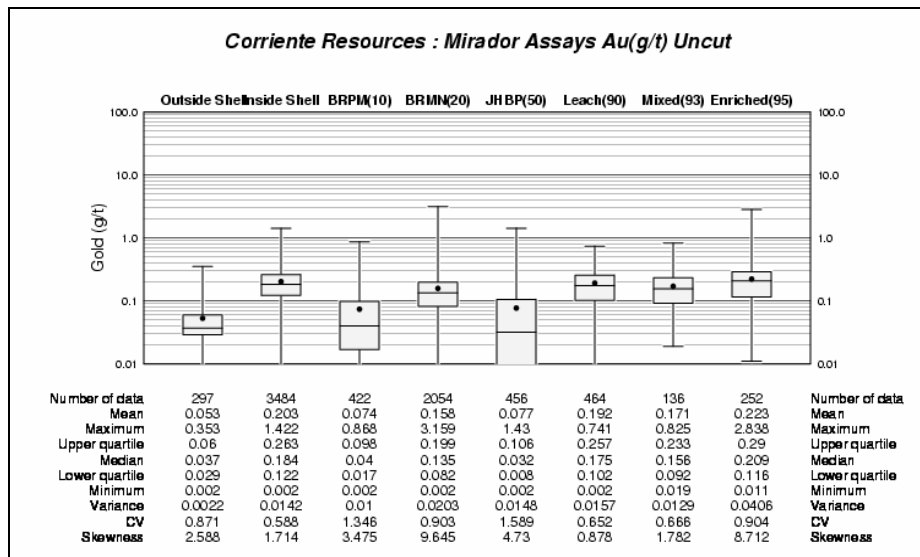


Figure 3-21: Box Plot of Au Assay Results by Litho Code





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Figure 3-22: 6 m Composite Box Plot, Cu %

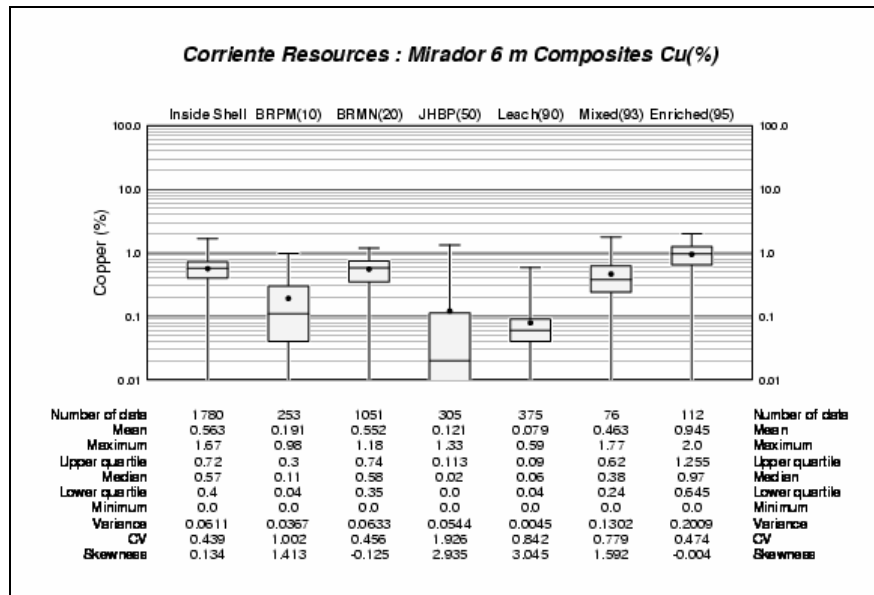
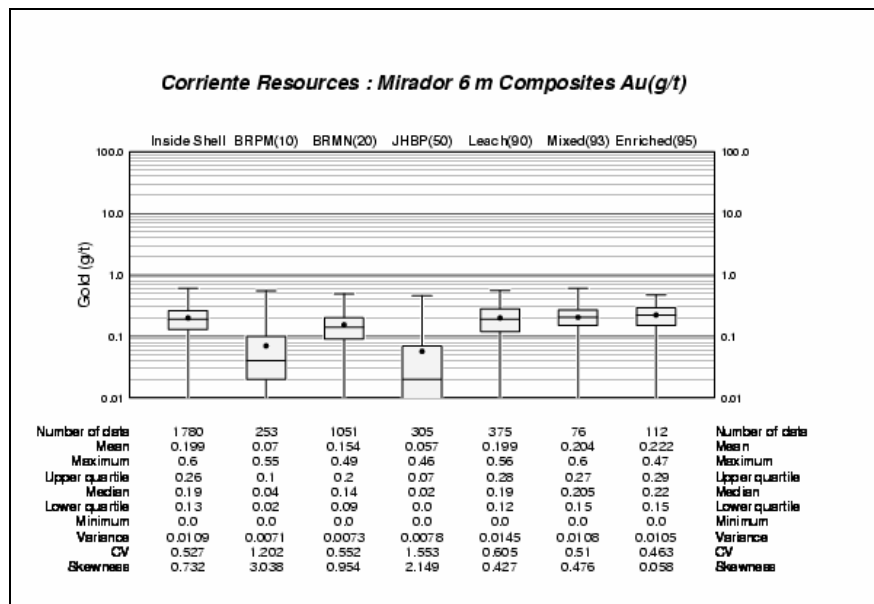


Figure 3-23: 6 m Composite Box Plot, Au g/t



Contact plots were generated to explore the relationship between grade and grade shell codes. The plots are constructed with software that searches for data with a given code, and then searches for data with another specified code and bins the grades according to the distance between the two points. This allows for a graphical representation of the grade trends away from a “contact.” If average grades are



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reasonably similar near a boundary and then diverge as the distance from the contact increases, the particular boundary should probably not be used as a grade constraint. In fact, if a hard boundary is imposed where grades tend to change gradually, grades may be overestimated on one side of the boundary and underestimated on the opposite side. If there is a distinct difference in the averages across a boundary, there is evidence that the boundary may be important in constraining the grade estimation.

Contact profiles, or plots, were made for copper and gold across the grade probability shells. The chart for copper is shown in Figure 3-24 and for gold in Figure 3-25. These charts show that the two populations (inside and outside the grade probability shells) have a gradual grade change over a distance of approximately 25 m. Beyond this distance, the grade ranges of the two populations are distinct. Beyond the 25 m buffer, copper grades inside the grade probability shell are just above 0.60% while outside of the shell the grades are below 0.40% Cu. Beyond the 25 m buffer, gold grades inside the grade probability shell are just above 0.225 g/t while outside of the shell the grades are below 0.150 g/t Au.

Figure 3-24: Contact Profile – Inside and Outside of Cu Grade Probability Shell

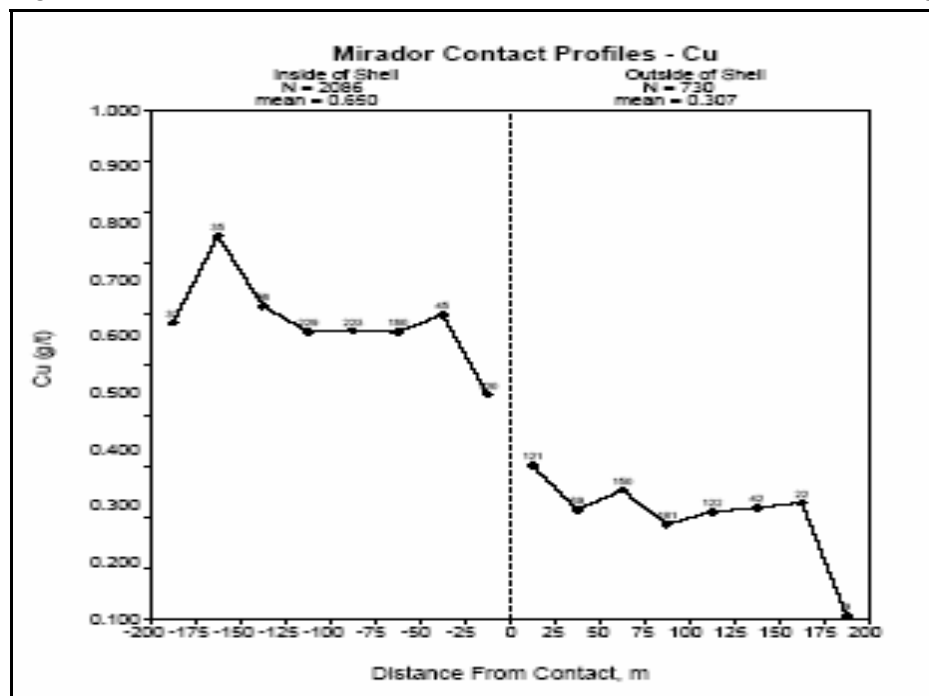
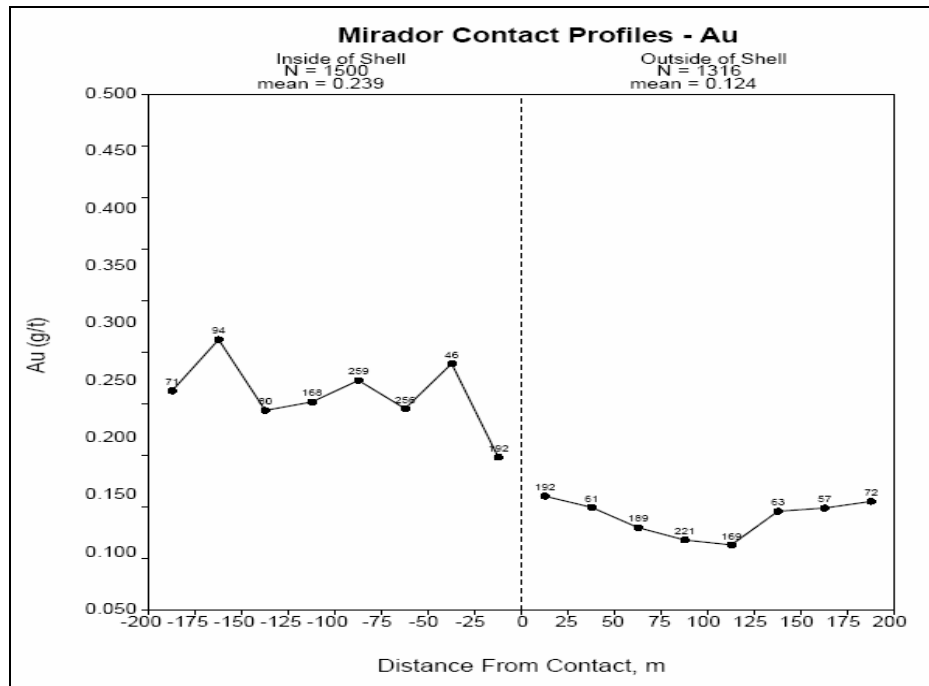




Figure 3-25: Contact Profile – Inside and Outside of Au Grade Probability Shell



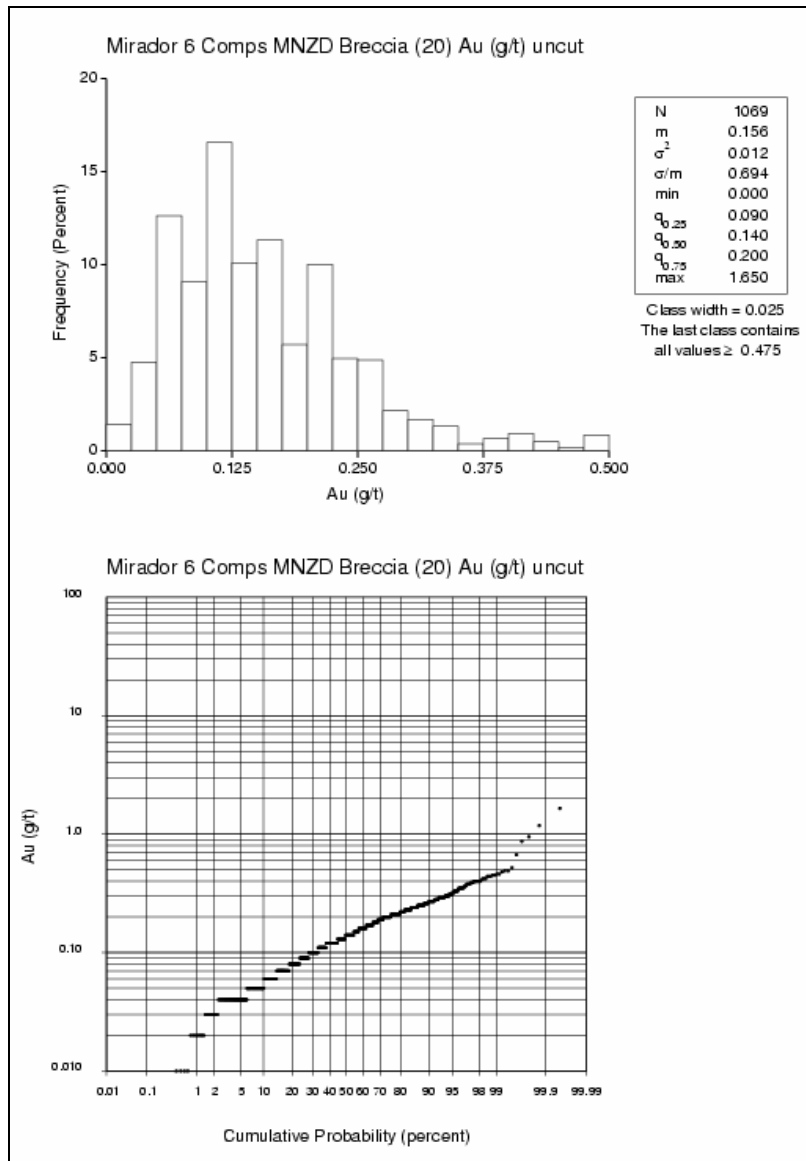
3.10.3 Evaluation of Extreme Grades

Extreme grades were examined for copper and gold composite values, mainly by histograms and probability plots. Generally, the distributions do not indicate a problem with extreme grades for copper. Gold showed extreme grade values (Figure 3-26) where a clear break in the sample trend is observed at 0.60 Au g/t. AMEC implemented a gold grade cap at the break value (0.60 g/t) to the assay data prior to grade interpolation.



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Figure 3-26: Histogram – Probability Plot of 6m Composites, Au g/t



3.10.4 Variography

Variography is the study of the spatial variability of an attribute. AMEC prefers to use a correlogram, rather than the traditional variogram, because it is less sensitive to outliers and is normalized to the variance of data used for a given lag. The correlogram ranges from -1 to +1, although models are usually made over the interval [0,1], where 0 represents no correlation (statistical independence) and 1 represents perfect correlation.



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Correlograms were calculated for copper and gold for all hypogene samples and then for the composites inside the grade probability shells. The approach to correlogram model development is to calculate a relatively large number of sample correlograms in several directions using composite samples. Directional sample correlograms are calculated along horizontal azimuths of 0°, 30°, 60°, 90°, 120°, 150°, 180°, 210°, 240°, 270°, 300° and 330°. For each azimuth sample correlograms are also calculated at a dip of -30, -60 in addition to horizontally. Finally, a correlogram is calculated in the vertical direction. Using the 37 sample correlograms, an algorithm determines the best-fit model. This model consists of a nugget effect; single or two-nested structure variance contributions; ranges for the variance contributions; and the model type (spherical or exponential type). After fitting the variance parameters, the algorithm then fits an ellipsoid to the ranges from the directional models for each structure. The anisotropy in grade variation is given by the two ellipsoids.

Variography was completed for gold and copper on composite data from the main mineralized unit within and outside the grade probability shells. Only hypogene material was investigated (due to paucity of data in the supergene domains). Variogram model parameters and orientation data of rotated variogram axes are shown in Table 3-4. The correlograms are included in this report in Appendix A for gold and copper.

Table 3-4: Variogram Results, ZYZ Rotations and Rotated Variogram Axes

Metal and Structure	Structure	Nugget Effect	Sill	Axis	Orientation (Azimuth/Dip)	Gemcom Rotation Type	Rotation Angles	Range
Cu	1 st	0.100	0.585	X	221/0	-	-	43
		-	-	Y	131/35	ZYZ	-42, -36, -89	72
		-	-	Z	312/54	-	-	150
	2 nd	-	0.315	X	65/9	-	-	301
		-	-	Y	336/-1	ZYZ	16, -10, 8	126
		-	-	Z	254/81	-	-	200
Au	1 st	0.190	0.365	X	160/23	-	-	47
		-	-	Y	44/46	ZYZ	2, -55, -61	25
		-	-	Z	268/35	-	-	22
	2 nd	-	0.445	X	94/2	-	-	79
		-	-	Y	4/0	ZYZ	7, -2, -11	244
		-	-	Z	263/88	-	-	200

Note: Models are spherical. ZYZ rotation is right-hand-rule

The copper correlograms showed ENE-WSW trending steeply dipping structures while the gold correlograms showed NS trending, subvertical structures. These observations match the property geology where observed trends are in similar orientations.



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The nugget effects, or random variation components of spatial variation, tend to be moderate. Copper variograms generally have nugget effects of 0.10 or was modelled as 10% of the total variability. The gold variograms had a nugget effect of 0.19 or accounted for 19% of the variability.

3.10.5 Grade Model and Interpolation Plan

The Mirador grade block model limits are (in UTM coordinates) 783500 to 786005 east, 9603000 to 9605505 north and 2000 to 200 m elevation. Block size is 15 m in the easting and northing directions and 12 m in height.

The geology and grade solids were used to assign rock codes to the block model as listed in Table 3-5. Blocks above the topographic surface were tagged as air and blocks outside of the Outer Shell were tagged as waste. A percent model was used to control the volume of blocks inside the solids. The data analyses demonstrated that the post mineral dykes should be treated as separate domains, representing unmineralized material and were treated as waste.

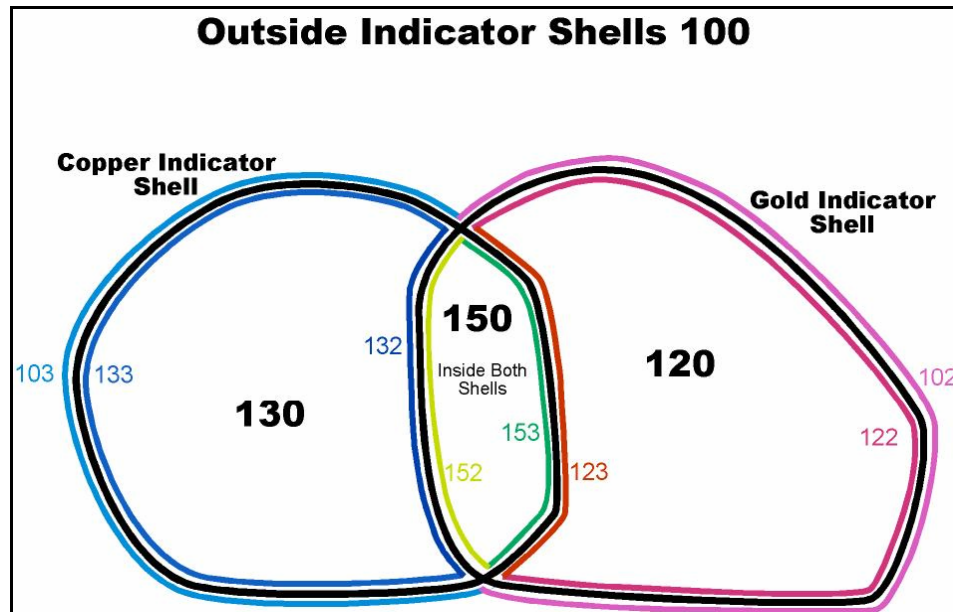
Table 3-5: Block Model Rock Codes

Rock Codes		Zone Name
90	Leached	
93	Mixed	
95	Enriched	
100 series	Outside grade probability shells but inside krige shell	
120 series	Au grade probability shell	
130 series	Cu grade probability shell	
150 series	Inside both Au and Cu grade probability shells	
900	Waste	
999	Air	

The grade probability shells were used as soft boundaries for interpolation purposes. This means that the blocks around the boundary of the indicator shells were able to see the composites from holes found both outside and inside the shells. A FORTRAN program was written to identify blocks that were within 30 m of the shell boundaries and they were given a lithology code based on their position relative to the shell boundaries. A schematic diagram of the lithology coding system is presented in Figure 3-27.



Figure 3-27: Lithology Codes for Sulphide Blocks



Lithology codes in the block model were used to make certain block selections for interpolation runs. Table 3-6 lists the lithology codes used to control the blocks used for interpolation runs. These block selections were necessary as composite files were made to contain the composites within these shell selections. Composite files were also made that contain all of the composites for the blocks coded in the proximal wall areas of the grade probability shells to be allowed to use data on either side of the shell boundaries.

Table 3-6: Lithology Groups for Grade Interpolation Runs

Interpolation Run	Lithology Codes Used
Inside Cu grade probability shell	130, 132, 133, 150, 152, 153
Outside Cu grade probability shell	100, 102, 103, 120, 122, 123
Inside Au grade probability shell	120, 122, 123, 150, 152, 153
Outside Au grade probability shell	100, 102, 103, 130, 132, 133

Values for copper, capped and uncapped gold, and bulk density were interpolated into the block model using ordinary kriging (OK), inverse distance weighting to the eighth power (ID8) and the nearest-neighbour (NN) methods. Interpolation was restricted by rock code. A minimum of five and maximum of 20 composites were used, with an additional parameter for maximum composites per drill hole set to four. This forces the interpolation to require a minimum of two holes within the search ellipse before a grade value can be assigned to a block.



The search ellipsoid for material in the hypogene was oriented as defined by the variography. Search ranges comprised 400 m along the long axis, orientated at 045°, 200 m down the dip direction (at -80° to the NW) and 300 m vertically. Block discretization is 4 x 4 x 3.

The interpolation of grade for copper and gold in the supergene blanket area used an isotropic search to find samples within a certain radius of a block centre. The search radii were 150 m in the X and Y directions and 50 m in the Z direction. A minimum of two holes within the search ellipse was required for grade to be interpolated into a block. The data were interpolated using inverse distance weighting to the second power (ID2). A minimum of three composites and maximum of 12 composites were required.

Bulk Density Interpolation

There are 665 bulk density values for the MNZD (hypogene) rock type with an overall average of 2.64 g/m³. This is a sufficient quantity of data for interpolation of the values into the block model. Inverse distance to the third power ID3 was used with sufficient ranges to fill all the blocks. A minimum of three and maximum of eight values were used for each block.

The supergene rock types contained only limited density data: 21 Leached zone, 22 Mixed zone and 46 Enriched zone measurements. These values were averaged and the result was tagged into the blocks for the appropriate rock type. The average bulk density values for the Leached, Mixed, and Enriched zones are 2.42, 2.47, and 2.51, respectively.

The distribution and summary statistics for the bulk density values by rock type are presented in Appendix A.

3.10.6 Model Validation

Visual Inspection

AMEC completed a review of the Mirador resource block model. The model was checked for proper coding of drill hole intervals and block model cells. The coding was found to be properly done. Gold and copper grade interpolation was examined relative to drill hole composite values by inspecting the sections and plans. The checks showed good agreement between drill hole composite values and model cell values. Examples of representative sections containing block model copper and gold grades, drill hole composites and grade probability shell outlines are included in Appendix A.



Global Means

AMEC also checked the block model estimates for global bias by comparing the average copper and gold grades from the model ID8 with means from OK and NN estimates. The NN estimator produces a theoretically unbiased estimate of the average value when no cutoff grade is imposed and is a good basis for checking the performance of different estimation methods. Results in Table 3-7 show no evidence of bias in the estimate.

Table 3-7: Comparison of OK, ID, and NN Global Mean Values

Zone	ID8 Cu (%)	OK Cu (%)	NN Cu (%)	ID8 Au (g/t)	OK Au (g/t)	NN Au (g/t)
Hypogene	0.50	0.50	0.50	0.161	0.162	0.160
Supergene						
Leached	0.07	0.07	0.09	0.165	0.170	0.181
Mixed	0.35	0.34	0.36	0.174	0.163	0.191
Enriched	0.86	0.81	0.91	0.213	0.199	0.236

Swath Plots

AMEC also checked for local trends in the grade estimates by plotting the results from the OK, ID, and NN estimate results on easting, northing and elevation swath plots. The OK and ID estimates should be smoother than the NN estimate and the ID should be smoother than the OK estimate. The NN estimate should fluctuate around the ID and OK estimates on the plots. The results for the copper grade inside the grade probability shell are presented in Figures 3-28 to 3-30, and they show close tracking between the three estimates and no local trends. The plots for copper values outside of the grade probability shells and all gold results are found in Appendix A.

Change of Support

The final check on the block model performed by AMEC was to check the model for smoothing through the Discrete Gaussian or Hermitian polynomial change-of-support method described by Journel and Huijbregts (Mining Geostatistics, Academic Press, 1978). The distribution of hypothetical block grades derived by this method is compared to the estimated model grade distribution by means of grade-tonnage curves. The grade-tonnage curves allow comparison of the histograms of the two grade distributions in a format familiar to mining. If the estimation procedure has adequately predicted grades for the selected block size, then the grade-tonnage curves should match closely. If the curves diverge significantly, then there is a problem with the estimated resource.



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Figure 3-28: Eastings Swath Plot – Cu % Inside Cu Grade Probability Shell

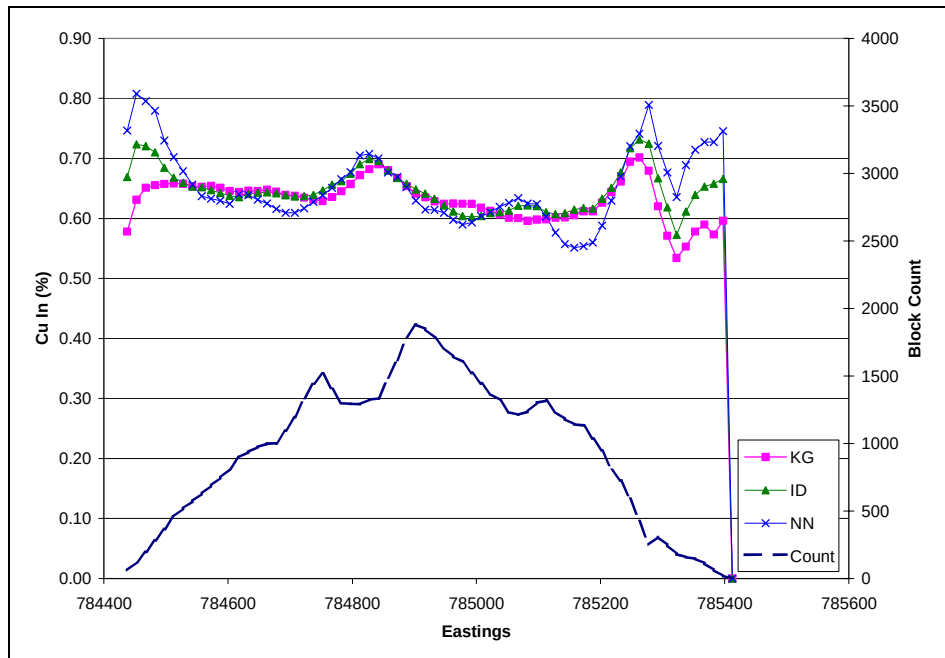


Figure 3-29: Northings Swath Plot – Cu % Inside Cu Grade Probability Shell

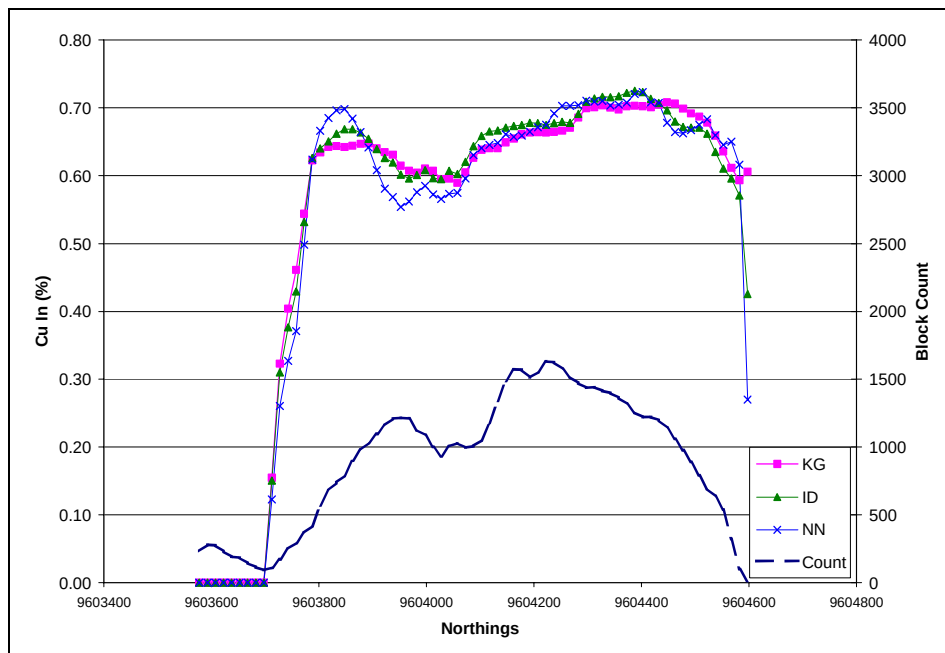
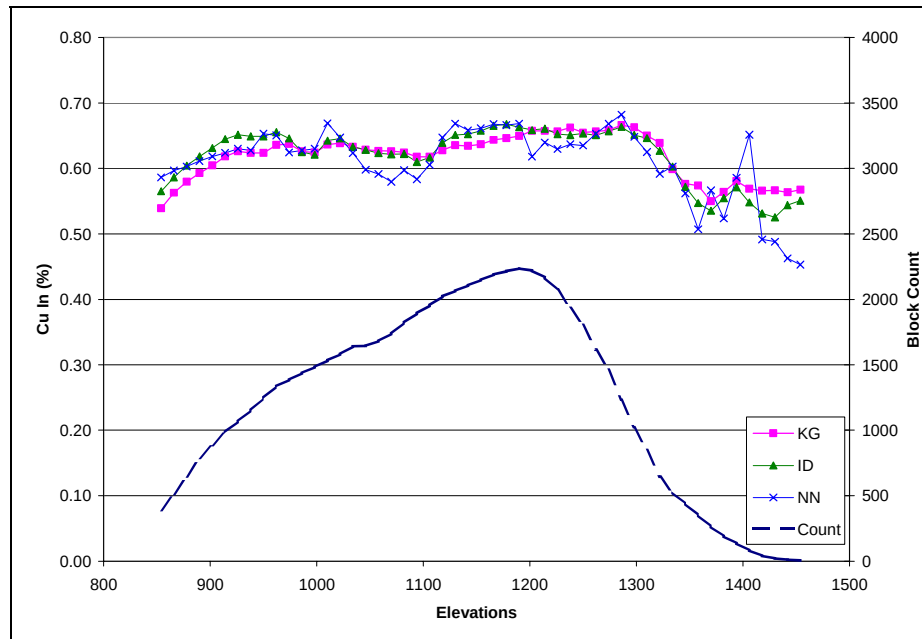




Figure 3-30: Elevations Swath Plot – Cu % Inside Cu Grade Probability Shell



This method uses the “declustered” distribution of composite grades from a nearest neighbour or polygonal model to predict the distribution of grades in blocks. In this case the blocks used in the model are 15 m x 15 m x 12 m. The unadjusted polygonal model assumes much more selectivity for ore and waste than is actually possible in mining practice, since many sample-sized volumes are averaged together within a block. This means that part of the sample-sized volumes in the block may be ore (above the mining cutoff) and part may be waste. Hence, the distribution of the grade of the blocks is not likely to resemble the distribution of grades from composite samples derived from the polygonal estimate. The method assumes that the distribution of the blocks will become more symmetric as the variance of the block distribution is reduced, i.e., as the mining blocks become bigger.

The histogram for the blocks is derived from two calculations:

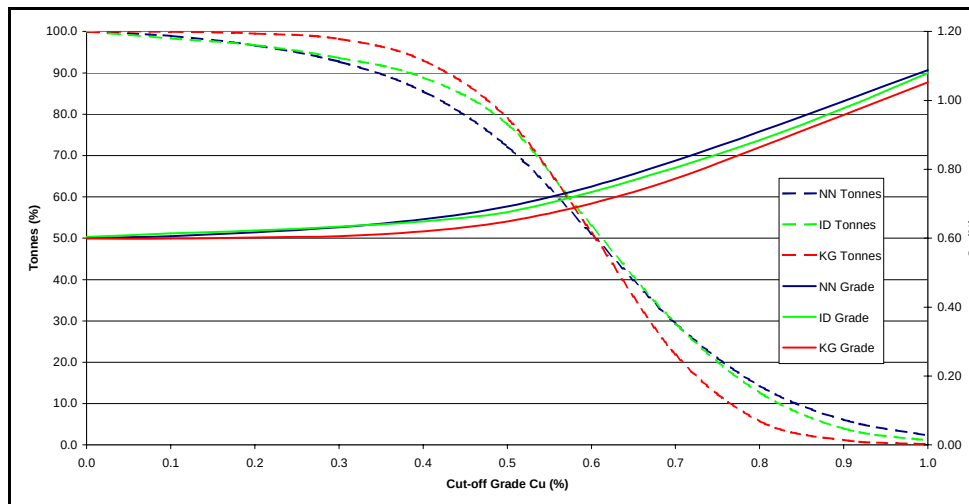
- The block-to-block variance (sometimes referred to in statistics as the between-block variance), which is calculated by subtracting the average value of the variogram within a block from the variance for composite samples (the sill of the variogram)
- The frequency distribution for the composite grades transformed by means of hermite polynomials (Herco; hermite correction) into a less skewed distribution with the same mean as the declustered grade distribution and with the block-to-block variance of the grades.



The distribution of hypothetical block grades derived by the Herco method is then compared to the estimated grade distribution to be validated by means of grade-tonnage curves.

The distribution of calculated 15 m x 15 m x 12 m block grades for copper inside the grade probability shell is shown with dashed lines on the grade-tonnage curves in Figure 3-31. This is the distribution of grades based on 15 m blocks obtained from the change-of-support models. The continuous lines in the figures show the grade-tonnage distribution obtained from the block estimates. The grade-tonnage predictions produced for the model show that grade and tonnage estimates are validated by the change-of-support calculations over the likely range of mining grade cutoff values (0.4% to 0.6% Cu).

Figure 3-31: Change of Support Grade Tonnage Curve, Cu %



3.10.7 Resource Classification and Summary

The mineral resources of the Mirador Project were classified using logic consistent with the CIM definitions referred to in NI 43-101. The project mineral resources were classified into one of two categories; Indicated or Inferred mineral resources. Review of the model relative to the drill hole data, in sections and in plans, showed good geologic and grade continuity in areas where sample spacing is around 100 m. The current drill hole spacing is nominally 100 m with some areas approaching 75 m. Combined with spatial statistical work and investigation of confidence limits in predicting planned annual production work, AMEC decided that blocks covered by this data spacing may be classified as Indicated mineral resources. In the hypogene mineralization two drill holes needed to be within 100 m of the block centre or a single drill hole within 35 m to be classified as Indicated mineral resources. Supergene



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mineralization needed to have two drill holes within 75 m of a block centre or a single drill hole within 35 m. Locally a second pass utilizing a one-hole rule was necessary to prevent a “striping” effect of isolated Inferred mineral resource blocks amongst mostly Indicated mineral resource blocks.

All interpolated blocks that did not meet the criteria for Indicated mineral resources were assigned as Inferred mineral resources if they fell within 200 m of a drill hole composite in the hypogene mineralization or 150 m if in the supergene zone.

Table 3-8 contains the results of the resource estimation for the Mirador deposit as of 23 September 2004. The estimate is made by mineralization type. The resource estimate result for Mirador is being declared using the 0.40% Cu cutoff. For comparative purposes, additional copper grade cutoffs equal to 0.50, 0.55, 0.60, 0.65 and 0.70% were also tabulated and they are included in Table 3-9. The resources are reported to a depth of 850 m elevation, which is approximately 500 m below the surface.

Table 3-8: Mirador Deposit Mineral Resource Summary, 23 September 2004

Zone		Tonnes	Au (g/t)	Cu (%)
0.4 Cu % Cutoff				
<i>Indicated Mineral Resource</i>				
	Mixed	1,300,000	0.23	0.57
	Enriched	6,700,000	0.24	0.99
	MNZD	301,700,000	0.20	0.65
Total	Indicated Hypogene	301,700,000	0.20	0.65
	Indicated Supergene	8,000,000	0.24	0.92
<i>Inferred Mineral Resource</i>				
	Mixed	-	-	-
	Enriched	1,200,000	0.25	0.83
	MNZD	313,900,000	0.17	0.56
Total	Inferred Hypogene	313,900,000	0.17	0.56
	Inferred Supergene	1,200,000	0.25	0.83



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Table 3-9: Mirador Deposit Mineral Resource Summary – Comparative Cutoffs, 23 September 2004

Zone	Tonnes	Au (g/t)	Cu (%)
0.5 Cu % Cutoff (Cu ID)			
<i>Indicated Mineral Resource</i>			
Mixed	800,000	0.25	0.65
Enriched	6,600,000	0.24	1.00
MNZD	254,800,000	0.21	0.68
Total			
Indicated hypogene	254,800,000	0.21	0.68
Indicated supergene	7,400,000	0.24	0.96
<i>Inferred Mineral Resource</i>			
Mixed	-	-	-
Enriched	1,200,000	0.25	0.83
MNZD	215,300,000	0.18	0.62
Total			
Inferred hypogene	215,300,000	0.18	0.62
Inferred supergene	1,200,000	0.25	0.83
0.6 Cu % Cutoff (Cu ID)			
<i>Indicated Mineral Resource</i>			
Mixed	500,000	0.28	0.76
Enriched	6,200,000	0.24	1.03
MNZD	182,400,000	0.22	0.73
Total			
Indicated hypogene	182,400,000	0.22	0.73
Indicated supergene	6,700,000	0.24	1.01
<i>Inferred Mineral Resource</i>			
Mixed	-	-	-
Enriched	900,000	0.22	0.89
MNZD	93,300,000	0.20	0.71
Total			
Inferred hypogene	93,300,000	0.20	0.71
Inferred supergene	900,000	0.22	0.89
0.65 Cu % Cutoff (Cu ID)			
<i>Indicated Mineral Resource</i>			
Mixed	400,000	0.25	0.75
Enriched	6,000,000	0.25	1.05
MNZD	140,600,000	0.23	0.77
Total			
Indicated hypogene	140,600,000	0.23	0.77
Indicated supergene	6,400,000	0.25	1.03
<i>Inferred Mineral Resource</i>			
Mixed	-	-	-
Enriched	800,000	0.25	1.00
MNZD	56,900,000	0.22	0.76
Total			
Inferred hypogene	56,900,000	0.22	0.76
Inferred supergene	800,000	0.25	1.00
0.7 Cu % Cutoff (Cu ID)			
<i>Indicated Mineral Resource</i>			
Mixed	200,000	0.35	1.05
Enriched	5,800,000	0.24	1.05
MNZD	101,600,000	0.24	0.80
Total			
Indicated hypogene	101,600,000	0.24	0.80
Indicated supergene	6,000,000	0.25	1.05
<i>Inferred Mineral Resource</i>			
Mixed	-	-	-
Enriched	800,000	0.25	1.00
MNZD	40,400,000	0.23	0.79
Total			
Inferred hypogene	40,400,000	0.23	0.79
Inferred supergene	-	-	-



3.11 Conclusions and Recommendations

AMEC reviewed the drilling and sampling data from the Mirador Project to obtain a sufficient level of understanding to assess the mineral resource estimate for the Mirador deposit. AMEC's general conclusions from this review are as follows:

- The geology for the Mirador Project is well understood. The Mirador deposit is considered to be a copper-gold porphyry system. The copper and gold mineralization encountered at the Mirador deposit is thought to be a calc-alkaline type porphyry copper system. Supergene zones of enrichment are weakly developed beneath a superficial leach cap. Mineralization at Mirador can be divided into early molybdenum, early copper $\pm$ gold, and late copper-gold stages. Chalcopyrite occurs as mainly irregular masses between clasts in the early breccia and as fine disseminations in a halo zone around the breccia. A late stage series of 1 to 2 cm quartz sulphide stringers transect the deposit and are subvertically emplaced. AMEC believes more angled drill holes should have been drilled due to the presence of various geologic features having sub-vertical contacts (e.g., post-mineral dykes, late quartz-sulphide stringers).
- Of the 91 drill holes included in this resource estimate, 50 were not surveyed for hole deviation (Table 3-1). Corriente surveyed the last 20 drill holes that were completed and should continue to do so with all future drilling campaigns.
- The database used to estimate the mineral resources for the Mirador Project consists of samples from 91 core drill holes drilled by Corriente between 2000 and 2004. The data underwent a 5% check of the assay results to original assay certificates and no errors were found.
- Sample security issues need to be addressed by Corriente where sample numbers should be assigned through the use of standard sample books and not include the hole number, sample depth or give unique identifiers to the QA/QC samples. The sample shipping bags should be secured with security tabs and emplaced in a locked box prior to release from the project site.
- A review of the QA/QC database revealed a paucity of QA/QC samples for the 1st, 2nd, and 3rd drilling programs, so 5% of the samples were selected for reanalysis. These samples were selected by AMEC and were selected randomly to ensure coverage over time and the deposit area. The samples were analyzed with the benefit of SRMs and duplicates. The reanalysis results indicated that the original results were of the appropriate grade for inclusion in the resource estimation.
- The pulp duplicates indicate that the 100 g pulp size currently being submitted for fire assay analysis for gold is too small and any future samples should be increased to a 250 g pulp.



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- The Mirador resource model was developed using industry-accepted practices. AMEC validated the model estimates and found them to reasonably estimate the grade and tonnage for the Mirador deposit.
- The classification applied to the resource estimation at the Mirador deposit adheres to the guidelines set out in the CIM definitions referred to in NI 43-101 into Indicated and Inferred mineral resource categories.



4.0 MINE PLAN

4.1 Introduction

This section of the report describes the mine plan developed by AMEC for the Mirador Copper Project. The complete mine schedule is included in Appendix B and the annual pit plans are included in Appendix D.

The geological model was divided into four copper/gold-bearing zones: leached, mixed, enriched, and mineralized. After resource modelling was completed, these four zones were combined into a single zone per block to reflect the mineability of each block, on a diluted basis. The mine plan treats leached material as saprolite and all other material as sulphide.

Conventional open pit shovel-truck methods will be used for mining. The mine plan is based on a contract mining company providing ore to a conventional copper concentrator to support an average milling rate of 25,000 t/d (9.125 Mt/a). Waste stripping will vary by year, from a maximum of 39,000 t/d in Year 1 down to 7,000 t/d in the final full year. The average waste stripping rate is 20,000 t/d.

Mining occurs in three phases: a starter pit mined for eight years, an intermediate pit mined through nine years, and the ultimate pit mined to the end of the 12 year mine life. Overlaps of the pits occur to balance waste stripping and ore feed grade.

4.2 Open Pit Optimization

To run the pit optimization, an NSR model was created within Gemcom®. The calculated NSR values are based on copper values only, and do not include gold. Gold was included in some optimizations but was not seen as a dominant economic factor. Only model blocks carrying ore grades within the Measured and Indicated category were assigned a revenue value. Blocks carrying grades in the Inferred category were treated as waste, and referenced as Waste-Inferred in this report. This model was then imported into the Whittle 4X® mine optimization software package. A series of pit shells were created at various net smelter return (NSR) values utilizing a varying revenue factor between 0.6 and 0.84, these pit shells were then analyzed to select the best one for pit design. The best pit shell had the maximum NPV at the stated criteria. The Whittle® Milawa algorithm was used to generate production forecasts at throughputs of 20,000 t/d, 25,000 t/d, 30,000 t/d and 35,000 t/d, and assumed three pit phases. The shells were loaded back into the mine planning package and evaluated for accessibility and overall pit footprint. For the purposes of



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this study, Corriente chose 25,000 t/d, based on company objectives at this time, and design work completed to date. The discount rate used in all Whittle work was 10%.

The parameters used to create the NSR and optimization models are summarized in Table 4-1.

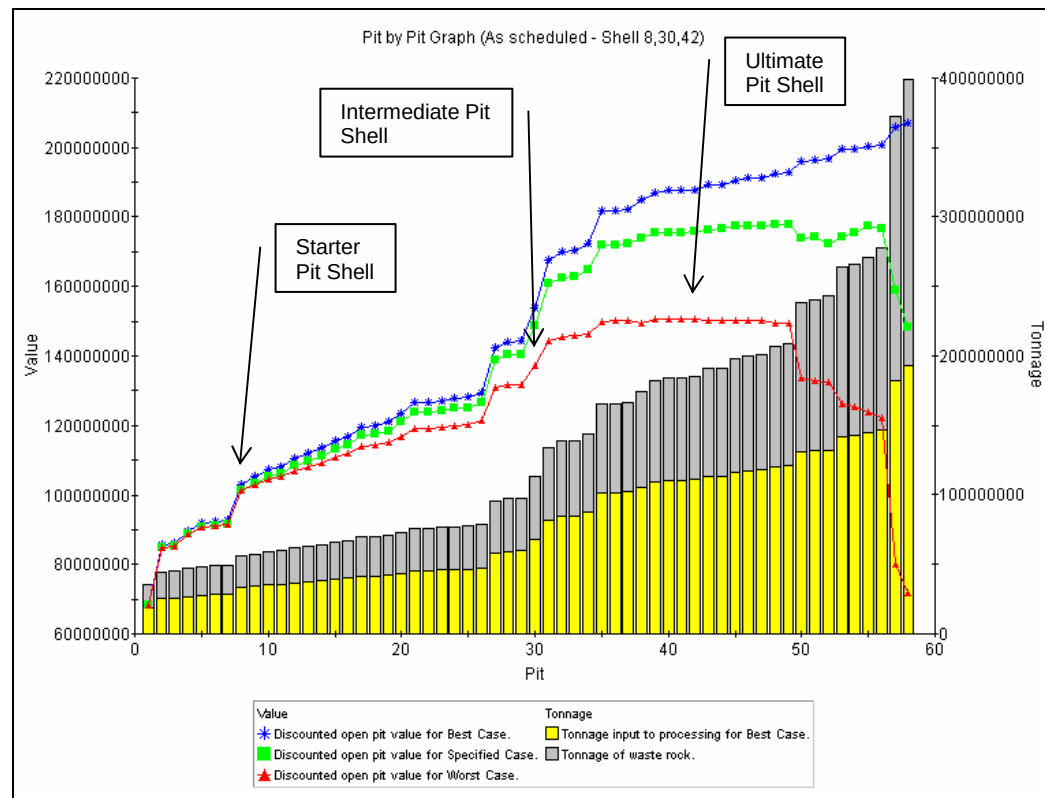
The Whittle forecast output for the series of generated pit shells is shown in Figure 4-1. The Worst Case mining forecast simulated mining the shell in a top-to-bottom manner, a scenario that is unlikely to occur in even the most challenging environment. The Best Case mining forecast mines in a shell-by-shell manner, which is far too optimistic to occur and due to mining width, is not feasible. The Specified Case allows mining each specified pit shell. In this case, three shells were chosen to represent the starter, intermediate and ultimate pit shells. These pit shells are then forecast using the Milawa algorithm, which allows for simultaneous mining of multiple phases in an optimized manner. It is expected that the Specified Case can be achieved with proper scheduling, and therefore significantly greater gross value than the Worst Case.

Table 4-1: Pit Optimization and NSR Parameters

Parameter	Unit	Base Case
<i>Block Size</i>		
X	(m)	15
Y	(m)	15
Z – oxide	(m)	12
<i>Bulk Density</i>		
Ore & Waste	(t/m ³)	Variable from model
<i>Metal Price</i>		
Copper	(\$/lb)	1.00
<i>Primary Process Recovery</i>		
	(%)	Cu 91.4
<i>By-product Credits</i>		
		N/A
<i>Concentrate Grade</i>		
Copper	(%)	29.8
<i>Smelter Terms</i>		
Treatment Charge	(\$/dmt)	Cu 75
Amount Payable	(%)	96.5
Refining	(\$/lb)	0.075
<i>Transportation (all products)</i>		
Road freight & storage	(\$/wmt)	81.62
Ocean Freight	(\$/wmt)	Incl. above
Moisture	(%)	8.00
Losses	(%)	0.25
<i>Mining Recovery</i>		
	(%)	100
<i>Preliminary Costs</i>		
Process and G&A	(\$/t milled)	3.90
Ore mining @ 25,000 t/d	(\$/t mined)	0.89
Waste mining	(\$/t mined)	0.89
<i>Pit Slope Angles</i>		
	degrees	35 to 42
<i>Maximum Slope Height</i>		
	m	500



Figure 4-1: Whittle Nested Shell Output



Final operating costs were slightly different than the preliminary costs used for pit optimization. Mining costs increased to \$1.14/t mined based on contract mining, and process and G&A costs decreased to \$3.51/t milled. This change in costs would have the effect of lowering the marginal cutoff grade.

The optimization shape selected is well below the revenue factor 1 shape because of the restrictions placed on the overall wall height. Internal to the pit there is potential to pick up marginal tonnes between the \$3.90/t used for optimization and the final estimated process and G&A operating costs of \$3.51/t.

4.3 Open Pit Design

4.3.1 Geotechnical Evaluation

The geotechnical investigation report is provided in Appendix E. The following summarizes the key points that were used in the determination of the pit slopes and the cost estimates for the study.



For feasibility purposes, the following inter-ramp slopes are recommended by zone (where 0° due North, 90° due East):

- Zone I (330° to 60°) 35°
- Zone II (60° to 150°) 42°
- Zone III (150° to 180°) 40°
- Zone IV (180° to 270°).....38°
- Zone V (270° to 330°).....40°

The slopes suggested above are preliminary estimates only and may be modified with additional geotechnical drilling, during subsequent design work and trial excavations during the early stages of mining.

4.3.2 Design Parameters and Summary

The parameters used in the pit design, including the geotechnical data described above, are as follows:

- bench height, single benching 12 m
- berm width 8 m
- haul roads and pit ramps
 - total width allowance 22 m
 - running surface 18 m
 - berms and ditches..... 4 m
 - berm access from ramp bottom only
 - maximum grade 10%

The three pit phases share a final wall on the southwest of the pit. Stripping will start on the south highwall and progress towards the north. Waste from the pit will be hauled to the waste dump or used for haul road construction and maintenance. Waste dump construction will isolate saprolitic (leached) waste so that it is contained within a sulphide waste berm.

The primary road access to the pit will be via a new road, which climbs from the Wawayme River past the waste dumps to the northeast corner of the pit, with the crusher located to the northwest of the pit. All haul roads, both within and outside the pit, are designed to a minimum width of 22 m, except the lowest segments of the pit, which are 18 m (single-lane access).



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4.4 Pit Design Tonnages

The resource was originally modelled as four distinct rock types of variable grade and thickness. In order to calculate reserves with dilution, the rock types were combined into a single block model. Pit optimization was then used to determine the pit extents. A pit design, and three phases were developed, and internal to the designs the resources were tabulated, utilizing a declining cutoff grade.

The resources within the Mirador pits are summarized in Table 4-2.

Table 4-2: In Pit Resources by Material Type

	Tonnes (000)	Cutoff NSR (US\$/t)	Cutoff Cu (%)	NSR (US\$/t)	Cu (%)	Au (g/t)
<i>Indicated</i>						
Phase 1	38,294	6.67	0.44	9.74	0.751	0.241
Phase 2	29,807	5.84	0.41	8.23	0.635	0.211
Phase 3	42,526	5.46	0.35	8.00	0.617	0.198
Total	110,627	-	-	8.66	0.668	0.216
<i>Inferred</i>						
Phase 1	2,407	6.67	0.44	8.86	0.683	0.228
Phase 2	2,330	5.84	0.41	6.91	0.533	0.191
Phase 3	5,057	5.46	0.35	5.94	0.458	0.166
Total	9,794	-	-	6.89	0.531	0.187

Note: Cu cutoff shown as a NSR equivalent; actual NSR cutoff was completed in Whittle

4.5 Waste Material Handling

Waste material was categorized into the three types listed in Table 4-3. Material that was within the leached area was considered saprolite. Inferred waste was material above copper cutoff grade, but within the Inferred resource category. All other material was tabulated as sulphide waste.

Table 4-3: Waste Quantities and Type (t x 000)

Waste Type	Phase 1	Phase 2	Phase 3	Total
Sulphide waste	12,300	9,549	23,726	45,575
Inferred waste	2,407	2,330	5,057	9,794
Saprolitic (leached) waste	21,129	6,143	8,531	35,803
Total	35,836	18,022	37,314	91,172



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4.6 Mine Plan

4.6.1 Summary

Mining at Mirador will be undertaken by a local contractor with sufficient financial strength and mining experience to handle the requirements of the Mirador project. Corriente has negotiated with Mamut Andino S.A. of Guayaquil (Mamut) to provide a budgetary estimate of mining costs using the design basis and mine design provided by AMEC. The mine plan and budget estimate used in this feasibility study was developed jointly by Corriente and Mamut and is described below. A copy of Mamut's budget pricing is included in Appendix B.

Mining will be by conventional open pit truck and shovel methods. The primary mining fleet will consist of eight 140 t trucks and two 19.9 m³ front-end loaders. Support equipment will include bulldozers, graders, and excavators to maintain the surfaces of the roads, dumps and operating benches and the water collection system at the pit rim and in-pit.

Mill feed will be hauled to a gyratory crusher at the west side of the pit.

The annual mine production and mill feed forecast for the three mining phases is summarized in Table 4-4. Figure 4-2 provides a graphical representation of the forecast tonnages, and Figures 4-3 and 4-4 illustrate the feed tonnes and grades for the three phases.

Table 4-4: Mine Production Forecast

Year	Total Mill Feed (t x 000)	Grade			Total Waste (t x 000)	S.R.
		NSR (\$/t)	Cu (%)	Au (g/t)		
-1*	2,100	8.570	0.661	0.223	14,175	6.75
1	7,025	9.622	0.743	0.227	14,344	2.04
2	9,125	9.880	0.762	0.228	7,481	0.82
3	9,125	8.846	0.683	0.237	7,132	0.78
4	9,125	9.066	0.700	0.241	5,974	0.65
5	9,125	8.850	0.683	0.234	5,903	0.65
6	9,125	8.736	0.674	0.230	5,907	0.65
7	9,125	8.453	0.653	0.238	5,908	0.65
8	9,125	7.962	0.615	0.198	5,911	0.65
9	9,125	8.383	0.647	0.200	5,877	0.64
10	9,125	8.242	0.636	0.196	5,898	0.65
11	9,125	7.880	0.608	0.184	3,811	0.42
12	9,125	8.178	0.631	0.187	2,628	0.29
13	1,127	9.387	0.724	0.203	223	0.20
Total	110,627	8.662	0.668	0.216	91,172	0.82

*stockpiled ore for plant feed in Year 1



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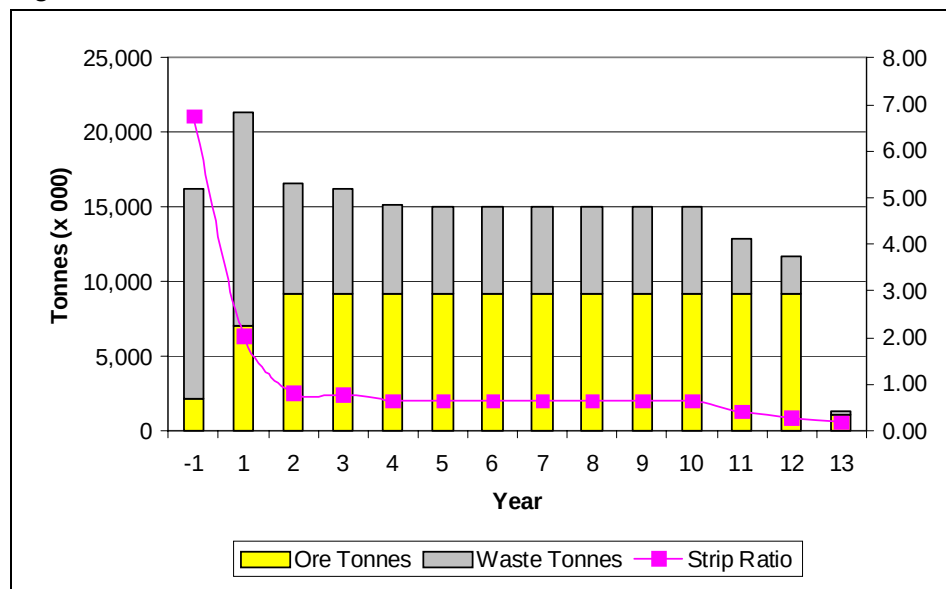
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4.6.2 Pit Sequencing

The deposit is to be mined in three phases. The starter pit is centred on a higher-grade section in the west portion of the deposit. Each subsequent pushback, or phase, adds additional depth at this high-grade portion, and expands laterally to the east. The west wall is shared between the starter (Phase 1), intermediate (Phase 2) and ultimate (Phase 3) pits.

The purpose of each of these phases is to accomplish different goals. The starter pit achieves a higher average grade (0.751% Cu) at a relatively high strip ratio (0.94). The intermediate pit provides a medium average grade mill feed (0.635% Cu) at a low strip ratio (0.60), and phase three pit provides a low average feed grade (0.615% Cu) and an average strip ratio of 0.82.

Figure 4-2: Mine Production Forecast





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Figure 4-3: Mill Feed Tonnage by Pit Phase

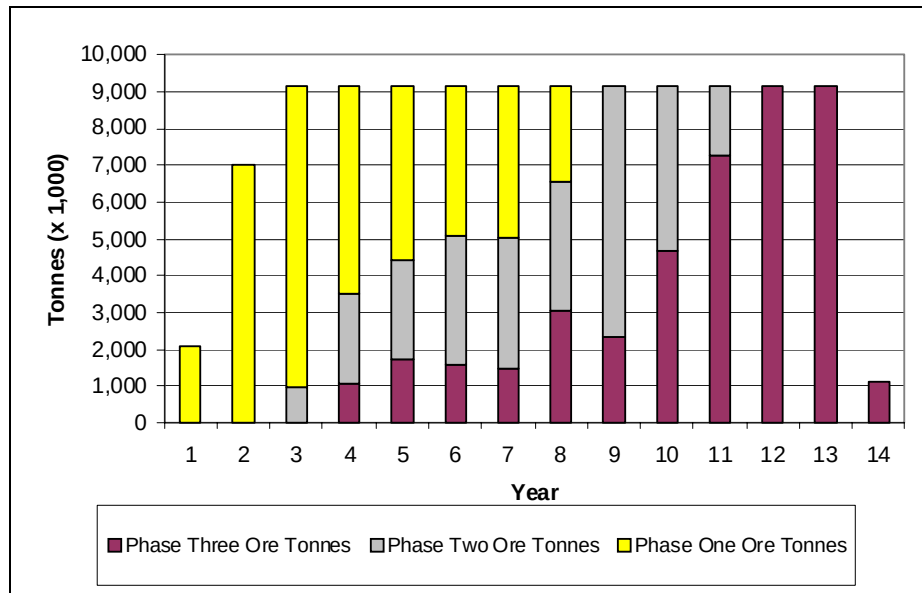
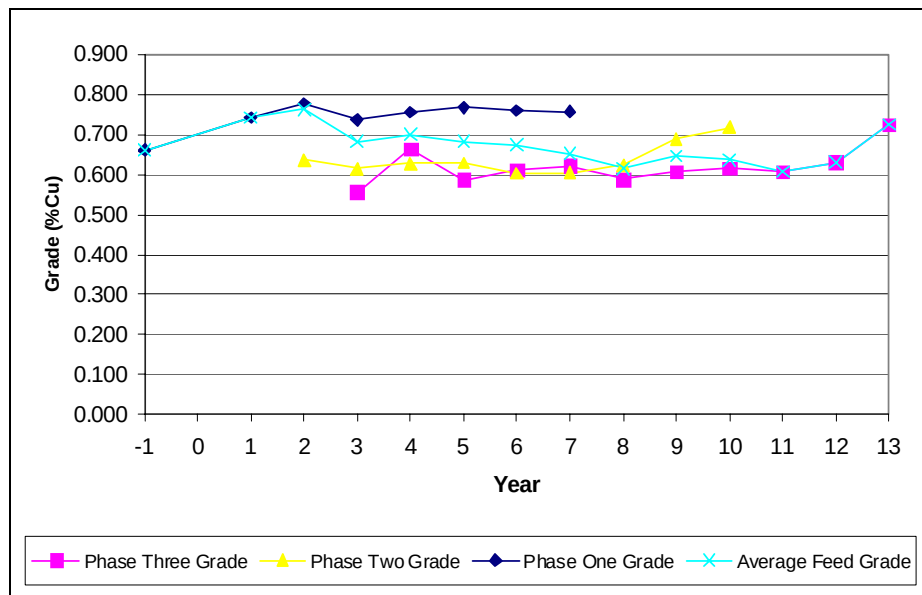


Figure 4-4: Feed Grade by Pit Phase



4.6.3 Pre-production Mine Development

Pre-production mine development is expected to be complete in the year prior to mill start-up. During this period, 14.18 Mt of waste material will be removed and 2.10 Mt of



mill feed stockpiled for plant feed during Year 1. Of the total amount of waste, nearly 70% is classified as saprolitic or leached rock.

A pioneering road will be developed to the 1592 m elevation. Material from above this level (up to 1652 m elevation) will be ripped and dozed to the ramp elevation for loading and hauling. This ramp system will extend down as a temporary in-pit ramp to the 1460 m elevation at the edge of the pit rim. It will then follow topography down to the 1300 m elevation to join the crusher east haul road. This road will utilize as much of the competent in-pit waste rock as possible.

4.6.4 Production Forecast

The mine production forecast is based on mining the pit in three phases. First quarter production of mill feed in Year 1 will be at 100% of the rated plant capacity, but 92% of this feed will be from stockpiles produced in Year -1.

The mine production forecast was prepared on a quarterly basis for the Year 1 schedule. The rest of the forecast is by year. The annual production target is based on maintaining the plant throughput at design capacity. Basic production parameters are as follows:

- 365 operating days per year
- 25,000 t/d of feed to the crusher
- maintain production for minimum 12 years.

4.7 Mining Equipment

4.7.1 Summary

The mine production forecast and general logistical considerations were the driving forces behind fleet selection. Emphasis was placed on using equipment with the flexibility to react quickly to changing conditions during mining operations.

Pre-production mining will have two main priorities. The first is to open up the higher-grade material source to provide continuous plant feed in Year 1. The second is to provide the highwall dewatering which will prevent as much water inflow as possible. This will include berm trenching, horizontal dewatering holes and diversion ditches at the pit contact.

The pioneering work will be done by a contractor fleet and will be designed for single lane traffic with frequent passing turn-outs.



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The required equipment fleet for the mine production period, from Years 1 to 13, is listed in Table 4-5.

Table 4-5: Production Equipment Fleet Requirements

Years	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
203 mm blasthole drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1
19.9 m ³ front-end loader	2	2	2	2	2	2	2	2	2	2	2	2	2	2
140 t haulage truck	7	8	8	6	6	6	7	7	8	8	8	8	8	8
300 kW track dozer	1	1	1	1	1	1	1	1	1	1	1	1	1	1
425 kW track dozer	2	2	2	2	2	2	2	2	2	2	2	2	2	1
160 kW motor grader	2	2	2	2	2	2	2	2	2	2	2	2	2	1
2 m ³ backhoe	2	2	2	2	2	2	2	2	2	2	2	2	2	2

4.7.2 General Operating Parameters

To determine the number of equipment units required for each major fleet, productivities were calculated based on estimated annual operating hours and mechanical availability. The estimate of annual hours available is shown in Table 4-6. To allow for “other” inefficiencies, as indicated in the table, a 50-minute operating hour was applied to all equipment. In addition a truck availability to the loader was applied to the loading units based on the Caterpillar FPC program.

Table 4-6: Annual Hours Available

Schedule	
Calendar days	365 d/a
Scheduled down days	-
Unscheduled down days	-
Available days	365 d/a
Scheduled	24 h/d
Scheduled	8,760 h/a
Fixed Delays	
Shift change (2)	0.5 h/shift
Coffee	0.33 h/shift
Lunch	0.5 h/shift
“Other”	0.67 h/shift
Total	2.00 h/shift
Shifts per day	2 shifts/d
Available	20.00 h/d
Available	7,300 h/a



The estimated mechanical availability of the equipment decreases with hours worked, as shown in Table 4-7. Equipment availability of no less than 92% is based on the premise of contracting the maintenance of the equipment fleet to the local Caterpillar dealer with a condition of the contract being a constant mechanical availability for each equipment fleet of 92%.

Table 4-7: Equipment Life and Mechanical Availability

	Life Hours	Mechanical Availability
Blasthole drill	60,000	90
Wheel loader	60,000	92
Haul truck	70,000	92
Track dozer	35,000	92
Motor grader	35,000	92
Excavator	30,000	92

4.7.3 Blasthole Drills

One 203 mm drill will be used for blasthole drilling. This drill size fits with the selected bench height of 12 m. It is not expected that drilling and blasting will be required in the saprolite rock. Blasthole drilling will be critical for proper grade control, and not drilling near the leached/mixed contact zone may result in lost copper. Wall control on the northern highwall is also an important consideration in the selection of a smaller diameter drill system.

Wall control will not include pre-shear drilling as the rock is expected to be very friable. Instead, a buffer row will be drilled and, by using smaller, better distributed explosive charges, it is anticipated that the final wall will remain intact with minor shattering of the joint systems. The blast pattern near the high wall will be the same as the production blast pattern.

Production drilling has been based on the production forecast, estimated drill factors, and calculated productivities. These productivities were calculated for sulphide ore and waste using the factors given in Table 4-8. The theoretical penetration rate was determined by comparing calculations from previous studies, calculations provided by suppliers and Mamut's actual operating experience with these drills in rock with similar density and hardness.

**Table 4-8: Blasthole Drill Productivity**

Production Drilling 203 mm Drill		
Effective penetration rate	35.0	m/h
Drill factor	179.3	t/m
Effective productivity	6,275	t/h

4.7.4 Loading Equipment

Two 19.9 m³ loaders are required for the mine life. Loading productivities were based on the parameters listed in Table 4-9. These parameters were applied to the average specific gravities resulting from the mine production forecast, and overall productivities were calculated. Annual loading productivities are summarized Table 4-10.

Table 4-9: Loading Parameters

Bucket size	19.9	m ³
Bucket fill factor	95	%
Average cycle time	0.5	min
Truck spot time	0.7	min
Truck availability to shovel	based on simulation	
Operating efficiency	83	%
Swell factor	40	%
Moisture	4	%

Table 4-10: Loading Equipment Productivities

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Average specific gravity	2,549	2,551	2,553	2,559	2,561	2,558	2,565	2,574	2,573	2,570	2,576	2,569	2,564	2,560
Theoretical productivity (t/h)	6,744	6,744	6,744	6,744	6,744	6,744	6,744	6,744	6,744	6,744	6,744	6,744	6,744	6,744
Effective productivity (t/h)	3,482	3,482	3,482	3,482	3,482	3,482	3,482	3,482	3,482	3,482	3,482	3,482	3,482	3,482

4.7.5 Haul Trucks

Haul truck sizes will be 140 t for all production. This size was selected to match the loading units and achieve production targets at minimum unit operating costs under design conditions. The fleet size will average eight trucks, over the life of the mine.

Mamut have assumed the actual full capacity of the trucks will be 138 t and may require the use of larger, lightweight truck boxes.



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Mamut calculated a haul profile to the crusher (for ore) and to the dump (for waste) for each bench and each sequence in the mine plan. A sample of these haul profiles was modeled using the Caterpillar FPC Haulage Simulation program to generate average haulage speeds for the different haulage situations. Using the average haulage speeds, a table of truck productivities for each bench and mine sequence was developed. The cycle time factors are shown on Table 4-11.

The productivities were then used to calculate the annual number of truck hours required to move the material from each bench in each sequence according to the production schedule developed by AMEC. It was assumed that the trucks would run on No. 2 diesel fuel, and so no derating factor was applied to account for reduced engine performance due to fuel quality. The average annual productivities for each material type are shown in Table 4-12.

Table 4-11: Haul Truck Cycle Factors

Fixed Cycle Times	Load Time		Dump Time
	Spot	Cycle	
Mill Feed	0.9 min	0.5 min	1.0 min
Waste	0.9 min	0.5 min	1.0 min
<i>Rolling Resistance</i>			
In pit – at shovel	3%		
In pit – on bench	2%		
In pit – ramp	2%		
Haul roads	2%		
Waste dump – ramp	2%		
Waste dump – on dump	2%		
Waste dump – at dozer	2%		
<i>Speed Limits</i>			
	<i>Speed Limit</i>	<i>Average Speed</i>	
Level – loaded	50 km/h	30 km/h	
Level – empty	50 km/h	35 km/h	
Uphill – loaded	Truck limits*	15 km/h	
Uphill – empty	Truck limits*	30 km/h	
Downhill – loaded	20 km/h	18 km/h	
Downhill – empty	20 km/h	20 km/h	
<i>Inefficiency Allowance</i>			
Dump	50 m		
Crusher	50 m		



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Table 4-12: Haul Truck Equipment Productivities

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Average specific gravity	2,549	2,551	2,553	2,559	2,561	2,558	2,565	2,574	2,573	2,570	2,576	2,569	2,564	2,560
Effective productivity (t/h)	318	333	288	345	336	320	291	170	251	252	237	216	189	180

4.7.6 Mine Support Equipment

The following complement of road construction and maintenance equipment will support the mine operations:

- two 160 kW class road graders to maintain mine site roads
- one 300 kW track dozer for dump maintenance, drill site preparation, road building, ditching, bench repair, and shovel cleanup
- two 425 kW track dozers for dump maintenance, drill site preparation, road building, ditching, bench repair, and shovel cleanup
- one 2 m³ excavator to assist in wall cleanup, ditching and maintenance of the water control system.

4.7.7 Ancillary Equipment

The initial ancillary equipment fleet required for mining is shown in Table 4-13. Mechanics trucks and auxiliary mechanical equipment will be provided in the contract with the Caterpillar dealer.

Table 4-13: Ancillary Equipment Fleet

Number	Description	Capacity
1	Fuel/lube truck	20,000 L
1	Blasting loader	n/a
1	Blasters' truck	n/a
1	Utility loader and tire manipulator	n/a
4	Lighting plant	n/a
1	Flat-deck truck	n/a
1	Man bus	40+ person
2	Small dump trucks	20 t
1	Drum vibrator	n/a
1	Water truck	n/a
6	Pickup truck	3/4 t



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The fuel/lube truck will service all tracked equipment in the field. The haulage trucks will be fuelled and lubed as required when running between the pit and the crusher.

The supplier will deliver bulk explosives to the blastholes where the Mamut crews will load the detonators and boosters into the holes and tie in the shots, necessitating the equipment listed above.

The crew cab trucks will be used to transport mine workers between the mine dry and their equipment. The six other pickup trucks will be used by personnel in mine management, engineering, geology, survey, and grade control.

4.7.8 Equipment Purchases

Mamut's purchase schedule for the production equipment is shown in Table 4-14.

Table 4-14: Production Equipment Purchase Schedule

	Year	-1	1	2	3	4	5	6	7*	8*	9*	10*	11	12	13
Blasthole drill	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Wheel loader	2	-	-	-	-	-	-	-	-	-	-	1	-	-	-
Haulage truck	8	-	-	-	-	-	-	-	-	-	2	4	-	-	-
Track dozer	2	-	-	-	-	-	-	-	-	1	1	-	-	-	-
Motor grader	2	-	-	-	-	-	-	-	1	1	-	-	-	-	-
Excavator	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-

* Replacement unit

4.7.9 Fuel Storage

Estimated diesel fuel consumption for the mine equipment is shown in Table 4-15. The fuel storage area is the responsibility of the mining contractor.

Table 4-15: Estimated Annual Diesel Consumption (L x '000)

	Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Fuel	6,278	6,764	5,776	5,596	5,017	5,051	5,317	5,104	5,497	5,774	5,731	5,431	5,823	758	

4.8 Drilling, Blasting, and Explosives

The primary blasthole drills will be diesel-powered rigs capable of drilling 203 mm diameter holes. Drilling requirements were calculated for mill feed and waste. No pre-



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shear blasting will be done, as a buffer row followed by mechanical wall cleanup is expected to yield better wall stability in these rock conditions.

Blasting operations will be affected by several factors, including wall control, groundwater, and heavy rainfall. The dewatering scheme incorporated into the mine plan anticipates that both surface ditching and sumps will be used to remove water ahead of mining in each area.

A number of modified operating procedures will be implemented during the rainy season. These could include minimizing the sleep time for loaded holes; ensuring that cuttings are mounded around the hole collars after loading to prevent water infiltration; and timing the blasting shots during the drier part of the day. These procedural changes would have limited or no impact on operating costs or number of personnel and thus have not been considered in Mamut's budget pricing. Because of the presence of groundwater and anticipated accumulation of precipitation it was assumed that a 50/50 split of ANFO/emulsion would be used for the blasting.

Blasting design and explosives consumption per hole are summarized in Table 4-16.

Table 4-16: Blasting Calculations

Parameter	Unit	Waste	Mill Feed
<i>Blasthole Diameter</i>	(mm)	203	203
<i>Blasting Design</i>			
Bench height	(m)	12	12
Subdrill	(m)	1.4	1.4
Depth	(m)	13.4	13.4
Burden	(m)	8.0	8.0
Spacing	(m)	10.0	10.0
Drill factor	(bcm/hole)	857	857
<i>Explosives</i>			
Stemming	(m)	3.5	3.5
Explosive column length	(m)	10	10
Explosive column mix			
ANFO	(%)	50	50
Emulsion	(%)	50	50
Total Blasting Agent per Hole	(kg)	151	151
Total Unit Consumption	(kg/bcm)	0.391	0.391



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The mining contractor will have the responsibility for operating the explosives plant, ANFO/Emulsion trucks and the delivery and placement of explosives in the blast holes. The blast crew will also place the detonators and boosters and tie in the patterns.

The explosives storage facility/magazine will be located approximately 500 m east of the ultimate pit rim south of the plant haul road.

4.9 Waste Dump Design

The waste dump design is described in more detail in AMEC's waste dump design report in Appendix B. The waste dump must be constructed to contain both saprolite and waste rock stripped from pit development. The mine schedule indicates that up to half of the waste material could be saprolite, although this volume also includes the highly weathered, but more competent saprolite rock. The mine schedule also includes inferred waste. It has been assumed that approximately one third of the total saprolite tonnage indicated in the Table 4-17 is competent saprolitic rock. A combined overall dry density of 2 t/m³ has been assumed for the materials.

Table 4-17: Mine Waste Schedule

Year	Waste - Saprolite (t)	Waste - Sulphide (t)	Waste - Inferred (t)	Total (t)	Waste - Saprolite (m ³)	Waste - Sulphide (m ³)	Waste - Inferred (m ³)	Total (m ³)
Year -1	9,859,990	3,908,009	407,367	14,175,367	3,286,663	3,597,336	203,684	7,087,683
Year 1	9,457,305	4,170,691	716,448	14,344,443	3,152,435	3,661,563	358,224	7,172,222
Year 2	4,322,909	2,315,119	842,225	7,480,252	1,440,970	1,878,044	421,112	3,740,126
Year 3	3,247,706	3,351,567	533,051	7,132,324	1,082,569	2,217,068	266,526	3,566,162
Year 4	2,376,511	3,208,883	387,902	5,973,295	792,170	2,000,526	193,951	2,986,648
Year 5	2,663,302	2,803,991	436,198	5,903,491	887,767	1,845,879	218,099	2,951,746
Year 6	1,706,029	3,430,527	769,786	5,906,342	568,676	1,999,602	384,893	2,953,171
Year 7	442,642	5,052,720	412,686	5,908,048	147,547	2,600,134	206,343	2,954,024
Year 8	552,803	4,556,202	800,978	5,909,983	184,268	2,370,235	400,489	2,954,991
Year 9	999,174	4,096,813	780,746	5,876,733	333,058	2,214,935	390,373	2,938,366
Year 10	175,088	4,057,600	1,665,392	5,898,080	58,363	2,057,981	832,696	2,949,040
Year 11	0	2,233,444	1,578,189	3,811,633	0	1,116,722	789,095	1,905,817
Year 12	0	2,178,355	450,360	2,628,715	0	1,089,177	225,180	1,314,358
Year 13	0	208,698	14,148	222,846	0	104,349	7,074	111,423

There is little information known about the foundation conditions at the waste dump site. Saprolite and saprolitic rock vary in depth throughout the area. Mapping information suggests that reasonably competent rock exists in the creek bottoms, but saprolite is expected to dominate the dump foundations. A streambed walk-through indicated that there is a colluvium layer overlying saprolite, which is underlain by weathered rock.



The saprolite characteristics have been estimated from published information. There could be considerable variations from these estimated values, which may be non-conservative. A summary of some of the key observations in each of the papers reviewed is contained in AMEC's report, in Appendix B. A summary of these key characteristics is presented below:

- engineering properties are influenced by the rock type
- contains micro and macro voids which influence strength and permeability
- relict structure will affect directional shear strength (may be important in foundation)
- excavation and compaction tend to break down particles and change gradational, index and compaction properties
- most in situ saprolites behave like an overconsolidated soil; loads increased beyond a threshold stress will result in increased compressibility and settlement
- pore pressures are usually low due to high void ratios resulting in moderately high permeability; pore pressure reduction under high load are generally rapid
- small scale strength tests may over estimate shear strength
- engineering properties will vary greatly over short distance
- drying and wetting of saprolite will change the compaction and index properties.

Assumptions regarding construction of the dump included the following:

- The dump will be segmented, with the upper one third of the dump volume consisting of saprolite, saprolitic rock and highly weathered rock, and the lower two thirds of the dump consisting of the majority of broken, but competent waste rock from the pit excavation.
- Diversions will be constructed upslope of the dump to divert surface runoff.
- Flow thru drains will be incorporated into the dump design. The drains will be constructed using competent clean rock fill waste from the open pit. The drains will be placed in lifts in the natural drainage valleys within the dump footprint. These will assist in draining the dump and will provide drainage for seepage water and unintercepted surface flow along the existing stream channels.
- The foundations below the competent waste rock are expected to be stripped of all organic materials, loose soils, and saprolite. Construction of the dump will consist of phased stripping and excavating down to a sound foundation within the footprint of the waste rock area. Only a limited amount of stripping should be completed in advance of dump placement as to not expose the foundation to precipitation and



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traffic. It has been assumed that the portion of the dump beneath the saprolite will not be stripped.

- The saprolite and saprolitic rock is assumed to be placed in engineered lifts and compacted. No foundation preparation is anticipated; however, clearing of trees and shrubs should be completed.
- Saprolite and competent waste will be placed starting at the interface between the two materials. The dump material would be placed working upslope with the saprolite and downs slope with rock from the interface. The dump should be constructed using horizontal lifts across the small valleys previously infilled with clean competent waste (flow thru drains). The stability analyses provide some guidelines for limiting heights and widths to maintain stability during the different years of operation.
- Minimum height lifts are expected to be placed over the foundation, and, depending upon response, the lift heights will then be adjusted. Bottom up construction will be necessary for the saprolite portion of the dump. The basic premise is to place and compact a minimal height of saprolite, then start building the waste rock berm downstream – similar, to how a dam would be constructed. The saprolite will be retained by the waste rock, which will act as a stabilizing berm.

Construction sequencing for the waste dump will consist of the following:

- Identification of a hard/competent rock borrow source for the flow through drains and road surfacing in saprolite.
- Construction of the flow through drains in down slope lifts along the valley bottoms.
- Stripping of organics and soft soil from the foundation of the competent waste rock dump. Organics and poor saprolite to be placed in alternating lifts with competent saprolite within the saprolite dump area, (i.e, one horizontal lift of competent saprolite across the dump lift, the next lift upslope should comprise of a 2:1 ratio of competent saprolite to organic waste, again to be placed the width of the dump. The next upslope lift will comprise of competent saprolite only, etc.).
- Placement of saprolite in 0.5 m high compacted lifts within the saprolite dump footprint. Use the observational approach to determine optimal lift heights.
- Sequenced placement of the competent rock below the saprolite toe is required for buttressing and overall stability of the dump. The saprolite foundation should be stripped immediately prior to waste placement. The extent of exposure will need to be determined in the field using the observational approach.



Dump construction for the saprolite will commence at the saprolite/competent rock interface and work up slope, while the competent rock dump area should work down slope as clearing of the foundation has been completed. The bottom edge of the cleared foundation should be free draining.

4.10 Mine Water Management

4.10.1 Pit Slope Depressurization

It is assumed that horizontal depressurization wells be installed to achieve the design wall angles. An estimate of the required drilling has been made and an allowance for materials has been included over the mine life. In addition to this, a tank drill with approximately 2,784 h/a with a 2-person drilling/dewatering crew was included.

4.10.2 Open Pit Dewatering

Water inflows to the open pit will be from two sources, groundwater, and precipitation. There are indications of artesian conditions in the north wall. The horizontal depressurization holes will reduce groundwater flow into the pit.

The precipitation component of the mine dewatering was estimated by AMEC based on the drainage capture area for the pit and the hydrology. Precipitation is expected to be heavy throughout the year. The pumping system will be composed of several diesel powered sump pumps and poly pipe. The system will be moved down the ramp system as the pit deepens. Sumps will be used to collect precipitation and groundwater at the lowest point in the pit for pumping to the sumps below the waste dump. The three-phase, or multiple push-back mining will be beneficial in this heavy precipitation climate. In the event that a deeper phase is temporarily abandoned due to excessive water on the pit floor, another mining phase will be able to sustain mill feed.

4.11 Risks and Opportunities

The feasibility study is based on a set of assumptions or design bases, which are the product of calculations and experience. In the course of developing a base case mine development plan, a number of optimization options present themselves. An understanding of the impact (cost and benefit) of these options will be pursued in the post-appropriation design phase of the project. The potential optimization opportunities to be pursued include:



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- The five geological types (Leached, Mixed, Enriched, Mineralized and Waste) were amalgamated into a combined block model. This allows for an estimate of dilution, as the block ore grade is a weighted-average of all the geological components. If typical blast-hole sampling of 12-m (bench height) composites is used for grade control, this should reflect the estimated grades, which will be achieved in mining. The risks involved in this are that the current models are optimistic in the estimate of thickness of the high-grade enriched ore zone, resulting in an overestimation of metal, particularly in the critical first two years. There are opportunities to improve the mill feed grade by:
 - Implementing a program that uses dozers to push off the leached and mixed rock and mine it as waste, and expose the enriched material for mill feed. This sorting would require face monitoring by geologists, combined with a more accurate blast-hole sampling program. It may be difficult to sort these rock types visually, but this could be examined as an alternative.
 - Careful field mapping of the waste dykes which can then be mined as waste, and will not dilute the mineralized mill feed along the contact with the dyke. It may be possible to map this waste/mineralized contact visually.
- The final schedule includes 9,794,000 t of Inferred resources in the waste. This material represents potential mill feed if the classification can be successfully upgraded from Inferred to Indicated with further exploration drilling. There is no indication of what portions could be upgraded, or if the economics of upgrading this material are positive, and therefore would require further study.
- A better estimate of overburden or saprolite should be established and a solid produced for use in further pit design work. The saprolite will require a flatter slope and wider berms, and will add waste tonnes to the current design. The new overburden solid would allow this flatter development to only occur in the poor saprolite.
- Further geotechnical investigation should be done to refine the slope design. Any steepening of the slope will have benefits to the overall operation.
- Evaluate the impact on highwall stability of utilizing small, closer spaced blast holes with less concussive effect (lower powder per delay). If the highwall can be steepened by a few degrees using more controlled blasting methods the total waste moved may be reduced.
- Review the impact of the lower operating costs and resulting increase of in-pit contained marginal material on the project economics.
- Review the waste dump sequencing to allow waste from the upper benches to be hauled to an adjacent, out-of-pit waste dump immediately to the east of the pit. This dump will reduce the haulage distance for the upper benches and increase the operational safety of the mine.



5.0 METALLURGY AND PROCESS DESIGN

5.1 Metallurgical Testwork

5.1.1 Summary

A significant amount of metallurgical testwork has been undertaken on mineralized samples from the Mirador porphyry copper deposit since 2001. SGS Lakefield Research (Lakefield), in Lakefield, Ontario carried out the main program of feasibility testing between December 2003 and September 2004. The groups responsible historically for the metallurgical testing aspects of the project are summarized below.

Geomet S.A., Santiago, Chile (May 2001)

- scoping batch rougher tests.

Resource Development Inc. (RDI), Co, USA (May 2002)

- scoping batch rougher tests.

Sumitomo Metal Mining Co, Limited, Japan (Jul 2003)

- scoping batch rougher and cleaner tests
- concentrate chemical and mineralogical analysis
- composite bond ball work index (BWI).

SGS Lakefield Research Limited, Lakefield, Ontario, Canada (Dec 2003 – Sept 2004)

- feasibility bench-scale flotation test program (batch and locked cycle flowsheet development and locked cycle variability testing)
- comminution testing (BWI, RWI, CWI, Ai and JK and SMC drop-weight testing) and modelling (JKSimMet)
- QEM*SEM characterization of composite samples for variability testing
- concentrate characterization and dewatering.

G&T Metallurgical Services Limited, Kamloops, BC, Canada (Apr – Sept 2004)

- supporting bench-scale flotation testwork
- mineralogical modal analysis.



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MinnovEx Technologies Inc, Toronto, Ontario, Canada (Jul 2004)

- comminution testing (SPI) and modelling (CEET).

In 2001 Geomet S.A. conducted a scoping rougher flotation test, on behalf of Billiton Chile, on an unidentified sample (Muestra 2) from Mirador.

In April 2002, Resource Development Inc. (RDI) conducted three batch rougher flotation tests also on unidentified samples.

In June 2003, Sumitomo Metal Mining Co., Ltd (Sumitomo) conducted an independent scoping level metallurgical program on five selected drill core samples. This testing included batch rougher and cleaner flotation, mineralogical and chemical analysis of concentrates, and a bond work index determination. The liberation characteristics of the ore were also investigated. AMEC reviewed this work and found it to be done to industry standard.

Overall, the results of the Geomet, RDI, and Sumitomo testwork showed the samples tested had relatively simple metallurgy and favourable commercial concentration potential. The subsequent follow-up feasibility work by Lakefield on 3,000 kg of split diamond drill core from eighteen drill holes across the ore body, and at various depths, has confirmed their conclusions.

The feasibility metallurgical testwork carried out by Lakefield was done under the direction of AMEC. Lakefield also provided samples to MinnovEx and G&T Metallurgical Services (G&T) to conduct SPI grindability testwork, and mineralogical and flotation quality control testwork respectively. Lakefield and MinnovEx also conducted grinding circuit evaluations using their JKSimMet and Comminution Economic Evaluation Tool (CEET) simulation models respectively.

Lakefield's testing was conducted in two main phases; the reports Flowsheet Development (December 2004 to March 2004), Report No.1, and Mapping and Recovery Variability (April to September 2004), Report No.2., are included in Appendix C:

Four master composites were produced from the core samples for an initial flowsheet and design criteria development program. This indicated the mill flow sheet for Mirador will be a conventional copper-gold porphyry flowsheet, with relatively coarse primary SAG and ball mill grinding to about 150 µm followed by copper rougher flotation, concentrate regrind to 30 µm, and cleaner flotation and dewatering. Metallurgical testing and mineralogical quantitative modal liberation analysis,



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conducted by G&T, supported the selection of the primary grind and regrind parameters.

A recovery and mineralogical variability mapping program completed during the third quarter of 2004 subsequently confirmed that the metallurgy and mineralogy of the ore body is quite simple and homogenous, and the samples tested responded consistently well to the conventional flowsheet and reagent scheme selected. Over 44 variability sub-composite samples were produced from seventeen drill holes and tested by hole and depth. Each sample was subjected to chemical and QemSCAN (Quantitative Evaluation of Mineralogy by Scanning Electron Microscopy) mineralogical analysis, grindability testing, and locked cycle flotation. Locked cycle concentrates were subjected to chemical, pyroforicity, and dewatering testing.

Chemical analysis of the head samples indicated a range of copper grades from 0.20% to 1.07%, with average overall grade of 0.67%. Gold grades ranged from 0.05 g/t to 0.43 g/t with an average value of 0.22 g/t.

Concentrates produced are predicted to average 29.2% Cu at a recovery of 91%. The average gold grade and recovery was 5.2 g/t and 47.2%, respectively. A gold behaviour model developed from the flotation test data suggests gold tracks chalcopyrite, pyrite, and gangue, with near equal weighting throughout the process. There is good reconciliation between the test gold recovery data and that predicted by quantitative mineralogy.

Chemical analysis of the individual locked cycle concentrate products indicated that no significant deleterious penalty element impurities were present and this is in good agreement with mineralogical mapping. Concentrate thickening and filtration testwork was conducted. The concentrates settled rapidly and no dewatering problems were identified. Pyroforicity results indicated the concentrate is not expected to be self-heating.

Grindability tests were conducted on the sub-composite intervals of core from individual drill holes. Two dedicated whole core geotechnical and comminution holes were also drilled and used for additional grinding testwork, including Bond Work and Abrasion Indices, JK drop-weight and MinnovEx SPI testing.

Most of the ore in the pit falls geologically in an alteration zone of intense gypsum depletion. This is indicated by low RQD data and poor rock quality observed in drill core boxes. Comparative Bond low energy impact (CWI) and drop-weight test data also indicates the +150 mm ore lumps will break relatively easily at low-energy, but that the resulting reduction may be small. On this basis, it is reasonable to assume the



SAG mill feed granulometry will be relatively finer than the copper porphyry industry average.

JK and SPI testing data ranked the samples in the medium range of resistance to impact breakage for SAG milling. The ore exhibits low to moderate abrasivity. The average Bond ball mill work index is about 14.5 kWh/t and ranks the ore in hardness to ball milling as moderately soft relative to other copper porphyry ores in Lakefield's industry database and with relatively low variability. The JKTech drop weight and SPI test SAG mill parameters, and ball mill work indices, were used in JKSimMet and CEET simulation software models to confirm the grinding circuit design basis, and there was good agreement between both approaches.

5.1.2 Ore Description

The Zamora batholith, which hosts the known porphyry mineralization of the region, forms the wall rocks of the Mirador porphyry system.

Within the mineralized zone, the intrusion comprises mainly granodiorite, with some leucogranite dykes along the west and southwest margins. The core of the intrusion is characterized by a circular area of breccia composed of clasts of the host porphyry and granite, and early mineral quartz-stockwork fragments. Copper grades within the breccia range from very weak to slightly below average for the deposit, depending on the amount of mineralized rock incorporated.

An early, hornblende-feldspar porphyry also intrudes the Zamora rocks as north-south discontinuous dykes. These dykes, ranging in age from syn- to post-copper mineralization, are more numerous and thickest along the southeast and northwest margins of the mineralization. Post-mineralization, hydrothermal breccia dykes are associated with the porphyry dykes and are more dominant along the southeast margin of the mineralization. In the mine plan, post-mineralized dykes are treated as domains of unmineralized waste material.

The copper and gold mineralization encountered at the Mirador deposit display many of the characteristics of calc-alkaline type porphyry copper systems. Mineralogically, the ore is quite simple. Copper is hosted predominantly, as chalcopyrite. Minor amounts of secondary copper mineralization occur, mainly as an overlying blanket zone of weakly developed supergene enrichment, of about 0 m to 22 m thickness, beneath a superficial leach cap. The leached material is not considered ore.

Mineralization at Mirador can be divided into early molybdenum, early copper $\pm$ gold, and late copper-gold stages. Molybdenum is associated with early quartz veining. Both copper-gold events are sulphide dominated. Chalcopyrite occurs as mainly



irregular masses between clasts in the early breccia (with some disseminated) and as fine disseminations in a halo zone around the breccia within the Zamora granite. The syn- to post-mineral porphyry dykes are variably and weakly mineralized by the late copper-gold disseminations and sparse sulphide veins.

Silicates are predominantly quartz, feldspars and micas, the three groups constituting 95.4% of the non-sulphide gangue, while clays (2.1%) and carbonates (1%) comprise the remainder.

Chalcopyrite occurs as mainly irregular masses between clasts in the early breccia and as fine disseminations in a halo zone around the breccia within the Zamora granite. The chalcopyrite grain size is quite consistent, averaging about 40 µm. The secondary copper mineral grain size is finer, averaging about 19 µm. Pyrite occurs widely throughout the deposit, but at quite low levels, and the pyrite to chalcopyrite ratio, at about 3:1 is not at a problematic level. Pyrite is much coarser, averaging 98 µm.

A late stage series of 1 cm to 2 cm quartz sulphide stringers transect the project. They are subvertically emplaced but their strike is unknown. These veinlets are sulphide rich and have elevated gold and zinc values.

The porphyry and the granite were impregnated with anhydrite stringers during the chalcopyrite-mineralizing phase. Meteoric fluids percolated down from the surface and hydrated the anhydrite. The hydration of anhydrite created gypsum and a corresponding 50% volume increase that shattered the rock. Gypsum is soluble and has leached from the rock leaving weakly cemented to open fractures. RQD values measured from core also indicate that rock quality is very low, the average being 38. The poor rock quality migrated downward in the deposit area as a hydration front. The base of this front corresponds approximately to the pit bottom in the mine plan and consequently rock quality is pervasively low throughout the deposit. Away from the hydration front, and below the base of the pit, the porphyry and the granite are less fractured and the anhydrite fracture filling is unaltered and the rock becomes more competent.

The Mirador porphyry system exhibits well-defined alteration zoning, with a large (approximately 4 km²), quartz-sericite alteration zone covering a large part of the wall rock and quartz diorite porphyry. This alteration has overprinted a potassic alteration whose remnants are small secondary biotite nuclei mainly in the quartz monzonite.

In geological structure the nugget effects, or random variation components of spatial variation for copper and gold, tend to be moderate. Copper variograms were modelled as 10% of the total variability and gold variograms accounted for 19% of the variability.



This is also supported by the metallurgical mapping program, which in addition to showing the mineralogy was somewhat homogeneous, also indicated reasonable grade continuity with relatively low variation between samples.

5.1.3 Metallurgical Samples

Sample Origin

Exploration work on the Mirador Project to date has included the completion of 24,350 m of core drilling in 91 diamond drill holes arranged in a grid on approximately 75 m to 100 m centres. Most drill holes at Mirador were drilled vertically using standard core tubes at sizes of NTVV (5.2 cm ø) and TVV (4.8 cm ø).

A preliminary review of the drill hole data, in sections and in plans, showed relatively good geologic and grade continuity across the deposit. Copper grades had a relatively uniform distribution with few extreme grades. Gold grades were more variable and generally decline with depth. Subsequent resource modelling by AMEC has confirmed this conclusion.

In late 2003 and early 2004 a total of about 3,000 kg of split diamond drill core were collected from 20 drill holes (about 22% of all the drill holes) and shipped to SGS Lakefield Research (Lakefield) in Lakefield Ontario, for feasibility metallurgical testing. The author completed a site visit to the Mirador Project between the 10th and 19th of November 2003 to develop and supervise the metallurgical sampling program. In addition, two whole core metallurgical holes were drilled and collected specifically for comminution and geotechnical testwork.

The resources are reported to a depth of 850 m elevation, which is approximately 500 m below the surface. Preliminary open pit planning work, conducted as part of an ongoing feasibility study, showed potential open pit bottoms to be around 1000 m elevation, 150 m above the deepest interpolated grade blocks. The entire length of all twenty drill holes used in metallurgical testing below the leached rock cap and above the pit bottom was sampled. No leached rock type was included the samples as this is regarded as waste in the mine plan. It was also demonstrated that post mineral dykes could be treated as separate domains, representing unmineralized material and these were also treated as waste. Therefore, dyke material was not included in the metallurgical program. This should be reviewed if a larger scale pit operation and higher bench heights are evaluated in the future resulting in less selective mining. However, the volume of potential dyke dilution material in the deposit appears relatively low (<10%) compared to the overall resource.



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A plan of the metallurgical holes sampled is presented in Figure 5-1. Overall, AMEC believe this represents a reasonable spatial distribution of the ore types and expected metallurgy across the deposit.

Split-core samples from eighteen holes were delivered to Lakefield. These holes are summarized in Table 5-1 and were selected to represent a good cross-section of the granodiorite and breccia zones in the ore body.

Table 5-1: Metallurgical Drill Holes

Granodiorite	Breccia
M06	M42
M44	M52
M46	M69
M49	M70
M55	M20
M59	M68
M60	M12
M49	M27
M51	M33

Two dedicated whole core geotechnical and comminution holes (M85 and M86) were also drilled and used for additional grinding testwork.

Within the zone of supergene-copper enrichment the sample length was two meters. Sample lengths through the zone of primary mineralization are three meters. The individual hole sample intervals were combined into composite samples according to the rock type (granodiorite and breccia) and ore type corresponding to depth; near surface enriched supergene (0 m to 22 m), top (mixed chalcocite 15% and primary chalcopyrite), and middle and bottom (both dominantly primary sulphide chalcopyrite).

Initial Evaluation of Rejects

An initial evaluation of the potential use of assay rejects for metallurgical testwork was done. Four separate composites were prepared with assay rejects and core from the same holes and intervals in order to evaluate the relative metallurgical responses. Comparisons of the flotation results are presented in Figure 5-2. In all four cases, the assay laboratory rejects responded relatively poorly to flotation, and reject copper recoveries were about 2 and 15% lower than those on freshly crushed drill core. Further, reagent (most notably lime) consumption was much higher for the assay laboratory rejects. This indicated that the reject samples had been compromised due to oxidation of the finely ground material. Based on these results, the assay rejects



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were deemed not suitable for use in the flotation program. All further testwork was conducted on core. Intermediate stockpiling of lower grade ore for future treatment is not considered in the current mine plan. If this is evaluated in the future the potential impact of oxidation effects on recoveries of this material should be considered.

Figure 5-1: Metallurgical Drill Hole Collar Location and Pit Outline Plan

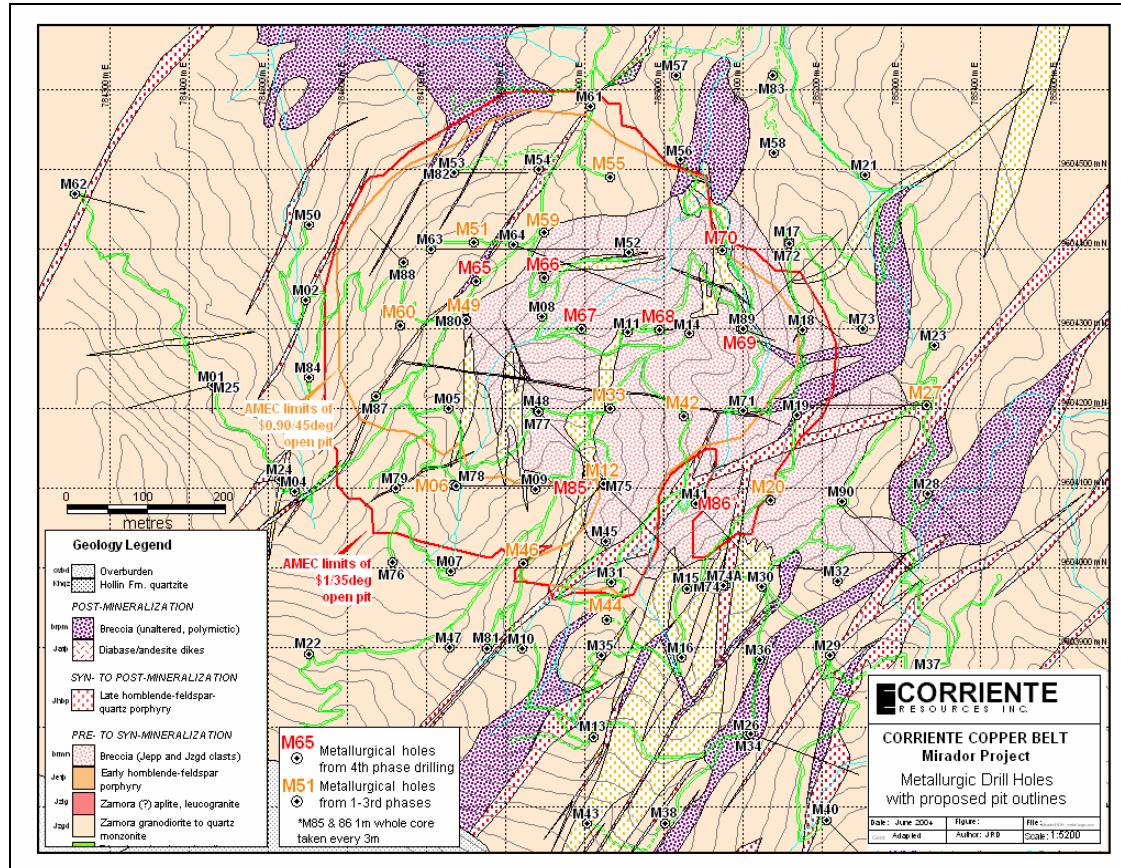
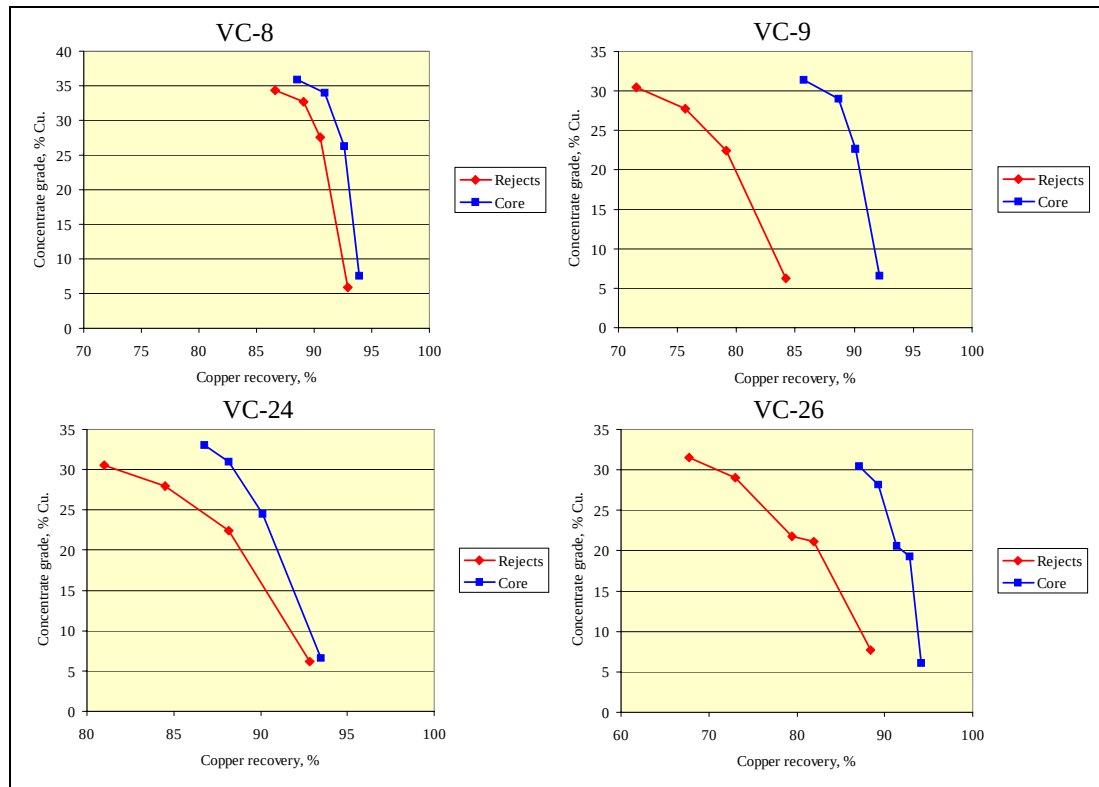




Figure 5-2: Cu Grade vs. Recovery from Flotation of Drill Core and Assay Rejects



Sample Preparation

Full details of the sample preparation and protocols are provided in Lakefield's reports. A flowchart summarizing the sample preparation procedure is presented in Figure 5-3.

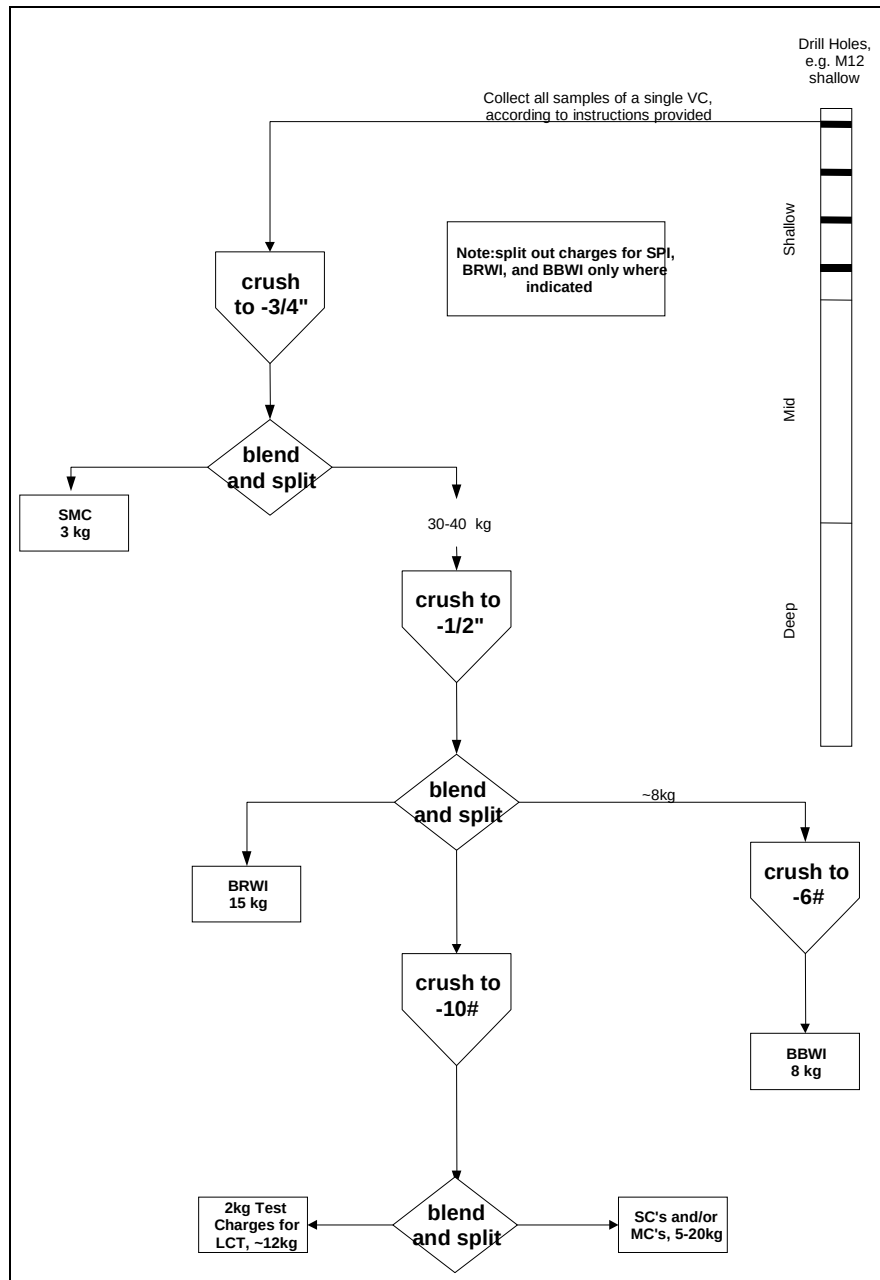
Eighteen split-core drill holes and two whole core hole intervals were delivered to Lakefield for metallurgical and grind testing respectively.

The metallurgical samples were subdivided into 44 variability composites for the mapping variability program as detailed in Table 5-2. Each variability composite contained samples from a single hole. The samples comprising each Variability Composite (VC) were crushed to $-\frac{3}{4}$ " and blended. A 2 kg to 3 kg sample was split out from each VC for SAG Power Index (SPI) testing at MinnovEx. In addition, coarse particles were removed for SAG Mill Comminution (SMC) drop weight testing with the test reject and any unused sample being returned to the blended composite. Each composite was then crushed to 100% passing 6 mesh. A specified weight from each VC was split out to form the Super Composites, as detailed in Table 5-3. Where enough VC sample remained, an additional 8 kg sample was cut for Bond Ball Work Index testing. The remaining samples were bagged and freezer-stored.



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Figure 5-3: Sample Preparation Flowchart





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Table 5-2: Metallurgical Variability Composites

Ore Type/Depth	Zone	Composite	Drill hole	Hole Depth (m)		(kg) Weight	Core assays	
				From	To		Au, ppm	Cu, %
Surface Supergene	granodiorite	VC2	M06	8	42	33.2	0.21	0.82
	breccia	VC18	M42	22	51	30.89	0.31	0.70
	breccia	VC29	M52	16	51	33.73	0.16	0.83
	breccia	Surface Comp	M69	0	23	34.00	0.17	0.56
	breccia		M70	0	26			
Shallow Mixed	granodiorite	VC3	M06	42	66	31.25	0.17	0.50
	breccia	VC8	M20	19	71	55.155	0.39	0.67
	breccia	VC19	M42	51	78	30.28	0.20	0.68
	granodiorite	VC22	M44	36	91	33.80	0.32	0.69
	granodiorite	VC23	M46	39	111	38.46	0.25	0.71
	granodiorite	VC24	M49	45	150	29.37	0.29	0.84
	breccia	VC30	M52	51	102	36.00	0.23	0.68
	granodiorite	VC33	M55	33	85	33.52	0.30	0.70
	granodiorite	VC36	M59	48	109	34.57	0.24	1.02
	granodiorite	VC40	M60	66	131	39.85	0.28	1.03
	breccia	VC44	M68	0	102	33.00	0.16	0.61
	breccia	VC47	M69	23	101	26.00	0.21	0.56
	breccia	VC50	M70	26	97	24.00	0.15	0.54
Mid Primary	granodiorite	VC4	M06	66	111	28.63	0.23	0.65
	breccia	VC5	M12	38	115	42.54	0.19	0.67
	breccia	VC6	M12	115	211	61.75	0.27	0.66
	breccia	VC9	M27	124	205	30.05	0.21	0.76
	breccia	VC11	M33	113	164	31.08	0.21	0.65
	breccia	VC12	M33	164	212	31.01	0.18	0.66
	breccia	VC20/21	M42	78	156	86.35	0.11	0.36
	granodiorite	VC25	M49	150	234	29.78	0.32	0.93
	granodiorite	VC27	M51	64	163	43.29	0.40	0.95
	breccia	VC31	M52	102	159	37.18	0.16	0.63
	granodiorite	VC34	M55	85	136	33.96	0.25	0.62
	granodiorite	VC35	M55	136	181	32.02	0.22	0.67
	granodiorite	VC37	M59	109	172	33.68	0.28	0.97
	granodiorite	VC41	M60	131	191	27.93	0.24	0.74
	breccia	VC45	M68	102	198	32.00	0.16	0.44
	breccia	VC48	M69	101	176	25.00	0.14	0.60
	breccia	VC51	M70	97	170	24.00	0.09	0.22
Deep Primary	breccia	VC10	M27	205	271	25.79	0.13	0.86
	breccia	VC13	M33	212	275	40.36	0.19	0.64
	granodiorite	VC26	M49	234	324	36.61	0.19	0.62
	granodiorite	VC28	M51	163	274	52.85	0.39	0.83
	breccia	VC32	M52	159	228	41.74	0.13	0.60
	granodiorite	VC38/39	M59	172	301	69.30	0.14	0.79
	granodiorite	VC42/43	M60	191	311	63.30	0.18	0.70
	breccia	VC46	M68	198	315	39.00	0.15	0.60
	breccia	VC49	M69	176	269	34.00	0.15	0.69
	breccia	VC52	M70	170	249	24.00	0.06	0.19
Ave.	-	-	-	-	-	-	0.21	0.68
Max.	-	-	-	-	-	-	0.40	1.03
Min.	-	-	-	-	-	-	0.06	0.19
SD	-	-	-	-	-	-	0.08	0.18



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The samples set aside for the Super Composites (SC) were blended to form the four composites used for the initial flowsheet development program. Each SC was crushed to 100% passing 10 mesh and rotary split into 2 kg charges and a reject fraction.

The two whole core grindability holes were combined into 20 m intervals.

Table 5-3: Metallurgical Super Composites

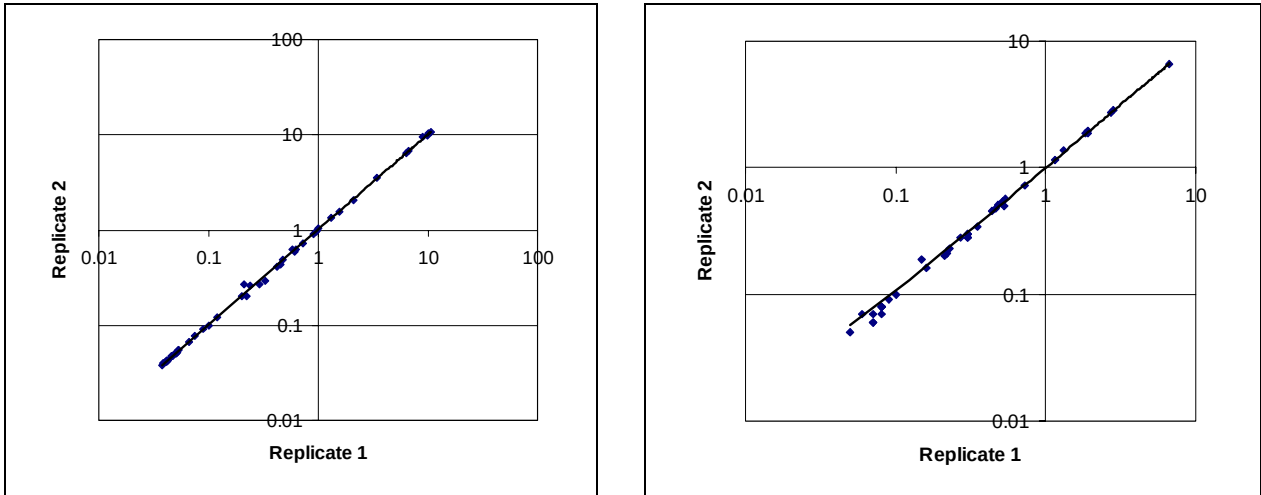
Super Composite	Ore Type/ Location	Variability Composite	VC Weight (kg)
SC-1	Breccia Top	VC8	15
		VC29	3
		VC30	10
		VC9	17
		VC11	5
SC-2	Granodiorite Top	-	-
		VC2	7
		VC3	2
		VC22	15
		VC33	10
		VC4	5
SC-3	Breccia Deep	VC34	11
		-	-
		VC12	5
		VC31	5
		VC32	16
		VC10	13
SC-4	Granodiorite Deep	VC13	11
		-	-
		VC24	15
		VC25	15
		VC35	8
		VC26	15

Quality Control

A replicate assay quality control programme was conducted as an ongoing evaluation of the assay data obtained during this project. The results are plotted in Figure 5-4 below. The replicate assay programme, determining to deviation in assay due to procedural variances in the laboratory, shows excellent assay repeatability. Relative assay deviations are typically 1% to 3%. Typical variability in these programmes on copper porphyry ores are 3% to 4%.



Figure 5-4: Results from Replicate Assay QC Programme



Copper: Assay Relative Deviations: >1% (fusion XRF), 1.2% (fusion XRF), 0.1-1%, 3.2%, <0.1% (AA), 1.6%

Gold: Assay Relative Deviation: 2.0%.

5.1.4 Sample Characterization

Chemical Analysis

Chemical analysis was conducted on the head samples from each of the SC and VC samples. Table 5-4 summarizes the results of this analysis.

In general, silver and molybdenum grades are low and no significant concentrate penalty elements were identified. Super composite grades are in reasonable agreement with the average mine grade. Generally low cyanide soluble copper levels indicate dominant primary copper speciation and this is consistent with the observed mineralogy.

Rock Analysis

Whole rock analysis of the SC samples is shown in Table 5-5. The composition is consistent with the granodioritic nature of the mineralized rock.



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Table 5-4: Chemical Analysis of SC and VC Samples

Method/Element	Super Composite No.				Variability Composites			
	SC 1	SC 2	SC 3	SC 4	Ave.	Max.	Min.	SD
<i>Fire Assay, g/t</i>								
Au	0.28	0.32	0.18	0.30	0.22	0.43	0.05	0.08
Au, replicate2	-	-	-	0.25	-	-	-	-
Au, replicate3	-	-	-	0.22	-	-	-	-
Ag	1.4	< 0.5	< 0.5	0.9	-	-	-	-
<i>XRF, %</i>								
Cu	0.76	0.64	0.64	0.72	0.67	1.07	0.20	0.17
Cu, replicate2	0.71	0.67	0.63	0.77	-	-	-	-
Cu, replicate3	0.70	0.68	0.57	0.68	-	-	-	-
Cu, reassay	-	-	0.56	-	-	-	-	-
Mo	0.008	0.001	0.003	0.005	-	-	-	-
<i>Cu Sequential %</i>								
H ₂ SO ₄	-	-	-	-	0.03	0.09	0.00	0.02
NaCN	-	-	-	-	0.08	0.33	0.01	0.07
Aqua Regia	-	-	-	-	0.49	0.86	0.14	0.14
<i>Cold Vapour AA, g/t</i>								
Hg	< 0.3	< 0.3	< 0.3	< 0.3	-	-	-	-
<i>ICP, g/t</i>								
Ag	-	-	-	-	4	35	<2	7
Al	72,000	75,000	73,000	68,000	68,477	79,000	45,000	6,170
As	< 60	< 60	< 60	< 60	78	180	33	53
Ba	1,000	1,200	1,100	1,400	1159	2,700	750	331
Be	0.52	0.52	0.52	0.44	-	<0.50	0.3	-
Bi	< 20	< 20	< 20	< 20	88	160	24	68
Ca	7,400	6,100	7,000	4,200	6619	25,000	140	5,036
Cd	< 2	< 2	< 2	< 2	3	5	2	2
Co	19	11	9	16	12	21	8	3
Cr	72	82	15	19	38	83	6	19
Cu	-	-	-	-	6,259	10,000	1,600	1,640
Fe	46,000	33,000	30,000	39,000	36,773	58,000	17,000	8,781
K	40,000	38,000	43,000	47,000	38,227	54,000	26,000	5,976
Li	< 40	< 40	< 40	< 40	7	9	5	2
Mg	7,300	7,700	6,400	6,000	6,345	8,800	3,400	1,358
Mn	530	410	330	320	445	1,000	84	204
Mo	-	-	-	-	29	60	5	14
Na	13,000	9,700	8,200	4,100	8,036	20,000	1,200	5,020
Ni	262	< 20	< 20	< 20	46	180	21	44
P	430	310	340	310	356	600	140	99
Pb	< 40	42	< 40	41	90	300	23	73
Sb	< 20	< 20	< 20	< 20	48	59	37	11
Se	< 60	< 60	< 60	< 60	<30	<60	<30	-
Sn	< 40	< 40	< 40	< 40	<20	<40	<30	-
Sr	170	130	140	120	142	310	60	64
Ti	2,100	2,400	2,200	2,100	2,139	2,500	1,500	218
Tl	< 60	< 60	< 60	< 60	<30	<60	<30	-
V	100	110	120	100	99	130	67	16
Y	5.8	5.4	4.4	4.7	6	8.7	3.1	1
Zn	200	140	130	180	181	860	28	204



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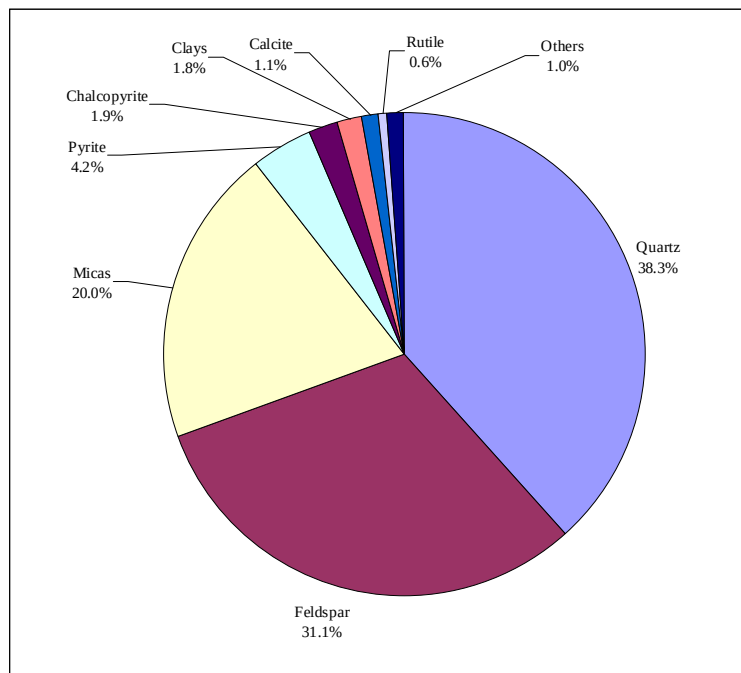
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Table 5-5: SC Whole Rock Analysis

Whole Rock Analysis	Super Composite No.			
	SC 1	SC 2	SC 3	SC 4
SiO ₂ %	64.7	68.5	69.0	68.0
Al ₂ O ₃ %	13.9	14.2	13.8	12.8
Fe ₂ O ₃ %	6.26	4.62	4.29	5.05
MgO %	1.22	1.32	1.12	1.00
CaO %	0.96	0.78	0.89	0.50
Na ₂ O %	1.35	1.39	1.21	0.57
K ₂ O %	4.78	4.52	4.89	5.54
TiO ₂ %	0.32	0.36	0.35	0.33
P ₂ O ₅ %	0.09	0.10	0.10	0.08
MnO %	0.07	0.05	0.05	0.05
Cr ₂ O ₃ %	< 0.01	< 0.01	0.01	< 0.01
V ₂ O ₅ %	0.03	0.03	0.03	0.02
LOI %	4.29	3.70	3.45	4.02
Sum %	98.0	99.6	99.1	98.0

The mean modal content of the Mirador composites is shown in Figure 5-5. Silicates are predominantly quartz, feldspars and micas, the three groups constituting 95.4% of the non-sulphide gangue, while clays (2.1%) and carbonates (1%) comprise the remainder.

Figure 5-5: Mean Modal Composition of Mirador Ore





Mineralogy

The mineralogy of each super-composite was studied by G&T Metallurgical Services in Kamloops, using manual point-counting modal analysis. Speciation data from these studies are shown in Table 5-6 below. The details of this study are provided in G&T's report.

Table 5-6: Simplified Super-Composite Modal Analysis

	Chalcopyrite	Bornite	Chalcocite	Pyrite	Non-sulphide Gangue
SC-1	1.74	0.01	0.09	5.92	92.24
SC-2	1.58	0.00	0.13	5.71	92.57
SC-3	1.96	0.01	0.01	6.44	91.58
SC-4	1.84	0.00	0.07	6.59	91.50

Considerable effort has gone into the characterization of the orebody variability using automated quantitative microscopy methods. Each of the Variability Composites were studied by QemSCAN (Quantitative Evaluation of Mineralogy by Scanning Electron Microscopy) analysis at Lakefield. This indicated the deposit is relatively simple and homogenous mineralogically.

Copper is hosted dominantly as chalcopyrite, in a primary sulphide zone that is open at depth, with minor amounts of secondary copper enrichment mainly developed as a shallow 0 m to 20 m thick chalcocite blanket zone overlying the primary sulphide ore. Overall mineralogically, it is reasonable to expect the ore will respond well metallurgically to conventional copper porphyry flotation processing at moderate grinds and with acceptably good commercial copper recoveries and concentrate grades. This is consistent with the good metallurgical performance achieved in flotation variability testing.

The variability of copper speciation with depth in the deposit is shown in Figure 5-6. On average more than 95% of the copper in the variability composites, is present as chalcopyrite.

More than 5% of the copper was contained in secondary copper minerals, in roughly one-third of the surface and shallow-zone composites studied. Virtually none of the deeper zone composites (>80 m depth) contains more than 5% secondary copper.

Grain size varies from composite to composite. An indication of this variation, and the mean grain size, is shown in Figure 5-7. Pyrite and quartz are the most coarse-grained components in the ore, although neither grain size approaches that of the optimum grind ($K_{80} = 150 \mu\text{m}$). The chalcopyrite grain size is roughly one-third that of the optimum primary grind, indicating that extensive liberation is not required for good copper rougher recoveries.



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Figure 5-6: Cu Mineral Speciation, % Cu as Secondary Minerals

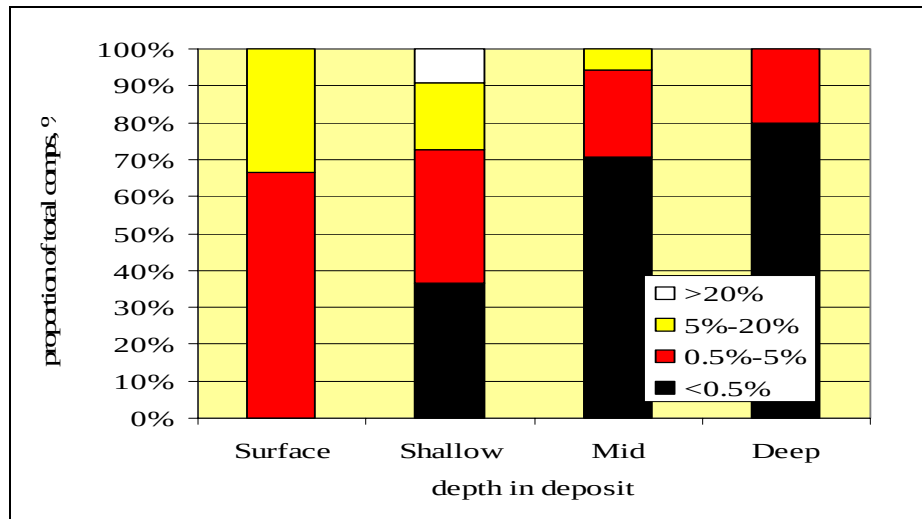
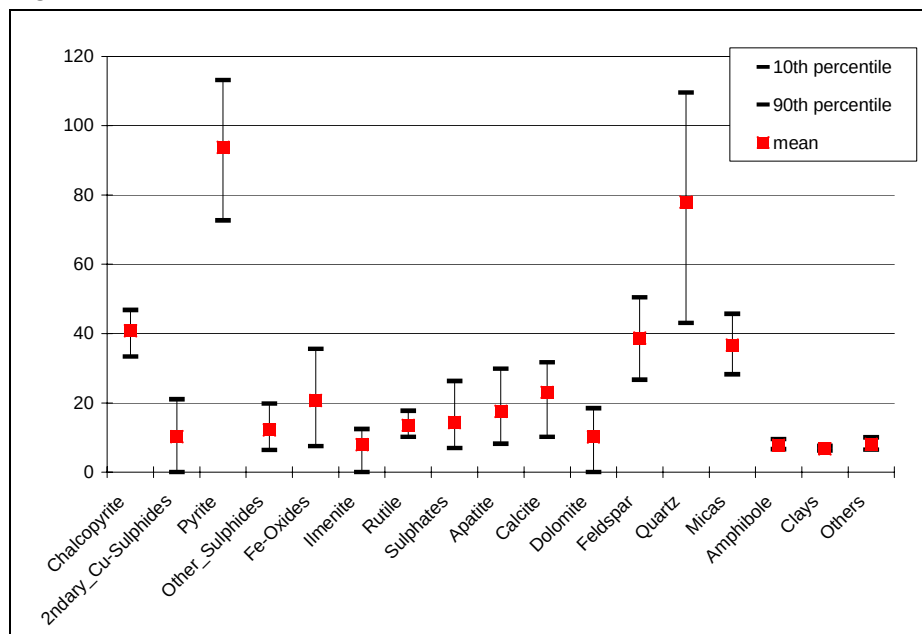


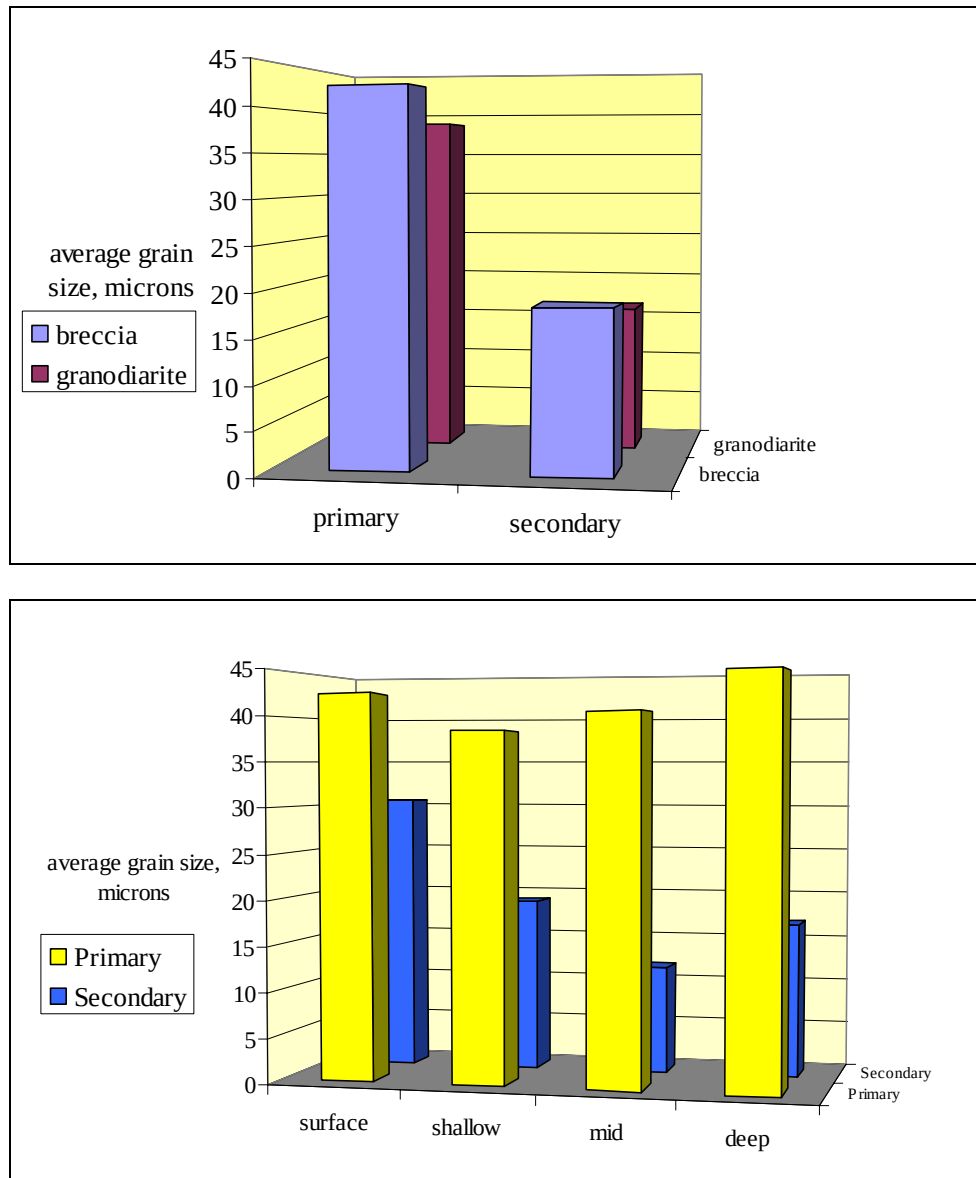
Figure 5-7: Constituent Mineral Grain Size Distributions



An analysis of the variation in copper grain size by lithology and depth within the deposit is shown in Figure 5-8. The average chalcopyrite grain size is 41 μm and although it is marginally coarser in the breccia, it is largely uniform throughout the deposit



Figure 5-8: Variation in Cu Mineral Grain Size with Lithology and Depth

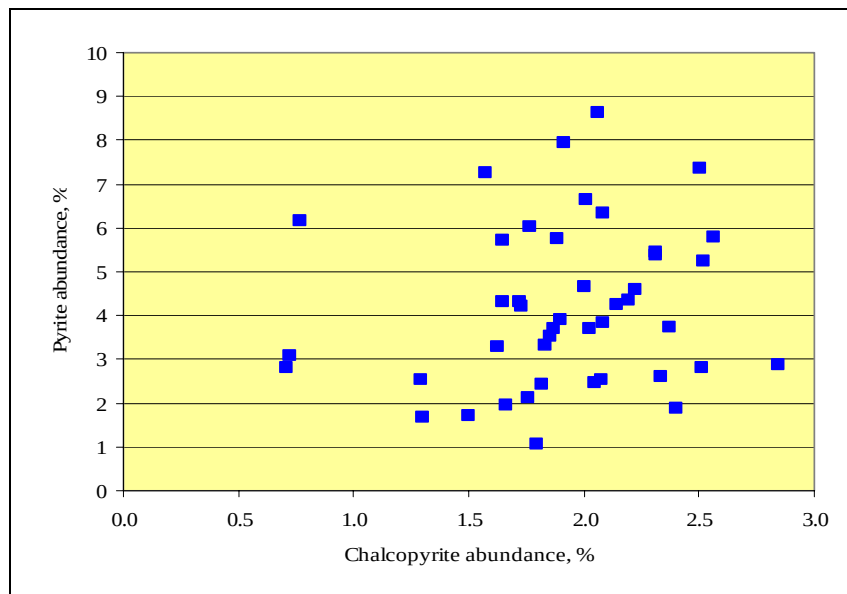


The secondary copper mineral grain size is moderately finer, averaging 17 μm , and requires a finer regrind. Secondary mineral grain size appears to decline with depth. To avoid activating pyrite during finer regrinding the rougher flotation will be operated in a more selective mode to reject pyrite. As a result, the early production of copper from the secondary copper zone requires a higher rougher flotation pH and a finer regrind scheme, and sufficient flexibility has been incorporated in the design of the mill lime and regrind systems to accommodate this.



Pyrite occurs widely throughout the deposit, but at relatively low levels, and the pyrite to chalcopyrite ratio, at about 3:1, is in the lower quartile of the industry, for porphyry deposits. Pyrite abundance varies widely and as indicated in Figure 5-9 is not linked with chalcopyrite abundance. It is also more coarsely disseminated than the copper minerals and can be expected to liberate first with quartz during grinding. The average grain size is 98 μm , relative to the chalcopyrite grain size, which is quite consistent and averages about 40 μm . Both these factors favour good flotation cleaner metallurgy even using the relatively neutral bulk sulphide rougher flotation mode of operation selected, and results in a relatively clean concentrate with a 30% Cu grade.

Figure 5-9: Comparative Chalcopyrite and Pyrite Abundance



5.1.5 Grinding

Grindability tests were conducted on selected metallurgical variability sub-composite sample intervals of split-core from individual drill holes, as well as whole core intervals from the dedicated geotechnical and comminution drill holes: M85 and M86. Geological data analyses demonstrated that the post mineral dykes should be treated as separate domains, representing unmineralized material and were treated as waste. Therefore, dyke material was not included in the comminution program. The lithology codes of the remaining mineralized material were confirmed using the drill logs.

The ore comminution characterization included:

- Bond rod mill work index (BRWI)



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- Bond ball mill work index (BBWI)
- Abrasion index (Ai)
- Crushing work index (CWI)
- J-K drop weight and SAG mill comminution (SMC) testing (an abbreviated form of JK test)
- SAG Power Index (SPI) tests at MinnovEx.

The JKTech drop weight testing provides ore specific parameters for use in the JKSimMet Mineral Processing SAG Grinding Simulator software whereas the SPI Tests provides data for input into MinnovEx's CEET software:

- JKTech SMC test DWI
- JKTech high energy or impact breakage $A \times b$
- JKTech low energy or abrasion breakage t_a
- MinnovEx SAG Power Index.

A full J-K drop weight test was conducted on hole M86 to calibrate the DWI for the SMC versus A and b correlations on similar ore, and to determine the t_a parameter. In addition where no BBWI was done a relative ball mill work index was calculated. All tests were conducted at Lakefield, except the latter, which were conducted at MinnovEx on samples provided by SGS Lakefield Research. The results of the grindability tests are summarized in Table 5-7.



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Table 5-7: Grindability Testing Summary

Location	Sample Composite	A x b	DWI (kWh/t)	ta	SPI (minutes)	RWI (kWh/t)	BWI (kWh/t)	AI (g)	CWI (kWh/t)
Drop weight holes	M86	48.2	5.50	0.57	-	-	-	-	-
	M86 (2-18)	-	-	-	73.9	13.1	14.9	-	5.2
	M86 (20-39)	-	-	-	63.7	-	14.9	-	-
	M86 (41-60)	-	-	-	71.4	13.7	14.8	-	4.8
	M86 (62-78)	-	-	-	70.4	-	15.1	-	-
	M86 (80-99)	-	-	-	62.1	13.5	14.9	-	6.2
	M86 (101-120)	-	-	-	70.2	-	14.5	-	-
	M86 (122-138)	-	-	-	66.4	14.0	14.3	-	4.4
	M86 (140-159)	47.6	5.48	-	62.7	-	15.1	-	-
	M86 (161-180)	-	-	-	97.6	15.3	14.9	-	5.5
	M86 (182-203)	35.2	7.51	-	88.4	-	14.3	-	-
	M85 (44-60)	-	-	-	23.8	8.6	13.2	-	4.2
	M85 (262-300)	-	-	-	82.5	13.9	16.0	-	3.7
Surface	VC-2	41.5	6.32	-	91.7	-	13.8	-	-
	VC-18	-	-	-	67.2	-	14.3	-	-
	VC-29	42.2	6.56	-	88.3	-	* 15.0	-	-
Shallow	VC-3	52.1	5.10	-	76.9	-	-	-	-
	VC-8	-	-	-	-	-	14.2	-	-
	VC-19	-	-	-	86.8	-	15.0	-	-
	VC-22	63.1	4.05	-	55.2	-	14.2	-	-
	VC-23	-	-	-	-	12.6	14.4	0.297	-
	VC-24	-	-	-	-	-	12.7	-	-
	VC-30	41.4	6.69	-	120.4	-	* 15.0	-	-
	VC-33	50.0	5.36	-	53.2	-	13.9	-	-
	VC-36	-	-	-	-	-	10.9	0.281	-
	VC-40	61.8	4.06	-	52.4	-	11.3	-	-
	VC-44	51.3	5.36	-	-	-	14.9	-	-
	VC-47	-	-	-	106.4	-	15.8	-	-
	VC-50	-	-	-	-	-	14.0	-	-
Mid	VC-4	41.0	6.49	-	78.0	-	13.7	-	-
	VC-5	-	-	-	-	-	13.0	-	-
	VC-6	-	-	-	-	-	13.9	-	-
	VC-9	-	-	-	-	-	14.3	-	-
	VC-11	38.9	6.73	-	103.3	-	* 15.1	-	-
	VC-12	35.5	7.34	-	81.8	-	-	-	-
	VC-20, 21	-	-	-	-	12.4	13.8	0.324	-
	VC-31	36.1	7.58	-	95.1	-	14.6	-	-
	VC-34	42.2	6.28	-	83.2	-	14.9	-	-
	VC-35	37.6	7.05	-	86.3	-	14.9	-	-
	VC-27	-	-	-	76.1	13.4	13.7	0.380	-
	VC-25	40.4	6.47	-	70.9	-	13.9	-	-
	VC-37	-	-	-	-	-	12.6	-	-
	VC-41	47.1	5.23	-	74.5	-	14.1	-	-
	VC-45	-	-	-	102.5	-	15.8	-	-
	VC-48	-	-	-	-	-	15.8	-	-
	VC-51	37.9	7.21	-	-	-	14.5	-	-
Deep	VC-10	48.5	5.36	-	71.6	-	* 15.0	-	-
	VC-13	40.2	6.65	-	76.2	-	14.9	-	-
	VC-26	-	-	-	-	-	14.8	-	-
	VC-28	37.7	6.67	-	85.2	15.1	14.9	0.406	-
	VC-32	40.0	6.54	-	79.4	-	14.0	-	-
	VC-38, 39	-	-	-	-	-	12.0	-	-
	VC-42, 43	44.3	5.82	-	68.8	13.5	14.6	0.417	-
	VC-46	38.6	6.77	-	99.4	-	16.3	-	-
	VC-49	-	-	-	-	-	15.8	-	-
	VC-52	-	-	-	-	-	13.7	-	-
Average		43.9	6.16	0.57	78.0	13.3	14.4	0.351	4.85
S.D.		7.3	0.96		17.8	1.69	1.07	0.058	0.85
S.D. Rel (%)		16.7	15.5		22.8	12.7	7.5	16.5	17.5
Min.		35.2	4.05		23.8	8.6	10.9	0.281	3.7
Max.		63.1	7.58		120.4	15.3	16.3	0.417	6.2
Median		41.5	6.48		76.6	13.5	14.5	0.352	4.8
75 th percentile		48.1	6.72		87.9	13.9	14.9	0.400	5.4
90 th percentile		51.7	7.28		100.3	15.0	15.7	0.412	5.8

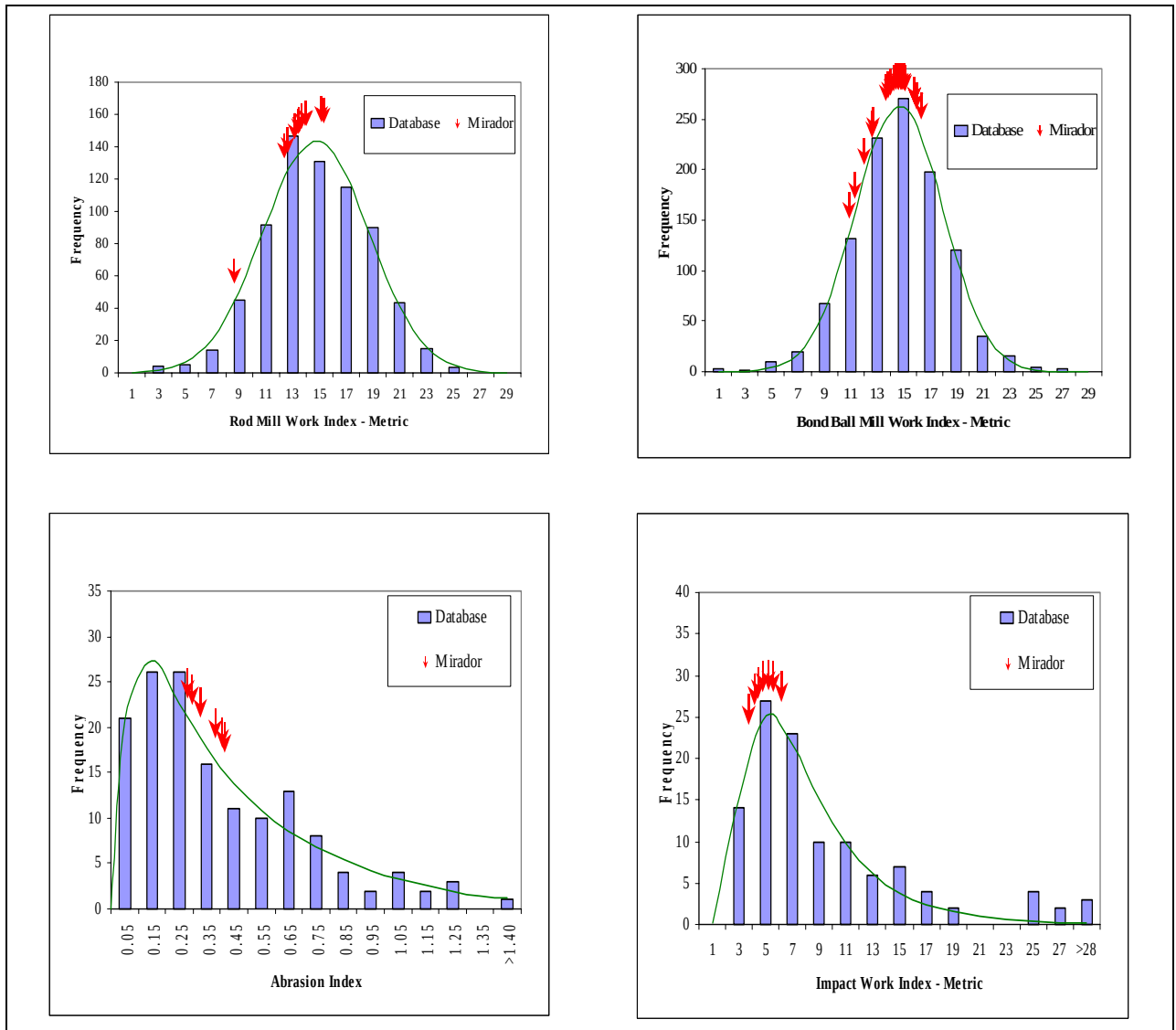
*VC 11 and VC-13, VC-29 and VC-30, as well as VC-10 and VC-31 were blended for the Bond ball mill grindability.
BWI indices in italics are estimated BWI's based on a 2 kg batch grind



Bond Work Index Testing

The results of the Bond Work Index studies are summarized in Figure 5-10.

Figure 5-10: Bond Work Index Test Summary





The following observations can be made relative to an industry database of projects:

- Seven samples were tested for Bond low-energy impact crushing. The crushing indices varied between very soft to medium. The average CWI was 4.85 kWh/t, which is in the soft range.
- Eleven samples were tested for Bond rod mill grindability at 14 mesh grind. The average BRWI was 13.3 kWh/t, and varied between 8.6 and 15.3 kWh/t. This ranks the ore in the very soft to medium resistance range for rod milling. The BWRI is also lower than the BBWI, suggesting a relatively low propensity to form critical size material during SAG milling. This is consistent with the crushing index, SPI and drop weight data and the results of subsequent comminution modelling.
- Twenty-five samples were tested for the Bond ball mill grindability at 100 mesh grind. The average BBWI was 14.5 kWh/t, and varied between 10.9 and 16.3 kWh/t. This ranks the ore hardness to ball milling as very soft to moderately hard. The BBWI also has a relatively low variability.
- Six samples were tested for abrasion index. The average Ai was 0.35, and varied between 0.28 and 0.42. This ranks the ore as moderately abrasive, which is consistent with its largely granodioritic quartz composition.

There does not appear to be significant variations in Bond hardness parameters across the ore body, or by depth, within the proposed pit.

SAG Comminution Testing

The frequency distribution of the SPI data relative to MinnovEx's database of other operations in the industry is summarized in Figure 5-11, higher values indicating a higher strength.

The JKTech industry database of frequency distribution for the low energy or abrasion breakage t_a parameter is shown in Figure 5-12. By comparison the t_a parameter measured for drop weight hole M86 was 0.56, lower values indicating a higher strength.

The JKTech industry database of frequency distribution for the DWI and A and b appearance functions versus the Mirador data are presented in Figure 5-13. The DWI is determined from SMC testing and is used to derive A and b parameters, a lower A x b values indicating a higher strength.



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Figure 5-11: Frequency Distribution of SPI Data

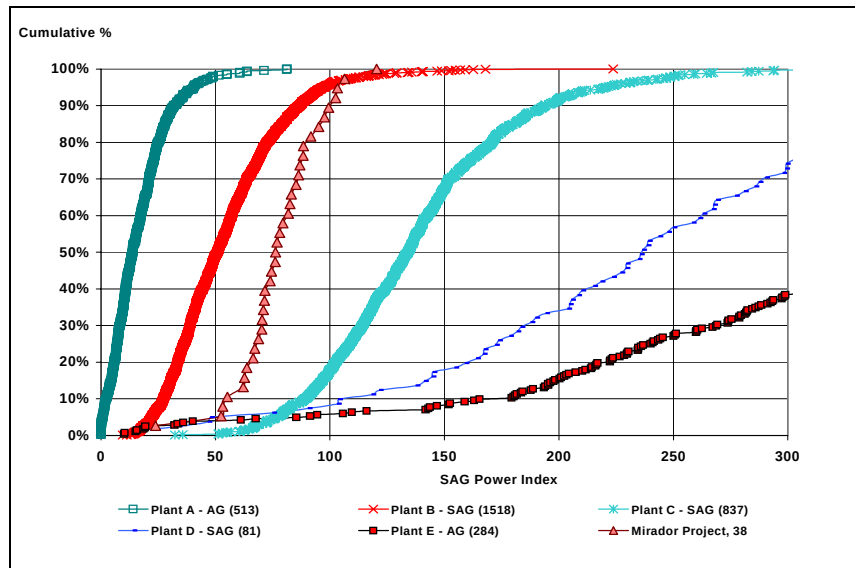
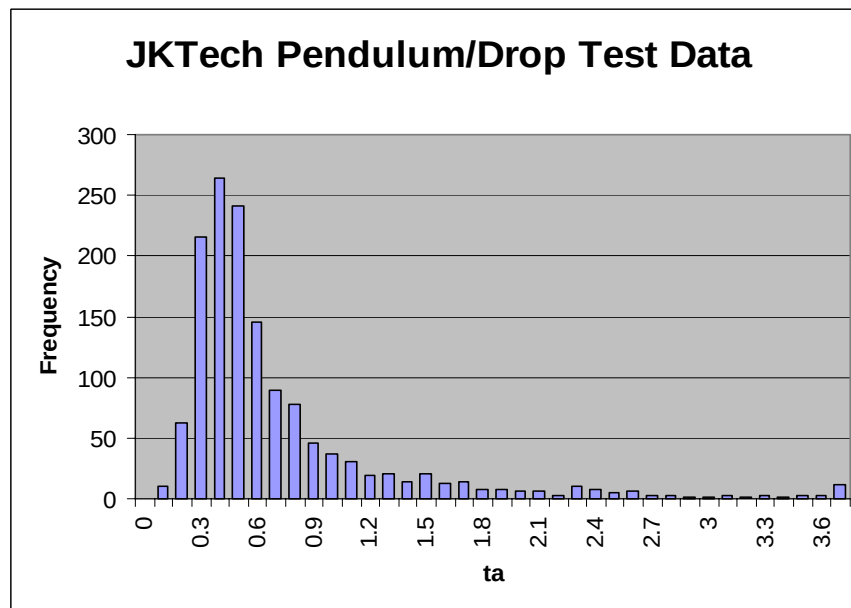


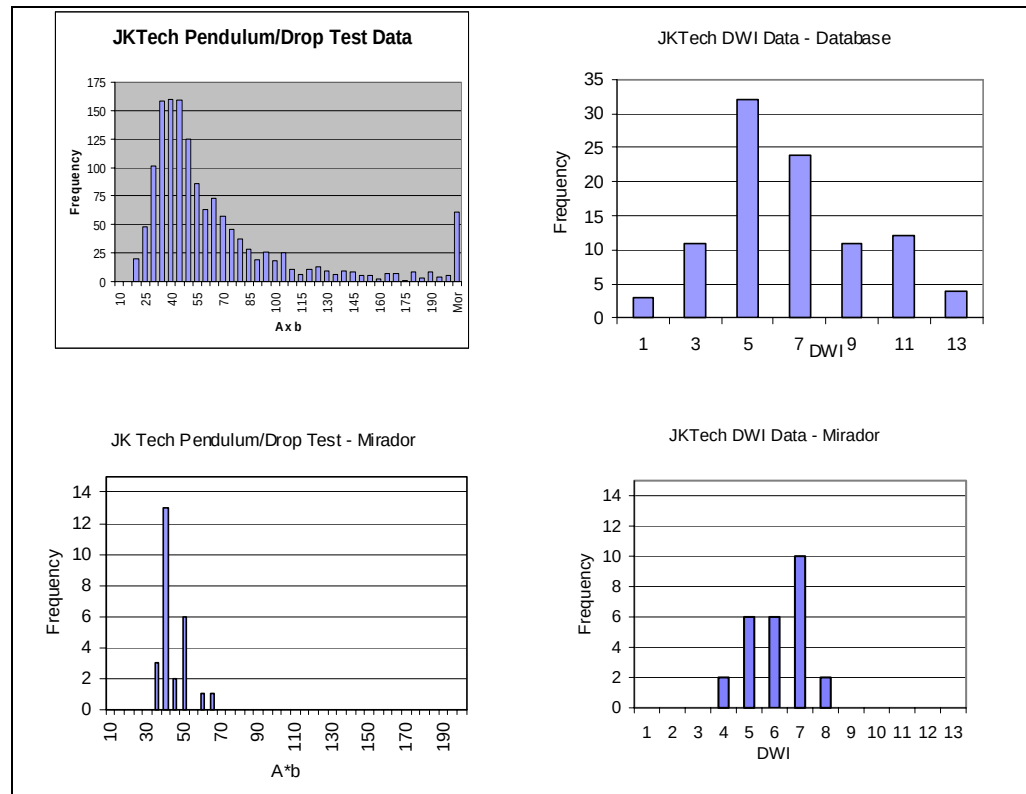
Figure 5-12: Frequency Distribution of t_a Parameter in JK Database





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Figure 5-13: Frequency Distribution of A*b and DWI – Database vs. Mirador



The following observations with respect to ore hardness for SAG milling can be made relative to JKTech and SPI's industry database of projects:

- The JKTech drop weight test and the SMC test results fall in the moderately soft to moderately hard range. The average of A x b and DWI are 43.9 and 6.16 kWh/t respectively. On average, the Mirador samples are slightly harder (59th percentile) than average with respect to impact breakage.
- The SPI values vary between 23 and 120 minutes. The median and 90th percentile are 78 and 100.3 minutes, respectively. This ranks the ore hardness in the moderately soft to moderately hard, which is in reasonable agreement with the JKTech data.

Mineral Fragmentation Characteristics

The RQD values measured from the core indicate that rock quality is very low through large parts of the deposit. The average RQD value from the drilling over the whole of



the deposit is 38%. The poor RQD at Mirador is the result of the shattering process due to the hydration of anhydrite mineralization.

Bond low energy impact data (CWI) relative to drop-weight test data also suggest that the material will break relatively easily at low-energy, but the resulting reduction may be small. This is consistent with the poor rock quality indicated by low RQD data in drill core, as well as geological structure indicating most of the ore falls in a gypsum depletion zone of alteration. It is also reasonable to expect the SAG mill feed granulometry will be relatively fine relative to other copper porphyry operations.

5.1.6 Flowsheet Development Flotation Testwork

Full details of this phase of flowsheet development testwork, including assays, met balances, and reagents are reported in Lakefield's Progress Report No.1. This section presents a summary of this data.

Batch Rougher Testwork

The effect of primary grind on rougher flotation each of the SC samples was investigated. The copper metallurgical results are presented in Figure 5-14.

A simple economic study was conducted on these results, using typical power costs and a net smelter revenue value for copper of \$0.90. This indicated that the optimum grind was about 140 μm to 150 μm . This should be reviewed if copper prices continue to improve. The results for gold metallurgy were less conclusive, as the grade/recovery relationship was marginally better for the coarsest grind in three cases.

Reagents

The effect of pH on rougher flotation was tested on each of the SC samples and the results are presented in Figure 5-15.



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Figure 5-14: Effect of Primary Grind Size on Cu Rougher Grade and Recovery

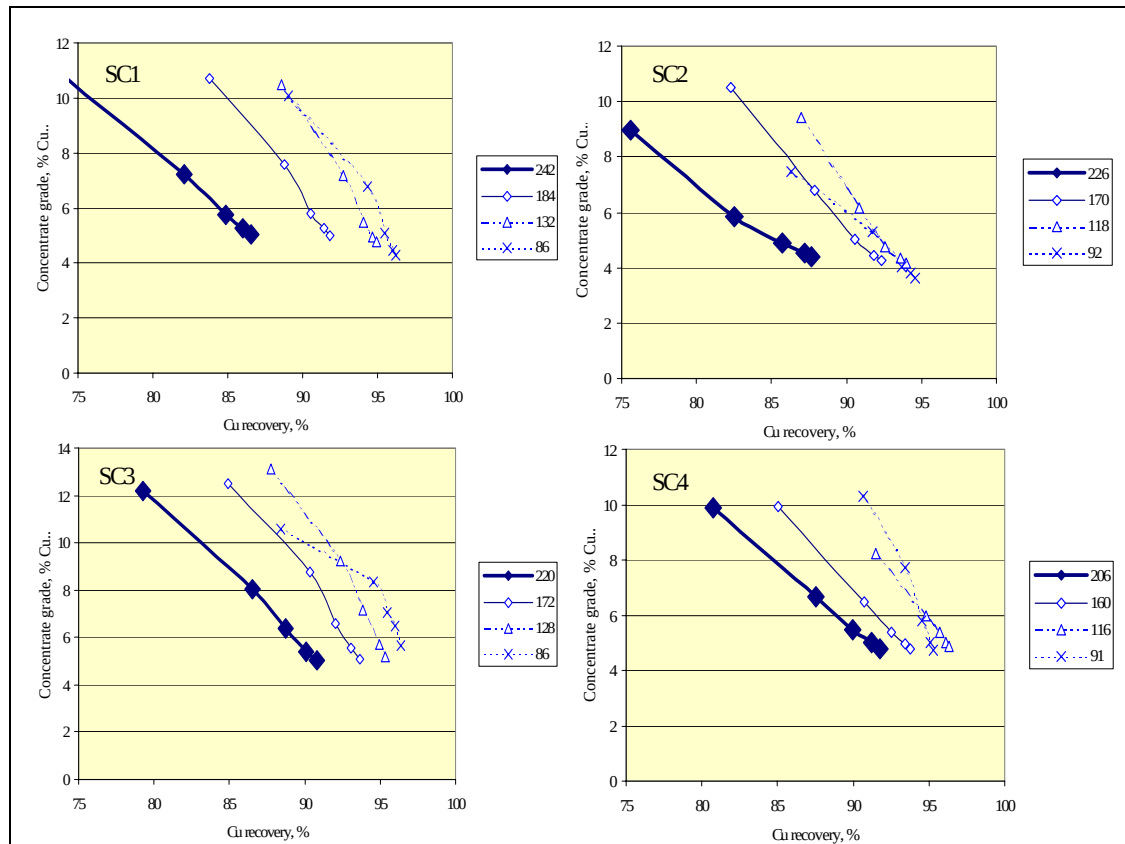
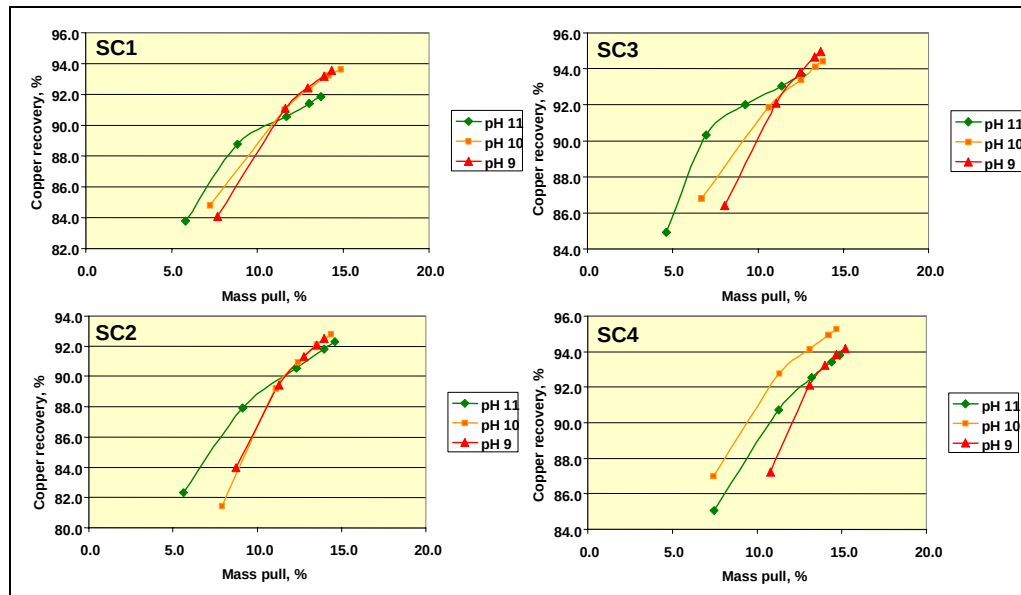




Figure 5-15: Effect of pH on Copper Recovery vs. Mass Pull

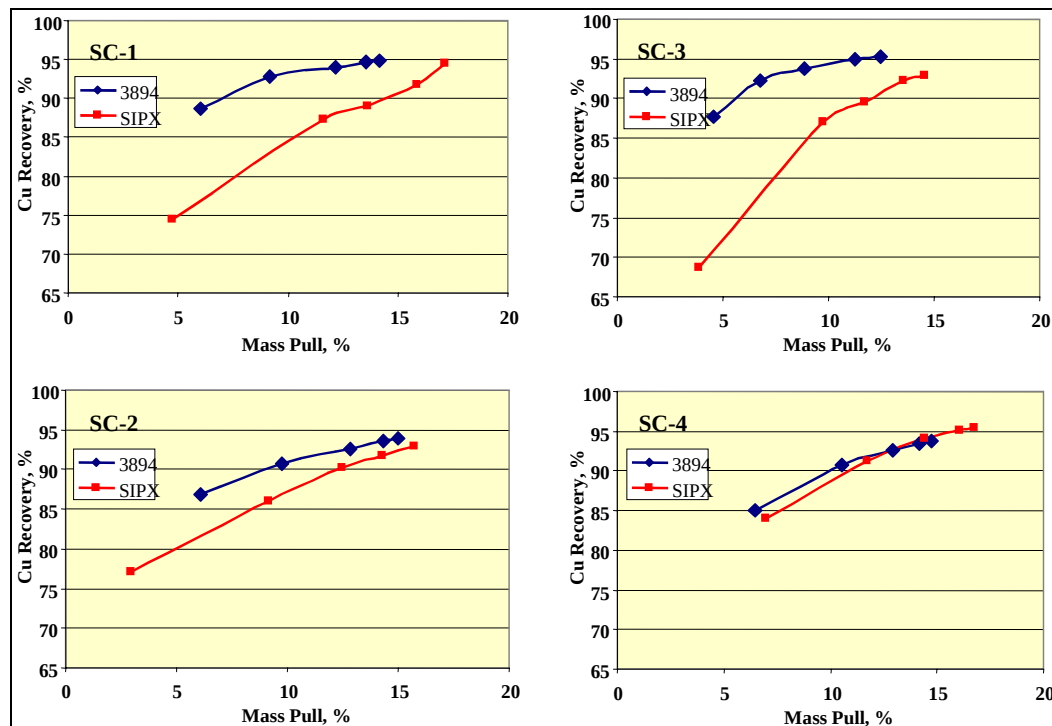


Rougher flotation is best conducted at a pH of 9.5 as pH levels in the 9 to 10 range yielded 1% to 2% better recoveries. This is supported by G&T modal data that shows between 3% and 10% of the copper is present as low-grade middlings in pyrite, and supports the benefit of floating middlings pyrite to maximize copper rougher recovery. This is also facilitated by the use of a more selective thionocarbamate collector. There was no significant pH effect on gold recovery.

The ore is relatively easy to treat, suggesting a feasibility collector optimization strategy would probably yield only relatively minor incremental metallurgical improvements. Therefore, collector suite fine-tuning was deemed largely better left for future optimization, possibly when the plant is in operation. In earlier studies Sumitomo had adopted the use of thionocarbamate Cytec 3894, a relatively selective and more expensive nitrogen-based collector, but not particularly strong. This was also found to be highly effective when used alone in these studies. Cytec 3894 was compared to a standard xanthate (sodium isopropyl xanthate) and the effect on copper flotation metallurgy is shown in Figure 5-16. This shows lower mass pulls. The reduced operating cost of the regrind and cleaner circuit would most likely warrant the extra cost of using Cytec 3894. No significant improvement in gold metallurgy was noted.



Figure 5-16: Effect of Collector on Rougher Flotation Metallurgy



Cytec 3894 may sometimes be used in actual plant operation in conjunction with a little xanthate because it is not a strong collector. However xanthate tended to raise mass pull rates in this test programme, therefore, it was not used in the locked cycle flotation confirmation testing.

Regrind Target and Mineral Liberation

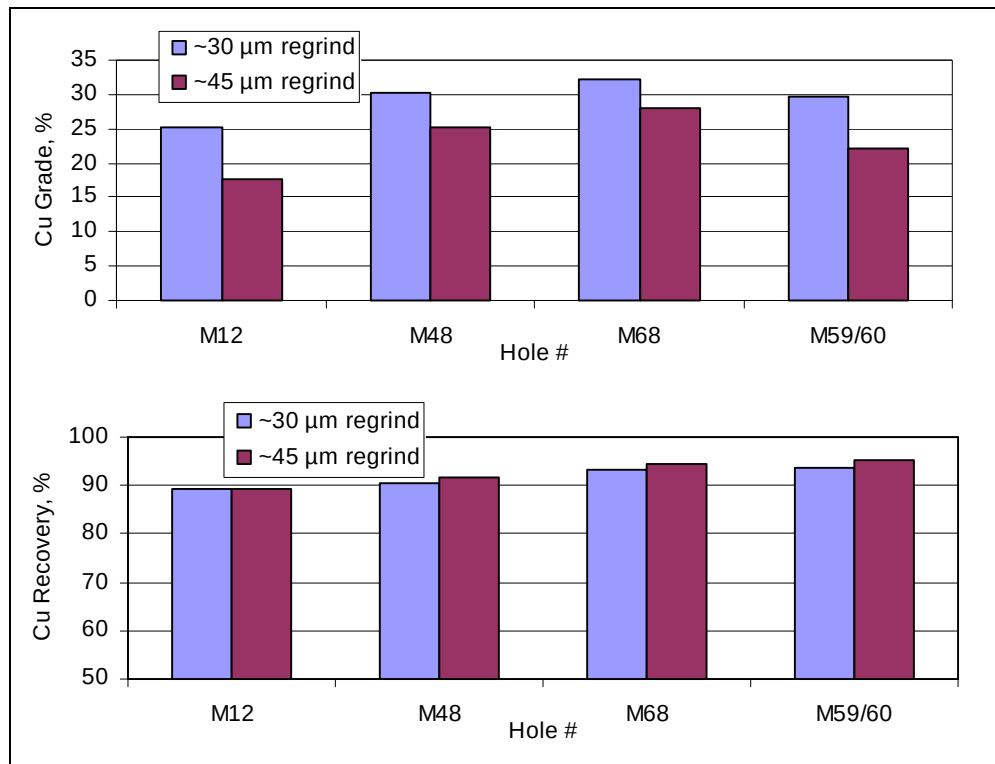
Mineralogy data clearly supports a need for concentrate regrinding, the primary grind of 80% passing 150 μm being considerably coarser than the primary copper sulphide grain size (averaging about 40 μm).

Batch regrind and cleaner testing on the four SC samples indicated the finest regrind of 80% passing 20 μm yielded better grade recovery curves and the highest final concentrate grades on two and three of the samples respectively. However mineralogical modal evidence from G&T and Lakefield both showed potential for the samples to yield excellent concentrate grades assaying 30% to 32% Cu following regrinding to 30 μm . It was believed that the very short regrind time required for the coarser regrind using the testing rod mill may have left some of the pyrite surfaces inadequately “cleaned” and therefore depressed the recovery. Subsequent testing confirmed this.



Based on an average primary copper sulphide grain size of about 40 μm , and liberation analysis, it is reasonable to believe that a regrind product size of 30 μm should be suitable for the majority of the ore because it is dominantly the primary sulphide type. During the mapping testwork phase an additional four locked cycle tests were conducted to investigate the effect of coarser grind on final concentrate grade and recovery. Figure 5-17 compares the results of the four tests done at a target regrind P_{80} of 45 μm with the initial tests done at a target regrind P_{80} of 30 μm . In all four cases the final concentrate grade decreases.

Figure 5-17: Comparison of Cu Grade and Recovery at Two Regrind Sizes



On average the coarser regrind had a P_{80} of 44.8 μm and a copper grade of 23.3%, while the finer regrind had a P_{80} of 27.0 μm and copper grade of 29.4%. The coarser regrind showed a slightly higher average recovery, 92.7% versus 91.6%, which was possibly due to a slight increase in rougher circuit mass pull during that round of tests. Gold showed similar upgrade behaviour to copper at the finer regrind, but recoveries marginally lower in three of the four tests.



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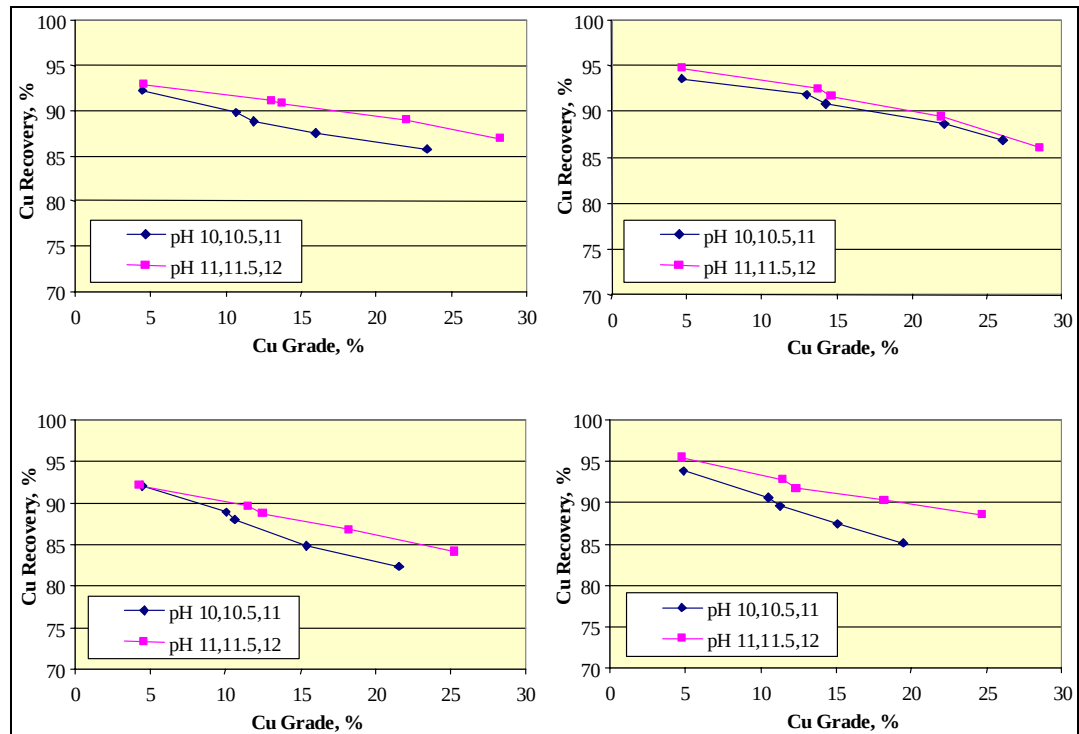
The secondary copper mineral grain size is moderately finer, averaging 17 μm , and requires a finer regrind. This ore type is processed early in the mine plan and therefore a target of 25 μm was selected for the basis of the regrind design.

Batch Cleaner Testwork

Batch cleaner tests were conducted to confirm the target regrind size and select the cleaner pH conditions. The object of the high pH, collector-starved environment in the cleaners is to liberate and reject attached silicates, and depress the previously-floated pyrite. If the floatability imparted on the pyrite in rougher flotation is not adequately depressed during regrinding and cleaning, it will continue to float. The role of lime as a pyrite depressant in cleaning is shown in Figure 5-18 from pH profile tests conducted on each of the super-composites. High pH is required to ensure adequate pyrite rejection.

In all cases, higher copper concentrate grades were produced by maintaining higher pH levels in the cleaners. In most cases, high copper recoveries were also achieved. For three of the four super-composites, gold recoveries are also higher when higher pH levels are used. Accordingly, higher pH levels were adopted for cleaner flotation.

Figure 5-18: Effect of pH on Cleaner Performance



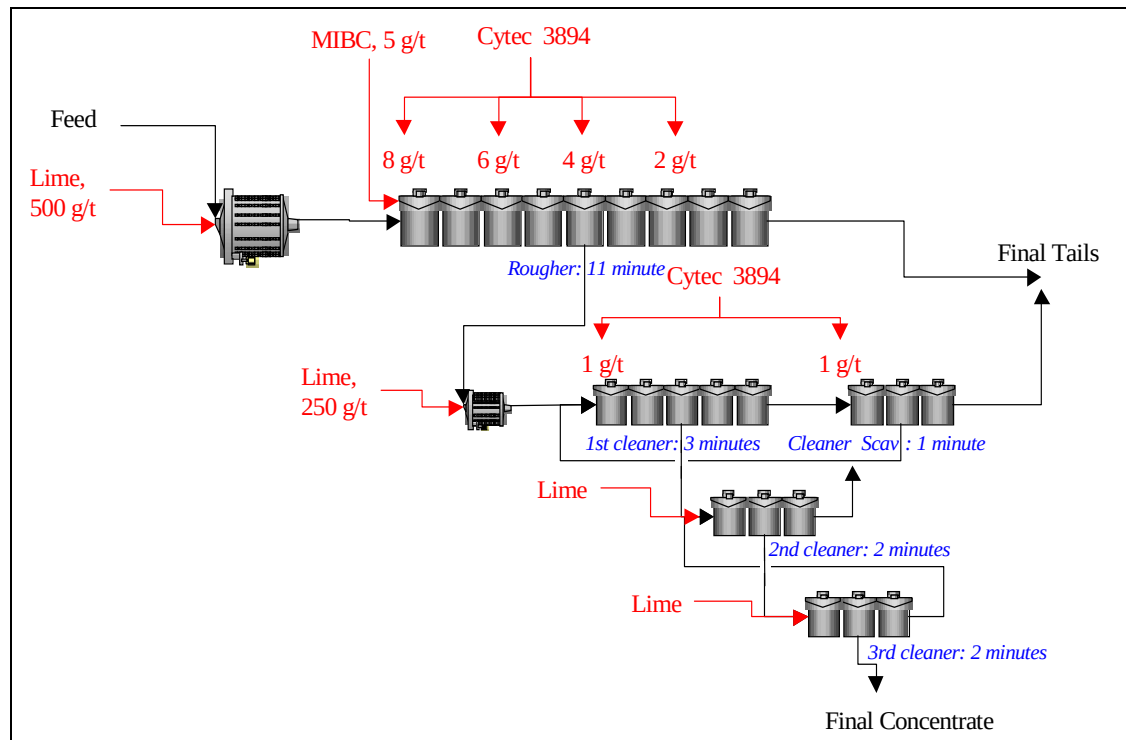


Concentrate regrinding to a product size of 80% passing 30 μm is adequate to yield, after three stages of cleaning at pH 11-12, a final concentrate assaying 31% Cu. Cleaner times in the laboratory were short, at 3 minutes (first cleaner), 1 minute (cleaner scavenger), and 2 minutes (2nd and 3rd cleaners). Overall, copper recoveries were 90%, on average. Gold recoveries were 49%.

Locked Cycle Confirmation Testing

Locked cycle tests were conducted on each of the supercomposites, using the flowsheet shown in Figure 5-19. The flowsheet consisted of a primary grind to a P_{80} of 150 μm , followed by rougher flotation at pH 9.5 to produce a bulk sulphide concentrate. The rougher concentrate was reground to a P_{80} of $\sim 30 \mu\text{m}$ prior to three stages of cleaning at pH 11.0, 11.5, and 12.0, respectively. The 2nd and 3rd cleaner tails were recycled to the feed of the previous stage, while the 1st cleaner tails went to a scavenger float. The scavenger concentrate was returned to the first cleaner feed, and the scavenger tail to the final tails. Reagent additions consisted of lime to control pH, Aero 3894 as collector, and MIBC as frother. Collector and frother additions were held constant between tests, whereas lime addition fluctuated in order to maintain the setpoint pH.

Figure 5-19: Locked Cycle Test Flowsheet





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The test results are detailed in Lakefield's No.1 Report. The tests exhibited minor scatter through the cycles, though reaching acceptable mass equilibrium after 2 to 3 cycles, and copper equilibrium after 4 cycles. Steady gold balancing was impeded by the low gold content in the ore and possibly, the presence of small amounts of coarser gold affecting assay reproducibility.

The results from locked cycle confirmation tests on each composite are shown in Table 5-8. On average, copper recovery was 90%, while the copper concentrate grade averaged 31%. Gold recovery was 49%. These results are consistent with the performance expected from batch testing and mineralogy.

Copper losses were primarily incurred in roughing (6%), while a further 3% to 4% were lost to the first cleaner scavenger tails. Gold losses were also mainly to the rougher tails, and probably consisted of gold occurring as ultra-fine inclusions in silicates. The gold lost in the cleaner tails were probably associated, primarily, with pyrite.

Table 5-8: Locked Cycle Test Results on Super-Composites 1 to 4

Super composite	SC-1	SC-2	SC-3	SC-4	Average
<i>Head Grade, %</i>					
Head grade, % Cu	0.70	0.62	0.64	0.72	0.67
Head grade, g/t Au	0.28	0.32	0.16	0.24	0.25
Copper concentrate grade, %	30.4	32.0	31.6	31.2	31.3
Copper recovery, %	90.8	85.6	92.2	92.5	90.3
Gold recovery, %	45.4	55.2	45.9	50.4	49.2
<i>Copper Losses</i>					
Rougher tails, %	6.9	8.9	4.8	4.4	6.3
Cleaner tails, %	2.3	5.5	3.0	3.1	3.5
<i>Gold Losses</i>					
Rougher tails, %	32.8	23.2	28.4	32.7	29.3
Cleaner tails, %	21.8	21.5	25.7	17.0	21.5

5.1.7 Variability Mapping Testwork

Full details of this phase of variability mapping testwork, including assays, met balances, and reagents are reported in Lakefields Progress Report No. 2. This section presents a summary of this data.

Process Mineralogy

Each of the Variability Composites were studied by QemSCAN analysis at Lakefield. Regression analysis was conducted to link copper rougher flotation recovery and final

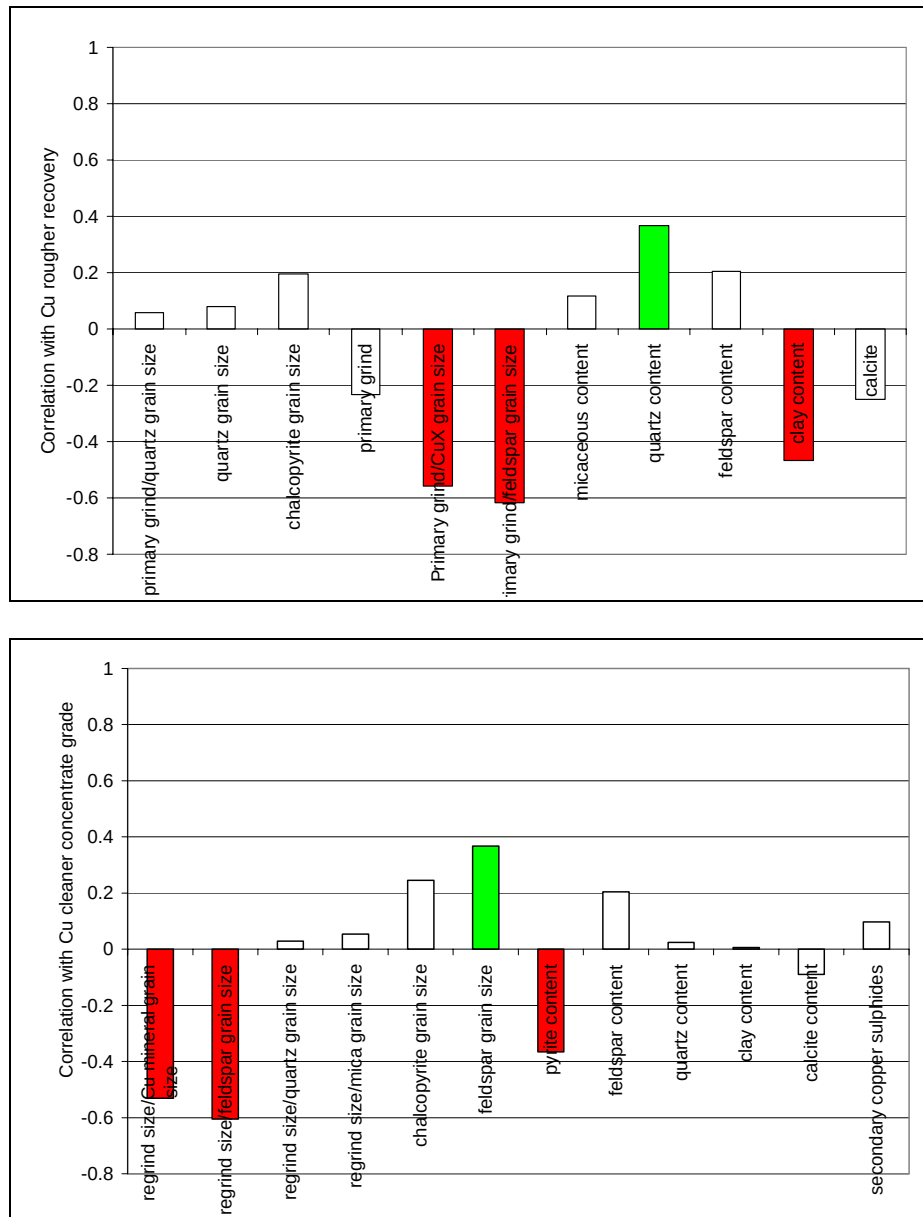


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concentrate grade with ore mineralogy. The results for the dominantly primary sulphide Mid to Deep Zone composites are summarized in Figure 5-20. The liberation results observed for these samples do not translate to the shallow and surface samples, probably due to their higher secondary copper content, which makes them more susceptible to ageing and masks the trends.

Figure 5-20: Correlation between Mineralogical Parameters and Copper Flotation Recovery and Final Concentrate Grade – Mid to Deep Zone Composites





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The following copper recovery mineralogical trends were observed:

- The ratio between the primary grind and the copper mineral average grain size appears to be a strong driver of flotation recovery, a strong negative correlation being evident.
- In tests where the primary grind P_{80} is relatively fine, compared to the average copper mineral grain size, copper rougher recoveries were the highest. This indicates copper mineral liberation is a primary driver of copper rougher recovery.
- Figure 5-21 shows the relationship between the ratio of primary grind P_{80} to mean copper grain size and recovery. As discussed earlier the average chalcopyrite grain size is about 40 μm . To maintain a target copper recovery of 91%, based on a rougher recovery of 94% net of average cleaner scavenger losses of about 3%, Figure 5-21 indicates the target primary grind should be 150 μm ($3.7 \times 40 \mu\text{m}$). This confirms the primary grind parameter established in batch and locked cycle flowsheet development testing.
- Gangue liberation, specifically feldspar gangue liberation is also strongly linked with copper rougher flotation recovery.
- Clay content may also be linked to copper rougher flotation recovery, with a higher abundance of clay having an adverse affect, and this is indicated in Figure 5-22. Clay levels in the samples ranged from 0.3% to 3.5%, with an average of about 1.8%, and at these levels recovery is not expected to be significantly affected.
- Assuming primary grind will be relatively constant in normal plant operation, any variation in copper and feldspar grain size will be a primary driver affecting overall recovery, with coarser mineralization yielding better copper recoveries. It has been shown earlier that the primary copper grain size is relatively consistent and therefore it is reasonable to assume copper recovery will be too.



Figure 5-21: Relationship of Ratio of Primary Grind P_{80} : Mean Cu Mineral to Copper Rougher Recovery

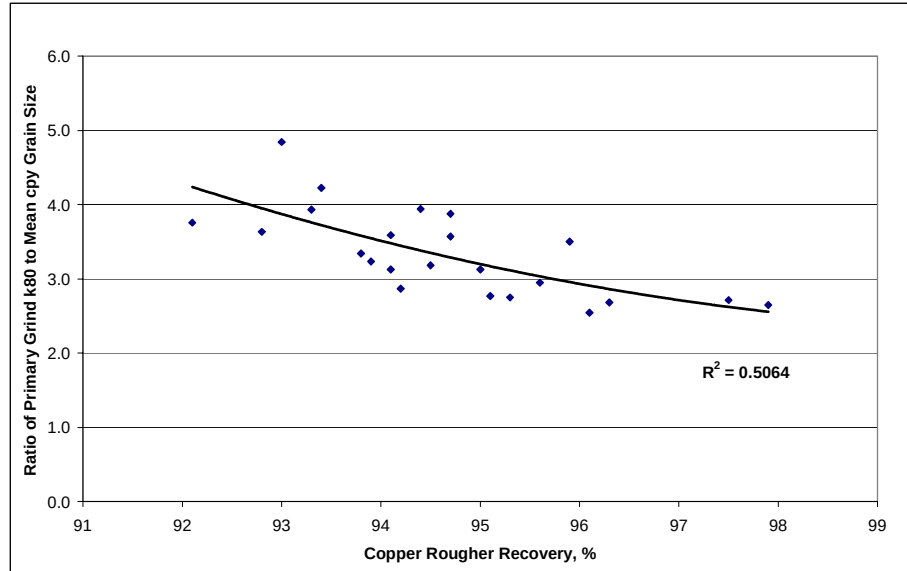
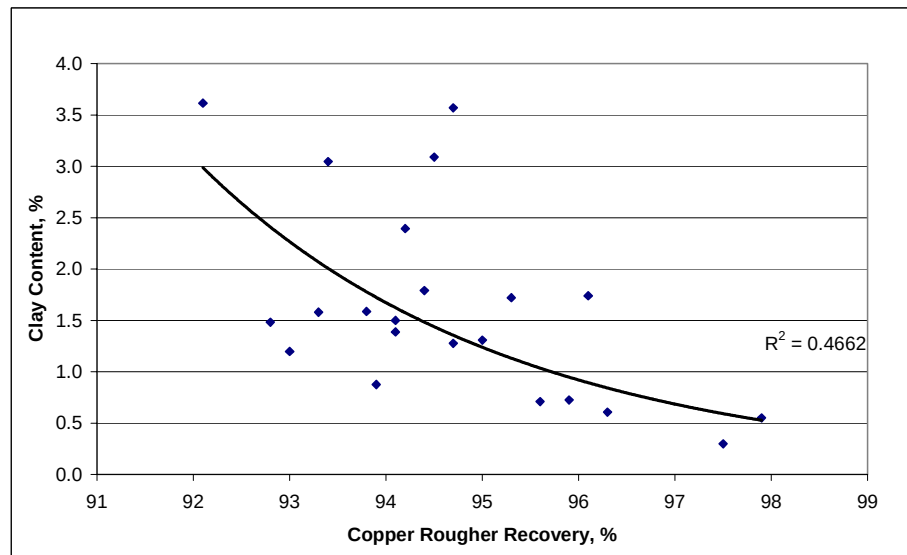


Figure 5-22: Effect of Clay Content on Rougher Recovery



The following final concentrate grade mineralogical trends were observed:

- Final concentrate grade is statistically linked to the ratio between regrind size and copper sulphide and feldspar grain size.



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- Adequate regrinding to liberate the copper minerals is an important factor behind maintaining high concentrate grades. The effect of the ratio of regrind size: copper mineral grain size on final concentrate grade is shown in Figure 5-23.
- To achieve a concentrate grade of 30% Cu, the regrind K_{80} needs to be about two-thirds that of the mean chalcopyrite grain size. Mineralogical mapping showed earlier that the chalcopyrite grain size is relatively consistent at about 40 μm , and it is reasonable to expect that a regrind target of 80% passing 25 μm (40 μm * 0.65) will be sufficient to maintain 30% Cu grades most of the time.
- The abundance of pyrite in the ore is also a factor behind achieved concentrate grade, albeit a weaker factor (see Figure 5-24). With more pyrite in the ore, pyrite depression becomes somewhat more important as it adversely affects concentrate grade. Pyrite abundance in the Mirador ore is relatively low in copper porphyry industry terms, and this can account for the generally weak influence of pyrite on average on concentrate grade. Batch testwork also showed the effects of higher pyrite grades on concentrate grade can be addressed using pH control in the rougher and cleaner circuit, as is done in normal plant operation.

Figure 5-23: Effect of Ratio of Regrind Size:Copper Mineral Grain Size on Final Cu Concentrate Grade

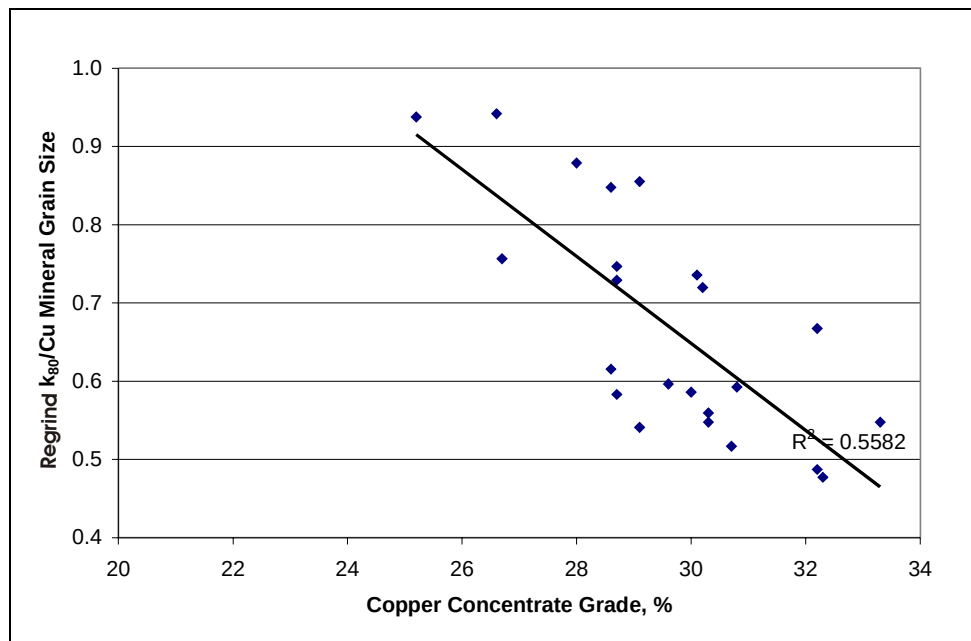
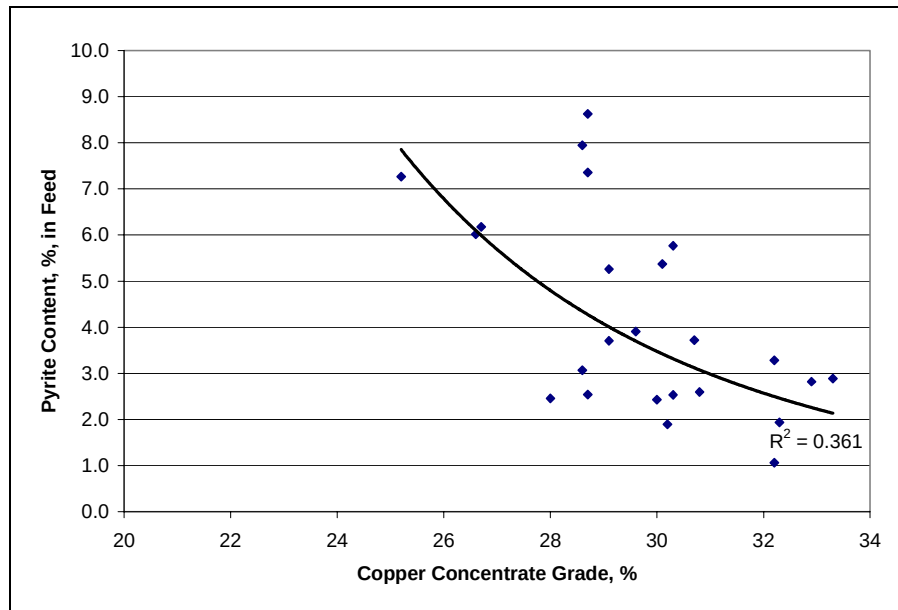




Figure 5-24: Effect of Pyrite Content of Feed on Final Cu Concentrate Grade



Locked Cycle Testing

This phase of the work consisted of three types of tests: 2 kg locked cycle tests to evaluate the variability of the orebody, 2 kg batch tests to provide supporting data, and 10 kg locked cycle tests to generate concentrate for dewatering and pyroforicity studies.

Forty-one locked cycle tests were conducted on the variability composites described earlier to evaluate the variability of the orebody. Testing was based on the locked cycle flowsheet and parameters shown earlier in Figure 5-19, derived during flowsheet development. Each cycle produced three final products: cleaner concentrate, cleaner scavenger tails, and rougher tails. The last cycle included three additional products consisting of the three recycle streams. All products were assayed for copper, gold, and sulphur in order to generate a metallurgical balance for the test. In addition, a metallurgical projection was also generated based on the final products from the last three cycles of each test. The projection represents the best estimate of grades and recoveries under steady-state locked cycle conditions. The results are summarized in Table 5-9.

Overall, the average results were in reasonable agreement with the earlier Super Composite testwork (shown within brackets). Copper grades averaged 29.3% (31.3%), while recoveries averaged 91.0% (90.3%). Gold recovery averaged 47.1% (49.2%).



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Table 5-9: Variability Composite Locked Cycle Flotation Results

Ore Type/ Depth	Weight Rec % to Final Conc	Assays, %, (g/t)		% Distribution			
		Cu	Au	S	Cu	Au	S
Surface-Supergene							
LCT-4	2.01	27.7	4.39	30.8	91.6	41.9	18.0
LCT-20	1.94	25.7	5.70	29.0	83.4	35.6	13.2
LCT-21	2.03	28.7	4.80	34.1	84.6	50.0	29.8
Ave.	2.0	27.3	5.0	31.3	86.5	42.5	20.3
Shallow-Mixed							
LCT-3	1.98	29.0	5.51	32.1	91.6	45.8	13.9
LCT-11	2.64	35.5	5.30	33.0	91.9	50.6	23.4
LCT-18	1.79	31.9	8.68	33.0	90.5	49.8	32.8
LCT-23	1.99	28.7	4.73	32.8	90.7	42.9	16.7
LCT-24	3.12	17.8	4.93	40.5	80.9	54.4	45.1
LCT-26	3.08	21.5	4.33	36.6	89.1	57.1	38.7
LCT-28	1.87	29.9	7.55	30.8	87.4	50.4	24.6
LCT-32	1.23	29.6	9.09	32.3	82.4	53.3	27.9
LCT-33	1.68	31.6	3.91	26.4	92.3	52.7	12.8
LCT-34	1.70	35.3	13.69	33.2	91.3	59.2	26.5
LCT-40	1.89	28.5	3.33	31.2	92.2	35.8	19.7
LCT-41	1.68	28.3	3.91	26.4	94.9	52.7	12.8
LCT-42	1.66	29.0	5.60	33.3	93.3	48.6	25.4
Ave.	2.0	29.0	6.2	32.4	89.9	50.3	24.6
Mid - Primary							
LCT-5	2.11	26.6	5.76	32.1	90.4	51.8	22.9
LCT-8	2.16	25.2	3.20	33.2	89.3	31.0	17.0
LCT-10	2.79	33.3	5.02	33.1	97.1	48.4	38.9
LCT-12	2.29	30.1	3.41	32.6	91.1	47.2	16.6
LCT-15	2.03	28.7	3.81	33.1	92.4	40.3	13.4
LCT-16	2.05	30.2	6.77	33.5	90.3	57.3	19.8
LCT-17	2.02	28.6	3.26	32.3	92.6	45.7	11.6
LCT-19	2.82	29.1	5.89	32.2	94.1	54.5	23.3
LCT-25	2.21	28.7	4.23	31.0	91.5	47.1	17.1
LCT-27	1.79	29.6	6.68	29.8	88.6	51.1	21.0
LCT-29	2.63	32.9	9.26	32.0	92.9	57.4	37.5
LCT-31	2.41	28.0	4.65	33.4	91.6	46.3	28.1
LCT-35	1.50	32.2	3.51	33.3	91.8	40.8	22.4
LCT-38	0.72	26.7	3.85	31.6	91.2	35.0	5.6
LCT-43	1.38	28.7	4.95	33.6	90.3	38.0	20.8
Ave.	2.1	29.2	4.9	32.5	91.7	46.1	21.1
Deep- Primary							
LCT-6	1.65	30.0	3.45	32.9	91.8	45.0	24.7
LCT-7	2.42	30.8	2.45	32.6	92.0	44.7	33.9
LCT-9	2.31	30.2	8.40	31.6	90.5	55.3	31.4
LCT-13	2.18	30.7	2.88	32.9	96.2	49.4	30.0
LCT-14	2.10	30.3	4.58	32.4	93.7	48.8	25.8
LCT-22	1.90	30.3	4.76	33.6	94.0	49.1	16.7
LCT-30	1.83	29.1	4.56	32.9	93.9	48.6	23.0
LCT-36	0.60	28.6	3.53	31.6	90.0	39.0	5.4
LCT-37	1.62	32.3	4.54	32.6	93.1	39.4	27.5
LCT-39	1.73	32.2	3.93	33.7	93.7	42.9	26.7
Ave.	1.8	30.5	4.3	32.7	92.9	46.2	24.5
Overall							
Ave.	1.99	29.3	5.18	32.4	91.0	47.1	23.0
Max.	3.12	35.5	13.7	40.5	97.1	59.2	45.1
Min.	0.60	17.8	2.5	26.4	80.9	31.0	5.4
SD	0.50	3.0	2.1	2.1	3.3	6.6	8.5

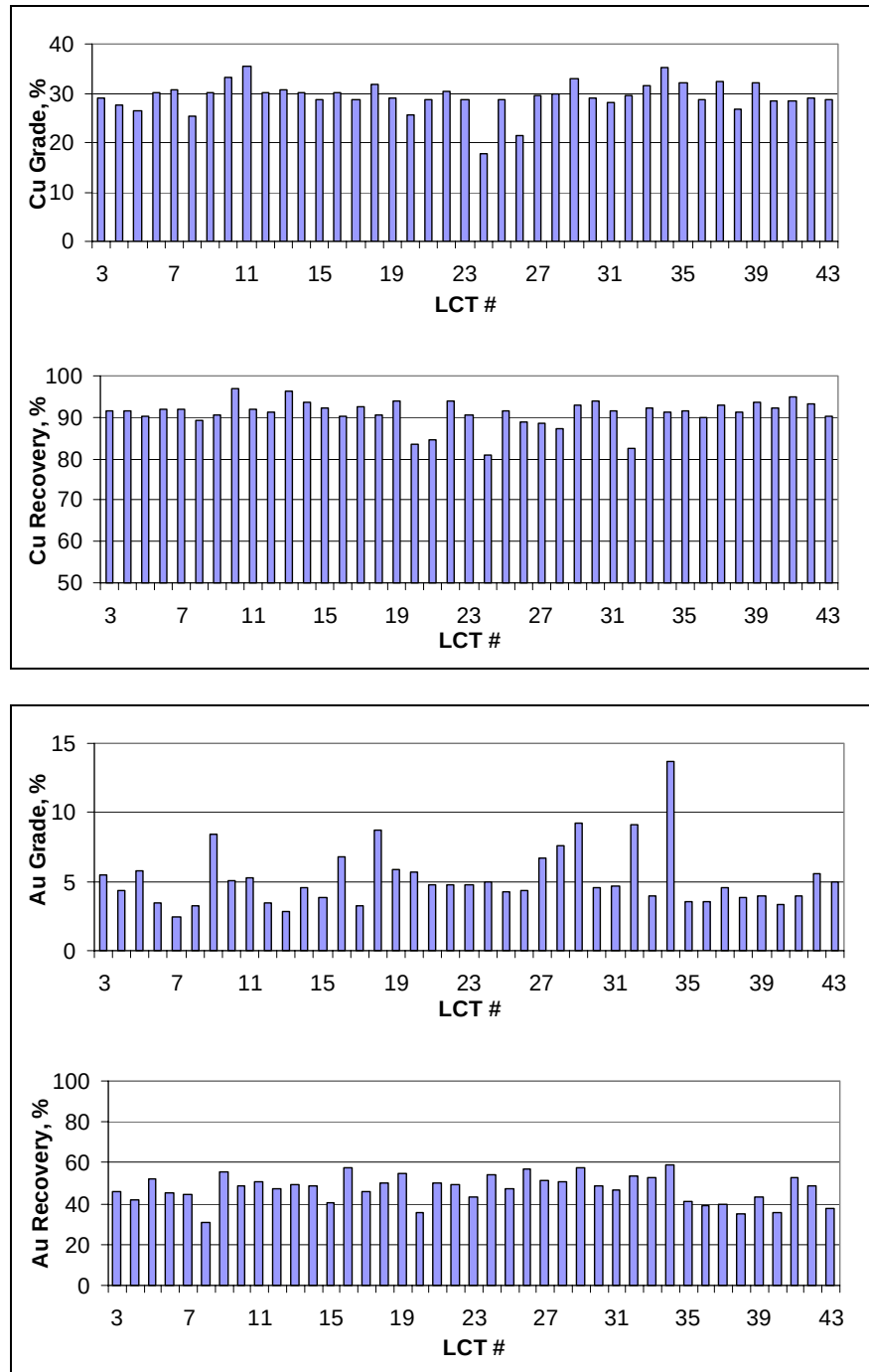


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Final concentrate grades and recoveries are summarized in Figure 5-25.

Figure 5-25: Grade and Recovery for the Variability Locked Cycle Tests





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The following overall LC final concentrate grade trends were noted:

- The overall copper grade average was adversely affected by two poor results, LCT-24 (VC-33) and LCT-26 (VC-24) produced copper grades of 17.8% and 21.5%, respectively.
- For the remaining 39 composites copper grades ranged from 25.2% to 35.5%, with an average grade of 29.8%.
- Both VC-33 and 24 are shallow granodiorites and it was believed that soluble copper from more dominant secondary copper mineralization in these samples resulted in pyrite activation during regrind, which in turn diluted the final concentrate grade. It was believed that by operating the rougher for these samples more selectively, by increasing the pH from 9.5 to 10.5 to depress more pyrite, the potential for pyrite activation in the regrind and cleaner circuit would be less and the final concentrate grade would improve.
- Subsequent batch testwork on VC24 and VC 33 at higher rougher pH's confirmed the above. The final concentrate grades improved to 30.7% and 32.1% respectively.
- In an effort to characterize the effect of pH on concentrate grade for surface samples, an additional composite VC33 was generated from the top eight intervals (25 m) of holes M69 and M80. The results of this are shown in Figure 5-26. This indicates the higher pH rougher strategy successfully mitigates the copper activation effects of secondary copper mineralization in near surface ore expected during early mining, and the plant lime distribution systems will be designed for this scenario. A subsequent locked cycle test was conducted at lower pH and produced a copper concentrate grade of 27.6% at over 94% recovery.
- The effect of feed copper head grade on mass pull to the final concentrate is shown in Figure 5-27. Increasing copper head grade results in steadily increasing mass pull. This is consistent with the relatively similar grades and recoveries observed between the variability composites. The two points that show a mass pull of greater than 3%, correspond to VC-24 and VC-26: the tests with poor copper concentrate grades.
- Gold grades in the final concentrate ranged from 2.5 g/t to 13.7 g/t and averaged about 5.2 g/t.

The following overall LC recovery trends were noted:

- The average copper recovery was 91% and ranged from 80.9% to 97.1%, with 33 of the 41 tests (80%) recovering 90% or better.



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Figure 5-26: Effect of Rougher pH on Grade and Recovery

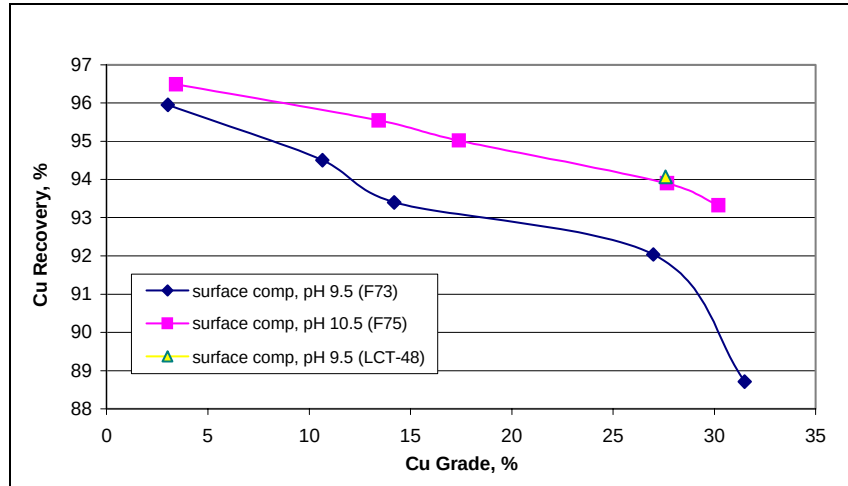
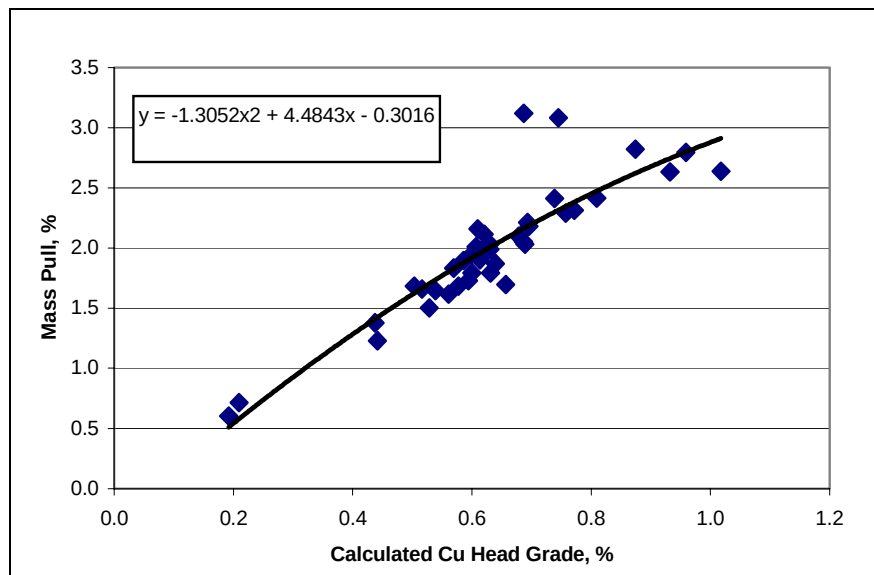


Figure 5-27: Relationship between Mass Pull to final Concentrate Grade and Head Grade



- Figure 5-28 breaks down the corresponding copper losses between the rougher tail (average copper loss: 6.2%) and the cleaner scavenger tail (average copper loss: 2.8%).
- The average gold recovery for the locked cycle tests was 49.2% and ranged between 31% and 59.2%.

Metallurgical performance trends by ore type and depth are presented in Figure 5-29.



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Figure 5-28: Cu Losses in the Rougher and Cleaner Circuits

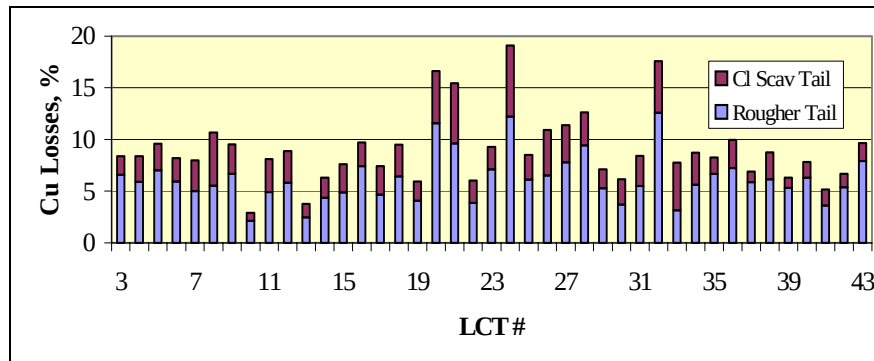
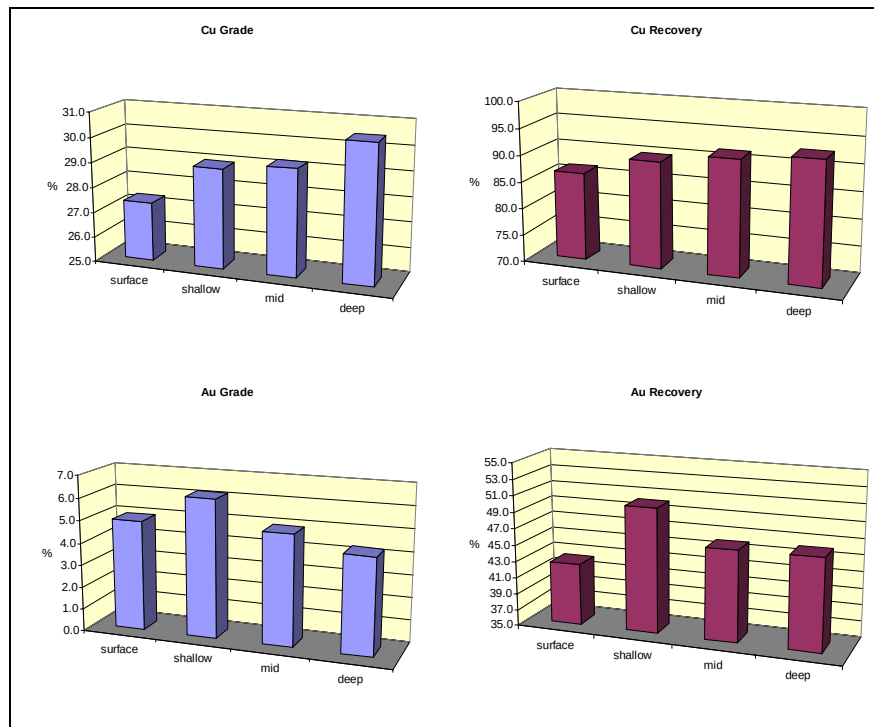


Figure 5-29: Comparison of Metallurgical Performance by Ore Type and Depth



The following metallurgical performance trends were noted by ore type and depth:

- Copper metallurgy generally improves by depth. This corresponds to the transition in ore type by depth from surface secondary copper supergene enrichment to dominantly primary chalcopyrite copper in the mid and deep zones.
- The surface composites showed the lowest average copper grade and recovery at 27.3% and 86.5% respectively, but it should be noted that this is based on only



three tests conducted at the standard locked cycle neutral rougher pH. It was shown earlier that the final concentrate grade of supergene material can be improved by increasing the rougher pH.

- Gold recoveries and grades are higher in the near surface material and decline by depth, and this trend simply mirrors the gold head grade profile.

5.1.8 Concentrate Dewatering

A series of four 10 kg locked cycle tests were conducted on new Master Composite samples prepared from the variability composites. A summary of the flotation results is provided in Table 11 of Lakefield's Progress Report No. 2. The metallurgy and equipment conditions were not optimized in these floats as the objective was to prepare bulk concentrate samples for subsequent settling and dewatering tests. The final concentrate grades and recoveries were lower partly because of too small of a flotation cell being used in the 1st cleaner stage. Therefore, these results should not be used to derive metallurgical performance projections.

Settling Tests

Concentrate settling tests were conducted on the combined cleaner concentrates from three of the locked cycle tests. Magnafloc 10 was added as flocculent in target doses of 5 g/t, 10 g/t, and 20 g/t. Rapid settling was observed for all samples under all flocculent dosages.

Filtration Tests

Several vacuum and pressure filtration tests were conducted tests were conducted on thickened concentrate from the settling tests.

Vacuum filtration resulted in relatively low filtration rates of up to 150 kg/m²/h, low cake thicknesses of 10 mm and high moistures of 18%. This is most likely due to fine nature of the concentrate.

Concentrate samples were sent to Larox for pressure filtration testing. This resulted in a recommended design filtration rate of 450 kg/m²/h and a cake moisture of 9%. The filtration rate determined by testing was discounted to account for declining cloth efficiency during normal life-cycle operation.



5.1.9 Concentrate Quality

Chemical Composition

A summary of the chemical composition of the individual locked cycle final concentrate products produced in variability testing is presented in Table 5-10. Chemical analysis indicates no significant deleterious penalty element impurities are present and this is in good agreement with mineralogical mapping results on head samples. The following observations can be made:

- Copper in concentrate ranged from 25% to 35.5% and averaged 29.2%, while gold varied from 2.5 g/t to 13.7 g/t. Silver content was variable, but on average exceeded the credit threshold expected from most smelters.
- Arsenic and antimony combined accounted for less than 700 g/t on average, well below the usual penalty threshold of copper smelters. Lead and zinc combined were less than 500 g/t. Fluorine and chlorine averaged about 100 g/t.
- Selenium ranged from 30 to 140 g/t and averaged 90 g/t. Some smelters occasionally attempt to impose modest penalties for selenium concentrations above 100 g/t.
- Silica and alumina combined concentrations averaged about 5% indicating the concentrates are relatively free of gangue contamination.
- Mercury concentrations ranged from 0.3 g/t to 24 g/t and averaged 3 g/t. Some smelters impose a penalty above 10 g/t, but 95% of the samples were below this level and it is reasonable to believe peaks will be blended out in operation.
- The S/Cu ratio of the concentrates averaged 1.13, and reflects a dominantly chalcopyrite mineral composition which is highly desirable to smelters.
- Overall the concentrates are regarded as relatively clean, have a simple composition and are not expected to attract any significant penalty charges. Their overall clean quality and favourable S/Cu ratio makes them relatively attractive and marketable to smelters.

Spontaneous Combustion

Standard self-heating (pyroforicity) tests were conducted on a combined concentrate sample produced from all four 10 kg locked cycle tests. The results indicated that the concentrate was not expected to be self-heating.

Transportable Moisture

The flow and transportable moisture limits for the concentrate were determined to be 12.1% and 10.9% respectively.



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Table 5-10: Cu Concentrate Quality

Period Parameter	Overall Life of Mine			Year 1-2 (Surface and Shallow Composites)		
	Max.	Min.	Average	Max.	Min.	Average
Cu, %	35.5	25.0*	29.2	35.5	25.0	28.0
Fe %	36.0	25.0	29.0	36.0	25.0	29.1
S, %	40.5	26.4	32.4	40.5	26.4	31.9
S/Cu	1.70	0.83	1.13	2.27	1.13	1.16
Au, g/t	13.7	2.5	5.2	13.7	3.3	5.6
Ag g/t	170	29	64	93	29	58
Al ₂ O ₃ %	3.2	0.5	1.3	3.2	0.5	1.7
SiO ₂ %	9.0	1.9	4.0	9.0	2.1	5.1
Al ₂ O ₃ % + SiO ₂ %	12.2	2.4	5.3	12.2	2.6	6.8
MgO %	0.28	0.05	0.15	0.28	0.05	0.17
Pb %	0.6	0.0	0.1	0.1	0.1	0.1
Zn %	3.5	0.0	0.4	2.6	0.0	0.3
Pb% + Zn%	4.1	0.0	0.5	2.7	0.1	0.3
Hg g/t	24.0	0.3	3.1	5.0	0.4	1.5
Bi g/t	43	-	-	43	-	-
Se g/t	140	30	91	120	30	77
Cl %	0.04	0.00	0.01	0.04	0.00	0.01
As %	0.19	0.00	0.03	0.09	0.00	0.02
Sb %	0.30	0.00	0.04	0.05	0.00	0.01
Ba %	0.05	0.01	0.02	0.05	0.01	0.02
K %	0.91	0.17	0.40	0.91	0.17	0.49
Ti %	0.15	0.03	0.08	0.13	0.03	0.09
Ca %	0.39	0.07	0.20	0.39	0.07	0.24
Mn %	0.04	0.00	0.01	0.03	0.00	0.01
Mo %	0.16	0.00	0.06	0.15	0.00	0.04
Na %	0.18	0.01	0.05	0.18	0.01	0.08
Ni %	0.06	0.00	0.01	0.05	0.00	0.02
Be g/t	0.20	0.04	0.07	0.2	0.0	0.1
Cd g/t	160	<3	44	140	6	21
Co g/t	120	16	35	120	18	39
Cr g/t	1200	6	189	950	8	224
Li g/t	<5	-	-	<5	-	-
P g/t	400	<30	177	400	<30	-
Sn g/t	<20	-	-	<20	-	-
Sr g/t	31	5	14	31	5	16
Tl g/t	<30	-	-	<30	-	-
V g/t	140	10	32	120	10	26
Y g/t	7.6	1.8	3.2	7.6	2.0	3.9
Th g/t	<60	-	13	<60	-	13
U g/t	<50	-	1	<50	-	1
Re µg/g	5.5	0.6	2.1	5.5	0.7	1.7
F %	0.04	0	0.01	0.04	0	0.01

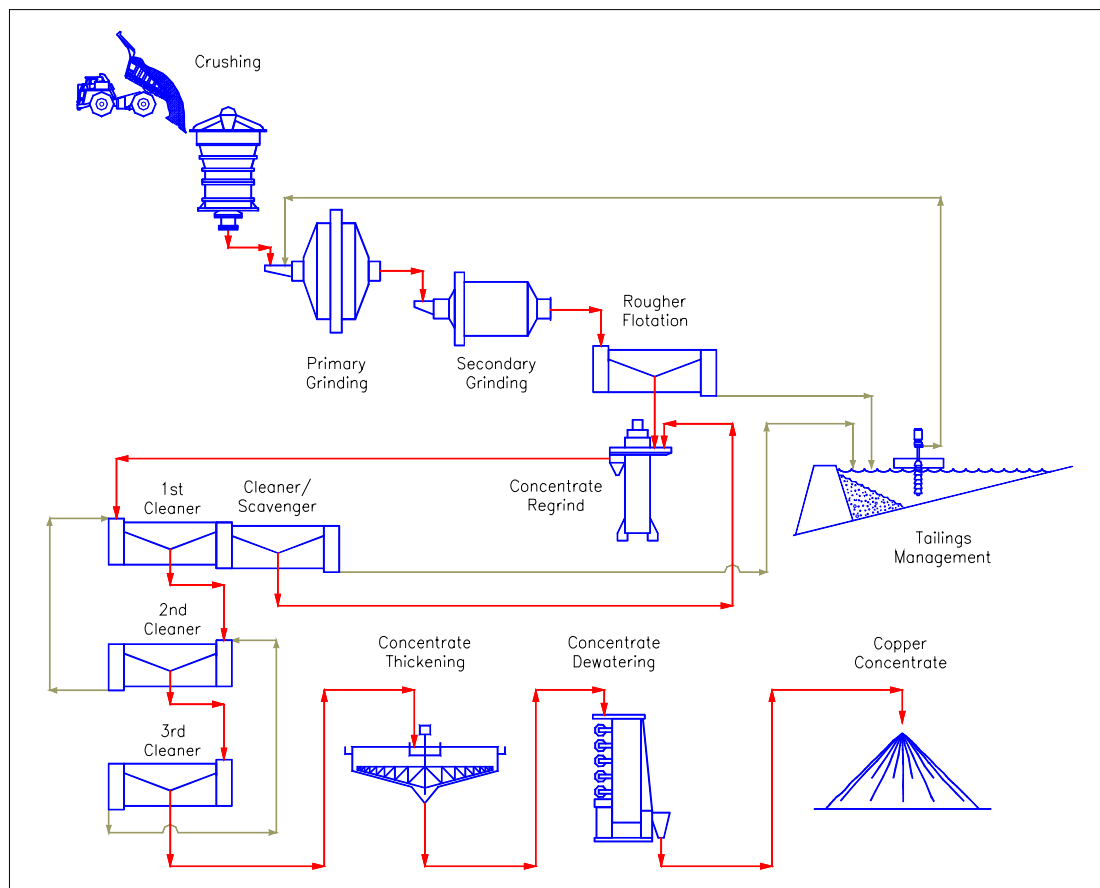


5.2 Flowsheet Development

The estimated mineral reserves included in the mine plan total approximately 111 Mt grading 0.67% Cu and 0.22 g/t Au. Silver and molybdenum are present but the grades are relatively low. Approximately 91 Mt of waste rock will be removed over the mine life, resulting in an average strip ratio of 0.8:1. The process will be designed to treat 25,000 t/d of material.

A simplified schematic drawing of the proposed flowsheet is provided in Figure 5-30. Run-of-mine open pit ore will be crushed in a gyratory crusher. The crushed ore will be processed by means of semi-autogenous and ball mill grinding followed by rougher flotation, regrind, cleaner flotation, and dewatering to produce copper concentrate. The concentrate will be trucked via the existing road network in the area to a port facility in Machala for shipment to smelters. Tailings from the process will be impounded in a tailings pond; water will be reclaimed from the tailings pond and reused in the process.

Figure 5-30: Simplified Mirador Flowsheet





5.2.1 Design Basis

Ore Types

The Mirador deposit was divided into four main ore types for testing based on copper mineralogy and depth:

- near surface – supergene (<5% of ore within 0 m to 20 m cap)
- shallow-mixed (includes the above zone since they will be mined concurrently)
- mid- and deep-primary.

There is a minor amount of leached material overlying the ore body in places, but represents unmineralized material in the mine plan and is treated as waste. Each of the ore types was also evaluated with respect to variation by the two main rock type zones, granodiorite, and breccia. Testing indicated no material metallurgical differences between these two lithologies, and the mid- and deep-primary ores. Mid- and deep-ore are dominantly chalcopyrite. The near surface and shallow-mixed ore contains relatively more secondary copper mineralization with a finer texture. In the initial years of operation, this dictates a finer regrind target and the use of a more selective rougher flotation mode of operation by operating at a higher pH. Lime is used to adjust pH and the lime system will be designed to accommodate the higher lime consumption expected in first year of operation. Thereafter, and for most of the mine life, the rougher flotation will generally be operated in a bulk flotation mode at a relatively neutral pH.

Overall, the ore is regarded as relatively simple and has a relatively low variability in metallurgical response within the proposed pit and by depth. The base of the current pit corresponds to an anhydrite alteration front, which is expected to result in a relatively fine SAG mill feed for most of the mine life. Below this front, hardness to SAG milling is expected to increase, but this material is not considered in this study.

Selection of Design Feed Grades

The design copper feed grades were selected with the objective of minimizing capital costs while accommodating the anticipated head grade fluctuations during the life of the mine. The mine plan indicates that average annual copper grade will be about 0.67% and will be relatively consistent, peaking at about 0.76% in the first two years of operation. The metallurgical variability samples copper grade averaged about 0.68% with a peak of about 1.03% for benches of some Shallow-Mixed Ore Type in Years 1 and 2 ore. The standard deviation in grade was estimated to be 0.18. The 80% confidence limit of 0.92% ($1.3 * 0.18 + 0.68\%$) was selected as the basis for the design



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copper grade. Head grade excursions above the design grade will be dealt with through in-pit and stockpile blending. General production and grade criteria are summarized in Table 5-11.

Design Criteria and Mass Balance

A detailed mass balance covering all streams within the process plant has been prepared based on daily throughput requirements, plant availability, and the design feed grades for copper. The mass balance representing the expected flows of solids, water, and slurry when processing ore at the Year 1 feed grades of 0.72% Cu is provided in Appendix C.

Table 5-11: Production and Grade Criteria

Parameter	Average	Design
Ore grade – % Cu average	0.67	0.92
Ore grade – g/t Au	0.22	-
Tonnes milled t/a	-	9,125,000
Operating d/a	-	365
Tonnes milled/calendar day	-	25,000
Ore specific gravity	-	2.7
Ore bulk density – t/m ³	-	1.6
Ore moisture content – %	-	5.0
Concentrate production – dt/a	187,000	-
Concentrate production – dt/d	536	677
Concentrate production – t/h	23.5	29.5
Concentrate % wt of feed	2.15	2.75
Concentrate grade – % Cu	30	-
Copper recovery – %	91	-
Gold recovery – %	47	-

The design criteria established for the overall ore processing facilities are summarized below and detailed in Appendix C. The flowsheet based on these criteria is described in the sections following.

5.2.2 Primary Crushing and Reclaim

The mine production rate and haul truck size indicated the need for a 42 x 65 gyratory crusher for the primary crushing duty. The primary crusher design is based on an availability of 75%, resulting in a design throughput rate of 1,875 t/h. The crusher will operate with an open side setting of 150 mm. Figure 5-31 depicts SAG feed granulometry obtained from surveys on porphyry copper circuits. Based on the

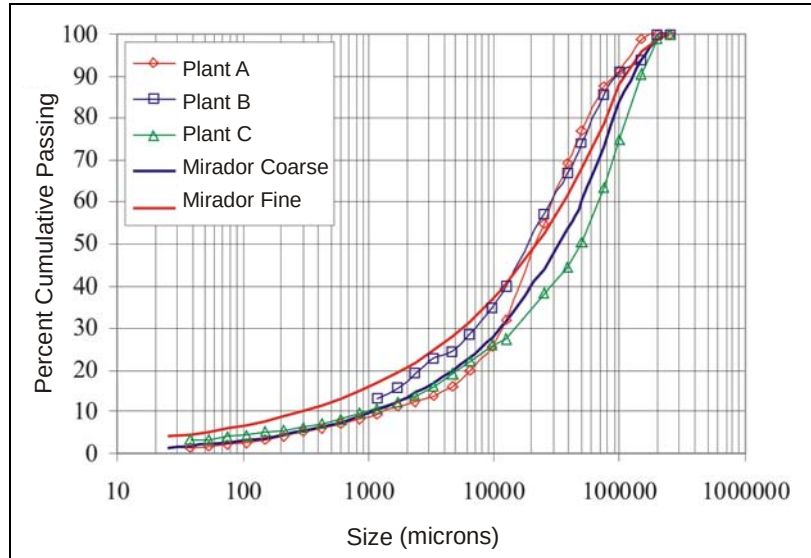


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fractured state of the drill cores, and the anhydrite alteration nature of the ore, it is reasonable to believe the fine feed will be more representative of the crusher product and SAG feed for Mirador. A summary of the main primary crushing, overland conveying, and coarse ore storage are presented below.

Figure 5-31: Copper Porphyry Plant Feed Size Survey Data



Primary Crushing

Crusher operation – d/wk	7
Crusher operation – h/d	24
Crusher availability – %	75
Daily capacity – t/d	25,000
Crushing rate – t/h design	1,875
Crusher type	gyratory
Crusher size – inches	42 x 65
Primary crusher OSS mm	150
Primary crusher product P80 mm	77
Dump truck capacity – t design	100
Primary crusher dump hopper – t	150
Primary crusher discharge bin – t	250
Primary crusher discharge feeder – type	apron



Overland Conveyor

Overland conveyor No.1 – size mm x m	1,219 mm x 4,000 m long x 430 m drop
Overland conveyor No.2 – size mm x m	1,219 mm x 6,000 m long x 55 m lift

Coarse Ore Stockpile

Live ore storage – t	25,000
Total ore storage – t	132,000
Number of reclaim feeders	3
Reclaim feeder type	apron

5.2.3 Primary Grinding

A JKSimMet model was developed for the Mirador circuit. The results of the drop-weight and SMC tests as well as the ball mill work indices were used in the model to describe SAG and ball mill hardness. The throughput target was 25 kt/d with an availability of 94% and final P_{80} of 150 μm . A 32' x 16' (14.25 EGL) SAG mill with 11,000 hp, followed by a 24' x 34.5' ball mill with 12,500 was selected.

The first series of simulations was performed on a SAB (SAG and ball mill) configuration which indicated the SAG mill was throughput limiting. The simulations were performed using two different SAG feed size distributions (F_{80} of 90 and 77 mm) and for two SAG mill ball charges corresponding to safe (12%) and aggressive (15%) design scenarios. For the SAG classification, two different grate sizes (50 and 75 mm) were simulated, and the screen aperture size was fixed at 12.7 mm. Because the circuit was mainly SAG mill limited a second series of simulations was performed on a SABC (SAG and pebble crusher and ball mill) configuration. The SABC flowsheet is presented in Figure 5-32. The simulations were performed with the grate size set at 75 mm to allow pebble extraction and still using the fine feed size (F_{80} of 77 mm). The pebble crusher was simulated with a design circulating load of 30%. The throughput rate was fixed to 25 kt/d, and the SAG ball charge was allowed to vary to handle the variation in hardness. The ball mill power requirement to achieve the 150 μm target was calculated for each simulation, and the results are presented in Table 5-12. The results indicate that the SABC circuit could achieve the required throughput of 25.0 kt/d 100% of the time, based on the samples tested. The predicted ball charge requirement for the various samples varies between 9% and 12%, with an average of 11%. The average SAG and ball mill power requirements are 6.3 kWh/t and 6.7 kWh/t, respectively, or 13.0 kWh/t overall. Full details of the simulations are provided in Lakefield's comminution circuit design report in Appendix C.



Figure 5-32: SABC Circuit

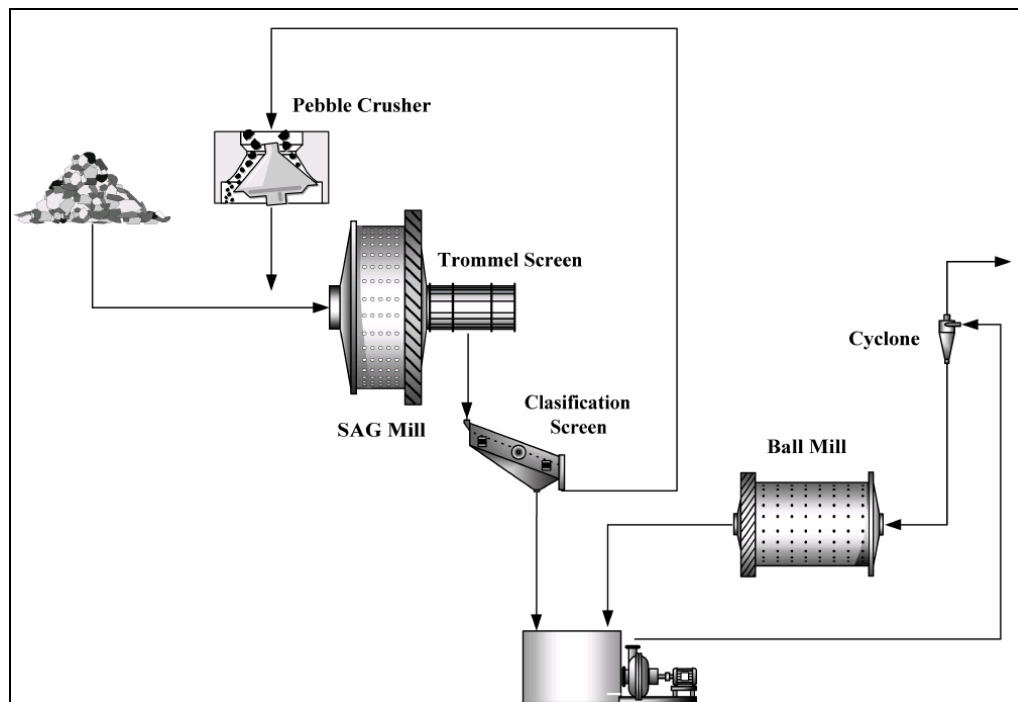


Table 5-12: Variability Simulation Summary, SABC – 25 kt/d and 150 μm P_{80}

	SAG Mill							Ball Mill				Overall		
	Ball		Throughput		T_{80} (μm)	Gross Power Requirement		Gross Power Requirement		P_{80} (μm)	Gross Power Requirement (kW)	Gross Power Requirement		
	DWI (kWh/t)	Charge (%)	(t/h)	(kt/d)		(kW)	(kWh/t)	BWI (kWh/t)	(kW)			(kW)	(kWh/t)	(WIO)
M86	5.50	10	1,100	25.0	2,001	6,787	6.2	14.5	150	7,410	6.7	14,197	12.9	15.9
M86:140-159	5.48	10	1,100	25.0	2,018	6,821	6.2	15.1	150	7,516	6.8	14,337	13.0	16.0
M86:182-203	7.51	12	1,100	25.0	2,264	7,346	6.7	14.3	150	7,150	6.5	14,496	13.2	16.2
VC-2	6.32	11	1,100	25.0	2,134	7,038	6.4	13.8	150	6,958	6.3	13,996	12.7	15.7
VC-29	6.56	10	1,100	25.0	2,095	6,971	6.3	15.0	150	7,487	6.8	14,468	13.2	16.2
VC-22	4.05	9	1,100	25.0	1,815	6,415	5.8	14.2	150	7,011	6.5	13,426	12.2	15.0
VC-30	6.69	10	1,100	25.0	2,120	7,014	6.5	15.0	150	7,582	6.9	14,596	13.3	16.3
VC-40	4.06	9	1,100	25.0	1,899	6,475	5.9	11.3	150	5,757	5.2	12,232	11.1	13.7
VC-11	6.73	12	1,100	25.0	2,166	7,144	6.5	15.1	150	7,433	6.8	14,577	13.3	16.3
VC-31	7.58	11	1,100	25.0	2,226	7,230	6.6	14.6	150	7,315	6.7	14,545	13.2	16.3
VC-10	5.36	10	1,100	25.0	2,021	6,804	6.2	15.0	150	7,544	6.9	14,348	13.0	16.0
VC-13	6.65	11	1,100	25.0	2,138	7,065	6.4	15.1	150	7,509	6.8	14,574	13.2	16.3
VC-28	6.67	12	1,100	25.0	2,250	7,279	6.6	14.9	150	7,699	7.0	14,978	13.6	16.7
VC-32	6.54	11	1,100	25.0	2,164	7,099	6.5	15.0	150	7,558	6.9	14,657	13.3	16.4
VC-42	5.82	11	1,100	25.0	2,096	6,954	6.3	14.6	150	7,384	6.7	14,338	13.0	16.0
VC-42	6.77	12	1,100	25.0	2,189	7,181	6.5	16.3	150	8,158	7.4	15,339	13.9	17.2
Average	6.14	11	1,100	25.0	2,100	6,977	6.3	14.6	150	7,342	6.7	14,319	13.0	16.0



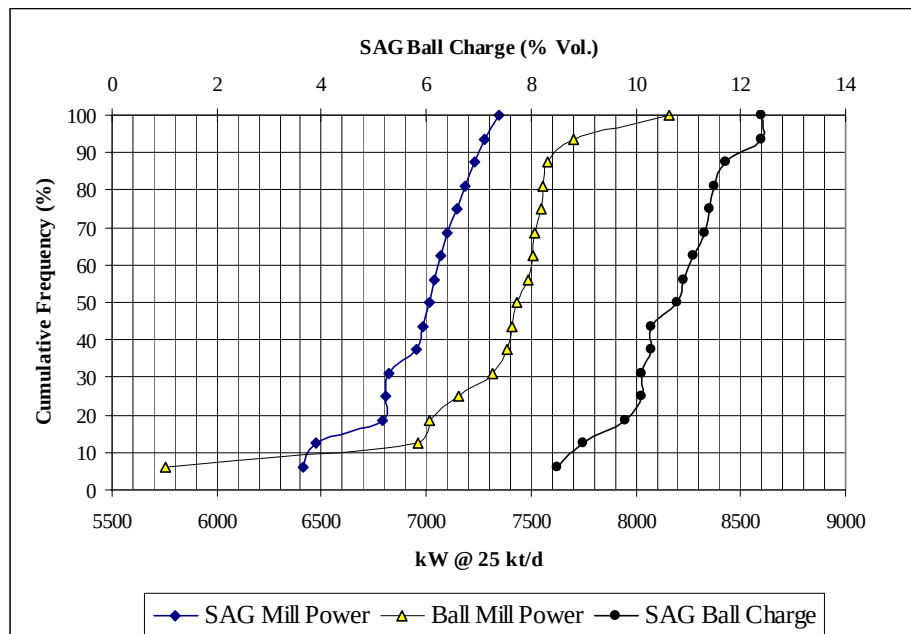
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Figure 5-33 presents the cumulative frequencies of SAG ball charge as well as power requirements for the SAG and ball mills for the SABC circuit configuration. The dispersion in power requirement is small with relative standard deviations of 3.8% and 6.9% for the SAG and ball mill, respectively. The median, 75th and 90th percentile of ball mill powers are relatively close, ranging between 10,000 and 10,250 hp. Therefore, the 90th percentile value was used for the ball mill design.

MinnovEx developed a CEET model for the Mirador circuit. The results of the SPI tests as well as the ball mill work indices were used in the model to describe SAG and ball mill hardness. Full details of this are reported in MinnovEx's grinding circuit evaluation report in Appendix C. These simulations were conducted at lower throughputs corresponding to an earlier design tonnage scenario. However, the results are in good agreement with the JK tech simulation conclusions at comparable tonnages.

Figure 5-33: SAG and Ball Mills Power Requirement



Grinding Circuit General

Operating days per year – d/a	365
Operating schedule – h/d	24
Grinding circuit availability – %	94
Number of grinding circuits	1



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Grinding circuit type	SABC
Feed rate – t/h	1,100
Ball mill work index (metric Wi)	15.0
Impact work index (metric Wi)	6.00
SPI index – minutes	78.0
Ai	0.35
JK A x b	35 to 63
JK ta	0.57

Primary Grinding

SAG feed size – K ₈₀ µm	77,000
SAG product size – K ₈₀ µm	1,800
SAG power applied (Ave.) – kWh/t	6.50
SAG power applied (Max.) – kWh/t	7.50
Number of primary mills	1
primary mill type	semi-autogenous
primary mill size	32' Ø x 14.25' EGL
primary mill horsepower	2 x 5,500
primary mill motors	WRIM
Primary mill ball charge – % vol – nominal operating	12
Primary mill ball charge – % vol – mechanical design	15
Primary mill classification type	vibrating screen
Primary mill screen aperture – mm bottom deck	13 x 50
Primary mill design circulating load – %	30
Primary mill pebble crusher	Cone
Pebble crusher feed size – mm	65
Pebble crusher product size – mm	13

Secondary Grinding

Number of secondary mills	1
Secondary mill type	ball mill
Ball mill work index (metric Wi) – design	14.5
Ball mill feed size – K ₈₀ µm – typical	1,800
Ball mill circuit product size – K ₈₀ µm	150



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Ball mill power applied (at pinion) – Ave. kWh/t	7.00
Ball mill power applied (at pinion) – Max. kWh/t	8.50
Secondary mill size	24' Ø x 34.5' long
Secondary mill horsepower	2 x 6,250
Secondary mill motors	WRIM
Secondary mill circulating load – %	250
Secondary mill classification	hydrocyclone
Cyclone diameter – mm	650
Number of cyclones	13 (11 op / 2 stby)

Regrinding

Rougher concentrate (new feed) – nominal t/h	143
Rougher concentrate (new feed) – process design t/h	166
Regrind mill work index (metric)	13.5
Regrind mill feed size – K_{80} µm	125
Regrind mill product size – K_{80} µm	25
Regrind mill type	vertical tower
Regrind mill total motor power installed – hp	2,000
Number of regrind mills	2
Regrind mill circulating load – %	150
Regrind mill classification	hydrocyclone
Cyclone diameter – mm	150
No of cyclones	28 (24 op / 4 stby)

5.2.4 Flotation

Rougher/Scavenger Flotation

Rougher feed solids – %	33.0
Rougher feed flowrate – m ³ /h nominal	2,660
Process design factor	1.10
Rougher/scavenger residence time – minutes	25
Aeration factor – %	10
Flotation cell volume – m ³	100
Number of cells	14
Number of lines	2



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Number of cells per line	7
Rougher concentrate – % weight of feed (nominal)	12.9
Rougher concentrate – % weight of feed (process design)	15.0
Rougher concentrate density – wt% solids	30

Cleaner Flotation

Cleaner No.1 feed flowrate – m ³ /h nominal	867
Cleaner No.1 residence time – minutes	20
Flotation cell volume – m ³	50
Number of first cleaner cells	8
Number of cleaner lines	2
Number first cleaner cells per line	4

Cleaner scavenger flotation time – minutes	20
Cleaner scavenger cell size – m ³	50
Number of first cleaner scavenger cells	8
Number of first cleaner scavenger cells per line	4
Cleaner scavenger concentrate density – wt% solids	25

Cleaner No. 2 flotation time – minutes	11
Cleaner No.2 feed flowrate – m ³ /h nominal	136
Cleaner No.2 volume (aerated) m ³	34
Cleaner No. 2 cell size m ³	5
Cleaner No. 2 No. cells	7
Cleaner No.2 concentrate density – wt% solids	30

Cleaner No. 3 flotation time – minutes	11
Cleaner No.3 feed flowrate – m ³ /h nominal	93
Cleaner No. 3 cell size	5
Cleaner No. 3 No. cells	5
Cleaner No.3 concentrate density – wt% solids	30

5.2.5 Concentrate Dewatering

The proportion of fresh feed weight recovered to the final cleaner concentrate will vary with the copper feed grades, as indicated by the relationship shown in Figure 5-26.



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Based on a design copper head grade of 0.92% the weight recovery of final copper concentrate will be about 2.75%. This results in a process design factor of about 1.25 over nominal production for the concentrate handling facilities.

Concentrate Thickening and Storage

Number of thickeners	1
Thickener type	conventional
Feedrate, nominal t/h	23.5
Feedrate, design t/h	30.0
Design factor	1.25
Thickener unit area – t/d/m ² – design	2
Thickener diameter – m	20
Thickener underflow density – % solids	60

Concentrate Pipeline Transportation

Pump type	slurry pump
Concentrate pumping flowrate – m ³ /h – design	30
Concentrate pumping % solids	56

Concentrate Filtration and Drying

Filter plant stock tank capacity – h (nominal)	16
Number of copper concentrate filters	1
Filter type	pressure
Filter feed – t/h nominal	23.5
Filter feed – design	30.0
Filter unit area – kg/h/m ²	450
Filter size – area m ²	72
Final concentrate moisture – %	9.0
Copper concentrate storage capacity – d	5
Copper concentrate storage capacity – t	2,760



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5.2.6 Tailings Handling

Process water will be re-used as much as possible around the plant. Tailings flow by gravity to the tailings impoundment.

5.2.7 Water and Air Services

Fresh water will not be used as process water unless the recycle/reclaim water supply is insufficient. Cooling systems are not considered to be consumption points because the water is continually re-used in a closed circuit.

5.3 Projected Metallurgy and Metallurgical Balance

A summary of the projected metallurgical recoveries and metal balance from Lakefield's testwork is provided in Table 5-13.

Table 5-13: Mirador Metallurgical Recoveries and Metal Balance

Ore Type Depth/Phase	Supergene (<20 m)	Shallow (Mixed) (0-5 yr) #1	Mid-Primary (6-10 yr)	Deep-Primary (11-15 yr)	Overall (0-15 yr)
Copper concentrate grade, %	27.4	30.0	29.2	30.5	29.8
Copper recovery, %	88.4	89.8	91.7	92.9	91.2
Gold recovery, %	44.2	48.1	46.1	46.2	47.0
Copper losses:					
<i>Rougher tails, %</i>	7.9	6.7	5.8	5.0	6.0
<i>Cleaner scav. tails, %</i>	3.7	3.4	2.5	2.1	2.8
Gold losses:					
<i>Rougher tails, %</i>	27.7	29.2	28.6	34.3	30.2
<i>Cleaner scav. tails, %</i>	28.1	22.7	25.3	19.5	22.9

5.4 Process Description

The flowsheets, plant layout drawings, and mechanical equipment list are shown in Appendix D.

5.4.1 Crushing

Run-of-mine (ROM) ore will be dumped from 140-ton mine trucks into the dump pocket of a primary 42" x 65" gyratory crusher. Crushed ore will discharge into a surge pocket below the primary crusher. Ore will be withdrawn from the surge pocket by a



1,370 mm wide x 6.80 m long variable speed apron feeder. The feeder will discharge onto a 1,525 mm wide x 26.1 m long crusher discharge conveyor, which will discharge onto the overland conveyor.

5.4.2 Overland Conveyors

Two overland conveyors in series will be used to transport coarse ore from the gyratory crusher to the coarse ore stockpile. The first conveyor starts at crusher area and will travel west and into the Rio Tundayme valley before transferring onto the second conveyor. The first overland conveyor is 1,219 mm wide x 4,000 m long with a vertical drop of 430 m, while the second overland conveyor is 1,219 mm wide x 6,000 m long with a vertical lift of 55 m. The covered conveyor will be equipped with walkways running along side the conveyor.

5.4.3 Coarse Ore Stockpile and Reclaim

The uncovered coarse ore stockpile will have a nominal live capacity of 25,000 t, or the equivalent of one day's operation at the throughput rate. Ore will be reclaimed from the coarse ore stockpile using three 1,219 mm wide x 7.2 m long variable speed apron feeders to feed the SAG mill feed conveyor. The 1,067 mm wide x 141 m long SAG mill feed conveyor will deliver ore to the SAG mill. An electronic weigh scale on this belt will send a signal to control the speed of the apron feeders to deliver the required tonnage.

5.4.4 Grinding

Ore will be ground in two stages to produce a product suitable for flotation. The first stage will be a 32 ft $\varnothing$ x 14.25 ft (effective grinding length) semi-autogenous grinding mill driven by two 5,500 hp wound rotor induction motors. The SAG mill discharge is classified with a vibrating discharge screen: the undersize passes through to the cyclone feed pumpbox where it is combined with the ball mill discharge, while the oversize is transferred via a series of conveyors and pebble crusher back to the SAG mill feed.

The cyclone feed pumpbox slurry is classified by a cyclopac consisting of thirteen 650 mm $\varnothing$ cyclones (11 operating, 2 standby), which produce a 150 μ m P₈₀ product. The cyclone underflow is sent to the second stage of grinding, which will consist of one 24 ft $\varnothing$ x 34.5 ft long ball mill driven by two 6,250 hp wound rotor induction motors. The ball mill discharges into the cyclone feed pumpbox. Two 28" x 26" variable speed pumps (one operating, one standby) will be used as cyclone feed pumps.



Cyclone overflow, at 33% solids and with a P_{80} of 150 μm is sent to the flotation circuits.

5.4.5 Flotation and Regrinding

Grinding circuit cyclone overflow will gravitate to a rougher flotation distributor that splits to two rows of seven 100 m^3 rougher flotation cells providing 25 minutes of residence time. Rougher concentrate will be combined with cleaner scavenger concentrate and pumped to the regrind circuit. Tailings from the rougher flotation circuit will gravitate to the rougher tailings box before it is discharged by gravity to the tailings management facility.

The primary regrind circuit will consist of two 1,250 hp regrind tower mills in closed circuit with a cyclopac consisting of twenty-eight 150 mm $\varnothing$ cyclones (24 operating, 4 standby). Cyclone underflow will return to the regrind mills, while cyclone overflow is combined with 2nd cleaner tailings and pumped to the cleaner flotation circuit.

The 1st cleaner flotation circuit consists of two rows of eight 50 m^3 cleaner/scavenger flotation cells (4+4) in series and provides 40 (20+20) minutes of residence time. The 1st cleaner concentrate is pumped to the 2nd cleaner circuit, while the 1st cleaner scavenger concentrate is returned to the regrind circuit. The 1st cleaner tailings will gravitate to the cleaner tailings box before it is discharged by gravity to the tailings management facility.

The 2nd cleaner circuit consists of one row of seven 5 m^3 flotation cells and provides 11 minutes of residence time. The 2nd cleaner tailings is returned to the 1st cleaner feed pumpbox, while the concentrate is pumped to the 3rd cleaner circuit.

The 3rd cleaner circuit consists of one row of five 5 m^3 flotation cells and provides 11 minutes of residence time. The 3rd cleaner concentrate represents the final mill copper concentrate, which is pumped to the concentrate dewatering circuit. 3rd cleaner tailings is combined with the 1st cleaner concentrate, and returned to the 2nd cleaner circuit.

Key samples from the flotation process will be sampled automatically and pumped to an on-stream analyser on the flotation floor. The five streams that will be sampled and analysed include the grinding cyclone overflow, rougher concentrate, rougher scavenger tailings, final concentrate, and cleaner scavenger tailings.



5.4.6 Concentrate Dewatering

Final concentrate will be thickened to 60% solids in a 20 m diameter concentrate thickener. The concentrate thickener overflow will be recycled to the process water tank. The thickened final concentrate will be pumped through a concentrate pipeline to the filter building.

At the filter building, the concentrate will be stored in a storage tank before it is pumped to the concentrate pressure filter, which has a discharge product containing 9% moisture. Filtrate is directed to the tailings management facility. The pressure filter product is conveyed to a concentrate storage area and trucked to market.

5.4.7 Concentrate Loadout

Covered storage for about 2,764 t of concentrate (5 days) is provided. Trucks will be loaded by front-end loader.

5.4.8 Tailings Disposal

Cleaner scavenger tailings and rougher tailings will flow by gravity through their respective floor launders into their respective tailings boxes. These two tailings streams will exit the facility by gravity to the tailings management facility.

5.4.9 Water Systems

The concentrator will use two kinds of water: fresh water and process water. Fresh water will be used for reagent mixing, process pump gland water, and cooling water for lube systems. The total projected use of fresh water at the concentrator is 90 m³/h. A dedicated fresh water tank with a retention time of four hours will be located at the concentrator. The source of fresh water will come from water wells located approximately 1 km away from the plant site.

Process water will be used primarily in the grinding, flotation, and regrind circuits. The total projected use of process water at the concentrator is 2,296 m³/h. A dedicated process water tank with a retention time of half hour will be located adjacent to the fresh water tank at the concentrator.

Reclaim water from the tails impoundment dam is used to provide the bulk of the process water requirements as well. The other source of process water comes from the copper concentrate thickener overflow, which amounts to 51 m³/h.



5.4.10 Reagent Handling Systems

The reagents consumed in this project are:

- quicklime, with an annual consumption of:
 - 11,500 t/a for first year
 - 6,700 t/a from Year 2 onwards
- collector No.1: Xanthate, with an annual consumption of 275 t/a
- collector No.2: Aerophine 3894, with an annual consumption of 185 t/a
- frother: MIBC, with an annual consumption of 185 t/a
- Flocculent, with an annual consumption of 10 t/a.

5.4.11 Plant Services

Air Supply

Both low-pressure flotation air and compressed air will be supplied to various areas of the plant.

Low-pressure air is required for the flotation circuits at the concentrator. This will be provided by two low-speed centrifugal blowers, each capable of delivering 30,000 m³/h. Only one unit will be needed at any one time, with the other serving as a spare.

Compressed air is required for the pressure filters at the filter plant. An air pressure of 2,000 kPa is required for diaphragm pressing, and 900 kPa is required for air-drying the concentrate cakes. The pressing compressor will supply a total 762 m³/h of compressed air, while the drying compressor will supply 1,476 m³/h.

Compressed air will also be provided to various outlets for general service throughout the plant. Use areas will include the lime dust baghouse shaker, pneumatic maintenance tools, and a coalescer that can supply instrument air as needed. Most instrument air will be provided by a dedicated oil-free compressor; the air supply will be processed through an electric air dryer before being delivered to the plant loop.

All blowers and compressors will be equipped with inlet filters to keep out ambient dust.



On-Stream Analyzer

An on-stream x-ray analyzer will be provided to measure the metal contents of various process slurries throughout flotation circuits. Slurry samples will be collected from the critical streams on a continuous basis by pressurized pipe samplers and pumped to the on-stream analyzer multiplexers. Sample flows exiting the multiplexers or analyzer chamber will be directed to a set of pumps for delivery back to the appropriate location in the flotation circuit.

To ensure proper metallurgical control of the flotation circuits, the following sample points will be established:

- flotation feed
- copper rougher concentrate
- copper rougher tailings
- copper cleaner-scavenger tailings
- final copper concentrate.

Automation

- Motor Control – Motor operations will be monitored remotely by the PLC. A remote input/output (I/O) rack in each MCC section will be connected to the motor panels so that status signals and amperages will be available on the plant communication bus.
- Interlocks – Most of the safety and process interlocks will be triggered through the PLC. Only pull-cord switches on conveyors will be connected directly to the motor starter control circuit. Interlocks for drives with auxiliary support systems (e.g., lube systems for primary mills, gyratory and pebble crusher, tower Mills, pressure filters) will be provided by the equipment supplier as part of the control circuit. This will include the necessary data outputs to the plant loop for remote monitoring of status and alarms.
- Instrumentation – Instruments will be electronic type with “smart” functions, whenever feasible. The use of intelligent transmitters will allow much of the basic control capabilities to be delegated to the instrument level, reducing the number and size of I/O points tied to the central controllers. This approach will also minimize field-wiring requirements and provide the capability for instrument diagnostics, remote calibration, and downloading of control algorithms.
- Field-mounted switches will be avoided as much as possible because they cannot provide feedback to the plant control loop.



- Cabling – Primary and final control elements will be connected to the field-mounted junction boxes with multi-conductors and multi-pair cables. The EEMAC-4X junction boxes receiving these signals will then connect to the distributed controllers via a fibre-optic-based data communications link.
- Controller Communications – Inter-controller communications will be carried out on a data highway using fibre-optic links. The highway will cover the mill facilities, crushing station, fresh water pumping stations, filter plant, and tailings pond area. The fibre-optic links will permit communication between the configuration, alarming and archiving consoles of the control network and the operator interface units (OIU). Personal computers will have local and remote access to monitor the status of the plant operation. Although technically feasible, remote activation of plant equipment will not be permitted.
- Communications Systems – The communications systems at the mill site will include a fixed-line telephone system, a local area network (LAN) for the site computers, portable radios for operators and in mobile equipment, and the fire protection system.

5.4.12 Process Control Philosophy

The control system for the processing plant will consist of distributed controller architecture tied by fibre-optic links to a central PC to provide OIU (operator interface unit), alarming and reporting capabilities. The mill instrumentation personnel will be responsible for programming the controllers and maintaining the hardware. Some of the major equipment will have dedicated on-board PLC-based control systems that can be accessed by the plant-wide control system. The general aspects of the major process control loops are described in the following subsections.

Crushing

The crusher package provided by the vendor will include a PLC tied to all protection alarms. The PLC will monitor the operating status and metre amperages of the equipment in the crushing area. Measured process variables will include crusher discharge pocket level, discharge feeder speed, and radial stacker conveyor position. Other protective devices will include traffic light switches to permit truck dumping into crusher feed pocket, low-speed and side-travel switches on the conveyors, pull-cord interlocks for the conveyors and the apron feeder, a high sump level switch, air compressor pressure and temperature switches, and a differential pressure switch on the dust baghouse.



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An air-conditioned control booth equipped with an OIU will be provided for the crusher operator. Cameras will allow the operator to monitor the status at the discharge apron feeder outlet, the transfer point between the two conveyors and the discharge point of ore onto the stockpile.

The data generated in the crusher area will be fed to the central plant control computer, which will allow the crusher operator to monitor and interface with the rest of the plant.

Ore Reclaim

Feeder speed will be used as the control variable to achieve consistent tonnage, based on feedback from a conveyor scale. Throughput will be determined by the SAG mill charge level and power draw.

Grinding

Grinding circuit control will be based on a local PLC/DCS hybrid controller with extended analog capabilities. The vendor packages for the SAG and ball mills will include PLC-based data gathering systems to collect and relay all equipment protection outputs to the plant controller. Basic protection will include data on flow, pressure, temperature, and motor RTD, as well as filter differential pressure switches on the lubrication systems. The DCS will pass all motor status and amperage information through to the MCC (motor control centre). The overall operating strategy for the grinding area will be to maximize the tonnage throughput until a power draw limitation occurs on one of the mills or the target grind size of 150 μm P₈₀ cannot be met.

The control system for the SAG mill circuit will control the throughput to meet the limitations of the downstream circuits, especially flotation circuit, when processing high feed grades. Basic stabilizing control measurements will include the SAG mill charge, as determined by trunnion bearing pressure and an acoustic device; and SAG mill discharge pumpbox level. Valves for water addition to the discharge pumpbox, SAG mill feed chute and vibrating screen deck will be automated.

In the ball mill circuit, control measurements for stable operation will include ball mill discharge pumpbox level, cyclone feed, slurry density and pressure, and cyclone overflow slurry density. Six of the cyclones will have remote operating capability (cyclones installed c/w automated isolation valves with limit switches) to maintain the pumpbox level within an acceptable range despite varying circulating loads and process tonnages. The cyclone feed pumps will be variable speed to maintain the



feed pressure to the cyclones within an acceptable range and meet the grind size target.

Regrinding

The required power draw for each regrinding application will depend on the tonnage of feed material presented to the individual Tower mills. The cyclone feed pumps will have variable-speed motor controllers to maintain the feed pressure to the cyclones for efficient classification. A pressure transmitter will provide the required feedback signal. One-third of the operating cyclones will be automated and have isolation valves equipped with limit switches to permit remote operation per the pumpbox level controller signal.

To ensure the proper grind is being attained, the P_{80} of cyclone overflow samples will be measured in the laboratory; the measured value will be plotted in a calibration curve against the calculated kWh/t draw when the sample was extracted. This will allow the power draw setpoint to be adjusted for each ore type processed, so that the system becomes fine-tuned over time.

Flotation

Each flotation cell will be equipped with a manual airflow control valve, and each bank will have an airflow metre and automatic valve to maintain the air volume, even under fluctuating demand conditions in other areas of the plant. Each bank will have an ultrasonic pulp level monitor to pneumatically actuate the dart valves in the cell connection and discharge boxes.

In the flotation circuit, pH meters will be provided for the first and fifth rougher cell, the first cell of the first cleaners and the second copper cleaner conditioner, the second cleaner-scavenger connection box, and the third cleaner connection box.

The metal content of five critical streams will be sampled and assayed continuously by the on-stream analyzer (OSA). Calculated recovery data will be provided to the operators to assist them in optimizing the metallurgy. The primary control variables for steady operation are airflow rates and pulp levels. Optimum reagent addition rates can be determined from the monitoring data.

The flotation control system will incorporate separate OIUs.



Concentrate and Tailings Dewatering

The concentrate thickener will be equipped with bed level sensors, rake motion sensors and torque transmitters and nuclear density gauges on the underflow pipes. A position monitor with limit switches will enable remote monitoring of the thickener rake elevation and direction of movement. The overflow pumpbox and sump pump pit will have conductivity-based high-level switches. Underflow pumps will have variable-speed drives.

The pressure filter will include a PLC-based control system supplied by the vendor that will monitor all protective and operational instrumentation and motor status. The entire filtration cycle will be controlled by a number of preset logic timers dictating the slurry feed duration, the pressing sequence and the air-drying sequence. The operators in this area will be responsible for the tailings handling operation as well as concentrate dewatering. One OIU will be provided to integrate the I/O and control requirements for all the functions within these areas.

Water Systems

The process, and fresh water tanks will be equipped with ultrasonic level controllers for on-line indication of the water inventory. Flow meters will be installed on the multi-stage horizontal pumps used to deliver water to the processing areas. Flow meters will also be installed on the fresh water well discharge and the reclaim pump discharges.

Reagent Systems

Diluted reagents will be distributed via a pressurized loops. Pressure feedback in the line pressure will control the addition valve. Small magnetic flow metres will be used to compare the actual flows with the required reagent quantities, as calculated from the consumption setpoints established by the operator.

The flotation operator will determine and control the reagent addition rates by assessing the assay information provided by the on-stream analyzer.

Plant Air Services

The plant air services will include compressed and instrumentation air supplies and the flotation blowers.

The compressors will be equipped with inlet air filters and will discharge into air receivers. The compressor motors will be equipped with protective instrumentation



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such as RTD. Motors above 150 kW will also have RTD's on their lubrication systems and vibration monitors. The air receivers will have pressure transmitters and automated drain valves.

Similar instrumentation will be provided for the flotation blowers. To maintain the air pressure at the required setpoint despite variations in air moisture, temperature, or flow requirements, a bypass line will be provided to vent excess air to the outside.



6.0 MINE SITE INFRASTRUCTURE

6.1 Site Location Options

The Mirador Project site is located in rugged terrain, characterized by low mountains with steep slopes, and narrow ravines associated with drainage systems. The area is heavily forested.

Six locations were examined as possible sites for process facilities, two sites were examined as possibilities for the crusher, and seven locations were examined as alternatives for the tailings facilities. These locations were examined by AMEC, Corriente, KP, and Merit personnel during field investigations.

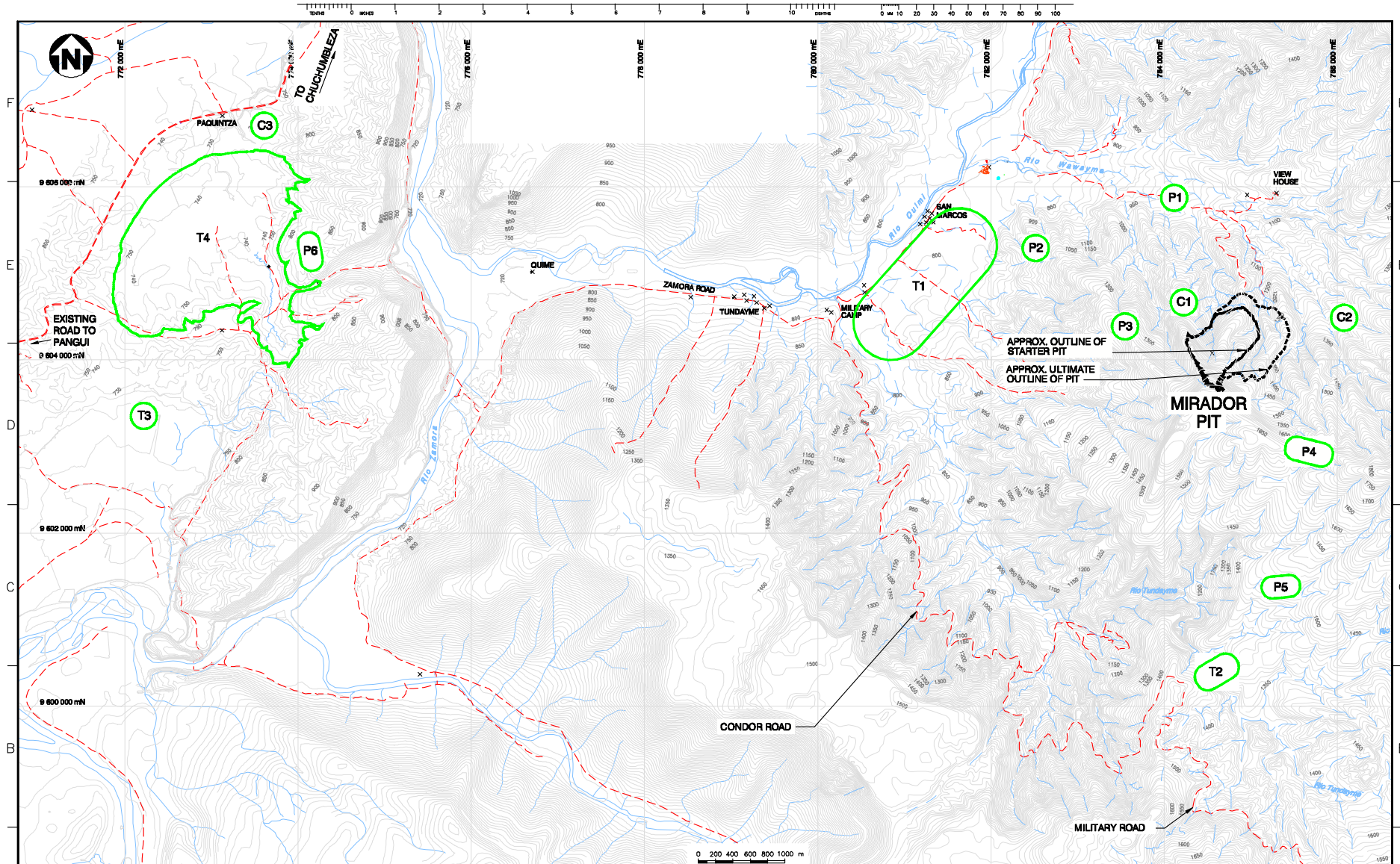
6.1.1 Tailings Impoundment Location

The following criteria were considered when locating the tailings impoundment site.

- **Dam Stability** – The tailings dam should be stable with minimal risk of failure considering foundation, flooding, seismic activity and construction materials.
- **Aesthetics** – The tailings dam should be located, constructed, operated, reclaimed and closed in a manner that minimizes the effects on the community and the environment as much as practical.
- **Economics** – Location and construction of the tailings dam should consider construction, operation, and closure costs.
- **Water Quality** – The location, construction, operation, reclamation, and closure of the dam should consider water quality from dam discharge and seepage.
- **Watershed** – The dam and impoundment should be located in a watershed that is adequate to contain the volume of tailings generated over the life of the mine and in a watershed that requires minimal routing or flow through of water runoff.

Knight Piésold in their report entitle Initial Tailings Storage Site Alternatives Study, Ref. No VA201-00078/1-1, dated 17 February 2003, and included in Appendix F, identified seven alternative tailings storage locations within 5 km of the Mirador deposit.

The most suitable site examined by Knight Piésold, site T1 on Figure 6-1, was rejected as it would have required relocation of the community of San Marcos. The other sites examined by Knight Piésold were felt to be unsuitable due to the need for major river diversions and/or unacceptable foundation conditions. An examination of the upper Tundayme Valley found a suitable site, confined by a bedrock ridge, where the Tundayme River flows through a steep-sided, narrow canyon, site T2 on Figure 6-1.



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PROJECT NO. PACKAGE CODE SCALE 1: 20 000												BY D/M/Y 23/04/21				AREA MIRADOR PROJECT ECUADOR		TITLE MIRADOR COPPER PROJECT FACILITIES LOCATION OPTIONS					
CLIENT DWG. NO.												DRAWING NO. FIGURE 6.1						REV.					



The volume behind the proposed dam was found to be adequate to hold the anticipated volume of tailings, with ample potential for future expansion. However, a tailings impoundment in this area would have resulted in a flow through dam structure. While water modelling showed the discharge water should be compliant with recognized discharge regulations, the concept was unique and was therefore rejected. A third location, site T3, was investigated approximately 11 km west of the mine area, in the Pangui valley. The positive features of the site over the other locations considered are:

- suitable foundation conditions with residual (sandstone) soils
- small catchment area and site characteristics that enable the construction of a zero discharge tailings impoundment
- the close proximity of the plant site to the tailings impoundment allows for separation and separate submerge disposal of the potentially acid generating high sulphide cleaner tailings (which are about 10.3% of the feed weight), thereby preventing oxidation
- the tailings embankment will be raised with centreline construction using cycloned tailings. This will reduce upfront capital costs due to the small starter embankment requirements
- minimizes displacement of residents and reduces visual and noise impacts to communities.

6.1.2 Crusher Location

To minimize mine truck haul costs, the crusher is located near the pit. Considering the topography there were only two possible locations, which were readily accessible from the pit, and provided a good alignment for the overland conveyor to the process plant. Location C1 was developed for process plant options P1, P2, P3, and P6, and location C2 was more suitable for plant locations P4 and P5.

6.1.3 Process Plant Location

Originally, sites P1, P2, and P3 were examined. P3 was proposed because it was close to the pit and the proposed crusher, thereby eliminating the need for an overland conveyor. However, P3 was rejected because of the amount of cut and fill necessary to create the platform for the plant and to construct the access road. P2 was examined as it was close to tails impoundment area T1 and had good topography. However, P2 was rejected because transportation of the ore to the plant, either with trucks or with an overland conveyor, would be more expensive than for location P1. Location P1 was selected as the most suitable plant location because the topography



was not difficult, the conveyor alignment from the crusher located near the pit was not difficult, access to the main highway, along the Wawayme river valley was easy, and the tails line would be a gravity line.

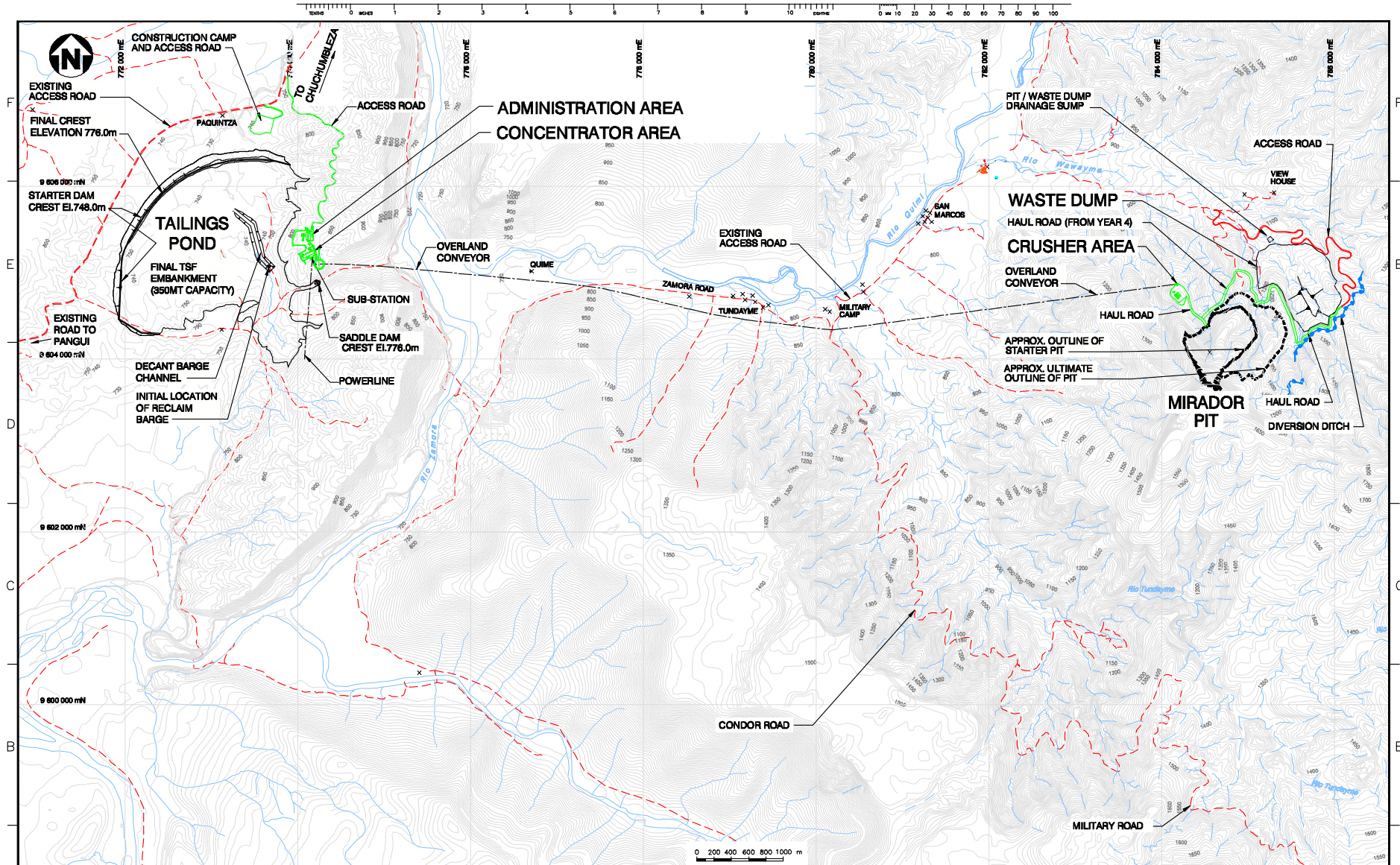
When tails impoundment area T1 was rejected, plant locations P1, P2, and P3 were rejected, as it would require pumping of tails several kilometres up hill. Therefore, locations P4 and P5 were examined. Both locations required construction of a new access road, the most suitable route being a branch from the existing Condor road. The main difference between the two locations was topography and distance from the tails dam. Location P4 was higher, located on the ridge above the pit between the Wawayme and Tundayme river valleys. This resulted in a longer access road and higher power requirements for the crushed ore conveyor and reclaim water pumps. Location P5, being lower, had reduced power requirements and a cheaper access road. As well, the site cut and fill was less for P5. Therefore, location P5 was selected as the most suitable location should the tailings impoundment area be T2.

When tailings site T2 was rejected and site T3 was selected, plant locations P4 and P5 were no longer suitable. Location P1 would have been suitable, but that would have required pumping tailings approximately 14 km to the discharge points. The other alternative was to convey crushed ore to a new process plant area located just above site T3. It was felt that conveying crushed ore was preferable from an operations and environmental perspective so plant site P6 was selected as the most suitable location.

The overall site layout is shown in Figure 6-2 and the project drawings in Appendix D.

6.2 Site Development

The administration, services, process, and concentrate filtration facilities will be located near the tailings facility, within easy access to the road to the coast. The crushing and mine maintenance facilities will be located on a bench about 1 km from the western edge of the ultimate pit. Ore from the crusher will be conveyed to the north and west to the concentrator in the Pangui Valley. The concentrator is located above the tails impoundment area, allowing for gravity flow of tailings and minimizing the pumping requirements for reclaim water.





6.2.1 Geotechnical Investigations

Preliminary geotechnical investigations have been carried out to assess foundation conditions at the various sites. Geotechnical characterization was carried out by surficial geological mapping; mapping of soil exposures in stream beds, hand-dug test pits and cut slopes, and borehole drilling. The borehole investigations were carried out using portable diamond drilling equipment, which yielded disturbed cores of soils and underlying bedrock. The cores were logged and photographed, and representative samples taken. Samples were shipped to the laboratories of Caminosca in Quito for geotechnical testing. The quantities of geotechnical investigations are summarized in Table 6-1, and the geotechnical reports are included in Appendix E.

Geotechnical investigations for the proposed Pangui TMF are currently underway. These investigations include 19 drill holes and 42 test pits to assess both embankment founding conditions and construction materials.

Table 6-1: Summary of Geotechnical Investigations

Component	No. of Drill Holes	No. of Piezometers	No. of Stream Beds/Test Pits Logged
Open pit	6	5 (vw)	0
Crusher	1	0	0
Waste dump	0	0	15
Plant/admin area	13	0	2
Tailings dam	19	8 gwm, 11 p	27
Tailings dam borrow areas	-	-	15

Note: vw = vibrating wire piezometer
gwm = 2" PVC groundwater monitoring wells
p = 1" PVC piezometers

Concentrator/Administration Area

The geotechnical investigation showed the ground in the concentrator / administration area is adequate to support the proposed structures. Colluvial soils covers the area ranging in depth from 1 m to 10 m. In the area proposed for the mills, the colluvial soil is about 1 m to 2 m deep, below which is residual soil transitioning to weathered sandstone. The water table varied from near surface to approximately 5 m below.

Crusher Area

No detail geotechnical field investigation was carried out in the immediate crusher area; however, geotechnical investigations were conducted for the pit area to the southeast. One borehole, M 62, is located on the same ridge as the crusher, about



400 m to the south. Based on a review of the core photographs, the elevation of competent rock is about 1,272 m. Since the bottom of the crusher will be at El. 1225, it is expected to be founded in good rock, if the geology at the actual crusher site is similar to that represented by borehole M62.

6.2.2 Site Preparation and Foundation Design

Administration Area

The in situ soils underlying the proposed maintenance, offices and accommodation facilities consist of granular soils. Footings for these structures can be constructed on the undisturbed dense soils on structural fill materials placed directly over top of these deposits.

Crusher Area

Relatively competent rock for the crusher structure foundation is present at depths of over 20 m to 25 m below ground surface. The cut for the crusher structure is in this range, therefore it is anticipated that the crusher structure will be founded on competent material with design bearing pressures in the order of 150 kPa to 250 kPa.

Concentrator Area

Relatively competent soils for building foundations is present at depths of over 5 m, therefore the entire area is stripped to below 5 m. Cut slopes are at 2H:1V due to the loose nature of the overburden soil.

6.2.3 Aggregate and Borrow Sources

The aggregate and borrow sources are summarized in Table 6-2.

6.3 Plant Site Drainage

Run-off from the concentrator/administration area will be directed toward the tailings impoundment facility, located directly downstream from the area.

Run-off from the crusher area will be directed to the sediment control ponds located below the waste dumps.



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Table 6-2: Summary of Borrow Sources

Application	Material	Estimated Volume (m ³)	Source	Processing	Extent of investigations
Concrete	Coarse aggregate	5,000	River deposits near confluence of Quimi and Tundayme rivers	Crush and screen	Visual observations
Concrete	Fine aggregate	5,000	River deposits near confluence of Quimi and Tundayme rivers	Screen	Visual observation
Sedimentation ponds	Compacted impervious fill	100,000	Open pit overburden stripping	Visually select finer materials	Laboratory testing carried out on samples from drill pads and drill core
Tailings starter dam	Low permeability soils	440,000	Source from within tailings impoundment	Visually select	Currently underway, 42 test pits and 19 drill holes
Plantsite fills	General fill	20,000	Plant site area stripping	Crush and screen	Drill holes and test pits

6.4 Site Roads

Two types of roads will be constructed at the site:

- haul roads for mining truck traffic hauling ore to the crusher and waste rock to the dump site
- access roads to provide service/maintenance vehicle access to the crusher area and the concentrator area.

6.4.1 Haul Road

The ore haul road is approximately 1 km and connects the open pit with the crusher. The road to the waste dump is approximately 1.6 km. The road is a minimum 22 m wide, sized to accommodate two-way traffic of mine haul trucks. The road will be constructed of mine waste, with drainage and culverts to direct run-off water away from the road and the waste dumps.

6.4.2 Site Access Roads

A new 2.6 km road will connect the concentrator/administration area to the main highway at Paquintza. Existing secondary roads connect the highway to the village of San Marcos, just below the deposit. From San Marcos a 3 km section of road up the Wawayme valley will be upgraded and a new 5.3 km extension will climb to the pit area, see Caminosca's report in Appendix E.



The access roads to the concentrator/administration area and to the crusher area will have a running surface of approximately 7.2 m and will be designed for highway trucks.

Secondary site roads to access wells, tailings facilities and the overland conveyor will be rough four-wheel drive roads design for infrequent maintenance access.

6.5 Site Facilities

The site facilities will be distributed over the two main areas of the property. The main entrance for the site will be via a gatehouse and weigh-scale at the concentrator/administration area, layouts are included in Appendix D.

6.5.1 Concentrator/Administration Area

- administration and engineering offices
- assay laboratory
- warehouse, and storage yard
- change house
- concentrate filter plant
- concentrate storage shed
- fuel storage
- gatehouse and weigh-scale
- coarse ore stockpile
- concentrator, including internal offices, shops and washrooms
- main substation
- reagent mixing
- first aid room (internal to concentrator)
- lunchroom (internal to concentrator).

6.5.2 Crusher Area

- primary crusher station
- heavy vehicle repair shops
- mine offices
- mine warehouse



- first aid room
- lunchroom
- washrooms
- fuel tank.

6.5.3 Concentrator/Administration (Dwg. A1-142314-100-C-0012)

Administration and Engineering Offices

The offices complex will have a total floor area of 1,250 m² and provide space for individual staff offices, meeting/training rooms, toilets, storage, and safety and first aid facilities.

Assay Laboratory

A separate, stand-alone, single-story building adjacent to the administration and engineering offices will house the laboratory. The laboratory will have 600 m² of floor space and be equipped with facilities for sample preparation, assaying, atomic absorption testing, and environmental testing along with supporting sample and chemical storage rooms. Offices are provided for senior laboratory personnel.

Warehouse and Storage Yard

A separate building is provided for warehousing activities. The warehouse will have an internal clear height of 8 m and be clad with sheet metal. Additional uncovered storage of about 2,000 m² will be provided in an adjacent fenced compound. No repair shop is provided at the plant area as all major repairs will be done by shops in the surrounding towns. Small work areas and shops will be provided in the concentrator.

Change House

The concentrator and maintenance personnel will be provided combined change facilities. Employees are provided with separate clean and dirty lockers, shower and lavatory facilities. Separate locker rooms are provided for hourly paid employees, staff, and women. The building also includes a deployment area for daily task assignment.



Fuel Storage

The fuel storage facility will be adjacent to the main access road, in the administration area. Diesel fuel and gasoline will be stored in tanks provided by the fuel supplier. The tanks will be in a lined and bermed containment area; the gross contained volume will be 110% of the tank capacity. A lined dispensing apron and spill collection sump will be provided where fuel is transferred to the storage tanks or loaded into haul trucks for transport to the crusher area.

Gatehouse and Weigh-Scale

A gatehouse and truck weigh-scale will be located at the main entrance to the administration area, for traffic and concentrate inventory control. The entire perimeter of the administration area will be fenced.

Coarse Ore Stockpile Structure

The coarse ore stockpile will be an open structure, with no cover. A concrete structure under the pile will house the reclaim equipment and provide a means to drain water from the pile. An emergency escape tunnel will be provided.

Concentrator Structure

The concentrator building will be a steel structure on reinforced concrete foundations. The building will have a roof and cladding extending part way down the walls, to limit rainwater inflow to the building.

Internal offices and rooms will be provided for operators, first aid, lunchroom, ablution facilities, and maintenance.

Reagent Mixing

Adjacent to the concentrator, the reagent mixing building is a steel frame structure on reinforced concrete foundations, with a roof but excluding wall cladding. The building will house the reagent mixing and holding tanks and distribution piping. Limited storage space will be provided as the reagents are brought over on a regular basis from the warehouse in the administration area.



Compressor/Blower House

The compressor/blower house is a concrete block structure adjacent to the concentrator. The structure is completely enclosed due to the noise associated with the compressors and blowers.

6.5.4 Crusher Area (Dwg. A1-142314-100-C-0013)

Crusher Station

The crusher station will be a concrete and steel structure set into the side of the hill. The central core housing the crusher will be concrete, the service section will be steel on reinforced concrete foundations. There will be a roof over the crusher dump pocket to support the maintenance crane and reduce the amount of water from rainfall entering the ore pocket and structure. A reinforced earth retaining wall will extend out from the crusher structure to stabilize the area around the structure.

Mine Maintenance Facility

A mine equipment maintenance, dry and mine office complex will be built by the mining contractor near the crusher and pit. The facility will have three main sections:

- the truckshop will be a high-roofed, open-sided, five-bay structure for maintenance of pit haul trucks, dozers and other heavy equipment
- the mine, office, and light vehicle shop will back onto the truckshop and have a lower roofline
- the mine office and dry will be a one-storied section.

Mine Area Fuel Storage

Place for a mine fuel storage facility will be provided adjacent to the access road, next to the maintenance facility. The fuel will be supplied by the mining contractor. Fuel will be stored in a lined and bermed containment area; the gross contained volume will be 110% of the tank capacity. A lined dispensing apron and spill collection sump will be provided where fuel is transferred to the storage tanks or loaded into trucks.



6.5.5 Utilities and Services

Sewage Treatment

Sewage from the administration area will be pumped to a treatment facility. The package will be sized for 110 people and will meet government discharge regulations.

Sewage from the crusher area and the concentrator area will be collected and sent to septic tanks and leach fields.

Potable Water

Potable water will be supplied to the two areas from raw water tanks via packaged water treatment plants and stored in a potable water tanks. Design of the water treatment plant, potable water tank and distribution system is based on a maximum site population for each area and an average daily demand per capita of 380 L. The treatment plants will be a prefabricated, skid-mounted package complete with sediment filters, ultra violet units, and chlorine addition.

Solid Waste Disposal

All garbage, including domestic and office refuse, will be disposed of in the mine waste dump area. Sludge from the sewage plant and the water treatment plant will also be placed in the mine waste dumps.

Fire Water

Firewater will be provided to all areas. The systems are sized to provide two hours of demand at 227 m³/h. The firewater tanks are situated to provide gravity feed to a dedicated firewater ring main around the facilities. Fire hydrants, hose reels, and hand held fire extinguishers will be provided at strategic locations throughout the site.

6.6 Water Supply

Fresh water is readily available at all times of the year at both sites. Approximately 10 m³/h is required for potable and service use at the crusher area and 90 m³/h is required at the concentrator / administration area.

Fresh water for the crusher area will come from creek catch basins located in the hills above. At the crusher area the diverted creek water will be gravity fed to combined firewater/fresh water tank located about 100 m above the facilities. Water from the



supply tank will flow by gravity to the facilities. Water for potable use will be drawn from the fresh water distribution system, treated, and distributed as required.

Fresh water for the concentrator/administration area will come from wells located about a kilometre away near the Zamora River. The water will be pumped to a fresh water tank at the concentrator from where it will be distributed around the site. The firewater tank at the concentrator/administration area will be located on the hill above the facilities. It will be filled during the construction period. Should the tank be emptied, it will be re-filled by connecting a portable pump to the concentrator reclaim water system and pumping up the fire water line to the tank.

Reclaim water will be used for the bulk of the process make-up water required at the concentrator. The reclaim water will be pumped to the reclaim water tank at the concentrator, from which it will be distributed for various uses.

6.7 Power Distribution

6.7.1 Power Supply

Electrical power demand for the project is estimated to average 28.3 MW. This is split between the following:

- crusher site.....0.6 MW
- concentrator 24.8 MW
- filter plant.....0.5 MW
- water supply and distribution 1.4 MW
- ancillary facilities 1.0 MW

The figure for the concentrator takes full credit for the regenerative capability of overland conveyor No. 1 at an efficiency of 93%.

A 13.8 kV power supply is to be provided by others at the concentrator site. From here it is taken to the main 13.8 kV switchgear line-up in the concentrator indoor electrical room. Power is distributed from here. One 13.8 kV overhead line feeds the reclaim water pumps and another the crushing plant with drop-offs to the fresh water wells, conveyor transfer station and mine site conveyor electrical room. A 13.8 kV cable feeds the mill site conveyor electrical room. Power is transformed at these sites to 4160 V and 480 V as required. 4.16 kV cable feeds are taken from the mill 4160 V MCC to the filter building and concentrate storage, administration building and repair shops. Further 480 V feeds are taken from these areas for low-level power requirements at the gatehouse, truck scale, warehouse, fuel tank farm, and laboratory.



The concentrator 13.8 kV switchgear line-up contains starters for the SAG and ball mill motors and feeder breakers for the 480 V mill unit substations feeding the low voltage MCCs.

Power factor correction is provided for correcting to approximately 0.95 lagging and has been accomplished by the use of two, individually switched, 3 MVAc capacitor banks connected to the bus at the main 13.8 kV switchgear line-up.

The single line diagram A1-142314-100-E-0001 shows the intended power scheme down to the low voltage busses, indicating transformer ratings and medium voltage motor starters.

It is assumed the 13.8 kV system is appropriately grounded by the power supplier at his mill site substation. The 4160 V and 480 V systems are resistance grounded with ground fault currents limited to 25 A and 5 A respectively.

6.7.2 Emergency Power

In the event of a power failure, automatic start, 480 V generators have been provided at the concentrator, crusher, and administration building to provide emergency power. Critical drives such as thickener mechanisms and underflow pumps may be run as well as some lighting, communications equipment and the control system. Drives that require connection to emergency power supplies are designated on the load list in Appendix D. The generator at the administration building will provide power to essential drives and services in the filter building, warehouse, gatehouse, and vicinity.

The generators have automatic transfer switches and 8-h capacity diesel fuel tanks.

6.7.3 Distribution Voltages

Step-down transformers are utilized to produce power at the following voltages for distribution to the electrical loads (other than the mill motors).

- site distribution 13.8 kV and 4.16 kV, 3 phase
- motors over 200 kW 4.16 kV, 3 phase
- motors 0.5 to 200 kW 480 V, 3 phase
- motors up to 0.5 kW 240 V, single phase
- lighting..... 240 V or 120 V, single phase

240 V and 120 V single phase supplies will be derived from 600 V to 240/120 V transformers located in the appropriate electrical rooms.



6.7.4 Electrical Rooms and Transformers

These are located as near as possible to the major electrical loads. The main ones are located in the concentrator, the crusher, and the filter plant site. 4160 V and 480 V step down transformers are outdoor, oil filled type with the exception of the 480 V at the concentrator which are indoor, dry type. The 480 V transformers at the mill site are incorporated into unit substations each feeding three MCCs.

6.7.5 Electrical Distribution Equipment

Distribution equipment will be rated as follows.

- site power distribution..... 13.5 kV in 15 kV rated switchgear and 4.16 kV in 5 kV rated switchgear
- motors over 200 kW 4.16 kV in 5 kV rated switchgear/starters, motors to be 4 kV rated
- Motors 0.5 to 200 kW..... 480 V motor control centres, motors to be 460 V rated.

The electrical distribution equipment will consist of switchgear, starters, and feeder breakers for the motor and non-motor loads in common lineups. Lighting and small power applications will be fed from power panels as required, located in the electrical rooms.

6.7.6 Mill Motors

The SAG mill is driven by two 4.1 MW and the ball mill by two 4.66 MW, 13.8 kV, wound rotor induction motors. Those on the SAG mill are capable of operating, at variable speeds. The inrush on these drives will be limited to 150% to 160% by the rotor resistance. They are connected to the mills without the use of a clutch. This is a less expensive option than that of using synchronous motors and has the advantage of a low inrush current. There is no inherent advantage of power factor improvement that synchronous motors provide so 6 MVAc of power factor correction has been provided at the 13.8 kV bus.

6.8 Communications

The Mirador communications system will be satellite based and provide the following:

- voice-over internet protocol for offsite telephone communications



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- a local area network on site, which can be expanded to provide a wide area network incorporating Corriente's head office in Vancouver
- 20 telephone trunk lines, which will support simultaneously 15 to 20 phone calls, a fax and a number of computers on the internet, with full quality service. Should this prove to be insufficient, the bandwidth can be increased in less than one hour, remotely by the service provider.
- Vancouver dial tone (no long distance charge). Other city dial tones can be provided should it be necessary.



7.0 TAILINGS IMPOUNDMENT

7.1 Introduction

Knight Piésold (KP) has undertaken a study of the proposed Panguí Tailings Management Facility (TMF) for Corriente Resources Inc., Mirador Copper project. Their report is included in Appendix F and summarized in this Section.

The Panguí TMF will ultimately be required to store 110 Mt of tailings from the Mirador copper mine. The mine will generate 25,000 t of tailings per day. The Panguí area was selected for further investigation as a potentially viable TMF site because it appeared to have suitable characteristics, including:

- topography suitable for a 110 Mt TMF, with the potential to expand to a 350 Mt TMF if a decision is made to expand the mine
- potential as a zero discharge facility
- located within about 10 km of the pit, with a potentially viable conveyer belt route.

A number of potential layouts for the Panguí TMF were considered and these were assessed based on the available topographical, geotechnical, and meteorological information. A site visit was also carried in order to determine there were no “fatal flaws” and none were found. The sections below describe the studies completed to date and the details of the proposed Panguí TMF.

7.2 Site Characteristics

7.2.1 Hydrometeorology

The meteorological conditions at the proposed Panguí TMF Site were estimated using the information found in the Caminosca report “Información Meteorológica e Hidrológica.”

The average annual rainfall at the Panguí TMF Site is assumed to be 2,000 mm. Monthly precipitation distributions and standard deviations are based on data from the El Panguí and Gualaquiza monitoring stations; these may be found in Table 7-1.



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Table 7-1: Precipitation and Evaporation

Parameter	Unit	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Precipitation														
Mean	(mm)	125	148	193	232	210	204	169	140	158	155	144	123	2,000
	(%)	6.2	7.4	9.6	11.6	10.5	10.2	8.5	7.0	7.9	7.7	7.2	6.1	100
Standard deviation	(mm)	41.8	68.0	60.8	72.0	49.2	61.2	44.0	67.2	43.5	56.4	62.5	39.3	55.5
C _v		0.34	0.46	0.32	0.31	0.24	0.30	0.26	0.48	0.28	0.37	0.44	0.32	0.34
Evaporation														
	(mm)	126	101	105	99	117	88	97	110	117	139	131	149	1,380
	(%)	9.1	7.3	7.6	7.2	8.5	6.4	7.0	8.0	8.5	10.1	9.5	10.8	100

Notes: 1. Monthly precipitation distribution at El Pagui TMF site is based on El Pagui and Gualaquiza stations. 2. Coefficients of variation at El Pagui TMF site are based on the average of those calculated at El Pagui and Gualaquiza Stations. 3. Standard deviation is calculated as C_v x Mean.

Average annual evaporation depth for the proposed Pangui TMF site is determined from the data available at the Yanzatza and Gualaquiza stations and is estimated to be 1,380 mm. The monthly distribution of evaporation, as a percentage of the annual total, is based on information available at these two regional stations and is presented in Table 7-1.

Caminosca has also estimated a runoff coefficient of 0.80 for the proposed Pangui TMF location. This is based on a unit runoff of 60 L/sec/km², as inferred from an analysis of several regional hydrometric stations.

7.2.2 Basic Geology

Figures 7-1 to 7-3 show the typical terrain in the area of the proposed Pangui TMF. The surficial geology of the proposed Pangui TMF basin consists almost entirely of residual soils (sandstone). The depth to competent bedrock is estimated at between 10 m to 20 m, but may be significantly more than this in the centre of the valley. Detailed site investigations are currently underway, including 42 test pits and 19 drill holes throughout the proposed Pangui TMF.



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Figure 7-1: Southern Ridge of Proposed Pangui TMF



Figure 7-2: South East Corner of Proposed Pangui TMF



Figure 7-3: Western Edge of Proposed Pangui TMF





7.2.3 Seismicity

A seismicity review for the Mirador project was recently conducted by AMEC Earth and Environmental, a Division of AMEC Americas Limited (AMEC E&E) and presented in their report "Mirador Project, Tailings Management Facility Preliminary Design Report," 23 November 2004. This study has been reviewed and appropriate seismic design earthquakes selected for the TMF for feasibility design.

Consistent with the current design philosophy for geotechnical structures such as dams, two levels of design earthquake have been considered: the Operating Basis Earthquake (OBE) for normal operations; and the Maximum Design Earthquake (MDE) for extreme conditions (ICOLD, 1995). Values of maximum ground acceleration and design earthquake magnitude have been determined for both the OBE and MDE.

The OBE is typically determined using the probabilistic seismic hazard analysis to select an acceptable hazard level, based on the probability of exceedance over the design life of the facility. This is often chosen as the earthquake that has a 10% probability of exceedance in 50 years, corresponding to a return period of 475 years.

Assuming a design operating life of 12 years for the TMF, the probability of exceedance is 2.5%. The maximum acceleration for the 1-in-475 year earthquake is 0.20 g. A design earthquake magnitude of 7.0 has been selected for the 1-in-475 year event, based on the findings of the AMEC E&E seismicity study. The TMF would be expected to function in a normal manner after the OBE.

An appropriate MDE for the TMF has been determined based on a hazard classification of the facility, with consideration of the consequences of failure. The hazard classification has been carried out using appropriate guidelines. For this study, criteria given by the Canadian Dam Association's "Dam Safety Guidelines" (1999) have been adopted. These criteria are presented in Table 7-2. Classification of the TMF is carried out by considering the potential incremental consequences of failure. That is, the consequences of failure are incremental to the impacts, which would occur from the earthquake event but without failure of the facility. The consequences of failure include life safety, economic, social, and environmental impacts.

The project site and the area around the TMF is consists of residual (sandstone) soils and relatively flat topography. The area around the TMF is used by local farmers for general subsistence crops and timber harvesting. Therefore, there is some potential for loss of life following a potential dam failure. If failure of the TMF resulted in the release of tailings and/or process water, it would likely have some environmental impact. The economic and socio-economic impacts would also likely be high. Accordingly, a HIGH consequence category (hazard classification) has been assigned to the TMF.



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Table 7-2: Canadian Dam Association's "Dam Safety Guideline" (1999)

Consequence Category	Potential Incremental Consequences of Failure ^[a]	
	Life Safety ^[b]	Socioeconomic Financial & Environmental ^{[b][c]}
Very High	Large number of fatalities	Extreme damages
High	Some fatalities	Large damages
Low	No fatalities anticipated	Moderate damages
Very Low	No fatalities	Minor damages beyond owner's property

Based on Canadian Dam Association (CDA) "Dam Safety Guidelines" (1999).

- [a] Incremental to the impacts, which would occur under the same natural conditions (flood, earthquake or other event) but without failure of the dam. The consequence (i.e. loss of life or economic losses) with the higher rating determines which category is assigned to the structure. In the case of tailings dams, consequence categories should be assigned for each stage in the life cycle of the dam.
- [b] The criteria which define the Consequence Categories should be established between the Owner and regulatory authorities, consistent with societal expectations. Where regulatory authorities do not exist, or do not provide guidance, the criteria should be set by the Owner to be consistent with societal expectations. The criteria may be based on levels of risk, which are acceptable or tolerable to society.
- [c] The Owner may wish to establish separate corporate financial criteria, which reflect their ability to absorb or otherwise manage the direct financial loss to their business and their liability for damage to others.

An appropriate MDE for the facility is selected, based on the consequence category and the criteria for design earthquakes presented in Table 7-3. For a HIGH consequence category, a Maximum Design Earthquake is taken as a probabilistic event with a return period in the range of 1,000 years to 10,000 years. Alternatively, the Maximum Credible Earthquake may be selected based on the findings of a deterministic seismic hazard analysis. The November 2004 AMEC E&E report states "*a reasonably conservative peak horizontal acceleration value of 0.34 g is recommended for design of critical facilities at the Mirador site.*" This represents a magnitude 6.9 earthquake at a distance of about 18 km from the site. For this study it is recommended that a magnitude 7.0 earthquake be assigned to the MDE with a maximum bedrock acceleration of 0.34 g. This is representative of a near-field maximum 7.0 earthquake within 20 km of the project site. Limited deformation of the tailings embankment is acceptable under seismic loading from the MDE, provided that the overall stability and integrity of the TMF is maintained and that there is no release of stored tailings or water (ICOLD 1995).

The maximum accelerations of 0.20 g and 0.34 g for the OBE and MDE events respectively are for ground motions in bedrock or firm ground through which ground amplification effects are negligible. Maximum accelerations within the dam will likely be higher due to amplification of ground motion through the foundation soils and overlying tailings.



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Table 7-3: Consequence Category and the Criteria for Design Earthquakes

Consequence Category	Maximum Design Earthquake (MDE)	
	Deterministically Derived	Probabilistically Derived (Annual Exceedance Probability)
Very High	MCE ^[a]	1/10,000
High	50% to 100% MCE ^{[b][c]}	1/1000 to 1/10,000 ^[c]
Low	— ^[d]	1/100 to 1/1000 ^[d]

Based on Canadian Dam Association (CDA) "Dam Safety Guidelines" (1999).

- [a] For a recognized fault or geographically defined tectonic province, the Maximum Credible Earthquake (MCE) is the largest reasonably conceivable earthquake that appears possible. For a dam site, MCE ground motions are the most severe Ground motions capable of being produced at the site under the presently known or interpreted tectonic framework.
- [b] MDE firm ground accelerations and velocities can be taken as 50% to 100% of MCE values. For design purposes the magnitude should remain the same as the MCE.
- [c] In the High Consequence category, the MDE is based on the consequences of failure. For example, if one increment fatality would result from failure, an Annual Exceedance Probability (AEP) of 1/1000 may be acceptable, but for consequences approaching those of a Very High Consequence dam, design earthquakes approaching the MCE would be required.
- [d] If a Low Consequence structure cannot withstand the minimum criteria, the level of upgrading may be determined by economic risk analysis, with consideration of environmental and social impacts.

It is recommended that a more detailed seismicity study be conducted for future TMF design work, to confirm the magnitude and associated maximum acceleration for the Maximum Design Earthquake. This would require more detailed probabilistic and/or deterministic seismic hazard analyses than have been completed to date.

7.3 Tailings Characteristics

7.3.1 General

Ore will be processed at a rate of 25,000 t/d in a copper concentrator plant located on the northeast bank of the TMF. Concentrate removal has been estimated at 2% of the total ore throughput, requiring approximately 24,475 t/d of tailings solids to be impounded in the TMF.

7.3.2 Physical Characteristics

It has been assumed that the final slurry will have a solids content of 31.5% and be comprised of 89% rougher tailings and 11% cleaner tailings by weight. The solids will have a specific gravity of 2.65 and a dry density of approximately 1.26 t/m³.



7.3.3 Chemical Characteristics

The geochemical characteristics of the Tailings have been tested by SGS Lakefield Research Laboratories and the results of these tests are presented in SGS Lakefield Research Report LR10703-002 (February 2005). Based on the results presented in this report the rougher tailings would be the most suitable for cycloned sands embankment construction, as they are non acid generating. The high sulphide cleaner tailings are the only tailings that showed the potential of being acid generating if allowed to oxidize.

The composition of the tailings when averaging from the super-composites for the rougher tailings and high sulphide cleaner tailings is as follows:

- 87% of the total feed reports to the rougher tailings (21,950 t/d)
- 10.3% of the total feed reports to the high sulphide cleaner tailings (2,575 t/d).

In order to prevent the high sulphide cleaner tailings from oxidizing, the plan is to separate the cleaner finer tailings component in a separate tailings line directed into the TMF pond so that it would be perpetually saturated and therefore will not oxidize. With the mill in close proximity to the pond this is practical.

7.4 Evaluation of Alternatives

7.4.1 Preliminary Desktop Assessment

In December 2004, Knight Piésold (KP) completed an assessment of five alternative configurations for a TMF in the Pangui Area (Letter Report Ref # V4-1219, 17 December 2004). The purpose of this desktop study is to determine if a zero discharge condition can be achieved given the physical characteristics of the proposed site.

It was determined that the site was suitable for a zero discharge facility, if the geotechnical conditions of the site were adequate for the construction of a large earth embankment. Corriente then commissioned a more detailed satellite survey of the area and KP conducted a site visit in order to confirm there were no "fatal flaws" with the concept.



7.4.2 Site Visit and Determination of Preferred Alternative

Site Visit

Mr. Jeremy Haile, P.Eng. and Mr. Sam Mottram, P.Eng. visited the Mirador site and proposed Pangui TMF sites during early February 2005. They were accompanied by an engineer and geologist from Caminosca, an Ecuadorian consulting engineering firm. Caminosca had recently completed test pitting, drill holes and surficial geotechnical investigations along the proposed Sabanilla to Mirador 138 kV transmission line route, which also ran along the ridge between the north and south Pangui TMF sites. Caminosca's previous studies and knowledge of the area were valuable in helping to assess the two sites. In general, the geology of the two sites can be summarised as follows:

North Site

The surficial geology of the north site consists almost entirely of residual soils (sandstone). The depth to competent bedrock is estimated at between 10 m to 20 m, but may be significantly more than this in the centre of the valley.

South Site

Similarly, the surficial geology of the south site consists mostly of residual sandstone soils, except that below approximately elevation 750 m the residual soils are overlain with varying thickness of alluvial deposits.

No fatal flaws were identified during the visit of the proposed Pangui TMF sites, although the presence of the municipal palm forest reserve on the south site was confirmed. The location of culverts and drainage ditches under the roads that surrounded the two sites were noted.

7.4.3 Determination of Preferred Alternative

The north site was determined to be the preferred site for the development based on the following:

- based on the available geotechnical information, the north site has better characteristics for the construction of the TMF
- the north site would not impact the municipal palm forest
- the north site is closer to the proposed plant site and therefore would be more economical



- the north site has suitable topography for the construction on an economically viable 110 Mt TMF
- the north site could also be expanded to accommodate a 350 Mt TMF, if the mine were to expand.

Further details of the assessment of alternatives are included in KP's letter report, dated 2 March 2005 (Ref # V5-0209).

7.5 Geotechnical Site Investigations and Testing

7.5.1 Introduction

The site investigation for the Pangui TMF will consist of the drillholes, test pits, and geotechnical mapping of the TMF area. At the time of writing, the geotechnical investigations were underway, but no detailed results were available and are therefore not discussed. However, the details of the site investigation and testing program that is currently underway is discussed in the sections below.

7.5.2 Site Investigations

The geotechnical site investigation will require the testing of the in situ ground conditions, laboratory testing of soils and installation of groundwater monitoring wells. The in situ testing of ground conditions will be accomplished by standard penetration testing (SPT), permeability testing (Packer testing). The laboratory samples will be supplied by the collection of undisturbed soil samples from selected depths in drillholes (Shelby tubes) and test pits (bulk sampling).

A selected number of drillholes (8), will be equipped to operate as long-term water monitoring wells, and the remainder of the drillholes will be equipped with standpipe piezometers for investigating the groundwater conditions. As much as practical, the test pits shall be excavated by a suitable track driven machine to determine the ground conditions in the TMF basin, the availability of local borrow material for embankment construction, and the foundation conditions in the embankment footprint. Additional aims of the test pitting program will be the location of natural material, suitable for use as concrete aggregate and as drainage material for dam construction. Where practical, geotechnical surface mapping of the TMF footprint, basin, and the slopes above the basin will be conducted to identify surficial ground conditions and any potential areas of instability.



7.5.3 Geotechnical Laboratory Testing

Soil samples from both the test pitting and the geotechnical drill holes will be tested. The soil samples will undergo the following analyses only where appropriate:

- moisture content
- grain size distribution, which will include wet sieving and appropriate hydrometer tests where a minimum of 20% of the total sample passes through #200 sieve (ASTM)
- Atterberg Limits (Liquid Limit and Plastic Limits)
- Triaxial tests, which will include both unconsolidated and consolidated undrained single stage tests with pore pressures on both in situ and remoulded shelly samples
- consolidation
- compaction (ASTM Standard Proctor)
- California Bearing Ratio (CBR)
- organic
- sulphate
- sulphide
- chloride
- alkali-reactivity
- abrasion.

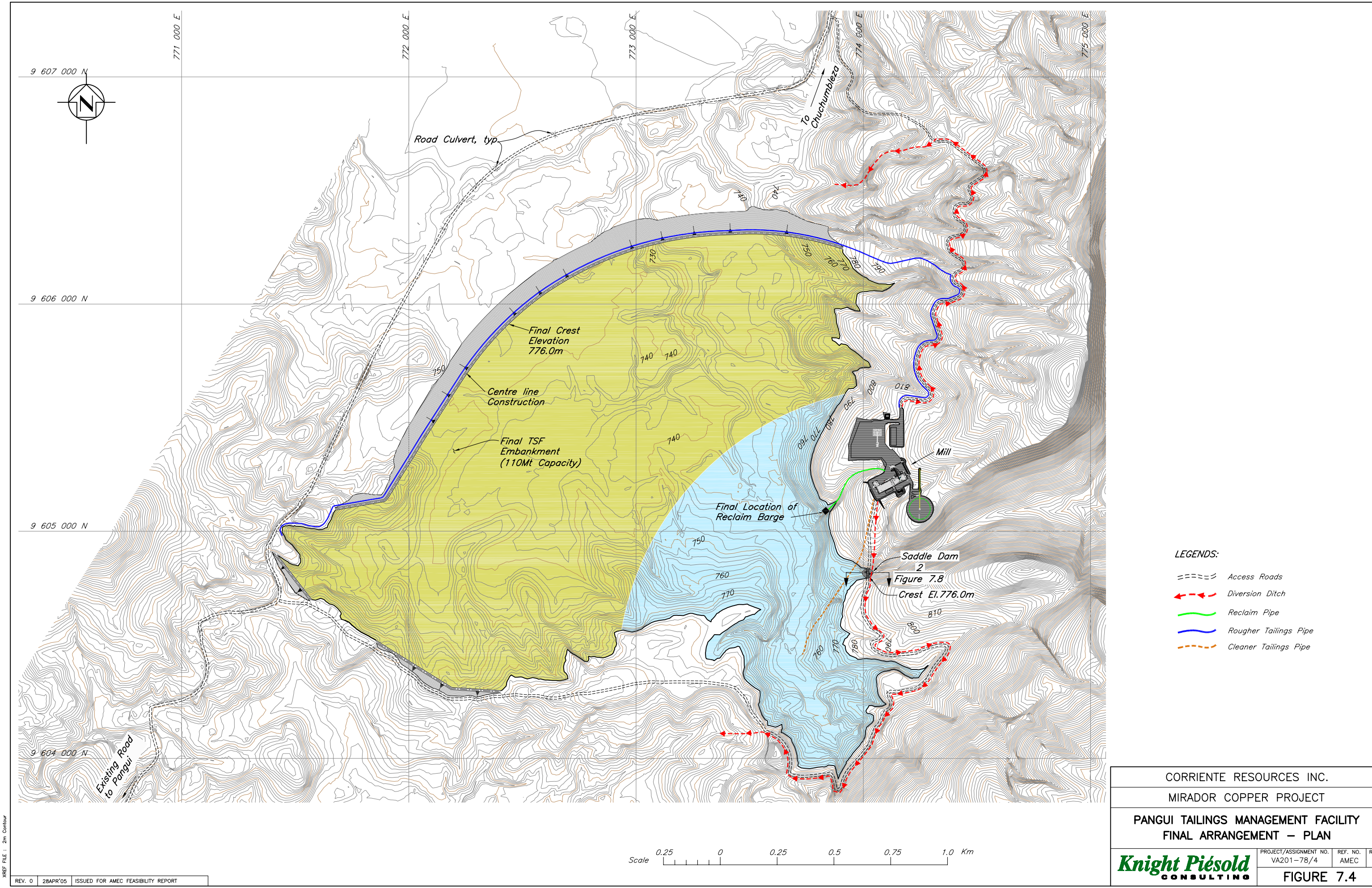
Sulphate, sulphide, chloride, alkali-reactivity, and abrasion tests are specified for potential concrete aggregate borrow source material only.

7.6 Design of Tailings Management Facility

7.6.1 Design Criteria

The final embankment, as shown in Figure 7-4, has been designed to accommodate the 106 Mt of solids produced over the 12-year life of the project. The principal design consideration is ensuring that the proposed facility meets the zero discharge criteria, meaning that no water will be released from the facility throughout the life of the project.

The basic criteria used for design of the proposed Pangui TMF are presented in Table 7-4.



- LEGENDS:**
- Access Roads
 - Diversion Ditch
 - Reclaim Pipe
 - Rougher Tailings Pipe
 - Cleaner Tailings Pipe

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**PANGUI TAILINGS MANAGEMENT FACILITY
FINAL ARRANGEMENT – PLAN**

Knight Piesold CONSULTING	PROJECT/ASSIGNMENT NO. VA201-78/4	REF. NO. AMEC	REV. 0
	FIGURE 7.4		



Table 7-4: Pangui Tailings Management Facility Design Criteria

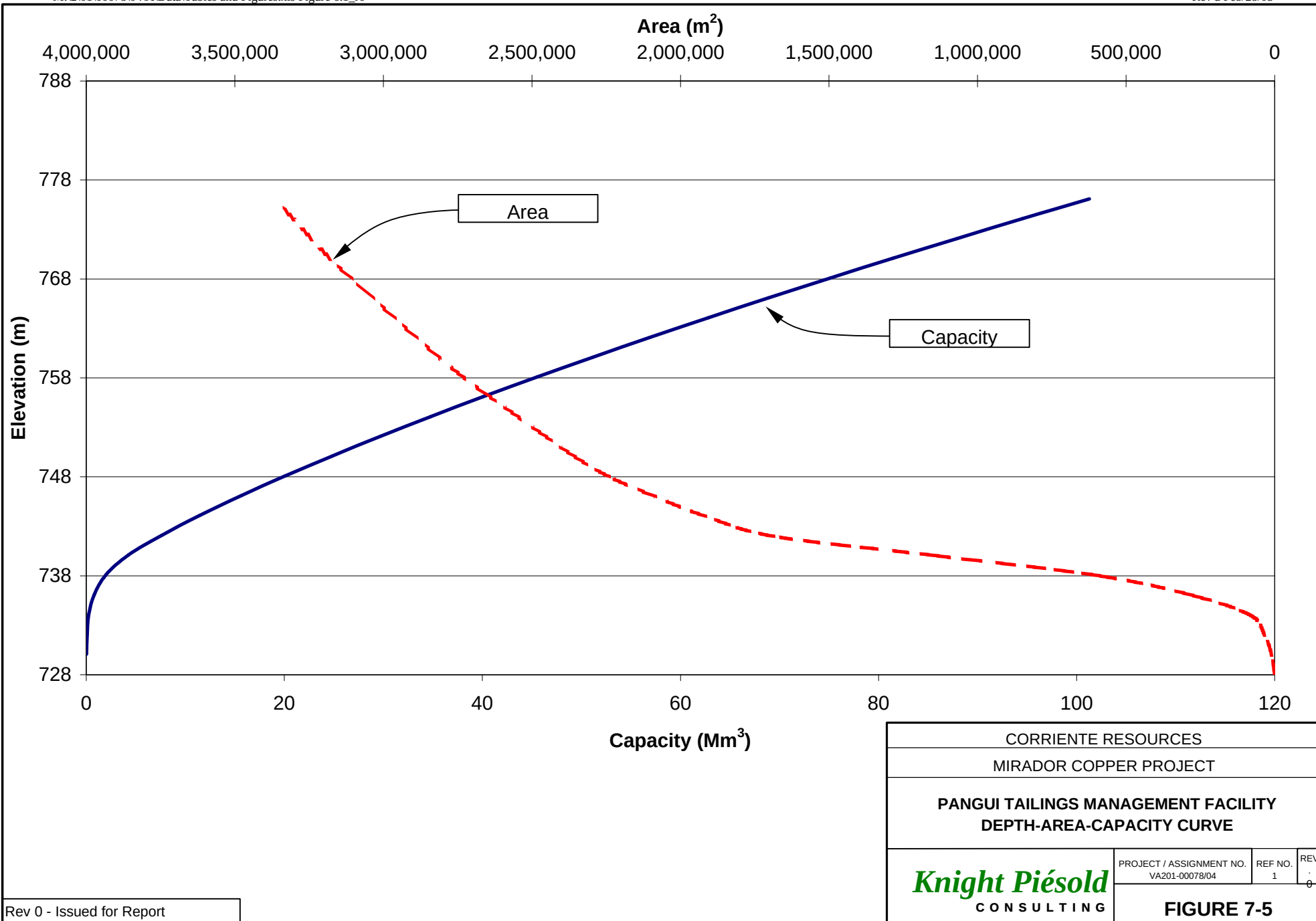
Water Management Design Criteria		
Total TSF catchment area	6.49	km ²
Undisturbed catchment area	3.12	km ²
Discharge criteria	no discharge	
Diversions	30 % of undisturbed area	
Seepage	None	
Water discharge to TMF with tailings	53,223	m ³ /d
Maximum reclaim to mill ¹ (% of process water input)	96	%
Tailings Management Design Criteria		
tailings discharge rate to TMF ² (365 d/a)	24,475	t/d
Mine life	12.0	years
Percent of solids in tailings (by weight)	31.5	%
Specific gravity of tailings solids	2.7	
Initial dry density of tailings	1.3	t/m ³
Average tailings consolidation rate	0.5	%/a

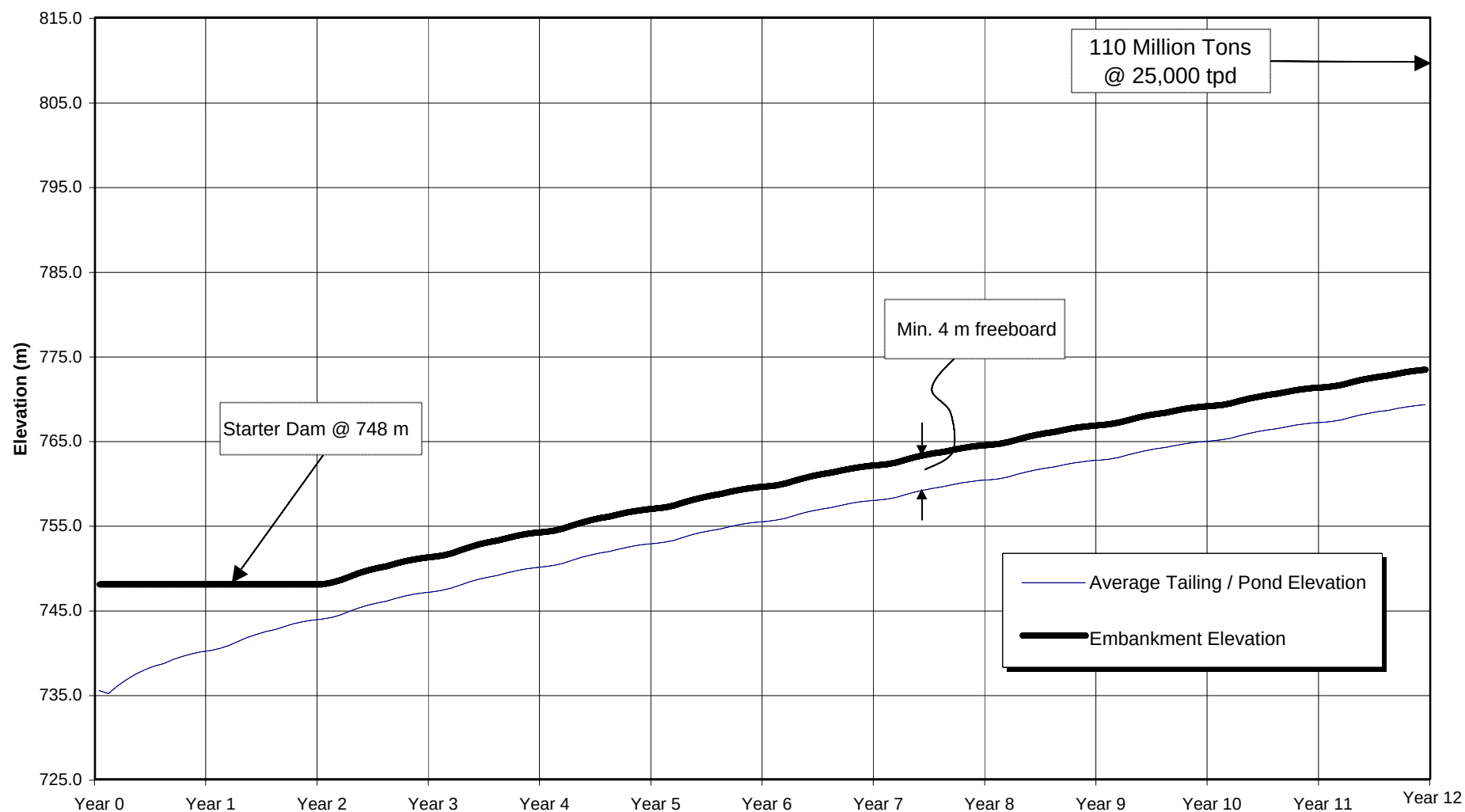
Note: ¹Assumes 2% water with ore and 2% fresh water make-up requirement. ²Assumes 3% to concentrate

7.6.2 Site Layout and Filling Curve

The Pangui TMF Site was designed using the most recent topographical mapping available, received from Corriente on 23 February 2005. The general arrangement shown on Figure 7-4 is the result of an optimization exercise that focuses on maximizing the TMF storage volume while keeping the embankment volume and catchment area as small as possible. A depth-area-capacity curve is shown on Figure 7-5.

A starter embankment will be built to an elevation of 748 m prior to the beginning of operations; this will provide about two years of storage at the proposed 25,000 t/d production rate. This elevation has been selected to provide ample time for commissioning of a cyclone system for coarse sand production and the procedures for placing and compacting the sand fraction. The main embankment will be raised to its ultimate elevation of 776 m using cyclone sand raises. An annual filling schedule is provided in Table 7-5, with Figure 7-6 showing the average tailings surface and required embankment elevation. In addition to the starter embankment, a small dam will be built at the southeast corner of the facility to an elevation of 760 m to provide a pond in which the cleaner tailings may be stored under a constant water cover. This small dam will eventually be submerged as the pond surface rises to its ultimate elevation.



**Notes:**

1. Simplified Filling Curve using horizontal surfaces only (elevations computed from total volume of solids + supernatant water).
2. Assumes 96% Process Water Reclaim.

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**PANGUI TAILINGS MANAGEMENT FACILITY
FILLING CURVE**

Knight Piésold
CONSULTING

PROJECT / ASSIGNMENT NO.
VA201-00078/4

REF NO.
1

REV.
0

FIGURE 7-6



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Table 7-5: Production, Filling, and Embankment Construction Schedule

	Daily Mining Rate (t/d)	Tailings Tonnage Stored in TMF ¹ (t)	Embankment Elevation ² (m)	Rate of Rise (m/a)	Required Split (%)
Year 1	25,000	7,573,649	748	Starter Dam	18
Year 2	25,000	15,840,285	748		8
Year 3	25,000	24,186,773	751	3	7
Year 4	25,000	32,578,173	754	3	7
Year 5	25,000	41,000,088	757	3	6
Year 6	25,000	49,444,756	759	3	6
Year 7	25,000	57,907,372	762	2	6
Year 8	25,000	66,384,691	764	2	5
Year 9	25,000	74,874,392	767	2	5
Year 10	25,000	83,374,735	769	2	5
Year 11	25,000	91,884,379	771	2	5
Year 12	25,000	100,402,254	773	2	5

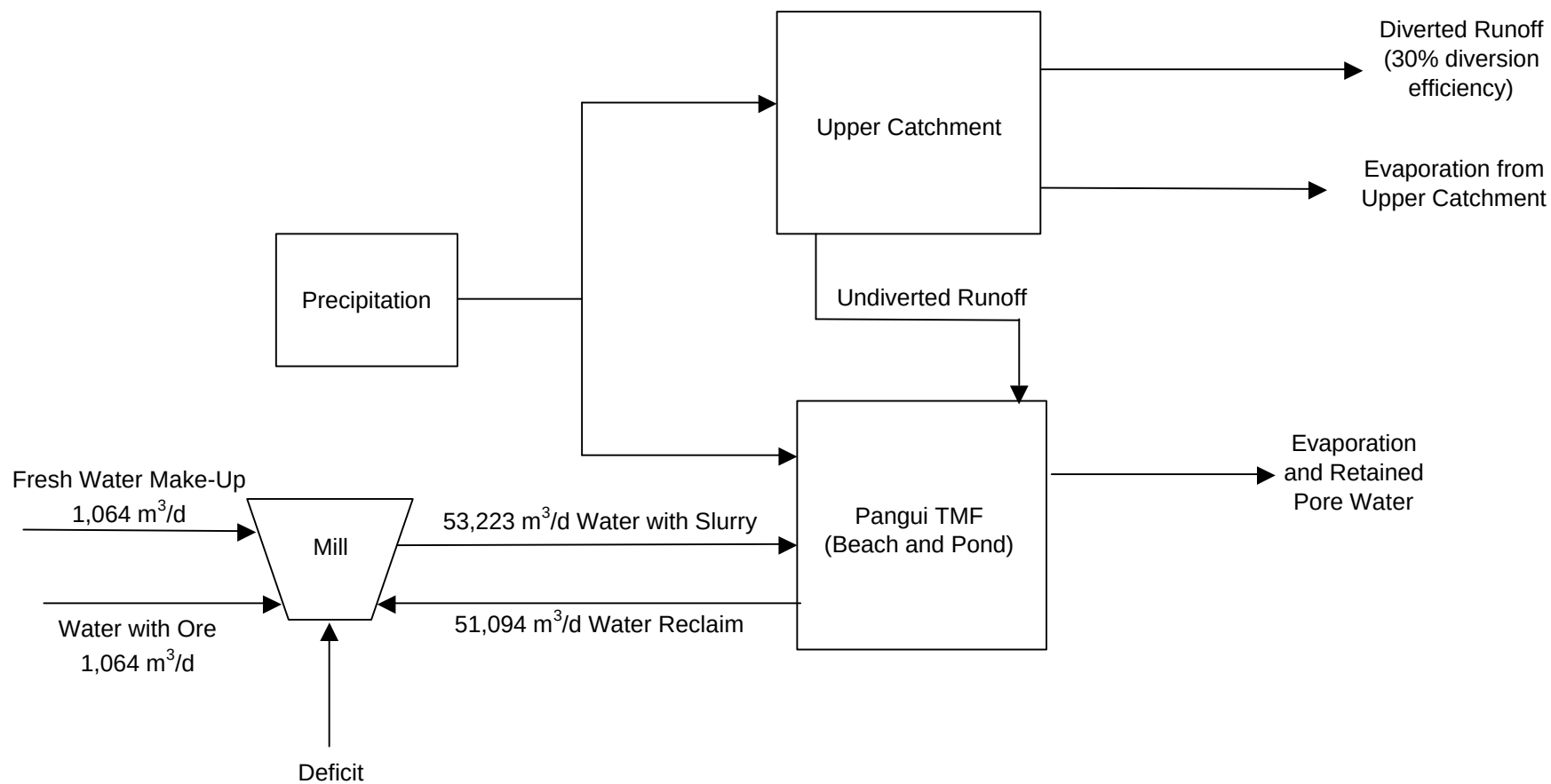
Notes: ¹Tonnage into TMF accounts for 3% removal as concentrate, as well as the required volume for embankment construction. ²Embankment elevation is calculated as average surface elevation in TMF with an additional 4 m freeboard.

The close proximity of the TMF to the Mill reduces the length of the tailings distribution pipework, resulting in decreased capital expenditures as well as reduced power requirements. Rougher tailings will be pumped north, along the eastern bank of the impoundment, where the coarse fraction will be used as embankment material with the cyclone overflow material being deposited into the TMF. Cleaner tailings (high sulphide cleaner tailings) will flow south through a separate pipe and be deposited sub-aqueously, thus preventing oxidation.

7.6.3 Water Balance

A water balance model was developed for the purpose of accurately tracking the volume of solids, entrained pore water, and supernatant water in the TMF. Inputs such as precipitation, evaporation, and production rates, are used to produce a monthly estimate of stored volume, average pond/tailings depth, and required embankment height. The design criteria used for the water balance modelling may be seen in Table 7-4. A schematic of the system elements, including inflows and outflows to and from the TMF are shown on Figure 7-7.

Precipitation and evaporation follow the monthly distribution pattern described above and a runoff coefficient of 0.8 is for the undisturbed portion of the catchment. In addition to the tailings characteristics provided above, a constant consolidation rate of 0.5% per annum is used as a simple estimate of how the solids will settle in the TMF.



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PANGUI TAILINGS MANAGEMENT FACILITY WATER AND MASS BALANCE SCHEMATIC			
<i>Knight Piésold</i> CONSULTING	PROJECT / ASSIGNMENT NO. VA201-00078/4	REF NO. 1	REV. 0
	FIGURE 7-7		



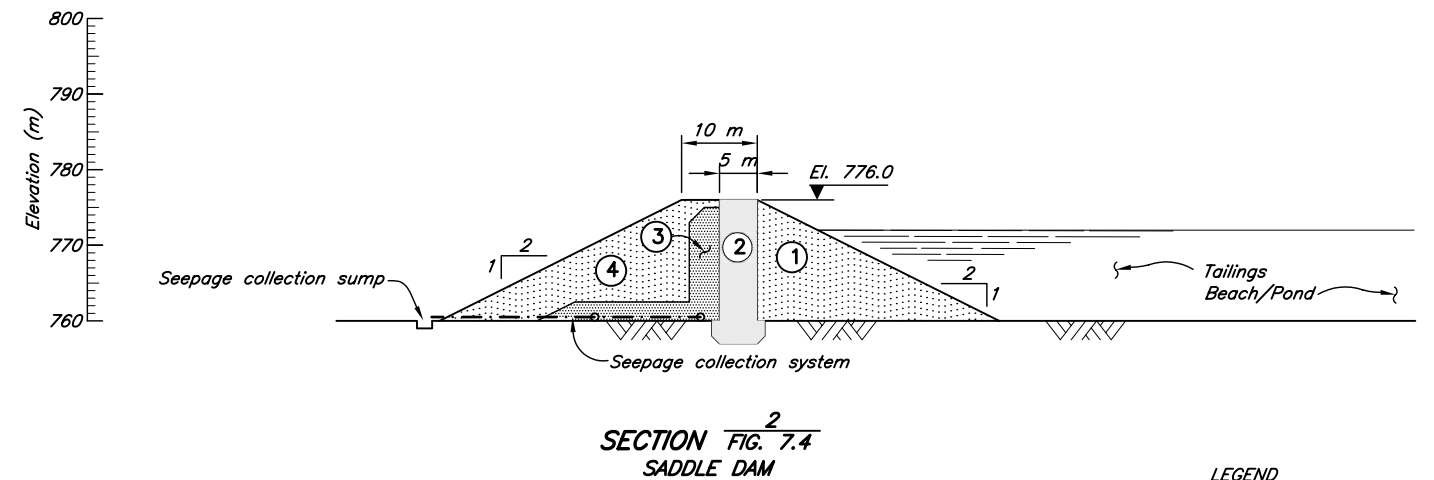
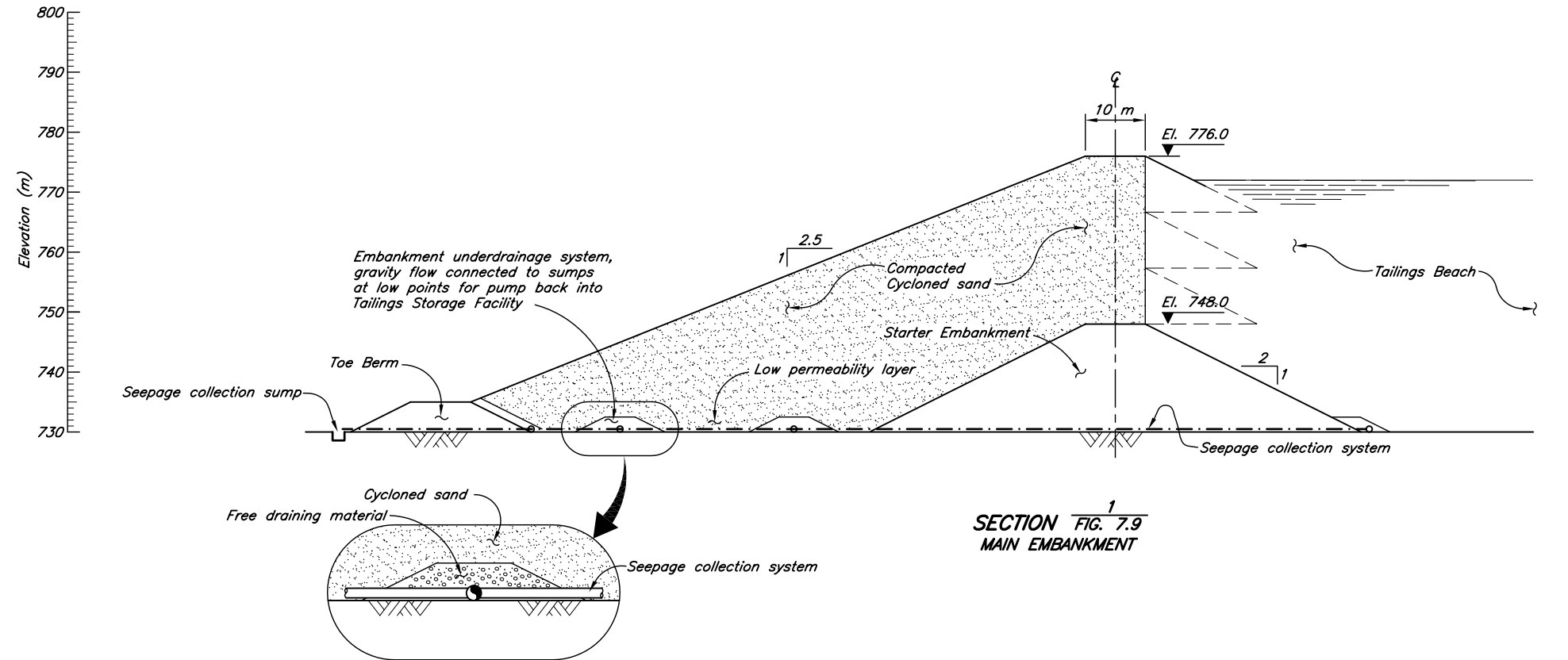
The proposed Pangui TMF is modelled as a zero discharge facility, with no water leaving the system except through evaporation and as process water reclaim. Geotechnical investigations are currently underway to determine the permeability of the proposed impoundment area to see if and how much seepage can be expected. The facility benefits from a relatively small catchment area, which allows it to operate in a favourable water deficit condition with only minor diversion ditching required. The model indicates that about 30% of the upper catchment runoff will have to be directed away from the TMF.

7.6.4 Embankment Design

The embankment will be raised using the centreline construction method, with a starter earth embankment and ongoing raises using compacted cycloned tailings. The starter embankment will be constructed to a crest elevation of 748 m, which provides an adequate storage volume for the first two years of mining operations.

Typical embankment cross-sections for both the main embankment and the starter dam are shown on Figure 7-8. Table 7-5 shows the required embankment elevations for each year of operation, as well as the average rate of rise (metres per year). The required split between fines and coarse tailings material varies from 18% in Year 1 down to 5% in Years 8 to 12; this appears to be within the anticipated range expected from the mill process.

Embankment foundation and toe drains will be installed as shown in Figure 7-8 and will consist of a drain pipe, drain material, sand and filter fabric. The sand will have a filter relationship with the tailings that will prevent tailings from entering the drain. The seepage collected in the drains will be delivered to the seepage collection sump, shown in Figure 7-9, and pumped back into the TMF.



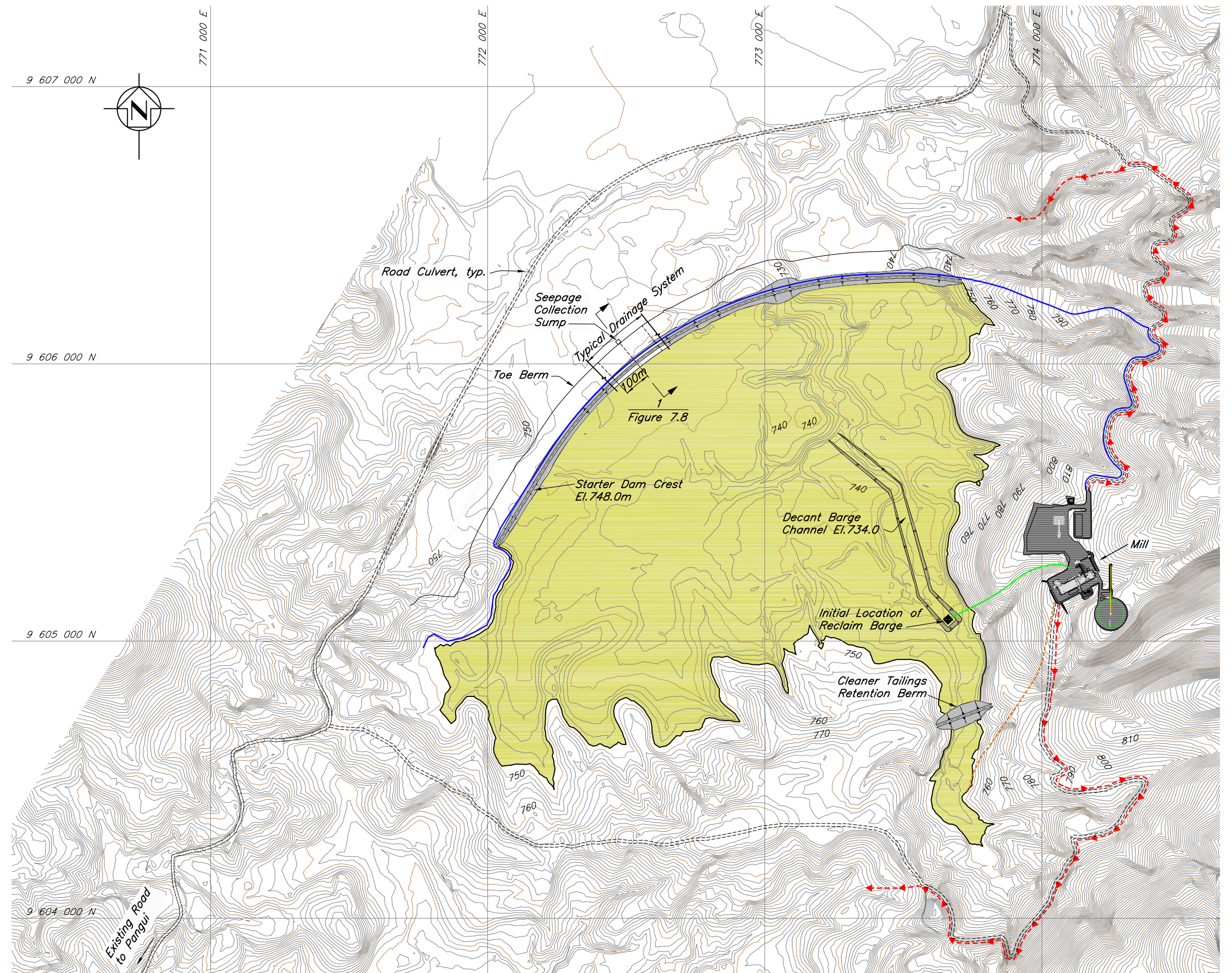
- LEGEND**
- ① Upstream shell
 - ② Low permeability core
 - ③ Filter Zone
 - ④ Downstream shell

Scale 10 0 10 20 30 40 50 m

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PANGUI TAILINGS MANAGEMENT FACILITY TYPICAL EMBANKMENT CROSS SECTIONS			
Knight Piésold CONSULTING	PROJECT/ASSIGNMENT NO. VA201-78/4	REF. NO. AMEC	REV. 0
	FIGURE 7.8		

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LEGENDS:

- Access Roads
- Diversion Ditch
- Reclaim Pipe
- Rougher Tailings Pipe
- Cleaner Tailings Pipe

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PANGUI TAILING MANAGEMENT FACILITY
YEAR 1 STARTER EMBANKMENT – PLAN

Knight Piesold
CONSULTING

PROJECT/ASSIGNMENT NO. VA201-78/4	REF. NO. AMEC	REV. 0

FIGURE 7.9

V:\00078\04\VA201\Figs\B11 1-1000 Plot 1-1 (PS) April 28 2005 By:WAL



7.6.5 Tailings Distribution Pipework

The tailings discharge system is designed to handle the tailings solids and process water from the plant based on a daily throughput of 25,000 t. As discussed above, the rougher tailings would be the most suitable for cycloned sand embankment construction, as they have been shown to be non-acid generating. The high sulphide cleaner tailings are the only tailings that showed the potential of being acid generating if allowed to oxidize. The approximate daily split of the two tailings streams is:

- rougher tailings = 21,950 t/d
- high sulphide cleaner tailings = 2,575 t/d.

In order to prevent the high sulphide cleaner tailings from oxidizing they will be separated from the rougher tailings and directed into the TMF pond through a separate tailings line as shown in Figure 7-4. In order to ensure that the high sulphide cleaner tailings will be perpetually saturated from start-up, a small Cleaner Tailings Retention Berm will be constructed, as shown in Figure 7-9. With the mill in close proximity to the pond this is practical.

The rougher tailings distribution pipeline will deliver the tailings from the plant site to the main embankment where the tailings will be cycloned, as shown in Figure 7-1. The coarse fraction will be used for embankment construction and the finer tailings will be delivered into the TMF.

7.6.6 Tailings Reclaim System

At start-up, a decant channel will be excavated as shown in Figure 7-9. The reclaim barge will be placed in this channel, which is in close proximity to the plant site. Water will be continuously reclaimed from this supernatant pond, brought back to the plant site, and re-introduced into the process.

7.6.7 Instrumentation and Monitoring

As part of the overall monitoring program, eight groundwater monitoring wells will be installed as part of the initial site investigation program. These wells are located downstream of the final TMF embankment toe and will continue to be monitored both during and post construction. These wells will be sampled to evaluate seepage water quality and may be converted to recover seepage if required.

Geotechnical instrumentation will be installed in the tailings embankment and foundation during construction and over the life of the project. The instrumentation will



be monitored during construction and operation of the TMF to assess embankment performance. The instrumentation installed will include:

- piezometers
- movement monuments
- slope inclinometers
- groundwater monitoring wells.

7.7 Risks

There are several potential risks that may arise during the detailed design and construction, including:

- the geotechnical investigations currently underway may determine that the assumptions regarding seepage (low permeability residual soils) in this report have underestimated the rate of seepage
- similarly, the geotechnical investigations may determine the foundation conditions for the embankment are different than assumed.

7.8 Recommendations

The main recommendation is the completion of the site investigation program for the Pangui TMF that is currently underway. These investigations will help to better characterize the site and design parameters and thus define or dismiss the issues raised as potential risks.

The hydrological and climate data gathering should be continued to provide ongoing site specific data. This will allow for further refinement of the water management measures during the detailed design.



8.0 OFF SITE INFRASTRUCTURE

8.1 Access from Site to the Coast

The Mirador deposit is approximately 418 km from the coastal city of Machala (see Figure 8-1). The route passes through the towns of Zamora, Loja, and Santa Rosa. Currently a loaded transport truck takes about 15 hours to make the trip.

The first 8.4 km of road (see Figure 8-2) is from the mine area to the Zamora River (see Figure 8-3) where an old existing barge ferries traffic across one vehicle at a time. It will be insufficient to take the construction loads and will be replaced by an upgraded barge service for construction.

The 6.7 km section of road from the Zamora River to the junction with the Amazon Highway is adequate for construction access to the minesite, overland conveyor and crusher and for mine operations vehicles and the mine services traffic.

The 100 km long section of road from the Amazon Highway junction to the town of Zamora is a wide, two-lane, well drained, gravelled road (see Figure 8-4). Twenty kilometres have been recently paved and a government program is ongoing to complete the paving.

The route from Zamora to the coast is a wide, two-lane, well-drained asphalt road (see Figure 8-5) in good condition.

While two of the bridges on route have been committed for replacement by the government, and a new barge service is needed to cross the Zamora River, the rest of the bridges and culverts along the entire route are in good condition and are part of the main east-west arterial route for heavy traffic crossing southern Ecuador. Bridges are generally rated for 40 t to 50 t capacity, of concrete construction, about 6 m wide on the gravel road and double lane on the asphalt section, see Figures 8-6 and 8-7. A complete review of the bridges on the route was conducted by Merit; its report is included in Appendix G.





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Figure 8-1: Project and Port Locations





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Figure 8-2: Typical Road from Mine Area to Zamora River and Junction with Amazon Highway



Figure 8-3: Zamora River Ferry Crossing





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Figure 8-4: Gravel Road to Zamora – About 7.3 m Wide



Figure 8-5: Asphalt Road from Zamora to Puerto Bolivar





Figure 8-6: Typical Bridge Crossing for Gravelled Amazon Highway



Figure 8-7: Typical Bridge on Asphalt Road from Zamora to Coast





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Road drainage is good and there are no overly steep sections. Elevations along the route from the site to the coast rise through several high points ranging from about 816 m at the plant site concentrate load out area to 2,370 m to 2,800 m and then down to Machala.

The road passes through a number of towns. While there are, in some instances, routes around some of them, it will be necessary to go through some of the smaller villages. In these cases, special care will be needed since people and retail outlets are often on the immediate perimeter of the road. Animals wander and chase traffic, and there are a number of electrical wires that cross from one side of the road to the other and appear to be low in some places. In the case of loads that are oversized in height, it will be necessary to raise the lines as the traffic passes but, in general, highway trucks will be able to pass unrestricted. The worst is at Piedras where traffic will need to use the back streets to by-pass the main throughfare (see Figure 8-8).

Figure 8-8: Congestion at Piedras



There are a number of small slide areas along the road route, but nothing that cannot be contained with gabions and road maintenance currently provided by the municipalities. There are no obvious high-risk areas even though the road winds considerably in some places. The visibility distance in both horizontal and vertical directions is good and there are many passing lane sections. Ice and snow are not hazards that need to be accounted for, although low cloud occasionally reduces visibility.





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It is not anticipated that any improvements to the existing road, culverts, or bridges need to be performed before the operations begin. However, operations will need to be in touch with municipalities on a regular basis to monitor the route maintenance. It is expected that some of the road upgrading and Zamora barge service upgrade will start before the main site construction activities start. However, the current condition of the roads will not prevent the main construction from proceeding.

8.2 Air Transportation

The closest airstrip is a military one in Gualaquiza about 45 minutes drive from the site. The asphalt strip is 2,075 m (6,800 ft) long and could be used by Hercules and 737 style of aircraft, see Figure 8-9. It is possible to use the strip for private traffic but it is not IFR rated.

Figure 8-9: Military Airstrip at Gualaquiza



8.3 Port Selection

8.3.1 Location and Natural Features

Machala is located in southwest Ecuador at similar latitude to the Mirador Project. It is the closest port city to the Mirador Project. Puerto Bolivar is located some 7 km south of Machala. It is accessible by road from the project site, 418 km away. Machala's industry is based on exportation of bananas and shrimps. The bulk of Ecuador's banana exportations go through Puerto Bolivar, see Figure 8-10.





Tides, Storm Surge, and Tsunamis

The shore depth at Puerto Bolivar is between 8 m and 10.5 m. The minimum depth in the channel is 9 m. The tide's amplitude is 3 m. Autoridad Portuaria de Puerto Bolivar (APPB) is in process of dredging the channel to 11 m.

The channel is naturally protected and storm waves combined with tide and storm surge do not have a significant effect which will be taken into consideration for detailed design of the slope protection.

The most recent Tsunami had no effects on APPB and no measurable impact at Machala. The site-specific risks of a tsunami at Machala have not been investigated. APPB is located 4.5 nautical miles from open sea and behind a row of small islands, and well protected.

Meteorological and Oceanographic (Metocean) Conditions

No independent check was made on Metocean conditions. Based on various sources of published or interview data, winds are generally mild. The speed of wind varies between 3 and 8 knots. The half speed of the current is 1 knot (0.5 m/sec).

The port is sheltered from seas and swell, but according to its experiences, the banana loading operation does suffer rain related downtime. As Corriente's operation will be largely automated, it is not expected that rain will affect the concentrate loading operation.

Predominant winds are from the Southwest. The weather is always hot, with strong annual rainfall and no fog.

From the sea buoy, 4.5 nautical miles from the port, depth is between 9 m and 12 m.

The topography of the area is low and regular ranging up to 5 masl.

Sedimentation and Maintenance Dredging

APPB is currently responsible for maintenance dredging. It is understood that annual sedimentation rates are low and that maintenance dredging has been required every 15 years or so. APPB is currently dredging a 30 m width of the 130 m wide channel. Dredging of the channel is financed with a \$0.05 DWT toll for each vessel passing in. Dredging at docks will be limited and easily achieved to 10.5 m.





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Figure 8-10: Puerto Bolivar



8.3.2 Infrastructure

Corriente Resources met with the Autoridad Portuaria de Puerto Bolivar (APPB) to discuss the feasibility and eventual installation and operation of a copper concentrate export facility on their concession. APPB has expressed interest in the venture.

APPB has six docks of which two are not utilized. APPB manages the concession for the government and is expected to expand the utilization and revenues of the port thru the development of further exports. APPB has offered Corriente the use of Docks 5 and 6, and Patios 14, 15, and 16 at its facility, see Figure 8-11.

Figure 8-11: Puerto Bolivar Site Plan





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- APPB is fully compliant under IMO (International Maritime Organization) and has certificates of Compliance of Port Facilities, issued under the International Code for the Security of Ships and of Port Facility (ISPS code).
- APPB has completed an EIA for their present port facilities. Plans for the concentrate loading facilities will need to be added as an amendment to this EIA.
- APPB has the required draft depth to allow free movement of vessels.

APPB appears to have most of the infrastructure and facilities Corriente would need.

8.3.3 Port Planning and Design Criteria

Cargo Throughput

Only concentrate will be exported. Corriente is not contemplating importation from Puerto Bolivar. It is estimated that exportation will be up to 18,000 tonnes/month.

Berth Occupancy

The existing APPB facilities consist of a pier of approximately 30 m x 100 m and 440 m of dolphins. Overall, port berth occupancy is in the range of 7%. Since Corriente will have its own berth, sufficient capacity exists for exporting the estimated 206,000 wt/a of mineral concentrate.

Environmental Controls

To prevent harmful emissions of mineral concentrate to the environment, handling facilities will be enclosed and exhaust air filtered. Concentrate will be received within an enclosed building, and truck-cleaning facilities will be provided within the building. Storage and reclaim systems will be within the building, with sufficient ventilation and dust collecting systems to ensure dust emissions meet regulatory discharge limits. Terminal conveying systems and transfer towers will be enclosed.

Concentrate Reception

Concentrate will be delivered in highway trucks on a continuous basis, seven days per week. At the maximum throughput rate of 206,000 wt/a, daily concentrate reception will average approximately 590 t/d; about 20 truck-loads per day.





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Concentrate Storage

Storage capacity is set at 15,000 t. At 7% of peak annual throughput, and 150% the predicted cargo load, this capacity is typical for a single product facility. Arrival of ships will be evenly spread out over the month to manage shed inventory.

Ship Sizes

Based on existing trade patterns for the west coast of South America, vessels are expected to range in size from small Handysize bulk carriers in the 12,000 DWT to 20,000 DWT capacity class to a maximum Handymax vessel of 47,000 DWT. A detailed analysis of the marine market identified that empty 33,000 DWT bulk carriers currently travel from Guayaquil to Peru. These 33,000 DWT vessels are Corriente's best opportunity to optimize freight rates.

The ship loading system will be designed for 33,000 DWT vessel sizes.

Table 8-1: Handymax Vessel

Vessel Size and Class	33,000 DWT Geared Bulk Carrier, Full Load
Maximum draft	10.8 m
Vessel size & class	33,000 DWT Geared Bulk Carrier, 10,000 t Load
Maximum draft	6.7 m

Shipment Lot Size

The concentrate marketing study commissioned by Corriente, estimated that copper concentrates will likely be sold in 8,000 wmt to 12,000 wmt lots. An analysis by AMEC of the draft requirements for various size ships, confirmed that a 33,000 DWT vessel could load 10,000 t of product and not be affected by tide or draft restrictions. Depending on timing, some delays from 3 to 6 hours might be encountered waiting for tides to access the channel.

Concentrate Material Properties

Assumed material properties for the design of materials handling equipment are summarized in Table 8-2.





Table 8-2: Design Concentrate Properties

Item/Property	Value
Material	Copper concentrate
Bulk density – loose	2.0 t/m ³
Bulk density – compacted	2.2 t/m ³ (for storage calcs.)
Angle of repose	40°
Drawdown angle	60°
Angle of surcharge	20°
Moisture content	6% to 10%

Conveyor Design

Concentrate ship loading rate will be 1,000 t/h.

Rates were set to minimize belt speed and conveyor size while maintaining sufficient throughput for efficient ship loading operations.

8.3.4 Design and Conditions

Port Structural Design

The wharf structure will consist of pilings tied into reinforced concrete pilecaps with a heavy, cast-in-place reinforced concrete deck. The soil borings show several very compressible soil types. This is common to the soil profiles for the wharf and the shed areas.

This soil disposition in both cases makes the installation of pilings necessary and the most economical solution.

In the case of the wharf the alternative of a solid quay is not practical since it would require the removal of several meters of the top layers of soft material which would have to be replaced by a better quality fill. This procedure would be much more costly and time consuming than the proposed piling system.

Pilings will also be driven for the storage shed to avoid large settlements of the upper layers of soft soil. Here also, replacing the soft soil with harder more stable fill would cost more and take longer than the proposed piling.

The loading system will include a fixed elevated conveyor and fixed shiploader, which will be adequately supported on the dock.





Navigation

Vessels arrive, stop and report at the “sea buoy” located 4.5 nautical miles from the port. From there, the tugboat contracted by the ship’s agent brings the ship in. There are several companies in the area offering pilotage and tugboat services. It is mandatory to use them. It is expected that such a service will not cost more than \$0.15 per DWT ship capacity.

Existing Conditions

Currently APPB offers the access to its complete infrastructure. The Mirador Project will be required to build the wharf, shed and loading systems. APPB currently has:

- a fenced, guarded and secured operation certified under ISPC
- access to electricity and fresh water
- appropriate water depth at dock side
- APPB currently manages the reception of 700 banana trucks per day, making Corriente’s 20 trucks relatively insignificant.

Port Facility Design

The proposed port facilities are shown in Figures 8-12 and 8-13.

General Design Concept

A 15,000 t concentrate storage shed will allow the unloading of 20 concentrate trucks per day. A front-end loader will feed the conveyor and stacking system for the two ships expected each month.

Concentrate Receiving

The concentrates received at the port will have moisture contents ranging from approximately 6% to 10%. To control fugitive dust emission, the entire concentrate receiving, storage, reclaim, and shiploading system will be enclosed. Handling facilities will consist of truck receiving scales, an enclosed truck dump (the shed), a truck cleaning bay and turnaround area, a 15,000 t storage shed and transfer conveyors to the berth. Ships will be loaded using a sealed conveying systems.





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Figure 8-12: Proposed Port Facilities – Plan

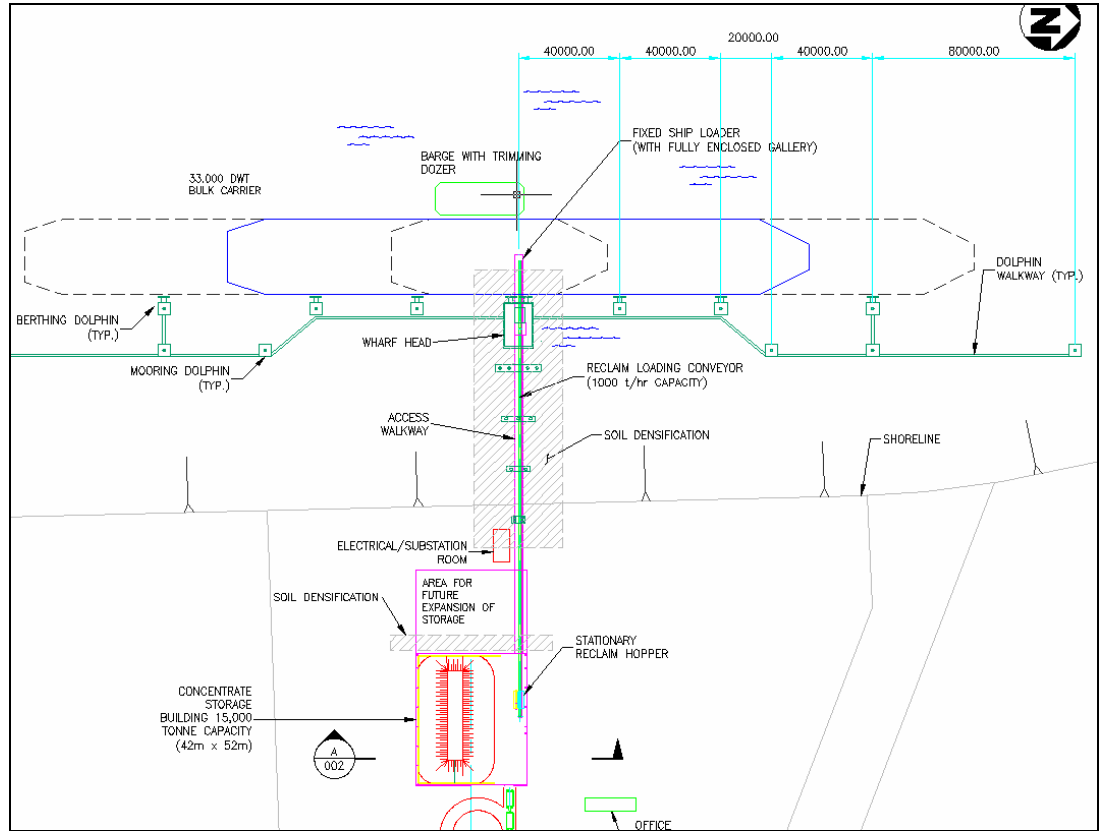
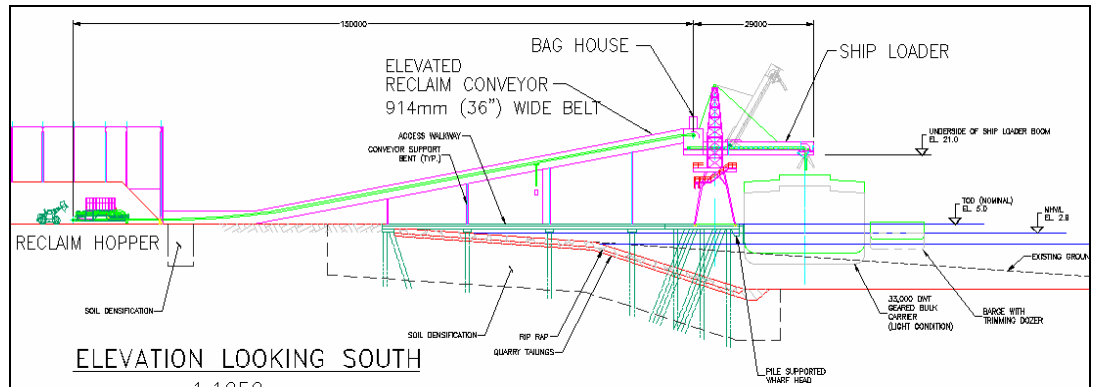


Figure 8-13: Proposed Port Facilities – Elevation





Concentrate will be delivered in highway trucks with a payload of 30 t on a 24-h basis, 7 d/wk. At the maximum throughput rate of 206,000 wt/a, daily concentrate reception will average 590 t/d; about 20 truck-loads. Upon entering the facility, trucks will pass a guardhouse, proceed to a truck scale and then continue to the storage shed. The concentrate will be dumped onto the enclosed shed floor where it will be re-handled by a 988 loader to optimize shed space.

Concentrate Storage Shed

The storage shed will be a 2,200 m² steel portal framed building with reinforced concrete buttressed walls to retain the concentrate piles. The shed will be fully enclosed and equipped with a ventilation system. Ground improvement is required to improve slope stability and the bearing capacity of the existing fills in this area. The shed will have concrete flooring supported by pilings.

The trucks will dump in the shed and be washed before leaving.

Concentrate Reclaim, Conveying, and Shiploading

Concentrate will be reclaimed by a front-end loader into a fixed hopper, centrally located along the north side of the storage shed. Reclaim rates will peak at 1,200 t/h. The reclaim hopper will be discharging onto a belt conveyor. The belt conveyor will be equipped with belt scale to record the weight of the concentrates. A sampling station will also be located near the shed. Collected samples will be shipped back to mine site for assay.

The conveyor will transfer concentrate from the shed to a transfer tower at the end of the pier. The conveyor will then discharge on the shiploader conveyor. A dust collecting system will be located in this tower.

The shiploader will be fully retractable (can be lifted to vertical) and equipped with a rotating chute, and remote hand-held controller for manoeuvring of the chute. The design loading rate will be 1,000 t/h.

The shiploader will be lifted and parked vertically after each concentrate shiploading operation is completed. This will leave the sea lane free in front and avoid accidental contact with any ships.

Once a ship's hold has reached its capacity, the shiploader will be lifted to the vertical to allow the ship to be moved, allowing access to another hold according to the loading sequence. Shiploading will then resume.





After the full cargo has been loaded, the ship's crane will bring on board a small dozer from a rented barge brought near the ship. The dozer will be put on board to trim the load according to the Captain's satisfaction and then returned by ship's crane to the barge. The Captain will then approve the reception of cargo and custom papers can be issued prior to ship's departure.

Ancillary Facilities

Some APPB infrastructure will be made available on a contract basis to the concentrate handling facility, including:

- main gate, guard post and site security
- truck scales
- complete access to Patios 15 and 16 for operators and administration
- power supply.

Additional ancillary buildings and infrastructure will be required and include:

- administration building, including washrooms (approx 15 m x 7 m)
- laboratory and sample storage rooms (approx 10 m x 7 m)
- open garage/maintenance shop for mobile equipment (approx 25 m x 7 m)
- fuel and lube storage for mobile equipment (approx 5 m x 5 m)
- wash water tanks for truck wash (100 m³ tank)
- collection and sedimentation tanks for truck wash water (packaged system)
- truck-washing equipment – pumps and water spray guns (packaged system).

These facilities will be installed by APPB or by a 3rd party port operator. As well, the contracted concentrate trucking company will use the Machala Port as a base, and will install any ancillary buildings that they may require as part of the trucking services.

Mobile Equipment

The following mobile equipment will be required and supplied by the 3rd party port operator:

- one front-end loader (CAT 988 or equivalent) for use as the primary reclaiming machine.
- one small dozer (D6 or smaller) to trim vessel's load.





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Port Operations

The concentrate handling facilities at APPB could be owned and operated in various ways, including:

- operation by APPB
- operation by Corriente
- operation by a third party (OTT).

Operation by Third Party

It is anticipated that a land and facility usage fee will be paid to APPB for the leasing of Dock 6 and Patio 15-16 area, and APPB's other facilities and services.

At the time of writing, no detailed proposal had been received from a third party operator. Fees for services are expected to be:

- an approximate fee of \$0.09/t of cargo DWT for the tugboats and pilot.
- a fee of \$0.05/DWT will be charged by APPB from start up of exportation for channel access.
- an estimated charge of US\$0.90/t will apply, assuming an annual average of 200,000 t of mineral concentrate. This charge excludes port tariffs, customs brokerage and shipping agents but includes stevedoring services.
- it is assumed that the mine operator will lease land on a long-term contract basis from APPB to at least 2035. This lease is estimated to cost \$6,000/month.

On a per wet concentrate tonne basis, and on the basis of 10,000 wt loads on a 33,000 DWT vessel, the estimated unit costs are:

Pilot fees	\$0.30/t
Channel access.....	\$0.165/t
Contracted loading charges	\$0.90/t
Leasing charges.....	<u>\$0.33/t</u>
Total port charges	\$1.70/t tonne of concentrate

The following special terms and agreements for a third party operation will need to be considered:





- Potential impacts to the environment will be studied and submitted to the port authority to be added as an amendment to the APPB EIA.
- The OTT operator will be responsible for operating, maintaining, and cleaning the facility, and will decommission and reclaim the facility following expiry of the contract.
- The OTT operator will provide, operation of the weigh scale, truck unloading for non-dumping trucks, truck washing, stacking, storing, reclaiming, shiploading, trimming, safety and environmental management.
- The OTT contract will be in effect for a minimum of six years, with optional extension periods of up to six years based on modifications to mine life.

8.4 Concentrate Transportation to the Port

Copper concentrate will be loaded onto highway capacity trucks at the administration area using a front-end loader. There will be some 20 trucks making the trip each day to the loading facility at Bolivar Port in Machala.

Companies have been contacted to provide proposals for the transportation of the concentrate from the mine site to the port of Bolivar and for the transporting the concentrate from the port to the refinery.

It has been estimated that, based on a maximum load per vehicle of 30 t, it will need a fleet size of 60 vehicles to manage the 20 vehicles per day required to transfer the 556 t of concentrate per day from the site to the port.

Budget pricing received for transportation of concentrate was US\$35.82/wmt.

8.5 Power Supply

In order to improve the reliability and reduce the costs of purchasing power from the Ecuadorian power grid, Corriente have purchased all the rights to the Sabanilla Hydroelectric Project in Ecuador, including all permits, studies, and designs completed to date. The mine will require 28.3 MW to be delivered to the Mirador plant site to meet its power requirements

The Sabanilla Hydroelectric Project is located in southern Ecuador, in the Province of Zamora Chinchipe, 40 km east of the City of Loja. The project will have an installed capacity of 30 MW and will operate as a run-of-river project with a capacity factor of about 81%. The project is located on the Sabanilla River, a tributary of the Zamora River.





The following summarizes the key project characteristics:

- mean annual flow, Sabanilla River at intake: 12.4 m³/s
- project design flow: 9.8 m³/s
- net head: 366 m
- installed capacity: 30 MW
- average annual energy produced: 212,000 MWh/a
- pelton units will be installed: 2 x 15 MW
- transmission line (138 kV) from Sabanilla to mine: 95 km
- it will also be connected to the Ecuadorian transmission grid to allow the sale of any excess energy the mine does not use
- it will also be able to operate in an islanded condition (i.e., if the Ecuadorian Grid goes down), and be able to supply energy directly to the Mirador Mine.

The overall project layout and concepts proposed are typical of a high head run-of-river hydroelectric project with low-pressure tunnel, high-pressure steel penstock and surface powerhouse.

The weir/diversion structure will include a 2 m high concrete sill with 3 m to 4 m high inflatable rubber dam. Part of the flow will be diverted into an intake structure that will include low-level intakes, screens, sluice gates, and sediment exclusion facilities. The banks of the valley at the proposed intake site are steep and the river is deeply incised with a narrow cross section. Therefore, access to the intake structure, during both construction and operation, will be via an access tunnel.

A D-shaped water power tunnel will be constructed using drill and blast construction technique and operate as a low-pressure conduit to transfer the water from the intake site to the high pressure steel penstock. The tunnel will generally be unlined with a finished concrete floor and adequate support where required. The rock quality is generally good and lining of the tunnel is only expected near the two portals. The surge facility will be located at the downstream end of the tunnel and will be located within the mountain, excavated vertically in rock.

The steel liner and high pressure penstock will run from the surge facility down to the powerhouse, which is located on the right bank of the Zamora River. The penstock will be buried and have a diameter of about 1.6 m.

The powerhouse layout is fairly typical of that required for Pelton turbines, with a concrete substructure and steel superstructure. The project switchyard will be located directly upslope of the powerhouse. A tailrace channel will discharge the water from





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the two units into the Zamora River, directly downstream of the Sabanilla River and Zamora River confluence.

Two 3-jet 15 MW horizontal axis Pelton units are proposed.

The powerhouse switchyard will include a step-up transformer (generation to transmission voltage) and a new 138 kV transmission line will be constructed from the Sabanilla Switchyard all the way to the Mirador Mine substation. It is also proposed that project will be connected into the National Grid and excess power produced will be sold on the SPOT market.

8.6 Employee Housing

The towns of El Panguí and Gualaquiza are a short driving distance from the proposed plantsite and administration facilities. With their developed infrastructure, these towns provide the opportunity for some employee owned housing or private rental accommodation. The construction camp facilities will be retained and maintained during mine operations and will provide alternative short-term accommodation for employees as may be required.





9.0 ENVIRONMENTAL AND PERMITTING

9.1 Environmental Baseline

Terrambiente, a Quito-based environmental firm, has completed an environmental baseline assessment for the Mirador Project. Baseline data collection commenced in March 2004 and has been ongoing through the study period.

The Mirador Project is located within the Cordillera del Condor, an area considered ecologically important because of its high biological diversity and presence of endemic species.

The topography of the project area is irregular, with altitude varying between 760 m and 1,500 masl. The climate is rainy subtropical with temperatures ranging between 10°C and 35°C.

The principal rivers in the area are the Zamora, Quimi and Tundayme rivers. Land use in the area varies from farming and grazing to secondary forests and remnants of first growth forests. Wood is harvested by the local inhabitants for use in construction and as a source of fuel for cooking.

The villages of San Marcos, Valle de Quimi, and Tundayme are found within the direct area of influence of the project. Within this area, the small village of Chuchumbletza and the town of Pangui are also taken into consideration due to their location near the tailings pond and processing facilities.

The town of Gualaquiza is located within the indirect area of influence.

9.2 Physical Component

9.2.1 Hydrography

The mine site is along a west-facing slope of the Cordillera del Condor, which forms the main drainage divide, and is east of the Andes. This shields the mine site from the Pacific Ocean wind currents from the west. It is exposed to weather patterns from the central Amazon River basin and the Atlantic Ocean on the east. The mine site drainage basin elevation ranges from 1,300 m to 2,100 m and drains into the Quimi basin. The high-altitude rain clouds surround the mine site and contribute to the total precipitation experienced at the project site.



In general, the mine site project area is located within the Quimi River basin. Within this basin are the Tundayme and Wawayme rivers. The Quimi River flows west to the Zamora River, which flows northeast to the Amazon Basin.

The tailings and processing site is located 10 kilometres west of the mine site. The elevations range between 760 masl and 825 masl. The site is located within the Zamora River Basin and contains smaller watercourses such as the Paquintza River, which flow to the north towards the nearby Chuchumbletza River.

Small streams in the area flow into the Zamora River and are used for watering cattle and also for domestic use. The presence of a swampy area and the hydrological conditions in the area make it probable that an unconfined superficial aquifer exists, which is protected by an impermeable soil layer that helps form the swampland. This hypothesis will be confirmed by drilling and the installation of piezometers.

9.2.2 Hydrology

Hydrologic parameters for the Mirador site were estimated using data from the nearest weather monitoring station at Gualaquiza, about 30 km to the northeast of Mirador. Adjusting for the elevation difference between the two sites, the mean annual precipitation at Mirador for the mean basin elevation was estimated to be 2,708 mm. This annual rainfall volume was applied to the open pit and waste dump diversion ditch mean drainage basin elevation of 1,750 m. The volume for the open pit and waste dump site was estimated to be 2,373 mm at a mean basin elevation of 1,350 m.

The meteorological conditions at the proposed Pangui TMF Site were estimated using the information found in the Caminosca report "Informacion Meteorologica e Hidrologica." The average annual rainfall at the Pangui TMF is assumed to be 2,000 mm. Monthly precipitation distributions and standard deviations are based on data from the El Pangui and Gualaquiza monitoring stations; these may be found in Table 9-1. Site specific hydrological studies are presently being developed for the Tailings Management Facility area and will be included within the final environmental impact assessment document.

9.2.3 Geology

Granodiorite and granite of the mid- to late-Jurassic Zamora batholith forms the mineralized host of the Mirador porphyry system. The mineralization consists mostly of fine-grained chalcopyrite disseminated evenly throughout the granite, together with gangue minerals pyrite and lesser magnetite. A thin enrichment blanket of chalcocite overlies the chalcopyrite mineralization.



A series of barren, northeast trending (mostly dipping steeply to the northwest), hornblende-feldspar-porphyry dikes cut the Zamora rocks. The flat-lying, cretaceous Hollin formation quartzite unconformably overlies the Zamora, with pre-Cretaceous erosion removing most of the early to mid-Jurassic Misahualli and Chapiza volcano-sedimentary rocks. These latter units are present only as hornfelsed pendants within the Zamora batholith.

All units display characteristic tropical weathering profiles, with saprolite thickest under ridges and streams generally cut down to bedrock. The larger river systems have thick colluvial and alluvial cover gravels that are weathered to clays at depth. The thickest cover sequences, in the Zamora and Quimi rivers, are related to fault movements.

Four fracture systems are found at the site: NNE, E-W, NNW-SSE and NE-SW. Of these, only the NNE and NE-SW appear to have related fault movement. The Quimi River appears to be controlled by a NE-SW fault, and, 12 km to the west, the Zamora River follows a N-S structure that has clearly tilted the Hollin strata. No significant or mappable faults were discerned within the porphyry system.

The tailings and processing plant site is located within the Hollin formation. It presents mainly well-cemented, finely grained quartz rich sandstones that appear to be dipping mainly to the west. Some porphiritic bodies were identified 1 km to the north of the river crossing and may correspond to an extension of the Zamora batholith.

9.2.4 Morphology and Soils

Four morphologic units were observed in the study area: Planar upper (ridge-top) surfaces (tabular to monocline), dissected relief on granitic basement, and lower valley fluvial terraces. The relief in the project site area is steep.

In the area of the mine, two principal soil types were found: Entisol, immature soils that lack the vertical horizon development, and Inceptisol, young soils that are more developed than entisols.

In general, the soils are pH neutral (6 to 7) with a low organic content. Arsenic, chromium, copper, lead, and vanadium, were detected in the soils in the proposed tailings management facility and open pit.

The tailings and processing site has an average altitude between 760 m and 825 m above sea level over 90% of its area, with the presence of low slopes and semi-flat terrain with small, rounded hills. To the north of the site are the slopes of the Cordillera, with a maximum altitude of 960 masl and where hillsides with steep slopes can be observed.



Analysis of these soils is currently underway for the same parameters as in the mine site.

9.2.5 Water Quality

Twelve background water quality stations were established for the Mirador Project. Another four stations have been established for the tailings site and the first water samples are currently being analyzed. Table 9-1 indicates the location of the stations for the project referenced geographically.

In general, results obtained regarding water quality in the area indicate that parameters are within Ecuadorian Drinking Water Quality Guidelines (SUMA), except for the presence of coliforms.

All of the water quality samples contained coliform levels high above regulated water quality limits. Samples taken downstream of the villages of San Marcos and Tundayme contained the highest concentration of coliforms. These villages lack sanitary treatment facilities.

Parameters such as metals within the water quality samples presented no anomalous readings. During the rainy season, soils from the steep sided terrain are washed into the river systems. This was confirmed by the high amount of total suspended solids detected in the river water samples. A total suspended solids (TSS) baseline sampling program is being completed to establish baseline TSS levels (likely exceeding SUMA and World Bank limits) during rainfall events.

Samples presented a low Biological Oxygen Demand (BOD), a result of steadily flowing water that does not allow local decomposition of any organic matter which may deposit along the river shores.

Some river water samples taken during dry weather conditions were tea coloured, characteristic of the presence of tannins. Foam was also observed in some of these samples. Such characteristics are related to the presence of organic matter along the drainage paths, and thus, are natural in origin.

It has been detected that the level of tannins and colour within the watercourses may depend on the recent precipitation and water flow level. The present water quality monitoring program is evaluating obtained results in order to establish a relationship between these factors.



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Table 9-1: Location of Background Water Quality Stations – Mirador Project

ID	Location	Coordinates
WQ1	Tundayme River, discharge from Tailings	784838 E 9600168 N
WQ2	Wawayme River, before operations discharge	784900 E 9605796 N
WQ3	Creek No Name – water from proposed Waste Dump Area	784840 E 9605794 N
WQ4	Creek No Name – water from east side of proposed Open Pit	784478 E 9605753 N
WQ5	Creek No Name – water from central part proposed Open Pit	783806 E 9605920 N
WQ6	Wawayme River, after operations discharge	781976 E 9606324 N
QW7	Cristalino River, reference point	784649 E 9610784 N
WQ8	Quimi River, before Valle del Quimi village	784372 E 9610772 N
WQ9	Quimi River, after Valle del Quimi and before San Marcos village	782156 E 9606961 N
WQ10	Quimi River after San Marcos and before Tundayme village	780482 E 9604926 N
WQ11	Tundayme River, before branching	780946 E 9603617 N
WQ12	Quimi River after Tundayme village	777707 E 9605171 N
WQ13	Tundayme River, before Tailings	789123 E 9537904 N
WQ14	Tundayme River, after Tailings	785639 E 9600293 N
WQ11A	Tundayme River, right branch after the conveyer crossing	780491 E 9604839 N
WQ15	Zamora River, after the mouth of the Machinaza	775119 E 9602056 N
WQ16	Zamora River, after the mouth of the Chuchumbleta	774790 E 9610261 N
WQT01	Stream that flows into the Zamora River and originates before the processing plant site	774100 E 9604813 N
WQT02	Paquintza River, after Tailings	773312 E 9606904 N
WQT03	Paquintza River, before Tailings	773784 E 9604561 N



9.3 Biological Setting

Baseline biological data was collected in five areas. These sites represent areas of direct impact by the project as well as a reference area, which will not be influenced by proposed mining activities.

The biological components sampled are representative of the flora and fauna of the following study areas:

Site 1 – Upper Tundayme River Basin

Site 2 – Tundayme River Basin

Site 3 – Open Pit and Waste Dump Site

Site 4 – Cristalino River Area – Reference Area

Site 5 –Tailings & Processing Site Facilities (in progress)

9.3.1 Mammals

A total of 45 species belonging to eight families and nine orders of mammals were identified. These represent 12% of the total mammals registered in all of Ecuador, and 32% of the species registered in the Cordillera del Condor. This area is considered a moderately biological diverse environment. Of the 45 species identified only three, comprising bats, are considered abundant.

Of the species identified in the area, six mammals are considered Vulnerable (VU) according to the IUCN Red List of Species, and the rest are considered under minor risk.

Some of the mammals like the otter, “machin” monkey, and tapir, are sensitive to habitat changes and may move out of the area when mining activities commence.

Analysis results for the tailings and processing site are currently being processed and will be included within the final environmental impact assessment document.

9.3.2 Birds

A total of 154 species of birds were identified during the baseline program. This represents 12.5% of the birds registered in the Ecuadorian Eastern Region and 9% of the birds in Continental Ecuador. In general, the study area contains a relatively low



diversity and abundance of bird species. Six of the 154 species identified during the baseline study are endemic to the area. According to the “List of Birds in Risk of Extinction in Ecuador” (Granizo, et al., 2002), 18 species are considered rare, five species are endangered, three species are considered vulnerable, and two are considered almost endangered. Some of the birds identified in the area migrate from North America while others migrate within Ecuador.

Analysis results for the tailings and processing site are currently being processed and will be included within the final environmental impact assessment document.

9.3.3 Reptiles and Amphibians

A total of 31 species of amphibians and 12 species of reptiles were identified in the study area, corresponding to 63% and 52% of the total amphibians and reptiles registered in the Cordillera del Condor. Fifteen of the amphibian species identified need water to reproduce while 16 of these do not depend directly of water resources.

The majority of the amphibians identified are nocturnal (96%). Two species of amphibians are registered as endemic to the eastern flanks of the Cordillera.

Analysis results for the tailings and processing site are currently being processed and will be included within the final environmental impact assessment document.

9.3.4 Insects

A total of 38 species of insects were sampled during the base line field program. This included seven species of diurnal butterflies. The numbers sampled indicate a high diversity of butterflies in the project area.

Analysis results for the tailings and processing site are currently being processed and will be included within the final environmental impact assessment document.

9.3.5 Fisheries Resources

The rivers sampled for fisheries resources included the Cristalino, Quimi, Tundayme and Wawayme. A total of 24 species from 10 families of fish were identified during the baseline study. The study area consists of aquatic systems characterized by the presence of matting, secondary brooks and clear water rivers with sandy bottoms, with medium to small rocks, a few of woods and leaves, a considerable stream, and conditions that determine a low fish diversity.



No irregular fish behaviour was observed at any of the sampling sites, nor alterations in pigmentation or appearance.

Analysis results for the tailings and processing site are currently being processed and will be included within the final environmental impact assessment document. The sites sampled include: the Paquintza and Chuchumbleta rivers and local drainages at the site.

9.3.6 Benthos

The water quality in the rivers in the project area is relatively homogeneous, indicating similar physical and chemical water conditions. A large number of macroinvertebrate species were sampled from the rivers indicating a high diversity and good water quality in the area.

Analysis results for the tailings and processing site are currently being processed and will be included within the final environmental impact assessment document.

9.3.7 Flora

A total of 583 species of flora were identified during the baseline study. It is important to mention that during the study, at least three new species for the cordillera and Ecuador were registered within the humid flank forest as well as the low mountainous bush forest. Of these three species, one is a palm tree which is a possible new species to science, but studies are still inconclusive.

Analysis results for the tailings and processing site are currently being processed and will be included within the final environmental impact assessment document.

9.4 Ecology

The ecology of the high, moderate, and low land areas are closely linked, and disturbances within a specific zone will affect other ecological zones.

In general, the project area has a variety of habitats, due to the climatic and geomorphological characteristics of the area. Due to the climatic conditions and the availability of food, altitudinal migration patterns have been observed for some groups of species.

Analysis results for the tailings and processing site are currently being processed and will be included within the final environmental impact assessment document.



9.5 Environmental Impact Assessment

9.5.1 Potential Impacts to the Physical Environment

Open Pit Mine and Waste Dump Impacts

The open pit mine will be developed with pit walls up to a height of over 400 m. Over the life of mine (one year pre-production development and 12 plus years of production mining), the operation will remove a total of 202 Mt of material, including 111 Mt of ore and 91 Mt of waste. The primary potential impact of mining will be on water quality, both in sediment and potential acid mine drainage from oxidation of exposed wall rocks.

The mine waste dump will contain about 91 Mt of mine waste material, consisting of about 55 Mt of sulphide-bearing wastes and 36 Mt of saprolite. The sulphide-bearing wastes are expected to be acid generating materials. These acid-generating wastes will present the most significant potential impact of the project, as the drainage from these dumps will likely become acidic and contain dissolved metals. If acid generating materials are not properly managed, drainage will likely present a risk to the water quality of the Wawayme River and downstream on the Quimi River.

Tailings Management Facility

The construction of the tailings management facility in the Pangui area will involve covering about 400 ha of mainly farming plots surrounded by secondary growth forest. The site will serve to provide permanent impoundment for about 106 Mt of tailings over the life of the mine. The Tailings Management Facility (TMF) will be developed by constructing a starter embankment from material excavated from within the TMF basin. The final embankment will be raised during the operating life of the mine. The raising of the embankment is achieved by cycloning the tailings and placing and compacting the cycloned tailings in lifts, using the centerline construction method

Potential negative impacts of the TMF are as follows:

- potential contamination of the Paquinzta and Chuchumbletza rivers by the tailings process water, as well as, Zamora river due to stream flow into the Zamora
- potential acid generation of tailings on closure of the TMF
- potential for dam instability leading to failure of the impoundment
- loss of habitat in tailings impoundment area
- change of natural drainage patterns



- potential soil instability on surrounding slopes of the impoundment
- potential contamination of local ground water
- potential change of hydraulic behaviour of local ground water
- visual impact of landscape changes.
- noise disturbance from the plant areas and surrounds, as well as on roads due to increase of vehicle circulation
- increase of dust in air or mud (according to season) near to road due to vehicle circulation
- exhaust emissions from engines.

Site Development Negative Impacts

Development of the project will entail considerable terrain disturbance, with potential sediment generation from construction disturbance and slope instability. The steep sided terrain, unstable soils, and periods of prolonged heavy rainfall promote erosion. This is evidenced by the elevated levels of suspended sediments detected in river water quality samples during rainfall events. Stripping of the vegetation for the development of the mine and exposure of the soils will promote the release of suspended sediments. Some of these areas will include:

- road cuts
- tailings dam and borrow pit development
- plantsite excavation
- waste dump foundations
- diversions.

9.5.2 Potential Negative Impacts to the Biological Environment

The natural ecology in the project area contains important ecological resources. While the secondary forests provide refuge and food for a portion of the local area wildlife, the primary growth forests with their dense canopies provide habitat for other flora and fauna. Development of the mine and construction of the tailings management facility will alter the ecosystem balance affecting biodiversity in the area. Wildlife will move out of the disturbed areas and migrate to other areas.



9.6 Proposed Mitigation Measures

In order to minimize the impact from mining activities on the physical environment, Corriente is developing a water management plan and closure and reclamation plans for the various phases of project development. The plans will address the impacts from the construction, operation and closure phases.

9.6.1 Open Pit Mine and Waste Dumps Mitigation

Diversions will be designed and constructed to route flow around and away from the pit to maintain water quality in the area and to minimize impacts to operations. Runoff within the pit area and overflow exceeding the ditch design volume or failures will be collected, pumped or routed out of the pit by ditches, pipes or other means.

If water quality is in compliance within effluent limits, the water will be conveyed and discharged. If the water quality is not in compliance within effluent limits, the water will be treated by active or passive means to bring the water into compliance and discharged.

Pit water will be diverted or pumped and routed out of the pit into the Wawayme River directly when in compliance or treated and discharged if necessary.

Studies are currently under way and will continue as the project develops, to assess the rate and potential severity of acid drainage from the waste dumps and open pit. Humidity cell testing is currently being carried out. As well, hydrological studies are ongoing, with a network of weather monitoring and rain gauges, as well as stream gauges, to help in better estimation of flows in the various drainages on site.

It is expected that water treatment may be required to mitigate acid mine drainage, particularly from the waste dumps. If the pH or other parameters are below the regulatory limits, the following generally accepted treatment schemes, alone or in combination, will be used to bring the discharge into compliance, as necessary.

Passive treatment systems based on the physio-chemical mechanisms such as adsorption, ion exchange and metabolic uptake will be used as a first mitigation step with the following options:

- discharging water into sediment channels and weep berms, using organic soils and plants to reduce metals and improve pH
- utilize natural wetlands and/or construct wetlands, using microbial activity, algae and vascular plants to reduce concentration of metals in the water



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- bactericides such as ProMac bioreactors, and other bioremediation techniques may be used to inhibit acid generation in the waste dumps, facility pads, and fill areas that may be acid generating
- metal tolerant plants may be used for reclamation and as a control mechanism as necessary and acceptable in the vegetation scheme.

If the discharge from any mine facilities (pit, waste dumps, tailings pond) are below the regulatory pH, and passive treatment systems are not adequate, active chemical treatment will be used to mitigate impacts. Typical bases or alkalis used to treat acidic water are:

- limestone (CaCO_3)
- quicklime (CaO)
- hydrated lime (Ca(OH)_2)
- soda ash (Na_2CO_3)
- caustic soda (NaOH)
- sodium bicarbonate (NaHCO_3)
- ammonia (NH_3).

Depending on the water chemistry, flocculants and coagulants may be used to reduce the metals concentration prior to neutralization. Aeration may be also used as a pre-treatment to reduce iron and manganese.

The alkali treatment requirements will vary over the life of the mine dependent upon concentration of heavy metals (Al, Mn, Zn, Cu, Fe). The concentration of metals in the pit wall and waste rock will vary over the life of the mine requiring different dosage levels of alkalis to raise the pH and precipitate the heavy metals prior to discharge to the receiving waters.

The application processes may be relatively simple, flow through hoppers dissolving soda ash briquettes; drip or trickle systems discharging NaOH based on manual or automatic flow measuring systems; semi or fully automated water treatment plants.

As new technology and treatment methods develop during the life of the operation, they may be utilized to mitigate impacts and maintain compliance.

As the waste dumps are constructed, any final surfaces will be capped with impervious soil or geomembrane liners to reduce infiltration, and hence reduce the volume of water that would have to be treated. Containment and monitoring procedures will also



be developed to avoid leaching or spilling of treatment chemicals out of the system to during all the project phases.

9.6.2 Tailings Management Facility Mitigation

The Pangui TMF has been designed as a zero discharge facility and includes the following mitigation measures:

- A starter embankment will be built to an elevation of 748 m prior to the beginning of operations; this will provide about two years of storage at the proposed 25,000 t/d production rate. This elevation has been selected to provide ample time for commissioning of a cyclone system for coarse sand production and the procedures for placing and compacting the sand fraction.
- The main embankment will be raised to its ultimate elevation of 776 m using cyclone sand raises, using the centreline construction method.
- In addition to the starter embankment, a small dam will be built at the southeast corner of the facility to an elevation of 760 m to provide a pond in which the cleaner tailings may be stored under a constant water cover. This small dam will eventually be submerged as the pond surface rises to its ultimate elevation.
- The TMF is designed to accommodate the 106 Mt of solids produced over the 12-year life of the project. The principal design consideration is ensuring that the proposed facility meets the “zero discharge” criteria, meaning that no water will be released from the facility throughout the life of the project.
- The geochemical characteristics of the Tailings have been tested by SGS Lakefield Research Laboratories and the results of these tests are presented in SGS Lakefield Research Report LR10703-002 (February 2005). Based on the results presented in this report the rougher tailings would be the most suitable for cycloned sands embankment construction, as they are non acid generating. The high sulphide cleaner tailings are the only tailings that showed the potential of being acid generating if allowed to oxidize. The composition of the tailings when averaging from the Super-Composites for the rougher tailings and high sulphide cleaner tailings is as follows:
 - 87% of the total feed reports to the rougher tailings (21,950 t/d)
 - 10.3% of the total feed reports to the high sulphide cleaner tailings (2,575 t/d)
- In order to prevent the high sulphide cleaner tailings from oxidizing the plan is to separate the cleaner finer tailings component in a separate tailings line directed into the TMF pond so that it would be perpetually saturated and therefore will not oxidize. With the mill in close proximity to the pond this is practical.



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- The proposed Pangui TMF is modelled as a zero discharge facility, with no water leaving the system except through evaporation and as process water reclaim. The facility benefits from a relatively small catchment area, which allows it to operate in a favourable water deficit condition with only minor diversion ditching required. The model indicates that about 30% of the upper catchment runoff will have to be directed away from the TMF.
- As part of the overall monitoring program, eight groundwater monitoring wells will be installed as part of the initial site investigation program. These wells are located downstream of the final TMF embankment toe and will continue to be monitored both during and post construction. These wells will be sampled to evaluate seepage water quality and may be converted to recover seepage if required.
- Geotechnical instrumentation will be installed in the tailings embankment and foundation during construction and over the life of the project. The instrumentation will be monitored during construction and operation of the TMF to assess embankment performance. The instrumentation installed will include:
 - piezometers
 - movement monuments
 - slope inclinometers
 - groundwater monitoring wells.

9.6.3 Site Development Mitigation

Road cuts and fill areas will be designed to be stable for safety and to minimize environmental impacts of erosion and water quality. Cut and fill slopes will be revegetated as soon as practical. Drainage controls will be designed to safely pass the peak runoff from a 30 year storm event. Controls include but are not limited to bridges, ditches, cross drains, and ditch relief drains.

Drainage pipe and culverts will be designed, installed and maintained to prevent erosion at the inlet and outlet, to prevent uncontrolled drainage over the road surface and fill, and to be non-clogging.

Road surfaces will be designed, surfaced, and maintained for the traffic volume, speed, and weight of vehicles using the road. Non-toxic and non-acid forming materials will be used for roadbed and surface construction.

Corriente will carry out an active program of concurrent reclamation/revegetation of disturbed surfaces during mine operation.



9.6.4 Chemicals and Hazardous Materials Mitigation

Corriente will comply with Ecuadorian hazardous materials handling and disposal regulations. The following general guidelines will be followed and detailed in the Environmental Management Plan. World Bank guidelines and international standards will generally be followed unless they are in conflict with Ecuadorian regulations which will take precedence.

Hazardous Materials Storage

Hazardous materials will be segregated and stored using best management practice including but not limited to the following:

- the storage areas will be designed to adequately and safely store a sufficient quantity over a prescribed time limit, one to three months
- the storage area will be properly designed to contain and prevent contamination of the environment, particularly soil and water
- floor, curbing, walls and roofs will be designed to adequately contain spills and protect the storage area from weather
- spill kits, protective equipment, and other necessary equipment will be in the storage area or approximate to the storage area to clean and mitigate spills
- fire prevention systems will be designed appropriate and adequate to the material being stored
- only containers that are in good condition will be used
- containers or liner materials will be compatible with the waste being stored
- incompatible (e.g., bases and acids) materials will not be stored in the same container and will be stored in safe manner and distance to prevent accidents
- to provide a safe work area, walls, dikes, berms, or separate facilities of other means will separate incompatible materials
- drums, containers, and storages will be properly labelled, marked, placarded and secured
- sufficient storage space between containers will be allowed for safe access and handling of containers
- no smoking, fire prevention and management practices will be developed specific to the materials being stored.

No cyanide, mercury or other acutely hazardous material or wastes are currently planned to be used or generated at the operation.



9.6.5 Waste Management Plan

The Waste Management Plan will be developed in consideration of World Bank and North American guidelines. Corriente understands that Ecuadorian regulations and guidelines have not been fully developed. As Ecuadorian regulations are developed, they will be integrated into the Environmental Management Plan and will supersede international guidelines as necessary.

Waste Identification

Facility designs and processes will be reviewed to identify waste streams. Discussion and review will include operations, maintenance, shipping/receiving, purchasing, and contractors. If available and practical, material safety data sheets (MSDS) and the workplace hazardous material information (WHMIS) system will be utilized to screen and classify waste streams. If necessary, waste streams will be tested to confirm proper classification and handling. Periodic review of operations and processes will be used to update and revise the waste management plan, again, including and involving the operations teams in the procedure.

Non-hazardous waste is any garbage, refuse, or sludge that is not ignitable, corrosive, reactive, toxic, acutely hazardous or listed as a hazardous waste.

Hazardous waste is a waste that is ignitable, corrosive, reactive, toxic, acutely hazardous or listed as a hazardous waste. Listed wastes include commercial chemical products such as arsenic pentoxide and sodium cyanide. An ignitable waste is a liquid and has a flash point less than 60°C; or burns so vigorously and persistently that it creates a hazard; or is an ignitable compressed gas. Corrosive wastes are aqueous and have a pH less than or equal to 2, or greater than or equal to 12.5. A reactive waste is a waste that reacts violently in water, forms explosive mixtures in water, generates toxic gases when mixed with water, or is capable of detonation or explosion at standard temperature and pressure. A toxic waste is a waste containing listed constituents such as arsenic, barium chromium, lead, mercury, etc., and exceeding concentration limits specific to the constituent.

Waste Minimization

The following procedures will be used to minimize wastes prior to initial operation start-up and as an ongoing program during operation.

- product review, selection, and substitution – whenever possible, non-hazardous materials will be used in lieu of hazardous materials



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- inventory control – inventories will be kept to a workable minimum to prevent expiration of dated products (shelf life) and generation of wastes
- procedure and process changes – alternative methods of processes will be developed to reduce generation of high volume wastes
- waste segregation – waste streams will be properly segregated and handled to minimize cross contamination of wastes
- good housekeeping and training – training programs and housekeeping standards will be developed, implemented, and tracked to reduce wastes.
- employee training and awareness – the waste minimization procedures will be a part of employee training and incentive programs.

Inspection Program

An inspection program will be developed to:

- inspect mine facilities for proper waste segregation, storage, and disposal
- inspect waste storage sites and document the volume of waste stored, type of waste, storage facility conditions, etc.
- inspect spill kits, protective equipment, reorder and replace as necessary
- perform periodic inspections of off site transporter including procedures, training, equipment, spill kits, records, and employee awareness
- perform periodic inspections of off site disposal and recycling sites including procedures, training, equipment, spill kits, records, and employee awareness
- review inspection findings with operation, transporters, and off site contractors to correct deficiencies, maintain awareness and communication, and to recognize negative or positive performance.

Spill Procedures Control Countermeasures Plan

As the facility design and layout are being finalized, a spill plan will be developed and include the following components:

- facility description, receiving area, above and below ground storage of fuels, lubricants, reagents, etc.
- spill prevention measures
- spill control, countermeasures, response, containment, cleanup and reporting
- plan implementation, improvements, inspections, records, training, and security.



Additionally the plan will include designs for an on site spill remediation area. The design will incorporate:

- volume of fuels and lubricants transported
- potential spill volumes
- frequency of spills
- remediation
- staged remediation area and volumes required
- climatic conditions and controls.

Recycling

Waste oil, lubricants, batteries, and antifreeze are typical waste streams amenable to recycling. Specification (not exceeding allowable levels of As, Cd, Cr, Pb, flash point and halogens) waste oil may be rehabilitated on site and reused in equipment, blended and used as a fuel, blended with ANFO and used in the blasting operations, or used as a motor or heating fuel. Similarly the used oil may be sent off site for similar recycling options. Lubricants and greases may be reclaimed and recycled for fuel materials. Anti-freeze may be recovered and reused on sit or sent off site for rehabilitation and reuse.

Off-site Disposal

Waste streams that cannot be recycled on site or off site will be disposed of on site or off-site. Non-hazardous materials and wastes may be disposed of in on site cells or the waste dump. Hazardous materials and waste streams may be treated by chemical, biological, or physical treatment to render the material non-hazardous. Hazardous materials may be disposed of in off site treatment facilities using the previously described treatment processes, or off site facilities that use other processes including incineration, land farming, or land filling.

9.6.6 Mitigation of Potential Biological Impacts

Corriente recognizes that there may be some loss of significant habitat and is developing mitigation measures, including:

- The development of a reserve zone (botanical garden) to preserve endangered flora and fauna species and the procurement of buffer areas to preserve ecological resources within the primary growth forest.



- A protected forest covering part of the concession rights, agreed with the Ministry of Environment to be a major part of a bi-national forest area.

9.7 Project Permitting

Ecuador's environmental legislation is extensive and their requirements for early stage operations (i.e., exploration are well defined.) Ecuador is one of the few Latin American countries that have adopted an EIA process for exploration activities. Argentina, Chile and Peru have adopted a similar process to conduct environmental assessments for early stage exploration.

Under Ecuadorian Mining Law and Mining Environmental Regulations, the Ministry of Energy and Mines handles the environmental approval system for new mining projects. Mining concession holders are required to complete environmental impact studies and environmental management plans to prevent, mitigate, rehabilitate, and compensate for environmental and social impacts as a result of their activities. These studies would be approved by the Ministry of Energy and Mines Sub secretary of the Environment.

The environmental approval process is summarized as follows:

- Proponent files a Project Description and Terms of Reference (ToRs) regarding how the EIA will be developed with respect to the conditions within the project area as well as the project description with the Ministry of Energy and Mines (MEM) and Ministry of the Environment (MoE). The ToR for Mirador has been reviewed and approved by MEM and MoE.
- These ToRs need to be approved by MEM and published.
- Environmental baseline studies and environmental impact assessment are completed by proponent in accordance with the ToR.
- The EIA is presented to the local affected communities and input to the EIA is requested. A most important section of the EIA is the Community Relations Plan (PRC) that includes all social mitigations planned by the company to be discussed and agreed with the communities. Corriente will have community meetings in Valle del Quimi, San Marcos, Tundayme, El Pangui, and at the Ministries of Energy and of the Environment. The EIA is updated to acknowledge community input.
- The EIA is submitted to MEM who reviews within a 45-day period after which the ministry will request Corriente to respond to any comments and questions regarding the EIA.
- Corriente will have a 30-day period to submit responses to all comments and questions.



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- The MEM will then take another 30-day period to revise the documentation and pronounce its satisfaction with all information, obtaining in this way the Approval for the EIA.
- Once the EIA is approved, proceedings towards granting of the Environmental License starts. It is estimated that another 30-day period is needed to prepare and grant the Environmental License.
- Submission of EIA to the Ministry of the Environment will take place at the same time as with the Ministry of Energy and Mines. Approval times are expected to be less than MEM.

Table 9-2: List of Major Permits required for the Project

Permit	Granting Institution	Requirements	Estimated Time for Approval
Environmental License	Ministry of Energy and Mines/ Ministry of Environment	Approval of EIA by both Ministries. Payment of license fees.	30 days after approval of the project and payment
Permit to Discharge	Ministry of Environment	Approval of EMP, payment of fees, compliance with EMP and regulations.	Valid for two years. To be obtained after one year of operations. Estimated time to obtain the permit: is 30 days.
Permit to Modify Water Courses	National Council for Hydrological Resources (Consejo Nacional de Recursos Hídricos)	-	-
Permit to Use and Transport Explosives	Joint Command of Logistics Management/Naval and Air Zone Command Squad (Dirección de Logística del Comando Conjunto/ Comandos de Brigada y de las Zonas Naval y Aérea)	Compliance with safety regulations	-
Health and Safety Permit	Ministry of Labour	Presentation of Company's Health and Safety Plan.	Estimated time to obtain the permit is 30 days.



10.0 SOCIO-ECONOMIC

10.1 Location and General Characteristic of the Area

The Mirador project is located within the provinces of Zamora Chinchipe and Morona Santiago. Three parishes are located within the mine operations area: the El Guisme and Tundayme parishes in the county of El Pangui and the Bomboiza parish in the county of Gualaquiza.

The Tailings Management Facility (TMF) is located in the county of El Pangui, in the parish of El Pangui. The site is specifically located to the north of the neighbourhood of Santa Cruz (Figure 2.2) and south of Paquintza (which are two neighbourhoods of less than ten houses).

The density of the population in the project area is low and poverty indices are high. There is a scarcity of infrastructure and public services.

The population in the region is composed primarily of immigrants from other parts of Ecuador. At the TMF site, the population is a mixture of colonists and Shuar who live in a similar manner to the colonists.

10.2 Social Analysis of the Tundayme, El Guisme, and Bomboiza Parishes

Poverty affects six out of ten people in these parishes. There are no pre-schools in the area and illiteracy is high. On average, people attend school for less than six years.

There is a lack of basic public and health services in the parishes.

The distribution of the native Shuar population in the area is as follows:

- 15% in Tundayme
- 20% in El Guisme
- 66% en Bomboiza

10.3 Initial Social and Population Analysis of the Mirador Project Area – TMF Site

The study area is located in the parish of El Pangui and specifically in the neighbourhoods of Santa Cruz and Paquintza. These neighborhoods are bordered to the west by the Chuchumbleta – El Panqui road.



The population of the area is a mixture of colonists and Shuar. Paqintza is the only organized Shuar community within the study area and is composed of eighteen families.

The Santa Cruz neighbourhood has 16 families. Each family is made up of an average of five people, and most have a small farm that they work on producing coffee, plantain, yucca, and rice.

Cattle raising provides the families with between 2 to 5 L/d of milk, with an average of seven to 15 cows and between three to five young bulls on each farm. Milk is sold at \$0.25/L. Cheese is also produced for self-consumption and for sale.

With respect to education, schools have one teacher working within buildings that are in generally good condition. In the area are the following schools: Miguel Avanchi school with 17 students, and the Paqintza school with 23 students.

It is important to indicate that Santa Cruz has a catholic church.

All of the communities communicate with each other via third order roads. The main Amazonian Highway, which passes through these communities, is used by all public as well as private vehicles for travel to Loja, Zamora, Cuenca, Macas, and the southeastern region.

The most important town nearest to the study area is El Pangui, which is the county seat and is located approximately 5 kilometres to the southwest of the TMF site.

In regards to the use of soil, approximately 80% of the land is dedicated to pastures and crops, 10% is secondary forest, 5% is the Los Hachales Reserve and 5% is for other uses. The land in the low-lying areas is swampy.

Shuar land lies mostly to the north of the study area and none of the mine infrastructure will be on the Shuar community land.

10.4 Population Characteristics in the Mirador Project Area –Mine Site

The population distribution in the communities or neighbourhoods present in the area is summarized in Table 10-1.



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Table 10-1: Population Distribution and Age Range in the Project Area

Community	Men	Women	Total Population
Valle del Quimi	48	32	80
San Marcos	29	30	59
Tundayme	72	73	145
Las Maravillas	18	16	34
Quimi	47	47	94
Chuchumbleza	66	62	128
Churuwia	16	15	31
Varias	15	14	29
Total	311	289	600

The age ranges of the population are:

Age	Population (%)
0 to 11	36
12 to 24	30
25 to 64	30
65 y +	4

Illiteracy rates for the population over the age of 18 in the project area communities are presented in Table 10-2.

Table 10.2: Rates of Illiteracy in the Project Area Communities

Community	(%)	Illiterates	Population Older than 18
Valle del Quimi	17.6	6	34
San Marcos	50.0	17	34
Tundayme	19.7	13	66
Las Maravillas	7.4	2	27
Quimi	12.5	5	40
Chuchumbleza	10.2	6	59
Churuwia	30.0	3	10
Varias	4.5	1	22
Total	18.1	53	292

10.5 Ethnicity

Of the population in the project area, 85% is of mixed race and 15% are native Shuar. The majority of the population in the area is originally from other provinces within



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Ecuador. Migration into the area has diminished in the past years. Of the present population, approximately 50% of the families have relatives that have moved from the Mirador area, of which some have moved out of the country.

In contrast with other Shuar federations, the Shuar community in the project area has integrated with the non-native population and have lost their cultural identity. In comparison with other Shuar communities in the provinces of Zamora Chinchipe, and Morona Santiago, the majority of the Shuars in the area do not speak their native language or live in a manner typical of Shuar communities. Only the Churuwias, a Shuar community located in the lower valley of the northern Quimi River area, have maintained their cultural identity. This group has broken ties with other Shuars of this province and Ecuador. The Shuars of this region do not belong to other Shuar federations nor are they part of the FICSH (Federación Interprovincial de Centros Shuaras) or the Shuar Federation of Zamora Chinchipe.

Table 10-2: Distribution of the Native Shuar Population in the Project Area

Community	(%)	Native Population	Total Population
Valle del Quimi	-	0	80
San Marcos	-	0	59
Tundayme	20.7	30	145
Las Maravillas	-	0	34
Quimi	30.9	29	94
Chuchumbleza	-	0	128
Churuwia	100.0	31	31
Varias	-	0	29
Total	15.0	90	600

It should be mentioned that the Paquintza Shuar community is present at the TMF site. Although these Shuars do not belong to the Shuar Federation, they do have an organization in Zamora.

10.6 Employment and Productivity

In general, there is a lack of employment and personal income is low. Productivity from farming is marginal and used primarily for local consumption. Approximately 40% of the households earn less than US\$100 per month, while the remainder of the population earns between US\$100 and US\$300 per month.

10.7 Existing Infrastructure

The existing infrastructure in the area is minimal and is as follows:



Mine and Conveyor Belt Corridor Area:

- roads range from non-existent or very rough to good secondary access; most travel in the area is on paths
- there are no potable water treatment plants nor sewage treatment facilities in the area; raw sewage is discharged directly to the rivers in the area
- houses located far from the populated areas lack electricity
- one telephone booth was recently installed in Tundayme
- sixty-nine percent of the population own their homes while 12% rent; approximately 90% of the houses in the area are made of wood.

TMF Site:

- a net of secondary roads run through the area
- the Main Amazonian road is located directly to the west
- the nearby town of El Pangui offers basic services such as food, lodging, and communications
- an estimated number of four houses are located directly within the TMF site.

In the vicinity of the TMF are located the schools of Santa Cruz and Paquintza, various housing for Santa Cruz and the Shuar centre of Paquintza. A road from the main Amazonian highway exists up to Santa Cruz, and third order roads are used for communication between these two communities.

10.8 Population Needs

Responses to interviews with community members clearly indicate, from both high percentages of the responses and constant repetition, that the lack of employment is the most pressing problem in the communities.

It has been observed that the levels of employment are low since there are no sources of employment. For this reason, most families depend on the income generated by temporary jobs, such as day labour, and the small surpluses from their production of agriculture, cattle, or other animals.

10.9 Opinion of the Population Regarding the Mirador Project

There are 12 organizations with influence in the county of El Pangui. Seven of these are social organizations, such as cooperatives and associations. One is a religious



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organization represented by the local catholic mission and the other four represent different government agencies. The most important government authority is the Municipality of El Pangui. The Shuar Association and the Association of Coffee Growers maintain a position against the proposed Mirador Project due to a sentiment of lack of trust towards the technical development of the project, environmental protection, and employment generation.

To obtain a social base of support for the Mirador Project amongst the inhabitants of the Communities Directly Related to the Project (CDRPs) an "Immediate Intervention Plan in the Communities of the Mirador Project" was designed and implemented by Corriente. Its strategies included the dissemination of information and the education of the communities with respect to the activities of Corriente, the Mirador Project and its operation perspectives.

The process included visits to each of the homes during which information was conveyed and a favourable reflection on industrial mining activity, in particular, the expectations of the Mirador Project, was generated. The process undertaken was a positive precedent in Corriente's community relations program. Corriente demonstrated a willingness to establish dialogue and to assume serious commitment and participation with community development.

10.10 Archaeological Component

A total of 788 lamp (or shovel) tests were performed in the Mirador Mine Site Area within the zones of direct influence of the project. Of these, only 87 contained cultural material such as ceramic fragments. Lyrthic fragments were also registered, but actual usage by historic settlers in the area will only be confirmed after analysis. Carbon dating will be carried out to assess the ages of the various cultural material.

Of note, one complete unbroken ceramic pot and two fragmented pots were found within the Mirador area.

Preliminary analysis indicates a medium sensitivity archaeologically for the mine area, in comparison with the mountain top areas of Mirador, which indicate a high sensitivity due to higher level of evidence of historical occupation. Still, some sensitive areas that will be affected have been identified within the operations areas.

The archaeological assessment for the TMF site is underway and results will be included within the final environmental impact assessment.



10.11 Preliminary Results of the Environmental and Social Impact Assessment

Corriente has established an active community consultation program in the region. It has held a number of public meetings to explain the project and to obtain input from the local communities.

Corriente has also established a pilot nursery program that contains many flora species of the area as well as an orchid nursery. This project is in its initial stages but can serve, during the construction and operation of the project, as a botanical reserve to be used during the rehabilitation, regeneration and reforestation programs, as well as, a source for environmental training and an example of sustainable management of resources for the local population.

10.12 Potential Social Impacts

During construction there will be a need for approximately 730 construction personnel at the peak period. Construction is expected to take 21 months. During operations there will be a need for between 150 to 200 persons. There will be a requirement for heavy equipment operators, mechanics, process operators, administrators, and others. Some of these will come from the local communities while other more skilled workers will have to come from other areas. The requirement for workers in both the construction and operations phases will strain the infrastructure in the local communities.

Construction, while providing opportunities for local unskilled labour, is temporary. The area may experience an influx of persons from other parts of Ecuador who are looking for employment opportunities. This may create tension between the local area residents and the newcomers. Expectations in the local communities are high and include improvements to the local road network to the provision of sanitary and potable water treatment systems.

10.13 Mitigation Measures

The mitigation measures proposed for the development of the Mirador project are based on the following social policies to be applied respectively and specifically within the direct influence area once all preliminary social analysis are terminated and main socio development programs defined:

- respect culture, traditions and values of individuals and groups of individuals related to the project



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- sustainable development in the social, environmental, economic, cultural, and institutional components of the communities within the area of influence of the project
- synergy with local plans, such that Corriente will develop the company plans taking into account local plans and objectives
- community territory acknowledgement by Corriente within their range of action and in accordance with national and local laws
- participation and communication, Corriente recognizes communities as groups of interest and is committed to undertake participation and communication programs within established dialogue, mediation and conflict resolution mechanisms
- transparency, as Corriente will share its information with communities in the area
- health and safety, it is a commitment of Corriente to take all necessary actions to protect the health and safety of its personnel, contractors and people under the area of influence of the project.



11.0 CLOSURE PLAN

11.1 Introduction

11.1.1 Objectives

This section reviews the Preliminary Mine Closure and Reclamation Plan for the Mirador Project, the complete plan is included in Appendix H. The purposes of this plan are to lay out the closure measures and estimated closure costs as input to the project feasibility study, and also to form part of the Environmental Impact Assessment to indicate the post-mining reclamation activities that will mitigate potential environmental impacts. In general, site reclamation objectives will include the following:

- protection of air, surface water, ground water, flora, and fauna
- protection of public and worker health and safety
- restoration of wildlife habitat
- design and restoration of a post-mining topography that is comparable with pre-mining conditions
- restoration of the area in an aesthetically acceptable manner that generally blends within the surrounding habitat
- establishment of post closure vegetation that is suitable and comparable to pre-mining conditions.

The Mirador Project will be permitted under Ecuadorian permitting procedures. This closure plan will form part of the Environmental Impact Assessment documents being prepared in support of the permit application process.

The Mirador Project is being developed to conform to Ecuadorian environmental standards as well as to the guidelines laid out in the Equator Principles. The Equator Principles are a framework for financial institutions to manage environmental and social issues in project financing. The Equator Principles have been endorsed by most of the significant international banks that are involved in mine financing, and in general reference the standards of the World Bank Group.

11.1.2 World Bank Group Guidelines

From the World Bank Group Pollution Prevention and Abatement Handbook, Base Metal and Iron Ore Mining, following are the guidelines for mine reclamation and closure plans:



“The Mine Closure and Reclamation Plan should cover reclamation of tailings deposits, waste rock deposits, any open pit areas, sedimentation basins, and abandoned mine, mill and camp sites.

Mine reclamation plans should incorporate the following:

- *return of the land to conditions supporting prior land use, equivalent land uses or other acceptable uses*
- *elimination of significant adverse effects on adjacent water resources*
- *use of waste rock for backfill and of topsoil (or other acceptable materials) for reclamation to the extent feasible*
- *contouring of slopes to minimise erosion and runoff*
- *planting of native species and of other species that are environmentally acceptable, to prevent erosion and to encourage self-sustaining development of a productive ecosystem on the reclaimed land*
- *post-closure management of AMD and tailings; reduction of AMD formation by sealing off pyrite-containing waste from oxidation and percolating water*
- *budget and schedule for pre- and post-closure reclamation activities*
- *sealing and securing of all shaft openings and mine adits on closure of the mine.*

Money should be reserved over the life of the mine to cover the costs associated with mine closure. The amount of money and the type of financing required will depend on a number of factors such as the projected life of the mine, the nature of the operations, the complexity of environmental issues, the financial and environmental management capacity of the borrower or project sponsor, and the jurisdiction in which the mine is located. The mine reclamation and closure plan, the timing of its submission, and financing of activities under the plan, should be discussed and agreed on with the borrower or sponsor as early as possible.”

Corriente intends to follow applicable and practical World Bank closure guidelines. Any exceptions or deviations will be justified and documented in the detailed closure plan and/or during final site reclamation and closure.

11.1.3 Ecuador Guidelines

The Ecuadorian environmental regulations for mining activities state that all mining title holders must comply with their Environmental Management Plan and must present a financial guarantee to the Subsecretary of Environmental Protection of the Ministry of Energy and Mines which will hold the guarantee until the complete closure of the mine and for one year after.



Under Article 67 of the environmental regulations, mine closure plans must be presented for any phase of the operation and must include a description of the infrastructure that will be dismantled including all equipment, camps and others in accordance with the company's Environmental Management Plan and Environmental Audit. Every year a mining titleholder in Ecuador must complete an Environmental Audit until their mining title expires.

The regulation states that the disturbed areas will be rehabilitated in accordance with the environmental studies that were presented to the Ministry. Rehabilitation plans should take into account future uses for cultural and recreational activities. The mining titleholder is also responsible for damages to the natural environment and contamination to areas that occur after closure of the operations.

11.1.4 Corriente Commitment

Corriente is committed to reducing residual environmental effects at the site upon closure. Reclamation work will form an integral part of the mine plan and will be carried out progressively, as practical, during the life of the project. The mining and water management plans have been developed to facilitate progressive reclamation. All surface facilities have been designed to minimize reclamation requirements following mine closure and to enhance the natural recovery of the areas affected by mining. It is anticipated that a significant portion of the proposed reclamation will be carried out during the operational phase of the project. Concurrent reclamation will be used to minimize the disturbed areas during the mining stages of the operation. Corriente has incorporated, where applicable, in this Preliminary Mine Closure and Reclamation Plan the guidelines set out in the World Bank Pollution Prevention and Abatement Handbook. Accordingly, Corriente's plan will comply with the conditions of mining permits, regulations, and industry standards. The following principles have been established to guide the development of the overall plan:

- plan and implement procedures in accordance with all applicable regulations
- apply cost-effective and appropriate closure and reclamation practices to reduce environmental risks and allow traditional use of the land
- conduct studies to predict post-closure environmental effects
- maintain a program of progressive closure and reclamation as an integral part of project operations
- incorporate new reclamation and mine waste management methods and procedures.



11.1.5 Status of Closure Plan

This plan was prepared prior to Corriente receiving Ecuadorian environmental approvals for construction and operation of the project. Therefore, it is considered preliminary since some elements of the plan, such as post-closure monitoring requirements and associated cost estimates, may require revision upon project approval. Further, the specific details of the Mine Closure and Reclamation Plan will evolve as mining progresses, and so the plan will be updated periodically during the mine life. The final plan will be generated several years before mine closure. The key closure and reclamation issues for the project are as follows:

- Acid rock drainage (ARD) will be minimized by placing a cap of compacted overburden materials over the mine waste rock dump, revegetating the dump surface to aid in controlling erosion, and by maintaining runoff diversions around the waste dumps. The successful implementation of this strategy will minimize the volume of water that will require ongoing collection and treatment. Collection and treatment of acidic drainage from the waste dumps will continue for as many years as required, until the levels of acidity and metals abate to such that they will be acceptable for release or that can be adequately treated by passive systems.
- The tailings impoundment will be drained, covered with native soils and revegetated. Diversion ditches will be constructed around the perimeter. All roads not required or not useable by the community will be decommissioned and revegetated. Drainage collection piping system and sumps will remain in operation where required and a permanent spillway will be constructed to safely discharge run-off.

11.2 Progressive and Final Reclamation Measures

11.2.1 Concurrent Reclamation

Construction Phase

All borrow pits, quarries, equipment and storage areas utilized during construction, but not required during mine operations, will be closed out and reclaimed at the end of the construction phase of the project. All cuts and fill created during construction, including roads, plantsite, tailings line, diversion ditches, and sedimentation pond dikes will be vegetated to assist in erosion protection.

Prior to disturbing areas, all practically available and suitable soil will be salvaged and stockpiled for redistribution during ongoing reclamation and final closure of the mine.



Operations Phase

During operations, areas will be regraded, revegetated, and stabilized, as soon as possible.

Waste dumps will have a cap of impervious material applied as soon as final surfaces are created, at which time they will also be vegetated. The waste dump will be constructed to its final slope angles where possible.

Soil salvage will be incremental and concurrent with each phase of waste dump construction, pit development and waste rock dump development. Soils will be tested for chemical and physical suitability to ensure suitability for reclamation. Soil stockpiles will be located as close as feasible to areas of reapplication. Stockpiles will be constructed with stable side slopes and access ramps. Stockpiles will be stabilized and seeded as soon as practical to prevent erosion and reduce leaching of soil nutrients. An annual inventory will be maintained of disturbed areas, volumes of topsoil accumulated, and volumes distributed.

A revegetation plan will be developed to determine species, planting schemes, seeding, and planting rates, seedbed preparation, seeding and planting methodology, and treatments as necessary. The species selection and revegetation rates will be based on pre-mining vegetation types.

The tailings cover will be constructed in stages, with the initial cover constructed on the sandy beaches around the perimeter of the impoundment. This will minimize the potential for dust generation from the tailings surface. Subsequent stages will extend the cover further over the fine tailings in the centre of the facility. Tailings dam borrow pits will be resloped as necessary, covered with stockpiled topsoil, and vegetated as soon as they are depleted or no longer necessary.

Closure and Reclamation Phase

Final reclamation measures will be carried out as soon as final surfaces are created. The final tailings dam slopes will be ready for reclamation after the last dam raise is completed about two years before the end of mine life.

Major closure activities after the end of mine life will include removal of all facilities and infrastructure that is not planned to be left for other uses or is needed for closure maintenance. Final waste dump capping will be applied. The tailings supernatant pond will be drained, covered, and a permanent tailings dam spillway will be constructed.



Post operational reclamation will return the disturbed areas to the pre-mine conditions and habitat. Certain mine features may be left in place as permanent control measures to prevent environmental pollution, for long-term community use, or as a post-mining enhancement. Examples of facilities that may be left in place over the long term include:

- the administration buildings may be turned over to the local communities for their use and benefit
- water treatment facilities
- waste dump diversions.

11.2.2 Final Reclamation Measures

Open Pit Mine

The open pit mine will be allowed to fill with water when mining operations cease. The mine will have an ultimate depth of over 560 m on the high south wall, to about elevation 1,040 masl. The highest point exposed by the mine excavation will be about 1,600 masl. The lowest point of the pit wall will be on the north side of the pit, where the main haulage ramp will exit at about elevation 1,300 masl. Thus after the pit is flooded there will be a height of about 400 m of exposed wall. Some of the exposed wall materials could have the potential for acid generation.

At this stage of mine planning, it is not possible to develop precise models of the anticipated pit lake water quality that may result from oxidation or metal leaching of the exposed wall rock. A pit lake study will be completed at Year 5 of the operation, when the ultimate mine life and pit configuration are better known and after more extensive geochemical characterization of the pit wall rocks has been carried out. The objectives of the pit lake study would be to determine:

- groundwater flows
- time for the pit to fill
- depth of pit lake
- pit wall stability
- anticipated water quality of the pit lake
- any potential environmental impacts
- develop mitigation measures as necessary.



Potential scenarios may include:

- there may be sufficient flow into and through the pit that there will be adequate dilution so there is no significant impact from the wall rocks and pit lake
- a one-time treatment of the pit water may be required to adjust the pH before initial discharge begins, after which the models may indicate that long-term water quality will be adequate for discharge
- diversions may be left in place and maintained to reduce pit inflow and if desirable to reduce the rate of filling, and extend the time period before the pit would discharge
- should it be necessary, open pit discharges would be treated along with waste dump drainage, either by active or passive means.

Mine Waste Dump

The mine waste dump will be reclaimed as follows:

- Final dump slopes will be constructed to an angle of 2.5 horizontal to 1 vertical, both for overall stability and to provide stable slopes for establishment of vegetation, see AMEC report, Mirador Waste Dump Design Report, November 2005, included in Appendix B.
- A cap of compacted overburden or composite soil/geomembrane liner will be placed over the final dump surface with the objective of shedding water to reduce infiltration.
- The dump surface will be planted with appropriate local varieties of plants and shrubs to provide erosion protection.
- The closure runoff system will be upgraded to separate clean, uncontaminated runoff from the dump surface from any seepage flowing from within the dump. Seepage flows will be isolated so they can be captured and monitored. If suitable for discharge, the seepage flows will be released, and if necessary treated prior to discharge.
- Major diversions upslope of the waste dump will be maintained over the long term, to avoid erosion of the dump by stream flow. Where possible, permanent, durable diversions will be put in place, but as necessary regular monitoring and maintenance will be continued.

The need for potential ongoing treatment of dump drainage waters following mine closure, and the potential options that will be assessed, are discussed above. If any discharges from the closed pit or waste dumps are below the regulatory pH and



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passive treatment systems are not adequate, active chemical treatment may be used to mitigate impacts. Typical bases or alkalis used to treat acidic water are:

- limestone (CaCO_3)
- quicklime (CaO)
- hydrated lime (Ca(OH)_2)
- soda ash (Na_2CO_3)
- caustic soda (NaOH)
- sodium bicarbonate (NaHCO_3)
- ammonia (NH_3).

Depending on the water chemistry, flocculants and coagulants may be used to reduce the metals concentration prior to neutralization. Aeration may be also used as a pre-treatment to reduce iron and manganese.

The alkali treatment requirements will vary over the life of the mine dependent upon concentration of heavy metals (Al, Mn, Zn, Cu, Fe). The concentration of metals in the pit wall and waste rock will vary over the life of the mine requiring different dosage levels of alkalis to raise the pH and precipitate the heavy metals prior to discharge to the receiving waters.

The application processes may be relatively simple, flow through hoppers dissolving soda ash briquettes; drip or trickle systems discharging NaOH based on manual or automatic flow measuring systems; semi or fully automated water treatment plants.

Infrastructure

After mining and ore processing are complete, the crusher, concentrator, conveyors, pipelines, and support facilities will be dismantled. Inert materials such as steel, iron, concrete, plastic and wood will be disposed of, buried in on site disposal areas, or sold to scrap dealers for recycling.

Beneficial roads will be left in place to support community, military, and public access. Non-beneficial roads and paving will be ripped and covered with an adequate depth of fill and topsoil to support revegetation.

Power lines may be left in place if there is a beneficial use. Otherwise, towers and lines will be removed and disposed of on site or sold as scrap.



Any hazardous materials or chemicals will be treated to render non-hazardous, or transported off site to an authorized treatment and disposal facility.

Surface pipelines will be dismantled and either sold as scrap or buried on site. Buried pipelines will be flushed with clean water and left in place, or removed and sold as scrap, or disposed of in an on site disposal area.

As the facilities are removed, the areas will be ripped as necessary, covered with an adequate depth of overburden, topsoiled, and revegetated.

Tailings Impoundment

Upon completion of the mining operations the TMF will be designed for closure such that the tailings surface is rapidly made trafficable, and the potential for wind and water erosion is minimal. Closure and reclamation activities will be carried out with the objective of providing for long-term stability and an appropriate end use that requires minimal maintenance.

During the final years of operation, the extent of the supernatant pond will be minimized and will be removed during the first year following closure. Water will be removed from the surface pond, treated to an acceptable quality if required, and released to surface waters. The other main components of the closure and reclaim area are as follows:

- revegetation of the downstream embankment face
- diversion ditches will be constructed around the TMF perimeter
- construction of a closure spillway
- all roads not required for servicing of the TMF will be decommissioned and revegetated
- drainage collection pipework and sumps will remain in operation where required.

As required, ongoing maintenance of the above systems will continue.

The tailings will be covered with native soils excavated from local borrow sources. The tailings cover will be constructed in stages, with the initial cover constructed on the sandy beaches around the perimeter of the impoundment. This will minimize the potential for dust generation from the tailings surface. Subsequent stages will extend the cover further over the fine tailings in the centre of the facility. The tailings cover will be approximately 500 mm thick and re-vegetated directly after placement.



Administration Area

The administration area will be dismantled except for facilities that may be required in the post closure period, or those that may be left as legacies for the local community.

If there were a need to operate a water treatment plant beyond the end of mine life, certain facilities would be left in place to support its operation. These may include:

- water treatment plant
- power lines
- security gates and fencing.

Any such operating facilities that remained after operation would remain under the control of the operating company with full safety, security, and environmental protection measures in place.

Facilities that may be left as community legacies may include the office building or any other structure.

11.3 Environmental Impact Assessment

11.3.1 Open Pit Mine

The open pit mine will be allowed to flood upon closure. This flooding will take about 5 to 10 years, depending on actual groundwater inflow rates. The flooding will maintain a cover over wall and floor rocks up to the level of the pit outlet, at about 1,300 m. There will be potential for acid drainage from exposed wall rocks above the outlet elevation. The actual impact on water quality, and potential mitigating measures, will be assessed in a pit lake study to be performed in Year 5 of operation. It is expected that, with the significant streamflows in the region, the rate of oxidation of pit wall rocks will be such that water quality in the pit water discharge could be mitigated by dilution. However, in the worst case the pit lake discharge could necessitate collection and treatment. Even in this worst case, there would be no downstream impact to the receiving waters of the Quimi River as the discharge would be collected and treated if necessary.

The open pit will permanently alter the local topography and will remain visible from a distance after closure, so there will be a permanent visible impact. This impact will be lessened by vegetation growth over the longer term. The open pit will also result in loss of about 117 ha of natural ecosystem.



11.3.2 Waste Dumps

The waste dumps are likely to become acid generating with discharge of acidic waters, containing dissolved metals, from the dump. The dump discharges will be monitored and collected and treated as necessary. Treated drainage waters, treated by either an active water treatment plant, passive biological treatment system or other means of treatment, would be released to the receiving waters only after meeting agreed parameters for discharge. Thus, the acidic drainage will have no impact on the downstream waters provided the water quality monitoring systems and any required treatment systems are implemented and maintained for as long as necessary. Corriente will develop a financial mechanism for accruing funds over the life of the mine such that the ongoing treatment will be adequately funded.

The waste dump will have some visual impact. However, the location of the waste dump is on the lower slopes of the valley side in the mid-reaches of the Wawayme River valley and will only be visible from within the essentially unpopulated and remote Wawayme Valley. The waste dump is expected to support vegetation and to be covered in vegetation within a relatively short time period (5 to 10 years). The vegetation will effectively mask the waste dump visibility.

11.3.3 Plant Site and Infrastructure

The plant site and all other mine infrastructure will be essentially returned to pre-mining conditions and revegetated. This there will be little long-term impact of the plant site and infrastructure.

11.3.4 Tailings Impoundment

The tailings impoundment is expected to have no negative impact on water quality, as the tailings will be covered and drained. The tailings dam will remain visible after closure, as the downstream face of the dam will be about 40 m high. The dam will vegetate naturally over a period of years so that the visibility will remain for years after closure. Berms and vegetative schemes may be developed to mitigate aesthetic impacts during operation, reclamation, and closure.



11.3.5 Biophysical Environment

Air Quality

There are not expected to be any significant air quality impacts of the mine upon closure. The tailings, which can be a dust concern in dry climates, will be covered with vegetation.

Flora and Fauna

The main impact of the Mirador Project will be the loss of forest habitat. The habitat loss will be the areas of the tailings impoundment (385 ha) and the open pit and waste dump (188 ha). This lost habitat has the typical diversity of the rainforest of this region.

Water Quality and Aquatic Resources

The largest potential impact of the Mirador Project is on water quality. However, as stated above, if adequate measures are maintained to handle potential acid rock drainage, there should be no significant water quality impact.

11.3.6 Socio-Economic Impacts

Corriente will work with the communities in the area to enable all stakeholders to have their interests considered during the mine closure process. Typically, a closure of a mine will impact the economic condition of the closest community(ies). As part of their active Community Relations Plan, Corriente intends to work with the stakeholders to establish a targeted communication strategy that will reflect the needs of the stakeholders and interested parties. A consultative process involving local communities, regulators and other interested stakeholders will be established during the operation to determine the long-term use for the mine and all associated lands.

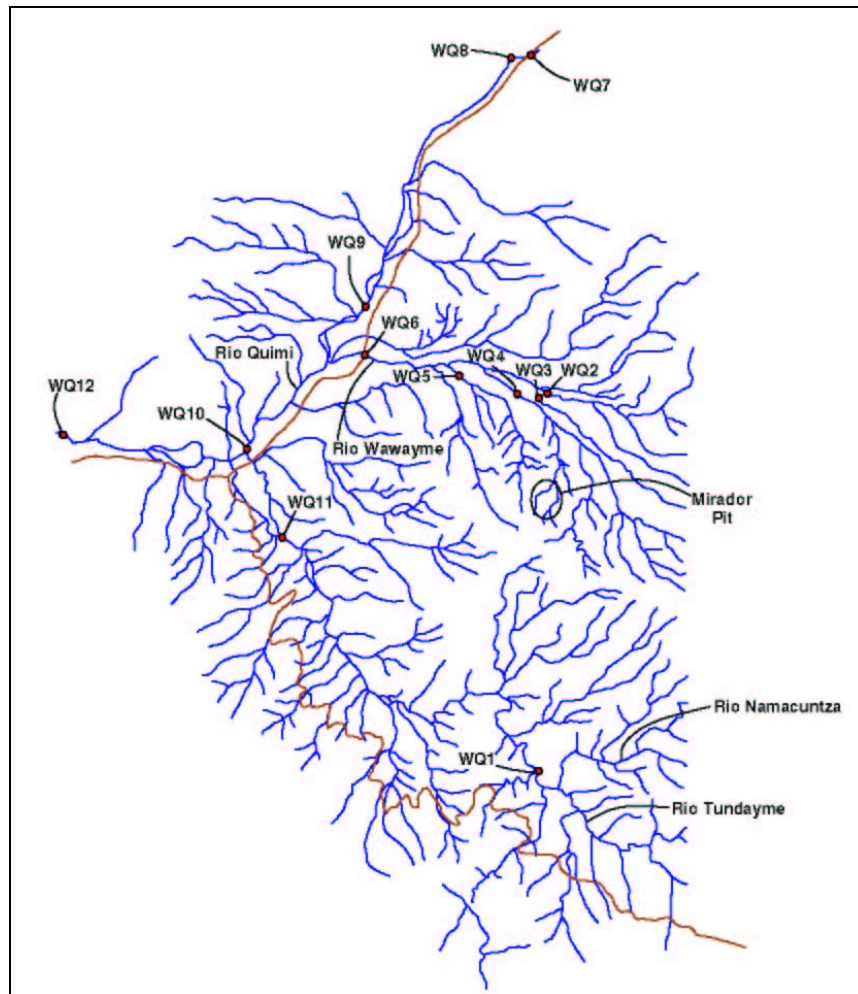
11.4 Post Closure Monitoring and Maintenance

11.4.1 Environmental Management and Environmental Effects Monitoring

A comprehensive water quality monitoring program has been developed as part of the environmental baseline study for the project. Water quality monitoring stations are being sampled at monthly intervals to provide an accurate picture of pre-mining water quality, see Figure 11-1.



Figure 11-1: Water Quality Sampling Locations



In addition to the water quality monitoring system, a climate monitoring station and a number of separate rain gauges at different elevations have been installed and are being used in the preproduction period to gather site specific data on precipitation, temperature, wind and evaporation. There is also a network of streamflow monitoring stations to develop a site-specific understanding of the correlation between rainfall and runoff. This data will be used to support detailed engineering of the project, and to provide baseline streamflow data to assess project impacts.

11.4.2 Environmental Monitoring during Operating Period

The water quality and streamflow monitoring programs established for pre-production will remain in place during the operating period. These will allow any impacts on water



quality and flows to be detected at an early stage and any mitigation measures put in place. In addition to the monitoring already in place, additional monitoring will include:

- seepage observation and measurement at the toe of the tailings dam
- regular inspection of diversions around the mine, waste dump and tailings area to assess any potential problems so that preventive maintenance can be performed
- tailings dam stability monitoring, including survey of fixed monuments to detect settlement or any horizontal movements
- water quality monitoring will be expanded as necessary to include sampling of open pit and waste dump drainages and tailings pond water quality.

Quarterly environmental reports will be prepared presenting all data on a cumulative basis. Annual reports will be prepared highlighting any significant changes or problems encountered in each period, and reporting on mitigative measures taken to deal with any problems.

11.4.3 Environmental Monitoring during Closure and Reclamation Period

Post-closure monitoring will be a continuation of the operational monitoring systems. All monitoring systems will be maintained for at least five years beyond the date of mine closure. At that time, the need for ongoing monitoring of each component of the mine will be assessed. Where there are ongoing water quality or other concerns, the systems will be maintained, and where the environmental systems are stable and have no negative impacts, monitoring will be discontinued or the frequency of monitoring will be reduced. The need for ongoing monitoring would then be reviewed at annual increments to assess further need.

In addition to the environmental monitoring of water flows and quality, the following regular inspection work will be continued in the long term:

- Waste dump diversion ditches that are designated as permanent will be inspected on at least a monthly basis.
- The progress of plant growth in replanted areas will be inspected annually by a qualified agronomist responsible for the site reclamation work, until such time the plantings are judged to be stable and self-sustaining.
- The waste dump cover will be inspected on at least a quarterly basis for the first three years post closure, and on an adjusted lesser frequency as appropriate in later years.



11.4.4 Post-Closure Maintenance

Post-closure maintenance will consist of:

- timely repairs to diversion ditches when any slope stability or erosion are noted to be threatening ditch integrity
- any remedial works noted to be required to the tailings management facility
- repairs of any eroded reclaimed areas and any additional works to maintain growth in reclaimed areas
- repairs to any erosion noted on the waste dump cover
- operation of the water treatment plant and/or passive water treatment systems.

11.5 Implementation Schedule and Cost Estimates

11.5.1 Implementation Schedule

Final closure plans for Mirador will be implemented as soon as the mine reaches the end of its economically viable life. The current mine plan calls for an operating period of about 12 years, after which final closure works would be started. Once closure work is initiated, it will be to Corriente's economic benefit to complete the work as quickly as possible. It is expected that the work will take about one year following the date of shutdown of operations to complete the closure works. Long-term post-closure programs (monitoring, maintenance and water treatment if necessary) will begin immediately upon mine shutdown.

11.5.2 Closure Cost Estimates

The cost of final closure and reclamation for the Mirador project has not been estimated in detail, as there are a number of significant unknowns with regard to the closure needs. In particular, the biggest unknown is the requirement for ongoing collection and treatment of acid drainage from the waste rock dump. Table 11-1 presents some indicative costs based on estimates in areas that could be quantified. The costs include an allowance for operation of a water treatment plant if the planned mitigation measures such as passive wetlands and other such techniques require additional support. A salvage value of \$5 million has been assumed for the plant and ancillary facilities. It is assumed that these will be sold for scrap or re-use, with removal costs paid by the purchaser. .



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Table 11-1: Mirador Project Indicative Closure Costs

Area	Estimated Cost (US\$)
<i>Direct Closure Costs</i>	
Tailings facilities	1,500,000
Waste dumps, roads and plant sites	500,000
Subtotal - Direct Costs	2,000,000
<i>Indirect Closure Costs</i>	
EPCM	100,000
Site facilities operation	500,000
Subtotal - indirect costs	600,000
Contingency @ 15% of direct and indirect costs	400,000
Total Closure Costs	3,000,000
<i>Allowance for Additional Closure Costs if Water Treatment Plant is Installed</i>	
Treatment plant incl. monitoring and maintenance	12,000,000
Total Closure Costs with Treatment Plant Allowance	15,000,000

11.6 Financial Security and Assurance

Corriente is committed to providing suitable financial security and assurance to cover the full cost of closure and reclamation of the Mirador project. It is Corriente's intent to enter into discussions with the responsible Ecuadorian authorities to reach agreement on an appropriate form and amount of security to be posted for the project.

At this time Corriente has not developed or indicated any preference towards a specific format for the posting of security against reclamation liability. Corriente remains open to consideration of a wide range of options, including but not limited to the creation of a reclamation trust, cash, letter of credit, insurance bond or a combination of these mechanisms and others that may arise as a result of future discussions with the authorities.

Corriente is committed to a program of progressive reclamation at Mirador. Progressive reclamation has been built in as an integral part of the mine plan. Consequently, Corriente intends to manage its reclamation liability at Corriente by initiating reclamation work at an early point in the mine life, thereby limiting the expansion of overall liability over time.

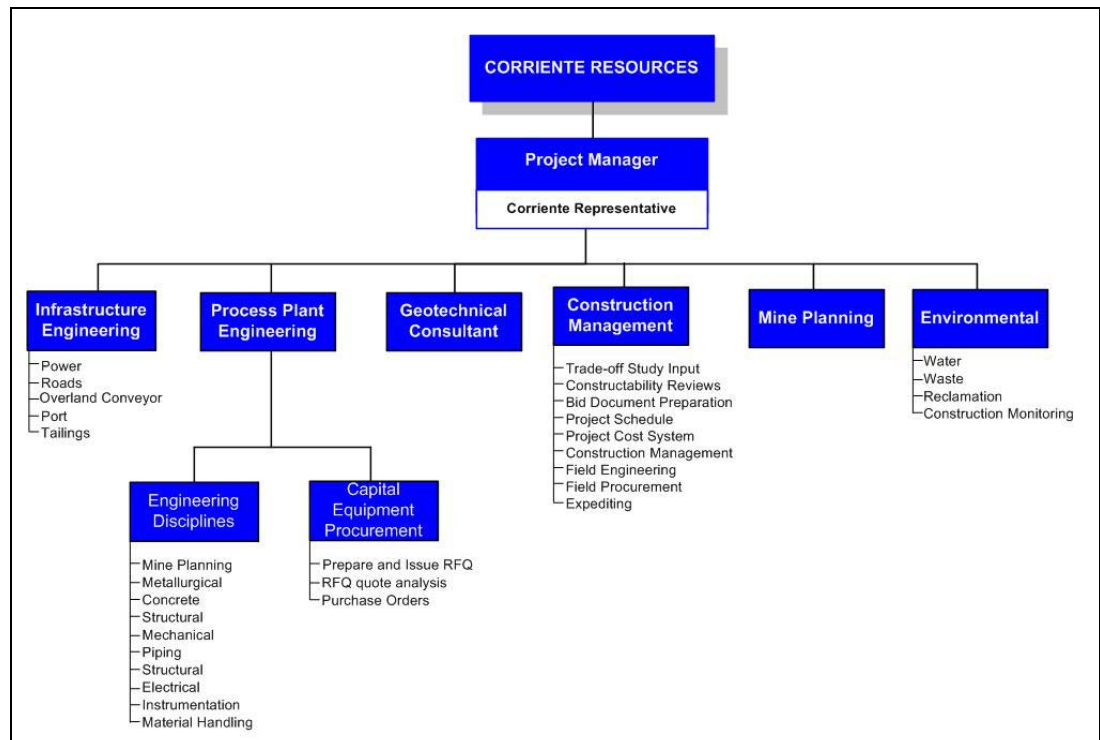


12.0 PROJECT IMPLEMENTATION

12.1 INTRODUCTION

A project group will be established by Corriente to undertake the execution of the Mirador Project. It will be comprised of the Corriente team working with outside specialist consultants to undertake the engineering, procurement and construction management, as well as much of the pre-production work. The proposed organization chart is shown as Figure 12-1.

Figure 12-1: Project Organization Chart



This section generally outlines how the project group will:

- undertake the engineering, procurement and construction programs
- develop the project schedule to include project milestones, critical dates and long deliveries
- establish a cost monitoring and control system for the project
- contract and administer the construction work
- make use of local resources.





The overall project capital program will take about 21 months from the scheduled project approval date to engineer and construct the facilities to mechanical completion when pre-operations can start.

The Mirador Project will use competitive tendering and negotiation to obtain the best quality material, equipment, and contractors for a fair market value. Emphasis will be placed on using local Ecuadorian labour, materials, and equipment wherever reasonable, provided it does not affect cost, schedule, or quality.

12.2 Objectives and Priorities

Key features for executing the Mirador Project are to:

- further optimize project costs following feasibility
- secure long delivery equipment orders before financing approval using a conditional purchase order with a cancellation clause
- establish a comprehensive project Implementation Plan and Procedures Manual at the onset of detailed design and construction
- start the detailed engineering, procurement and construction program immediately following the approval to proceed
- issue project documents in metric units; size construction materials in metric units; use the English language during the engineering and procurement stage with Spanish translations as needed
- establish a practical and effective project cost control system
- schedule start-up for month 21 (from development decision)
- incorporate a two-month ramp up during start up to reach maximum production levels
- comply with the laws and regulations of the Government of Ecuador
- maintain a high standard of safety so as to minimise incidents and accidents
- observe the requirements of Corriente's Environmental Protection Plan
- minimize disruption to the local communities.

12.3 Engineering Plan

After acceptance of the Feasibility Study and the associated trade-offs, and the issuing of permits, Corriente will select the engineering and procurement teams to undertake the design of the project facilities.





12.3.1 Project Organization

The engineering consultants will be responsible for the design work for their assignments from the start of the engineering program through to the end of construction.

Reporting to the project manager, their responsibilities will include:

- selecting their teams
- defining the project needs
- scheduling the project engineering and procurement work based on the overall project schedule needs
- establishing what design effort may be needed in the field to support the construction program
- monitoring and tracking project engineering and procurement progress for the capital equipment included within their scope
- solving any design problems that interfere with construction progress
- monitoring and controlling the engineering and capital equipment costs with Corriente
- ensuring that the engineering effort provides a practical, operation friendly and constructible facility using standard engineering practices and design methods used in North America and Ecuador
- communicating and coordinating with Corriente, construction management team, vendors and other associated parties as required to carry out the work in an efficient manner
- providing the engineering deliverables in a timely manner; drawing lists, equipment and electrical lists, specifications, drawings, etc.

12.3.2 Detailed Engineering

It is anticipated that the main engineering elements will be:

- mine planning
- site development and infrastructure
- material handling (including overland conveyor)
- process facilities design
- water, waste, and tailings design
- port facility design.





Combined with the details established during feasibility, there will be sufficient engineering and design completed in the first two months of the project execution, following financial commitments, to allow the bidding and evaluation of key construction contracts and placement of purchase orders for materials and equipment required for:

- grinding mills
- major electrical components
- crusher
- the commencement of construction on site during the first three months of the project.

No awards will be made for any equipment, material purchasing, or on-site work prior to the financial commitments being made, unless acceptable cancellation clauses are included in the agreement(s).

12.3.3 Design Standards

The project will generally be engineered to Canadian standards and regulations and will incorporate preferred Ecuadorian codes when applicable.

12.3.4 Constructibility

Constructibility, the optimum use of construction knowledge and experience in planning, design, procurement, and field operations to achieve overall project objectives, will play a key role in the project. Key supervisors from the construction management team will form a part of the project design team. A constructibility program will be developed to ensure that integration of all project elements is considered in every stage of the design and procurement, including practicality, cost, delivery, and equipment assembly.

12.3.5 Drawings and Specifications

The engineering group will produce drawings and specifications for use in procurement and construction.

12.3.6 Materials Management Support

The engineering group will be responsible for the material take-off of every item of equipment and material required to build the facilities. Database reports will be used





for monitoring progress during requisitioning, purchasing, and tracking materials from design through to delivery.

12.3.7 Construction Support

During construction, key members of the engineering group will be available for assisting the construction team in the resolution of design issues and changes required by field conditions. Project engineers and designers will be reassigned to the field during construction should a situation require site resolution of detail engineering matters.

12.4 Procurement

12.4.1 General

The engineering contractor's purchasing group will provide procurement of capital equipment, expediting, and inspection of all equipment if needed. The construction manager's group will organize purchases of bulk materials such as pipe, cable and cable tray. Orders will be placed with qualified onshore and offshore suppliers based on quality, price, and delivery.

Most equipment and materials will be procured from sources within North and South America. All efforts will be made to purchase locally where possible and practical to do so. Corriente will purchase as much of the capital equipment and materials as possible rather than leave it to the contractors.

12.4.2 Expediting

Expediting will be very important with materials and equipment coming from widely scattered sources. Engineering and construction management expeditors will maintain contact with all suppliers to maximize their performance. The use of third party inspection and expediting services will be used where appropriate to minimise the cost of expediting and inspection. This includes telephone contacts and visits to vendors' facilities. Expediting reports will be issued after each contact and information entered into the material control reporting system.





12.5 Logistics

12.5.1 General

Sources of equipment and materials, together with transportation and labour resource issues in Ecuador have been investigated and input to the capital cost.

12.5.2 Customs and Import Regulations

Regardless of value or mode of transportation, documentation for each consignment includes the commercial invoice, packing list, certificate of origin, and a bill of lading. Import matters will be handled by an independent expediting agency.

12.5.3 Freight

Incoming freight during construction and operations will come through the port at Guayaquil to the north of Machala. The port at Guayaquil has the appropriate unloading facilities on the dock to accommodate the heavy loads needed to be transported up to the mine construction site.

Dedicated customs brokers will operate from an office at the port to ensure that the paperwork has been received for inbound freight and the necessary clearances are obtained without delays at the port.

12.6 Construction Management

12.6.1 General

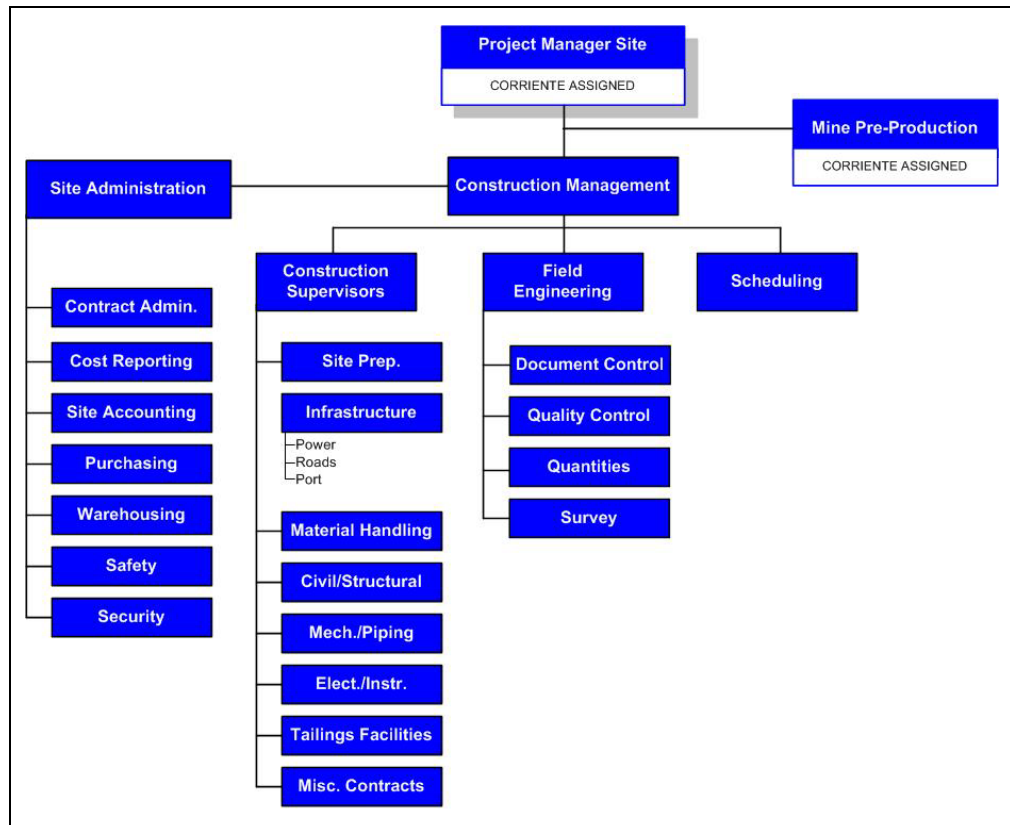
The Construction Management (CM) group will be responsible for the management of all field operations. The Construction Manager will be responsible to Corriente to effectively plan, organise, and manage the construction to meet quality, safety, budget, and schedule objectives.

The Construction Organization Chart (Figure 12-2) in this section shows the construction management team organization plan for the site. Individuals with proven abilities in a foreign environment will be placed into each position. Each will manage his or her part of the project in accordance with established procedures, in a professional manner, and with respect for the country, laws, customs, and people of Ecuador.





Figure 12-2: Site Construction Management Organization Chart



Items to be addressed during the very early days of the project include the sourcing of competent contractors; supply of concrete, cement, reinforcement; developing a camp site for site supervision; establishing construction power, access, and utilities; establishing construction access routes to the various construction location and clearing areas of interest.

12.6.2 Field Engineering

Surveying

The accuracy of the existing control system will be verified by the CM survey crew before construction starts. A listing of control data to be used will be made and issued to all contractors, with the stipulation that only the listed control shall be used for the project. Additional monuments will be set as needed.

Construction surveys will be checked by the CM crew prior to concrete placement or location of facilities.





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Quality Control/Quality Assurance

Contractors will furnish their own quality control program in accordance with contract documents. The CM field engineering team will employ QA/QC specialists who will be responsible for site quality control, including soils and concrete testing.

Document Control

While a commercial courier will be established to carry hard copies of construction documents, such as drawings and specifications, to the site, it is expected that facilities will be available at the site to download drawings and documents from the internet. The Corriente field office has a satellite phone and e-mail system that will be expanded. Drawing reproduction facilities will be installed in the field office.

The CM document control group will be responsible for receiving, storing, printing, and distributing construction drawings and specifications to the appropriate groups on the site.

Liaison with Detailed Engineering

Interpretation and design alternatives will be discussed between the field engineering representative of the CM staff and the engineer-of-record.

12.6.3 Personnel Matters

Visas

Corriente will assist all expatriate staff and their families to obtain the appropriate visas as required by Ecuador law.

Compensation Package

Salaries for CM staff will be based on a 60-hour workweek, with a 6-week-in and 2-week-out rotation.

Health and Medical

There are no requirements for special medical tests and inoculations required before entering or leaving Ecuador. A fully serviced first aid station will be established for the construction period.

An emergency evacuation plan will be established to provide rapid response and evacuation of medical cases in the event of an emergency requiring immediate hospital and/or specialist care.





12.6.4 Materials Management

As each shipment reaches the site, it will be unloaded and inventoried. Material Receiving Reports and Over-Short-and-Damage Reports will be generated and distributed to appropriate management and accounting groups by a representative of Corriente. All receiving information will be entered into the materials tracking system. After inventory, containerized materials will be returned to the container and secured until needed. Non-containerized equipment and materials will be stored in a site warehouse or behind a secure fenced area.

12.6.5 Construction Contracting

Basic Approach

It is planned to have the contractors originating from Ecuador and investigations indicate that there is a great interest and ability of contractors. Their experience has been generally associated with the oil industry and manufacturing plants and ports, but their labour pool is conversant with the trades needed to build the Mirador facilities. Contracts will generally be unit rates on civil, concrete, steel, mechanical, piping, and electrical/instrumentation. This will allow for competitive bid pricing and early mobilization. Lump-Sum contracts may be used for some ancillary facilities.

Contracting Plan

It is anticipated that the following major construction contract packages will be needed:

- infrastructure and site development
- concrete manufacturing and placement
- structural steel and pre-engineered buildings
- mechanical and pipe installations
- electrical and Instrumentation installations
- conveyor installation
- dam and diversion works
- yard services and utilities.

North American and Ecuadorian contracting companies have expressed an interest in providing their services for the mine pre-development work.





Contractors

Contractors will generally be Ecuadorian, employing and housing local labour whenever possible. From the costs received from contractors during the feasibility study capital cost estimate process, labour costs are lower than North American standards, but productivity is worse.

The normal workweek for the construction site workers will be 10 hours a day, 6 days a week; 60 hours per week. However, the construction site will be available for contractor work 7 days/week, 12 hours per day.

It is anticipated that there are about 1,329,000 man-hours of direct construction, excluding the pre-development of the mine and about 254,600 of indirect CM hours to be undertaken on the site during the capital program. There will likely be a peak of labour of about 729 persons during the construction phase of the project. The distribution of manpower is shown on Table 12-1 and Figure 12-3.

The construction contractors will be responsible for:

- accommodations and meals for their trades persons
- transportation for their workers
- site offices
- site management
- contractor surveying
- quality control.

Corriente will provide the following for the construction contractors, and the construction management staff:

- on-site first aid services
- security
- areas for site offices
- site utilities including power and water
- diesel fuel for construction equipment
- cement and concrete aggregate supplies
- supply of permanent materials
- quality assurance and quality audits.





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Table 12-1: Construction Manpower

Activity	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	Total Man Months	Total Hours
<i>DIRECTS</i>																							
Clearing Site	10	15	20	20	20	10	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Roads - Mine Access Road	10	20	20	20	10	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Roads - Pre 1: Access to Waste Dump	-	-	-	10	15	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Roads - Haul Road 1: Pit to Waste Dump	-	-	-	-	-	10	20	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Roads - Haul Road 2: Pit to Crusher	-	-	-	-	-	-	5	20	20	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Roads - Mill & Tailings Access Road	-	-	-	10	20	15	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Earthworks – Crusher	-	-	-	-	-	-	-	-	-	10	20	20	20	5	-	-	-	-	-	-	-	-	-
Earthworks – Mill	-	-	-	-	-	15	15	20	30	20	10	5	-	-	-	-	-	-	-	-	-	-	-
Earthworks – tailings	-	-	-	-	-	15	25	25	25	25	25	20	15	10	-	-	-	-	-	-	-	-	-
Earthworks at Overland Conveyor	-	-	-	-	20	30	30	30	30	30	25	15	-	-	-	-	-	-	-	-	-	-	-
Yard services at crusher	-	-	-	-	-	-	-	-	-	-	-	-	15	20	-	-	-	-	-	-	-	-	-
Yard services at mill	-	-	-	-	-	-	-	-	20	30	40	40	30	10	-	-	-	-	-	-	-	-	-
Tailings and reclaim piping	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	15	20	10	-	-	-	-
Batch Plant	-	-	-	-	-	-	-	6	6	6	6	6	8	8	8	6	6	6	3	-	-	-	-
Power lines from sub-station	-	-	-	-	-	-	-	-	-	-	-	10	30	30	40	-	-	-	-	-	-	-	-
Concrete at Mill	-	-	-	-	-	-	-	20	50	80	100	120	120	70	50	-	-	-	-	-	-	-	-
Concrete at crusher	-	-	-	-	-	-	-	-	-	-	-	-	25	50	70	40	-	-	-	-	-	-	-
Fresh water system	-	-	-	-	-	10	10	10	10	5	-	-	-	-	-	-	-	-	-	-	-	-	-
Structural steel and pre-eng buildings	-	-	-	-	-	-	-	-	-	-	-	20	40	40	40	40	40	40	30	20	-	-	-
Cladding/roofs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	20	20	15	5	-	-	-
Mechanical at crusher	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	40	50	30	-	-	-
Mechanical at mill	-	-	-	-	-	-	-	-	-	-	-	-	30	75	110	140	150	180	75	50	-	-	-
Piping at crusher	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	25	40	50	50	25	-	-
Piping at mill and in pipe fabshop	-	-	-	-	-	-	-	-	-	-	-	-	30	50	80	100	100	100	80	30	-	-	-
Piping at filter plant	-	-	-	-	-	-	-	-	-	-	-	10	20	30	5	-	-	-	-	-	-	-	-
Electrical at crusher, mill and filter	-	-	-	-	-	-	-	-	-	-	-	20	30	40	70	80	100	110	60	40	-	-	-
Admin services and finishes	-	-	-	-	-	-	-	-	-	-	-	10	15	15	15	15	-	-	-	-	-	-	-
Overland Conveyor Structural Works	-	-	-	-	-	-	20	40	60	60	70	100	70	20	-	-	-	-	-	-	-	-	-
Overland Conveyor Belting	-	-	-	-	-	-	-	-	-	-	-	25	25	25	25	25	20	-	-	-	-	-	-
Overland Conveyor Mechanical Works	-	-	-	-	-	-	-	-	-	-	-	10	50	75	80	70	50	10	-	-	-	-	-
Overland Conveyor Electrical Works	-	-	-	-	-	-	-	-	-	-	-	10	20	30	30	35	20	15	10	-	-	-	-
Port Facilities Earthworks	-	-	-	-	-	-	20	20	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Port Facilities Civil Works	-	-	-	-	-	-	-	-	-	10	40	40	30	-	-	-	-	-	-	-	-	-	-
Port Facilities Mechanical and Electrical	-	-	-	-	-	-	-	-	-	-	-	20	25	40	50	40	20	-	-	-	-	-	-
Mine Operations	-	-	-	-	-	-	-	-	-	19	19	19	19	19	19	19	19	19	19	19	19	-	-
Total Directs	20	35	40	60	85	115	105	146	221	295	325	400	597	652	652	610	605	585	582	349	164	6,643	1,328,600
<i>INDIRECTS</i>																							
Warehousing	-	-	-	-	-	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-
Safety	-	-	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	-	-
Security	-	-	5	5	5	10	10	10	10	10	10	10	10	10	10	10	10	10	8	6	-	-	-
Construction Management	3	4	6	9	9	10	11	11	11	13	14	14	14	15	14	13	9	9	8	7	7	-	-
Owner's representatives	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-	-
Engineers	-	-	-	-	-	-	-	-	-	3	3	3	3	3	3	3	3	3	3	3	-	-	-
Commissioning	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	6	6	6	-	-	-
Contractors supervision	6	6	6	8	8	12	15	20	20	20	20	20	20	20	20	20	20	20	10	6	-	-	-
Vendor reps	-	-	-	-	-	-	-	-	-	-	-	-	4	4	4	4	4	4	4	4	-	-	-
Catering for Expat camp	5	6	8	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	8	8	-	-	-
Maintenance Crew	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-	-	-
Total Indirects	22	24	36	43	43	53	61	66	66	71	72	72	76	77	76	75	71	77	76	61	55	1,273	254,600
Totals	42	59	76	103	128	168	166	212	287	366	397	472	673	729	728	685	676	662	658	410	219	7,916	1,583,200
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21		

Note: Not Including Sabanilla Power

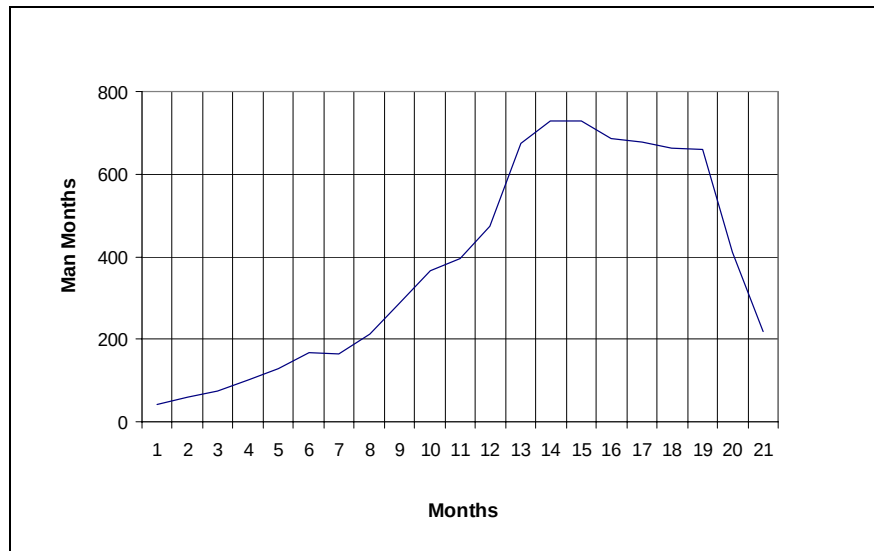




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Figure 12-3: Manpower Chart



12.6.6 Construction of Temporary Facilities

General

Corriente will provide the temporary construction facilities for the construction management staff, vendor reps, guests, etc., including accommodations, field offices, warehouses, laydown areas, and first aid facilities.

The contractors will provide their own accommodations for their employees to standards established in conjunction with Corriente.

Construction field offices, warehouses, laydown yards, and other construction facilities will be located as close as possible to the work area. The field offices and storage facilities will be patrolled 24 hours a day.

Living Accommodation

Adequate accommodations are available in the surrounding areas for the contractors workers, although they may prefer to set up camps. It is anticipated that during construction and mine pre-production the direct and indirect labour will peak at about 730 persons on the site over a three-month period. This includes both construction manpower needs, and the manpower requirements for pre-production mining.





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Existing Services

The site power is currently provided from the national power grid but will be supplemented in the remote construction areas such as the crusher area by diesel generator sets.

First Aid Facility

Corriente will provide a well-equipped first aid facility for use by all site personnel. This facility will normally be staffed 12 hours a day, with on-call services to ensure continuous coverage. The contractors will be expected to provide minor first aid stations for their workers at the site.

Site Testing

An Ecuadorian laboratory will test concrete and soils quality.

Communications

The communications framework to and from the project site is already partially in place, including accessibility for internet use. There will need to be some substantial upgrading for additional telephone and internet lines for construction which will also be used for operations.

Site Security

The site security program will be initiated during field mobilization. The specific program will be included in the field procedures and consists of the following elements:

- Vehicles requiring daily access to the site will require an authorization card, approved by the construction manager. These cards will be assigned to the vehicle for the duration of need and will be displayed where they can be seen from outside the vehicle.
- Workers bussed to the work site will get off the bus outside the control point and walk past the security guards and be picked up again by the contractors for distribution to their work areas.
- The site will be secured and patrolled 24 hours a day, seven days a week.

Access to the site will be controlled. All work areas, storage areas, office and camp areas, and hazardous areas will be fenced. Security guards will be stationed at the gates during work hours. Security guards will have reliable communications with the CM team at all times.





Fuel Supply and Storage

Corriente will furnish diesel fuel, storage, and dispensing facilities for all construction contractors.

12.6.7 Concrete

Approximately 15,600 m³ of concrete will be poured for the new facilities using locally available aggregates and a site batch plant.

Mix Designs

The proposed concrete mix designs for the various strength requirements will be developed by the appropriate contractor, using locally available aggregates and sand, and submitted for the approval of the engineer.

Quality Control

Concrete quality will be monitored by the independent quality control company on an as-required basis.

12.6.8 Health, Safety and Environmental (HSE)

A top-down commitment to HSE will be made and communicated to everyone associated with the project. Corriente will develop and issue a project safety program and an environmental control program based on its own policies and standards, internationally recognised requirements, and those required by North American and Ecuadorian law.

Construction contracts will require contractors to submit written health, safety, and environmental action plans that conform to the project HSE programs or to provide a written commitment to adhere to Corriente's HSE programs. Each contractor will be responsible for administration of its own HSE programs, and will be required to designate a management-level person who will be personally responsible to carry out the safety program and the environmental control program. Key elements of the program include new employee orientation and weekly HSE meetings.

Workers will be provided a safe place to work. Proper inspection, guidance, and strict adherence to safety policy will ensure that safe working conditions are maintained. The construction management team will insist that each contractor implement its own safety program in accordance with safety policies and procedures established for the project.





Construction activities will be carried out with an acute awareness of the environment and the need to comply with the environmental policies and programs of the project. Regular environmental audits will ensure compliance to the procedures.

12.7 Project Schedule

12.7.1 General

The project schedule is an essential planning tool for project success. It also carries resource allocations from the budget that show manpower and dollar value for each activity. Weekly and monthly status reports are provided to help determine actual progress and to flag activities that are late and/or on the critical path. Properly used, the schedule will alert management to the need for timely corrective actions to keep the project on course.

The schedule consists of a logic-networked critical path schedule based on all known project requirements.

Project Milestones

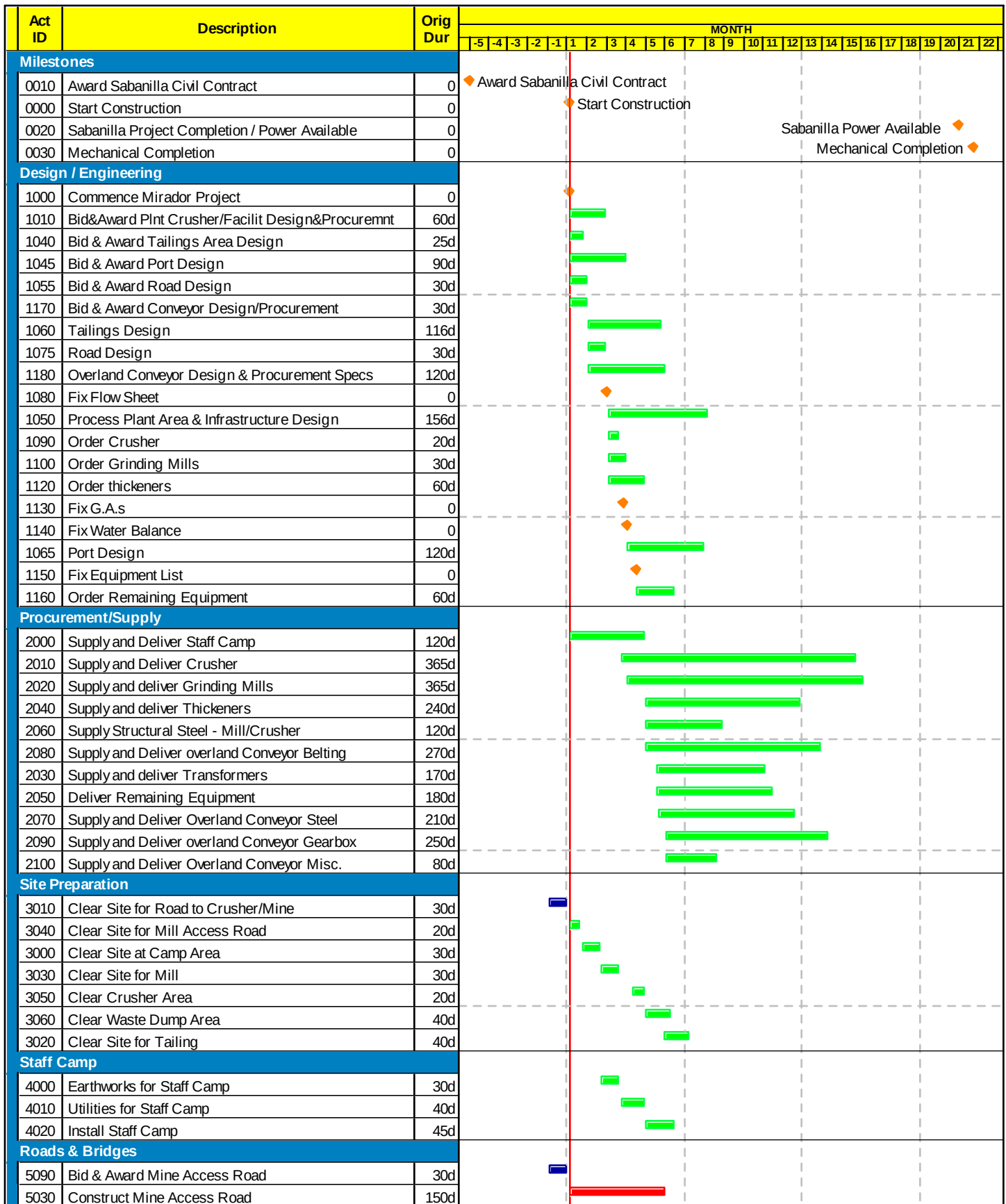
Major project element constraints that are mandated to ensure completion of the work within the prescribed period will be:

- pre-development mining activities
- project engineering activities
- project capital equipment purchases
- construction contract progress
- pre-commissioning.

12.7.2 Schedule

The project schedule, Figure 12-4, shows an estimated overall project duration of 21 months from the start of the detailed engineering to the completion processing plant ready for the dry runs of ore. The Sabanilla Power Station Project that will provide an average of 23.5 MW of hydro power to the project will need to start civil construction activity some five months ahead of the start of process plant design in order for it to be supplying power for mill run-in.

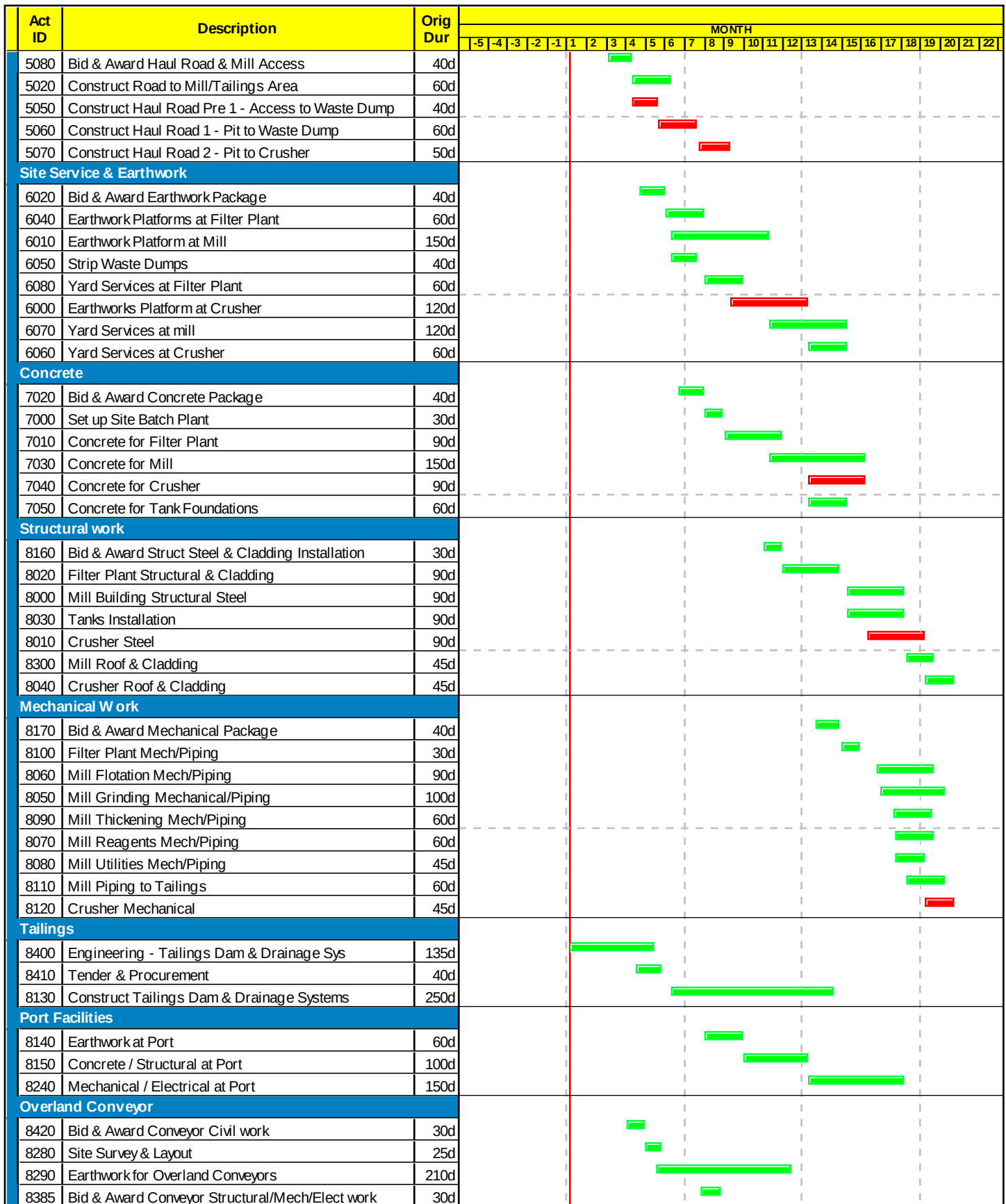


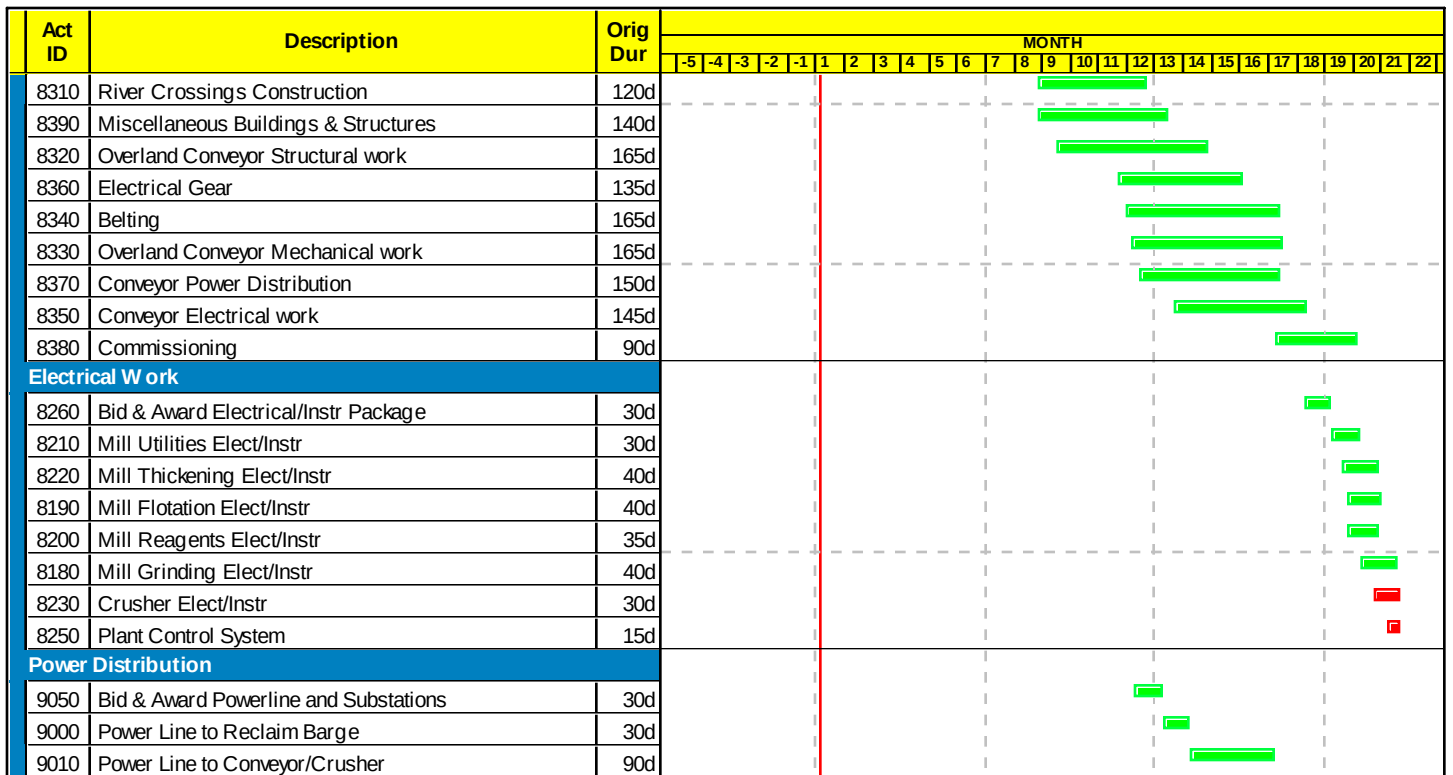


Start date 05JUL05
Finish date 24MAR07
Data date 05JUL05
Run date 19APR05
Page number 1A
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Mirador Copper Project Project Schedule

- Early bar
- Progress bar
- Critical bar
- Summary bar
- ◆ Start milestone point
- ◆ Finish milestone point







The schedule critical path, which determines the overall project duration as determined by the shortest logical chain of related events, must be followed to ensure completion of the project by the scheduled date. For the Mirador Project, the critical path will run through the project financing, approval to proceed, mine access road construction and crusher installation, as well as pre-development work in the open pit.

Engineering for the plant can proceed immediately upon the completion of project financing/approval to proceed. Procurement of major capital equipment will be done at this time also.

The key activities to commence plant operation are the procurement and delivery of major capital components, such as the crusher, mills, and electrical equipment. The milestones may be summarized as shown in Table 12-2.

Table 12-2: Key Milestones for Mirador

Description	Month No.
Project approval	0
Tailings facility engineering complete	5
Concentrator access road complete	6
Overland conveyor design complete	6
Mine access road complete	6
Earthworks at concentrator & filter areas complete	12
Earthworks for overland conveyor complete	12
Tailings facility complete	15
Mill building structural completion	18
Port facilities complete	18
Thickeners complete	19
Flotation mechanical / piping complete	19
Grinding mechanical / piping complete	20
Crusher facilities complete	20
Overland conveyor complete	20
138 kV power on-site	20
Mechanical completion	21

12.7.3 Project Initiation

Project approval and the beginning of construction activities are scheduled to occur right away in order to maximize the usable construction seasons. It is imperative that road access into the various parts of the site be completed early in order to facilitate the transport of construction materials and equipment to their work areas.





12.7.4 Engineering

Three engineering disciplines must be started right away. The first is geotechnical in order to confirm site conditions and provide the civil engineers with necessary information. The second is mechanical in order to facilitate the early ordering of long lead time equipment. The third is civil, in order to create specifications for early civil contracts, and design facilities, which will affect long lead-time delivery materials such as structural steel.

12.7.5 Long Delivery Equipment and Materials

There are a number of pieces of equipment in this project which typically have long delivery times associated with them, see Table 12-3.

Table 12-3: Long Delivery Items

Description	Typical Delivery (in weeks)
Gyratory crusher	52
Balls mills w/motors	52
Sag mills w/motors	52
Thickener	34
Agitators	28
Apron feeders	28
Substation equipment	24
Building structural steel	16

The longest delivery items should get priority in ordering to keep with the construction schedule.

12.7.6 Contracting

Unit price contracts will provide an opportunity for early mobilization.

The major contracts, which will be awarded during construction, are:

- infrastructure and site development
- concrete manufacturing and placement
- structural steel and pre-engineered buildings
- mechanical and pipe installations
- electrical and Instrumentation installations





- conveyor installation
- dam and diversion works
- yard services and utilities.

These contracts may be phased or split into smaller work packages in order to facilitate the early starting of some of them.

12.7.7 Key Construction Activities

This project schedule is driven by construction, and construction in turn is driven by site conditions.

There are three key activities which must start on schedule and which will dictate the success of bringing this project in on time. These activities are incoming primary power, process plant and conveyor design and infrastructure roads.

12.8 Project Controls

Project cost and schedule control is a key element of the project plan. The capital cost estimate is a detailed workup of equipment and materials quantities and prices, labour costs, freight costs, EPCM costs, and certain Owner's costs. When the feasibility study is approved by Corriente Resources, the feasibility study capital cost estimate is accepted and it becomes the project budget. The budget and the schedule then form the basis for project execution, against which actual cost and progress are measured. The project budget will be compared to ongoing project cost forecasting, and the schedule will be modified as engineering and construction moves forward. The project budget and schedule reflect basic project objectives, resources, and conditions unique to construction in Ecuador. When the project is released for engineering and construction, implementation of the plan will begin.

Actual progress costs, commitments, and changes will be tracked against the project Budget and schedule basis by project management. Cost variances will be calculated, based on comparisons of the project budget with the forecast (actual costs plus the balance to go) totals.

The budget status report and forecast will be developed by project-dedicated estimating, scheduling, and cost engineers in the field. Initially, consolidation of data will take place in the home office. Responsibility for this activity will shift to the site at the appropriate time.

Early identification of anticipated and actual changes to the estimate or the schedule will be used to develop change notice documentation to provide early warning of





potential cost and schedule impacts. The project controls system will provide weekly and monthly status updates of the project as it proceeds from start to completion. Any requests for significantly over-budget expenditures or significant variances in the schedule will require prior written approval by Corriente.

Cost and schedule control is the responsibility of each member of the project team. The execution plan, the schedule, and the budget form the basis and provide the means for monitoring project performance.

12.9 Commissioning and Handover

12.9.1 Pre-Operational Testing

Each process system or ancillary facility will be checked for compliance with the drawings and specifications, vendor data, and lubrication charts. Mechanical equipment will be checked for proper installation, alignment, and rotation. Conveyors will be run without load to verify belt alignment. Tanks and piping will be water/air tested. Electrical equipment and circuits will be checked for proper installation. Instrumentation circuits will be checked and instruments will be zero-calibrated. When all installations have been verified, each system will be run under no-load conditions. Permanent records will be maintained for each piece of equipment.

12.9.2 Commissioning

After each system has been checked and run successfully under no-load conditions, it will be test-run under load for a specified period of time. Instruments will be pre-calibrated to the extent possible. Any required adjustments will be made at this time.

12.9.3 Start-up

The actual start-up of each system will be performed by a team comprising of representatives of Corriente, construction manager, and contractors.

12.9.4 Handover

As the project systems are mechanically (and electrically) completed, they will be handed over to Corriente for pre-operational testing and shake-down, using the process flow diagrams and the piping and instrument diagrams. The construction management group will verify that each system is mechanically and electrically ready for turnover to Corriente. This transfer will be formally documented.





13.0 CAPITAL COST ESTIMATE

13.1 General

Merit Consultants International Inc. (Merit) has been contracted by Corriente to provide a bankable feasibility study level of capital cost estimate that is considered by Merit to be within a 10% accuracy of actual costs.

The total estimated cost to design and build the Mirador copper processing plant described in this report is US\$202,306,000. A summary overview of the estimate by area is presented in Table 13-1 and the capital cost estimate details are included in Appendix I.

Table 13-1: Summary of Capital Costs by Area, US\$

Area	Cost (\$ x 000)
<i>Direct Costs</i>	
Mine	20,108
Process plant	87,922
Utilities	6,920
Site preparation and roads	13,561
Tailings	8,474
Ancillary facilities	3,847
Total Direct Costs	140,832
<i>Indirect Costs</i>	
Project Indirects	39,362
Owner Indirects	4,376
Total Indirect Costs	43,738
Subtotal	184,570
Contingency	17,736
Total Project	202,306

All costs are expressed in first quarter 2005 US dollars, with no allowance for escalation, interest during construction, or taxes. All due allowances have been considered for items such as:

- cost increases in the price of materials and freight over the last two years (steel, cement, copper, and all general goods)
- availability of in-country labour and the prospect of initiating training programmes
- adequate experienced supervision during construction to supplement contractors
- in-country labour rates, provided by Ecuadorian contractors





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- understanding of the commitments to government in terms of standards and expectations
- degree of difficulty of the area in which the plant will be constructed; high precipitation and medium dense jungle
- environmental restrictions to be built in to the construction programme as evidenced through the EIA recommendations
- in-country methods of construction compared to those that may be used in North America, in terms of more labour intensive activities compared to machine supported ones
- understanding of the productivity of the labour force estimated to be in the order of half that experienced in North America provided experienced supervision is included
- understanding of the material resources of Ecuador for building materials
- understanding of the degree of heavy construction that is ongoing in Ecuador through the oil industry resulting in good, well run companies experienced in the type of terrain and activities that are required to build the Mirador Project
- logistics of freighting components from the ports to the site, where there is excellent highway service
- high level of information availability and quality of in-country consultants and contractors.

The estimate covers the direct field costs of executing this project, plus the indirect costs associated with design, procurement and construction efforts.

13.2 Basis of Estimate

The capital cost estimate is based on the following project data:

- mine plan
- design criteria
- flow sheets
- general arrangement drawings
- single line electrical drawing
- equipment list
- budget quotations from vendors for major capital equipment and buildings
- quotations from local in-country contractors
- quotations from in-country consultants familiar with the project





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- quantity estimates from studies (Overland Conveyor and Tailings Management Facility)
- regional climatic data
- in-house database.

Basic information has been received from several large contractors, some of whom have provided information regarding pay rates, typical crew make-up, housing requirements, the availability and skill levels of local workers, productivity factors, and the cost of construction equipment. The level of estimating accuracy and the contingency value have been based on the following premises:

- Santos CMI a large Ecuadorian civil contractor provided earthwork, concrete, and structural steel unit rates as well as construction labour rates for all trades and supervision. They also provided construction equipment rates.
- CIA Simar Constructores a large Ecuadorian contractor priced all infrastructure roads.
- Caminosca Caminos y Canales Cia. Ltda. a large Ecuadorian consultant with construction experience priced power lines and provided quantities and pricing for roads.
- Mamut Andino a large transportation and mining earthwork construction contractor in Ecuador provided unit prices for the mine pre-production work.
- CIPORT a large Ecuadorian port construction contractor provided the pricing for the port facilities required at Machala for the concentrate handling.
- All major capital equipment costs have been obtained from at least three internationally acceptable bidders.
- When productivity figures were not available through unit rates provided by in-country contractors, the productivities were extracted from actual construction statistics associated with North American projects of a similar nature and doubled to allow for supervised construction in Ecuador.
- AMEC provided all bulk material quantities for the processing facilities, the mine pre-production waste dumps and site services.
- Westmar Consultants Inc. provided material quantities, equipment pricing, and engineering costs for the Overland Conveyor.
- Knight Piésold Consulting provided material quantities, equipment pricing and engineering costs for the Tailings Management Facility.





13.3 Direct Costs

13.3.1 Quantities

Engineering material take-offs are based on “neat” line quantities derived from project drawings and sketches. Normal and acceptable allowances have been included for each discipline as appropriate. Conceptual quantities based on previous experience with the same size facilities were prepared where drawing information was not available.

Civil: All earthworks quantities, including mine pre-production stripping, have been calculated with no allowance for bulking or compaction of materials. Unit prices have been solicited from Ecuadorian civil contractors who have visited the site and are experienced with industrial work. Quantities have been based on existing plant elevations and topographic drawings at typically 2 m contours developed by ground survey teams and aerial mapping specifically for this project.

Concrete: Concrete quantities were determined from feasibility stage drawings and experience from previous projects of a very similar nature. The unit rates have been provided by Ecuadorian industrial contractors based on the assumption that local aggregates are used with an onsite batch plant.

Formwork: Formwork was estimated for each type of concrete classification and includes local supply, form oil, accessories and shoring, and stripping. No allowances have been made for re-use of forms although this will happen and improve the concrete prices during construction.

Reinforcing Steel: Reinforcing steel was calculated based on estimated weight per cubic meter of concrete for each type of classification based on projects of a very similar nature. The unit price includes the local supply of material, cutting and bending on site, accessories and installation.

Embedded Metal and Anchor Bolts: The unit price includes supply and installation of locally available carbon steel material including sleeves and anchors.

Structural Steel: Quantities were determined from feasibility stage drawings and experience from previous projects of a very similar nature. The unit rates have been provided by Ecuadorian industrial contractors. The weights shown include allowances for connections and base plates. The unit price includes steel purchase, detailing, fabrication and erection labour.





Mechanical Equipment: All large capital equipment is assumed to have to be imported, and was itemized and priced in accordance with the flow sheets. Budget quotations were obtained for all major items based on preliminary specifications. Installation hours were interpolated from similar North American projects and the number of hours doubled to reflect the expected productivity of Ecuadorian labour with good North American supervision in addition to local supervisory direction. The equipment that has not been quoted has been estimated from similar projects and the installation labour doubled from North American projects.

Mechanical (Plate work and Tanks): Plate work weights were calculated with allowances made for any necessary stiffeners, weirs, launders, etc. The unit prices include locally available plate purchase, detailing, fabrication and installation using North American labour then doubled to reflect what can be expected from Ecuadorian labour using North American supervision.

Piping: AMEC provided factored allowances for in-plant pipe, which includes for supply (not freight), shop and field fabrication, and installation with allowances for all hangers and supports.

Electrical and Instrumentation: Major electrical equipment has been assumed to be imported and prices were based on budget quotations. AMEC provided factored allowances for in-plant electrical systems and instrumentation, which includes for material supply (not freight) and installation. Lengths for overhead lines and high voltage cable were estimated from the overall site plan.

The main site substation has been provided for in the Sabanilla Power Project scope and not included here.

13.3.2 Direct Field Labour

Labour rates were provided by Ecuadorian contractors for a range of skill level in all disciplines, as well as supervisory rates from foremen to supervisor management. The all-in rates are based on the following criteria:

- base labour wage rate
- overtime premiums
- benefits and burdens
- workers compensation premiums
- travel allowances
- meal allowances





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- transportation from accommodations
- appropriate crew mixes
- small tools and consumables allowance
- field office overheads (included separately in Contractors Indirects)
- home office overheads
- contractors' profit.

Table 13-2 shows the utilized hourly labour rates for various trades.

Table 13-2: Labour Rates

Trades	Rates (\$)
Civil Earthworks	11.39
Concrete	6.00
Steel	7.42
Architectural	7.42
Mechanical	10.58
Piping	12.24
Electrical	11.54
Instrumentation	11.54
H.V.A.C.	12.24

13.3.3 Direct Field Materials

Bulk materials components are made up of locally available and imported quantities priced as FOB manufacturer. Freight costs to transport materials to site are included in the Indirects. Pricing is based on quantities provided by the engineer.

13.4 Indirect Costs

13.4.1 Temporary Construction Facilities and Services

Contractors' field distributable costs allowed for in the capital cost estimate but not included in the built-up labour rate or owner's costs are as follows:

- Contractors' mobilization and demobilization
- contractors' equipment
- construction field offices, furnishings, equipment
- contractor accommodations





- temporary power supply
- temporary water supply
- temporary heating and hoarding
- warehouse and laydown costs
- temporary toilets
- temporary communications
- first aid personnel and supplies
- on-going and final clean-up
- yard maintenance
- janitorial services
- site safety personnel and training
- material testing.

These indirect costs are not applicable to the major earthmoving costs, since unit prices submitted by contractors are “all-in” rates, inclusive of all direct and indirect costs.

13.4.2 Construction Equipment

It is expected that all construction equipment such as cranes, manlifts, welding machines, generators, etc., are available in Ecuador. An allowance has been made in the indirect accounts for construction equipment that will be used by the contractors except for the earthworks and concrete work where equipment is included in the contractor's unit rates. Specifically, concrete supply unit prices include supply and operation of the site aggregate crushing, screening and batch plant as well as the concrete transit trucks.

13.4.3 Construction Accommodation and Catering

An allowance is included for the construction and operation of a 75-man camp to house the expatriate supervisory staff, Owner's start-up people, vendor reps, and visitors.

Costs for the contractors accommodations on site have been included using an allowance of \$2 per man hour included in the labour rate, or about \$2.5 million assuming peak manpower of 700 to 750 contractor personnel. It is anticipated that some contractors will house their people in surrounding communities and bussed into the site on a daily basis. Some may be housed in contractor-built camps on the project





site. The labour rate includes all meal allowances for contractors' personnel. Catering cost allowances for the expatriate supervisory staff is included in the Indirect accounts.

13.4.4 First Fill and Spare Parts

Spares are based on vendor recommendations where available. Where vendor information has not been available, an allowance of 5% of the equipment purchase value has been included. Allowance has been made for the purchase of start-up grinding media and reagents.

13.4.5 Start-up and Commissioning

An allowance has been made for retention of vendor representatives for start-up, as well as a selection of ten people from the contractors' crews and the construction management staff for a period of about six weeks.

13.4.6 Freight

A freight allowance of 5% of the material costs has been included for all materials and equipment associated with purchases made in Ecuador. An allowance of 10% of the equipment and material costs has been included for freight that is imported. Equipment freight costs include packing, freight to port, marshalling costs, ocean freight, demurrage, and land freight to site.

13.4.7 Owner's Costs

Owner's costs have been developed in conjunction with Corriente and includes items such as:

- working capital where an allowance has been made for a full complement of operations staff for a three month period
- training programmes for operations staff available locally
- insurances
- head office staff assigned to the project
- Owners allowances for field operations offices and supplies
- Owner's travel costs during construction.





13.4.8 Engineering, Procurement, and Construction Management (EPCM)

Engineering and procurement costs are based on information from previous similar projects for the process facilities, and from quoted costs from in-country engineering companies for certain infrastructure portions of the work.

Construction management costs have been derived from calculations based on estimating the number of personnel required including transportation, supplies, and communications.

13.4.9 Taxes and Duties

No taxes have been included, but an allowance of 2.5% of the direct capital equipment cost has been included in the estimate for duties.

13.5 Contingency

The contingency amount is an allowance added to the capital cost estimate to cover unforeseeable costs within the scope of the estimate. These can arise due to presently undefined items of work or equipment, or to the uncertainty in the estimated quantities and unit prices for labour, equipment, and materials. Contingency does not cover scope changes or project exclusions.

Contingencies are included separately for the civil works, equipment and materials and the labour. Earthwork is considered to have the greatest risk and an allowance of 15% of the calculated costs have been assigned as a consequence. While ground and aerial surveys have been used to calculate quantities, the definition between earth and rock has been estimated from limited geotechnical drilling results and by supposition using knowledge of the area by Corriente's geological personnel.

Contingency levels for the Overland Conveyor and for the Tailings Management Facility studies have been developed by the study consultants and are a component of the project contingency.

Additional contingencies include 5% on the capital equipment and materials, and 10% allowed on the remainder of the direct and indirect estimates.

13.6 Exchange Rates

All estimates herein are quoted in first quarter 2005 US dollars.





13.7 Assumptions

The following assumptions were made in preparing the estimate:

- construction work is based on unit and fixed price contracts
- concrete aggregate and suitable backfill material are available locally and good potential areas were identified during the project team site visits
- actual soil bearing conditions will be adequate for the requirements of foundations included in the estimate. This has been confirmed through the surficial geology assessment made by Corriente based on their intimate knowledge of the area.
- construction activities will be carried out in a continuous program with no allowance for demobilization and remobilization
- the tailings area and other major areas of earthwork are assumed to be constructed over two seasons, where the first season is dedicated to water management
- labour productivities for the installation of materials and equipment are half of what can be expected in North America provided experienced supervision from North America supplements the Ecuadorian supervision. The initial baseline productivities have been derived from actual values achieved on projects of very similar construction. Earthworks activities retain the North American productivity values due to the nature of their high equipment utilization.
- all capital equipment is imported into Ecuador
- most bulk materials such as cement, rebar, steel, cable, cable tray, and piping is available in Ecuador
- buildings, other than the concentrator, will be constructed as Ecuadorian style buildings that make use of local labour, materials, and methods
- the project will employ locally available people to handle importation customs, transportation, warehousing, some purchasing duties, security, and safety.

13.8 Capital Cost Exclusions

- escalation
- scope changes
- interest during construction
- schedule delays and associated costs such as those caused by:
 - scope changes
 - unidentified ground conditions





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- extraordinary climatic events
- labour disputes
- permit applications
- receipt of information beyond the control of EPCM contractors
- cost of financing
- property costs and taxes
- sunk costs
- taxes
- sustaining capital, included in Section 15 Financial Analysis
- permitting costs.





14.0 OPERATING COSTS

14.1 Summary

Operating costs have been prepared in fourth quarter 2004 US dollars, and exclude:

- contingency
- allowance for escalation
- value-added (IGV) taxes
- import duties.

The operating cost estimates have been assembled by area and component, based on estimated staffing levels, consumables and expenditures according to the mine plan and process design. Mine operating costs are based on contract mining. Concentrate shipping costs are detailed in Section 15 Financial Analysis. Life-of-mine operating costs are shown in Table 14-1, and annual operating costs in Table 14-2. Detailed calculations and spreadsheets for operating costs are included in Appendix J.

Table 14-1: Operating Cost Life-of-Mine, US\$

	Labour	Expenses	Total Cost	\$/t Mined	\$/t Milled
Mine operations	11,966,000	198,565,000	210,531,000	1.14	1.90
Processing operations	28,179,000	289,784,000	317,963,000	1.71	2.87
Administration	36,093,000	34,302,000	70,395,000	0.38	0.64
Total	76,238,000	522,651,000	598,889,000	3.23	5.41

Table 14-2: Annual Operating Cost, US\$

Year	Mining	Processing	Admin	Total	\$/t Mined	\$/t Milled	\$/lb Cu
1	20,598,000	26,723,000	5,806,000	53,127,000	2.49	5.82	0.41
2	17,890,000	26,202,000	5,806,000	49,898,000	3.00	5.47	0.36
3	16,961,000	26,178,000	5,806,000	48,945,000	3.01	5.36	0.40
4	16,181,000	26,183,000	5,806,000	48,170,000	3.19	5.28	0.38
5	16,473,000	26,183,000	5,806,000	48,462,000	3.22	5.31	0.38
6	16,811,000	26,185,000	5,806,000	48,802,000	3.25	5.35	0.39
7	17,019,000	26,178,000	5,806,000	49,003,000	3.26	5.37	0.41
8	17,729,000	26,167,000	5,806,000	49,702,000	3.31	5.45	0.44
9	17,661,000	26,179,000	5,806,000	49,646,000	3.31	5.44	0.41
10	18,252,000	26,176,000	5,806,000	50,234,000	3.34	5.51	0.42
11	16,930,000	26,160,000	5,806,000	48,896,000	3.78	5.36	0.43
12	15,921,000	26,167,000	5,806,000	47,894,000	4.07	5.25	0.41
13	2,105,000	3,282,000	723,000	6,110,000	4.53	5.43	0.37
Total	210,531,000	317,963,000	70,395,000	598,889,000	3.23	5.41	0.40



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14.2 Basis of Estimate

14.2.1 Wage and Salaries

The typical average annual all-in costs for Ecuadorian personnel used for the operating cost estimates are shown in Table 14-3. The rates were provided by Corriente and include all burdens, bonuses, and allowances for overtime. The mandatory 15% before-tax profit sharing bonus is included in the financial analysis. The basis for these rates is presented in Appendix J.

Table 14-3: Ecuadorian Average Labour Annual Costs US\$

Area	Annual All-in Cost
<i>General Operations or Maintenance Labour</i>	
Unskilled	14,200
Mid skill/responsibility	19,900
High skill/responsibility	31,300
<i>General Office, Technical, or Management Labour</i>	
Unskilled	14,200
Clerical	12,900
Technical	19,900
Expert/engineer	31,300
Senior expert/engineer	39,800
Most senior expert/engineer	62,600
<i>Industry Specific Labour</i>	
Community relation superintendent	64,000
Environmental superintendent	64,000
Loss control superintendent	64,000
Geology superintendent	64,000
IT superintendent	64,000
Operations shift foreman	32,300
Maintenance planner	38,400
Maintenance foreman	54,500

A core group of expatriate management and speciality skilled professionals will be required for production and maintenance years of the operation as this expertise is not available within Ecuador. This estimated all-in costs for this group is summarized in Table 14-4, and are based on similar expatriate positions at other similar sites in South America.



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Table 14-4: Expatriate Management and Specialist Staff Annual Costs US\$

Position	Annual All-in Cost
<i>Mine</i>	
Mine superintendent	193,500
<i>Mill</i>	
General Operations Foreman	109,000
<i>Administration</i>	
General manager	292,500
Operations manager	251,250
Tax and finance manager	226,600
Environmental, health, and safety manager	226,600
Controller	226,600

14.2.2 Schedules of Work

The work schedule assumes production will operate 24 hours per day, seven days per week, 365 days per year. Operations and maintenance personnel will work six days on and three days off. Supervisory, engineering and administration staff and hourly labour will work five days on with two days off. Mining personnel will operate on two 12 hour shifts per day, on a four-day on, four-day off rotation.

14.2.3 Power Costs

Power will be supplied primarily from the Sabanilla power plant, with supplementary power from the national grid. The price of power for each source was provided by Corriente:

- Sabanilla power: \$0.048/kWh, at the Sabanilla substation
- Grid power: \$0.08/KWh, at the Mirador substation.

14.2.4 Other Costs

- consumption figures for crusher and mill liners and grinding media are derived from the abrasion index and power consumption
- reagent consumption levels are based on metallurgical testwork conducted by SGS Lakefield
- unit costs for Lime, aero 3894, xanthate, frother, and flocculant are based on budget pricing from South American suppliers
- diesel pricing is based on contract supply prices in Ecuador



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- costs for crusher and mill liners are based on budget pricing from equipment suppliers
- costs for grinding media are based on budget pricing from South American suppliers
- budget pricing was obtained for explosives
- other operating supplies and expenses are based on in-house experience with similar operations and on lump sum allowances.

14.3 Mining Costs

The annual direct mining cost results are shown in Table 14-5. Pre-production costs have been capitalized and included in capital cost estimate. Expenses are the contract mining costs and Labour is Corriente's costs to administer the mining contractor and provided mine planning and engineering services.

Table 14-5: Annual Direct Mining Cost, US\$

Year	Labour	Expenses	Total	\$/t Mined	\$/t Milled
1	987,000	19,611,000	20,598,000	0.964	2.257
2	987,000	16,903,000	17,890,000	1.077	1.961
3	987,000	15,974,000	16,961,000	1.043	1.859
4	987,000	15,194,000	16,181,000	1.072	1.773
5	987,000	15,486,000	16,473,000	1.096	1.805
6	987,000	15,824,000	16,811,000	1.118	1.842
7	987,000	16,032,000	17,019,000	1.132	1.865
8	987,000	16,742,000	17,729,000	1.179	1.943
9	987,000	16,674,000	17,661,000	1.177	1.935
10	987,000	17,265,000	18,252,000	1.215	2.000
11	987,000	15,943,000	16,930,000	1.309	1.855
12	987,000	14,934,000	15,921,000	1.355	1.745
13	122,000	1,983,000	2,105,000	1.559	1.871
Total	11,966,000	198,565,000	210,531,000	1.140	1.900

14.3.1 Basis of Estimate

The contract mining cost estimate was prepared by an Ecuadorian mining contractor based on Corriente's mine schedule prepared by AMEC. The operating costs details are included in contract mining pricing in Appendix E and include the following:

- supply, operation and maintenance of equipment
- supply of fuel



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- supply of explosives
- blasting, loading and hauling of ore and waste
- construction and maintenance of in-pit haul roads as required
- supply and maintenance of contractor's offices, shops, warehouse, first aid, fuel tanks and other required support facilities
- overhead and profit.

Excluded from the mining contractor's budget pricing are pit dewatering costs and grade control. These functions will be handled by a Corriente mine operations staff. Corriente will be responsible for the following:

- administration of the mining contract
- geology including grade control
- planning and scheduling
- surveying
- pit dewatering.

14.3.2 Mine Labour

The mine will be headed by Corriente's mine operations superintendent who will report to the operations manager. Reporting to the mine operations superintendent will be administration, geology, and engineering groups. The mining department, under the direction of the mine operations superintendent, will be responsible for the open pit operation.

The mine will operate on two 12-hour shifts per day, on a four-day on, four-day off rotation. Thus, there will be four shift crews, each directed by a shift foreman.

Corriente Personnel

Corriente's Staff and labour requirements over the mine life are listed in Table 14-6. Staff requirements were determined by assessing the pre-production mine plan and mine life activities. General role personnel are based on the operating activities and productivities.

The engineering department will be managed by a senior mine planning engineer who will be responsible for the annual mine plan. He will also manage a department of 13 personnel plus outside consultants for geotechnical and hydro geological services. He will be assisted by a mining engineer responsible for the pit water management, a



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surveyor, surveyor helpers (2), a field engineer who will be responsible for pit dewatering activities.

Table 14-6: Corriente Mine Personnel Requirements and Annual Costs

	Number	Annual Cost \$
<i>Administration</i>		
Mine superintendent, expatriate	1	193,500
Clerk	1	12,900
<i>Geology</i>		
Chief geologist	1	62,600
Grade control geologist	1	39,900
Samplers	4	66,000
Clerk	1	12,900
<i>Engineering</i>		
Senior mine planning engineer	1	39,900
Mining – geotechnical consultants		100,000
Surveyor	2	62,600
Surveyor helpers	4	66,000
Water management engineer	1	39,900
Hydro – geological consultants		100,000
Contractors		25,000
Field engineer	1	81,700
Tradesman	1	31,300
Helpers	2	39,900
Clerk	1	12,900
Total Personnel	22	987,000

Seven personnel will work in the geology department. This group will be responsible for updating the orebody model, calculating ore reserves and grade control. The chief geologist will be an Ecuadorian. He will be assisted by a grade control geologist four samplers to ensure that sampling for grade control is carried out in a timely fashion, and a clerk.

Contract Mining Personnel

The contract mining staff and labour requirements over the mine life are listed in Table 14-7.



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Table 14-7: Contract Mine Personnel Requirements

	Years	1	2	3	4	5	6	7	8	9	10	11	12	13
<i>General</i>														
Mine superintendent		1	1	1	1	1	1	1	1	1	1	1	1	1
Supervisors		4	4	4	4	4	4	4	4	4	4	4	4	4
Administration and camp maintenance		3	3	3	3	3	3	3	3	3	3	3	3	3
<i>Mine Maintenance</i>														
Mechanics		14	14	14	14	14	14	14	14	14	14	14	14	14
Warehouse		2	2	2	2	2	2	2	2	2	2	2	2	2
<i>Operations</i>														
<i>Loading & Hauling Ore & waste</i>														
Excavator operators		4	4	4	4	4	4	4	4	4	4	4	4	4
Haul truck drivers		16	20	16	16	16	18	18	20	20	20	20	20	20
<i>Drilling & Blasting Ore & Waste</i>														
Blasting		8	8	8	8	8	8	8	8	8	8	8	8	8
<i>Rippable Waste</i>														
Dozer operators		4	2	2	2	2	2	2	2	2				
<i>In Pit Road Maintenance</i>														
Grader operators		2	2	2	2	2	2	2	2	2	2	2	2	2
* Dozer Operators		(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)
Loader operators		2	2	2	2	2	2	2	2	2	2	2	2	2
Truck operators		4	4	4	4	4	4	4	4	4	4	4	4	4
Drum vibrators		2	2	2	2	2	2	2	2	2	2	2	2	2
Water truck operators		2	2	2	2	2	2	2	2	2	2	2	2	2
<i>Ancillary Equipment</i>														
* Dozer operators		(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)
Excavator operators		2	2	2	2	2	2	2	2	2	2	2	2	2
Lube truck operators		2	2	2	2	2	2	2	2	2	2	2	2	2
Trailer Platform operators		2	2	2	2	2	2	2	2	2	2	2	2	2
Emulsion mix truck operators		3	3	3	3	3	3	3	3	3	3	3	3	3
Total		79	81	77	77	77	79	79	81	81	79	79	79	79
Estimated absenteeism @ 5%		4	4	4	4	4	4	4	4	4	4	4	4	4
Total Contract Mining Labour		83	85	81	81	81	83	83	85	85	83	83	83	83

* Dozer splits time between pit road maintenance and other miscellaneous operations

14.4 Processing Costs

14.4.1 Summary

Processing costs include costs for operating and maintaining the concentrator, from coarse ore dump pocket at the primary crusher through to concentrate load out facilities, tailings management, tailings ponds plus process and reclaim water pumping. Processing costs below include hourly personnel, consumables such as reagents, power, operating supplies and maintenance supplies. The annual direct processing costs are shown in Table 14-8 with life-of-mine total and unit costs in Table 14-9 and area costs shown in Table 14-10.



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Table 14-8: Annual Direct Processing Cost and Unit, US\$

Year	Labour	Supplies	Power	Total	\$/t Mined	\$/t Milled
1	2,324,000	12,020,000	12,379,000	26,723,000	1.25	2.93
2	2,324,000	11,499,000	12,379,000	26,202,000	1.58	2.87
3	2,324,000	11,475,000	12,379,000	26,178,000	1.61	2.87
4	2,324,000	11,480,000	12,379,000	26,183,000	1.73	2.87
5	2,324,000	11,480,000	12,379,000	26,183,000	1.74	2.87
6	2,324,000	11,482,000	12,379,000	26,185,000	1.74	2.87
7	2,324,000	11,475,000	12,379,000	26,178,000	1.74	2.87
8	2,324,000	11,464,000	12,379,000	26,167,000	1.74	2.87
9	2,324,000	11,476,000	12,379,000	26,179,000	1.75	2.87
10	2,324,000	11,473,000	12,379,000	26,176,000	1.74	2.87
11	2,324,000	11,457,000	12,379,000	26,160,000	2.02	2.87
12	2,324,000	11,464,000	12,379,000	26,167,000	2.23	2.87
13	291,000	1,464,000	1,527,000	3,282,000	2.43	2.92
Total	28,179,000	139,709,000	150,075,000	317,963,000	1.71	2.87

Table 14-9: Processing Life-of-Mine Total Cost and Unit Cost, US\$

	Total	\$/t Mined	\$/t Milled
Staff personnel	4,594,000	0.025	0.041
Operations personnel	8,585,000	0.046	0.078
Maintenance personnel	8,896,000	0.048	0.080
Technical personnel	6,104,000	0.033	0.055
Processing consumables and supplies	139,709,000	0.753	1.263
Power	150,075,000	0.809	1.357
Total	317,963,000	1.71	2.87

Table 14-10: Processing Area Total Cost and Unit Cost, US\$

	Total	\$/t Mined	\$/t Milled
Primary crushing and conveying	13,290,000	0.072	0.120
Primary grinding	186,468,000	1.005	1.686
Flotation and regrinding	68,231,000	0.368	0.617
Tailings and water services	14,662,000	0.079	0.132
Concentrate filtration	7,614,000	0.041	0.069
Mill supervision and yards	19,903,000	0.107	0.180
Technical services	7,795,000	0.042	0.070
Total	317,963,000	1.71	2.87

14.4.2 Processing Labour

The processing operations will be headed by an expatriate mill operations superintendent who will report to the operations manager. Under his control will be four departments:

- supervision



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- operations
- maintenance
- technical.

The process work force build-up is summarized in Table 14-11.

Table 14-11: Processing Personnel Requirements

Years	1 to 13
Staff	
<i>Mill Supervision</i>	
Operations manager, expatriate	1
Chief metallurgist, expatriate	1
General operations foreman	1
Operations shift foreman	3
Maintenance foreman	2
Maintenance planner	1
Clerk	2
<i>Mill Operations</i>	
Crushing operator	3
Grinding operator	3
Flotation/regrind operator	3
Dewatering plant operator	3
Tailings and services operator	3
Reagent operator	3
Labourers	6
FEL operators	3
<i>Mill Maintenance</i>	
Crushing mechanic	3
Grinding mechanic	3
Flotation mechanic	3
Dewatering plant mechanic	3
Electrician	3
Instrument technician	3
Helpers	6
<i>Technical</i>	
Metallurgical technician	2
Chief chemist	1
Sample prep	2
Assayers	4
Environmental technician	2
Clerk	2
Total Personnel	75



14.4.3 Process and Ancillary Power Consumption and Cost

Power consumption is summarized by major areas in Table 14-12. Power consumption was calculated based on the projected motor power-draw for all equipment. Load and utilization factors were applied to all units to determine annual power demand and consumption. Power consumption was estimated at an average of 226,383 MWh/a.

The cost of power was averaged at \$ 0.0539/kWh, based on 80% being supplied from the new Sabanilla power plant and the remainder being supplied from the local grid. Note that line losses, at 5%, are included in the power calculations, because the Sabanilla power is priced at the Sabanilla substation; Mirador covers the cost of the line losses.

Table 14-12: Average Annual Power Consumption

Area	MWh/a
Primary crushing and conveying	7,382
Primary grinding	155,385
Flotation and regrinding	41,234
Tailings and water services	11,613
Concentrate filtration and drying	2,008
General and administration	8,760
Total site demand	226,383
Line losses	11,915
Total	238,298

14.4.4 Concentrator Maintenance Supplies Cost

Maintenance supplies costs are allowances, based on AMEC's experience with other similar operations, details are included in Appendix J.

14.5 General and Administrative Operating Cost

14.5.1 General

The G&A costs include administrative personnel, general office supplies, safety and training supplies, travelling, contracted consultant services, insurance, permits, security, camp, building maintenance (excluding the process buildings and the mine truckshop), environmental management, and employee transportation. These are costs not directly chargeable to the mining or process areas. G&A costs are summarized in Table 14-13.



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Table 14-13: General and Administration Costs, US\$

Area	Total Cost	\$/t Mined	\$/t Milled
Site labour	27,479,000	0.148	0.248
Site operating supplies and expenses	33,090,000	0.178	0.299
Quito office costs	9,826,000	0.053	0.089
Total	70,395,000	0.379	0.636

14.5.2 General and Administration Labour

The G&A labour cost estimate is based on manpower loading shown in Table 14-14.

Table 14-14: G&A Personnel Requirements

	Years 1 to 13
Staff	
<i>Quito Office</i>	
VP government affairs, expatriate	1
Tax and finance manager, expatriate	1
Accountant	1
Purchasing agent	1
Buyer	1
Executive secretary	1
Clerks	2
Janitor	2
<i>Site Administration</i>	
Operations manager	1
Operations manager secretary	1
Environmental health and safety manager	1
Controller	1
Information technology superintendent	1
Payroll supervisor	1
Payroll clerk	1
Warehouse supervisor	1
Warehouseman	6
Human resources superintendent	1
Community relations superintendent	1
Human resources clerk	2
Clerical/reception	2
General maintenance supervisor	1
General maintenance journeyman	6
General labourer	6
Janitor	4
Loss control/training manager	1
Safety technician	1
Nurse	2
Drivers	4
Loss control superintendent	1
Loss control guards	18
Total Personnel	74



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14.5.3 General and Administration Expenses

The G&A expenses are mostly allowances, based on Corriente and AMEC experience, details are included in Appendix J.



15.0 FINANCIAL EVALUATION

15.1 Summary

The Mirador Project was analyzed using a discounted cash flow approach assuming 100% equity in first quarter 2005 US dollars. Projections for annual revenues and costs are based on data developed for the mine, process plant, ancillary facilities, infrastructure, capital expenditures and operating costs. Estimated project cash flows were used to determine the pre-tax net present value (NPV) and internal rate of return (IRR) for the base case.

Results of the base case analysis indicate that the project has a potential pre-tax internal rate of return of 14.8% and a pre-tax net present value \$71.2 million at a discount rate of 8%. Effects of changes to commodity prices, capital cost and operating cost were examined in a sensitivity analysis. This analysis indicated that the project is more sensitive to changes in copper price than either capital or operating costs and is slightly more sensitive to operating costs than capital costs. The base case scenario has a projected payback period of approximately five years.

A complete list of model parameters used in the feasibility study and the printout of the model is in Appendix K.

15.2 Valuation Methodology

A discounted cash-flow analysis was used to value the Mirador Project. This method requires projecting annual cash inflows (or revenues), and then subtracting annual cash outflows such as operating costs, sustaining capital costs and taxes. The resulting net annual cash flows are discounted back to the date of valuation at a chosen discount rate, and totalled to determine the project's net present value. The date of valuation is assumed to be when the decision is made to proceed with project development identified as the commencement of detailed engineering. For discounting purposes, cash flow is accounted midway through the year.

An internal rate of return is calculated, equivalent to the rate of return at which net present value equals zero. The payback period is stated in terms of the number of years from the production start date required to pay back the initial capital investment, excluding sunk costs, based on the undiscounted cash flow.



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15.3 Silver

Silver was not included in the resource model as it was not part of the sampling program. For the base case financial analysis, silver was not included in the financial analysis as it could not be shown, by way of resource modelling, to be consistently present across the deposit. However, metallurgical testing results showed that silver was present in all metallurgical samples and in all concentrate samples, averaging 54 g/t in the first two years and 68 g/t for the remaining years. Therefore, the impact of silver was reviewed as a preliminary assessment. Including silver revenue in the financial modelling improved the Base Case IRR by over 1% and the NPV at 8% by over \$13 million. Corriente has embarked on a program of re-sampling to incorporate silver into the resource model.

15.4 Metal Prices

Prices used for the financial evaluation were:

- Cu: US\$1.00/lb
- Au: US\$400.00/oz
- Ag: US\$6.50/oz.

It was felt that recent copper prices do not reflect cycle average copper prices. Supporting this, major companies are using \$1.00/lb in their forecasts and marketing information available to AMEC indicates that is a reasonable level.

Projecting a gold price involves complex issues other than straight supply and demand, and as such, it was felt that a price of US\$400.00/oz, which is close to the current range, was the prudent path.

15.5 Smelter Terms

Smelter terms were provided by Corriente based on discussion with smelters in Chile and Peru. The smelter terms used in the model are shown in Table 15-1.

Table 15-1: Smelter Terms

Term	Copper	Gold	Silver
Payable Factor	96.5%	95%	90%
Deduction	1 unit	1 unit	N/A
Treatment charge	US\$75/dt	N/A	N/A
Refining charge	US\$0.075/lb	US\$6.00/oz	US\$0.40/oz



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15.6 Freight and Port Handling Charges

As described in Section 8, the concentrate will be trucked to Puerto Bolivar where it will be stored until loaded onto freighters for shipment to smelters. Corriente has obtained budget pricing for concentrate freight and handling, see Appendix G for details. The freight and handling charges are shown in Table 15-2.

Table 15-2: Freight and Port Handling Charges

Item	Charge (US\$/wt)
Freight charge from mine to port	35.82
Port Charges	1.70
Ocean transport charge	42.00

15.7 Sustaining Capital

Sustaining capital costs are costs, which at the time the estimate is prepared, are known to be required for sustainable production from the operation. Sustaining costs are not intended to cover future modifications or developments. Sustaining costs were developed for areas listed in Table 15-3. These sustaining costs were incorporated in the financial model.

Table 15-3: Sustaining Costs, (US\$ '000)

Item	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Total
Tailings	1,017	602	543	507	479	457	438	421	406	393	380	368	6,011
Water treatment	250	250	250	250	250	750	750	750	750	750	1,000	1,000	7,000
Total	1,267	852	793	757	729	1,207	1,188	1,171	1,156	1,143	1,380	1,368	13,011

Tailings sustaining costs are required to raise the elevation of tailings dam. The water treatment costs are an allowance included to cover the construction and operation of passive water treatment facilities. There are no mine sustaining costs as this is covered by the contract mining pricing.

15.8 Closure Costs

Closure costs are included to cover the dismantling of the plant and infrastructure, the modifications outlined in the closure plan discussed in Section 11, and the ongoing costs to provide water treatment. The estimated costs are approximately US\$15 million, plus a credit of US\$5 million for salvage.



15.9 Other Assumptions

The major assumptions made in development of the cash flow model used in this project evaluation are outlined below:

- valuation date of 01 January 2005 based on a two year pre-production and construction period and mill start-up of 01 January 2007
- use of mid-year cash flows for discounting purposes
- working capital assumes approximately three months of NSR for supplies, stores and accounts receivable; accounts payable is set at 15 days of operating costs
- royalties payable to Billiton amount to a flat 1% of NSR with an upfront payment of US\$2 million; no government royalties apply
- government taxes and any other profit sharing schemes excluded
- modelled on a 100% equity basis
- concentrate marketing costs were excluded
- concentrate losses allowed at 0.25% based on operating experience.

15.10 Inflation

No allowance was made for inflation of revenues or costs.

15.11 Economic Analyses Results

With silver excluded, the project has an expected pre-tax NPV of approximately US\$71.2 million at an 8% discount rate, a pre-tax internal rate of return of 14.8% and a payback of approximately five years, based on a mine life of approximately 12 plus years. With silver included, the project has an expected pre-tax NPV of approximately US\$84.8 million at an 16% discount rate, a pre-tax internal rate of return of 15.9% and a payback of approximately five years, based on a mine life of approximately 12 plus years. The variation in NPV with discount rate is shown in Table 15-4.

Table 15-4: Variation in Pre-Tax NPV with Discount Rate, (US\$ M)

	0%	5%	8%	10%
NPV – Base Case	254.6	122.2	71.2	45.1
NPV – Silver included	280.3	139.2	84.8	56.9



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15.12 Sensitivity Analyses

Base case sensitivity analysis is summarized in Table 15-5. For the range of changes examined, from 80% to 120% of the base case, the project NPV is most sensitive to copper price and is more sensitive to operating cost than capital costs.

Results of the net present value (8%) and internal rate of return for metal prices, capital cost and operating cost are illustrated graphically in Figures 15-1 and 15-2.

Table 15-5: Base Case Pre-Tax Sensitivity Analysis

Parameter	Variation (%)	Value	IRR (%)	NPV at 8% (\$ M)
<i>Average Copper Price</i> (\$/lb)	-20	\$0.80	-	-86.2
	-10	\$0.90	7.2	-7.5
	0	\$1.00	14.8	71.2
	10	\$1.10	21.3	149.9
	20	\$1.20	27.2	228.6
<i>Average Gold Price</i> (\$/oz)	-20	\$320	13.5	55.9
	-10	\$360	14.2	63.6
	0	\$400	14.8	71.2
	10	\$440	15.5	78.9
	20	\$480	16.2	86.5
<i>Operating Cost</i> (\$/t ore)	-20	\$4.33	20.2	135.4
	-10	\$4.87	17.6	103.3
	0	\$5.41	14.8	71.2
	10	\$5.95	11.9	39.1
	20	\$6.49	8.7	7.1
<i>Capital Cost</i> (\$ M)	-20	\$161.8	20.1	107.1
	-10	\$182.1	17.2	89.2
	0	\$202.3	14.8	71.2
	10	\$222.5	12.7	53.3
	20	\$242.8	10.9	35.3



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Figure 15-1: Base Case Pre-Tax NPV Sensitivity @ 8%

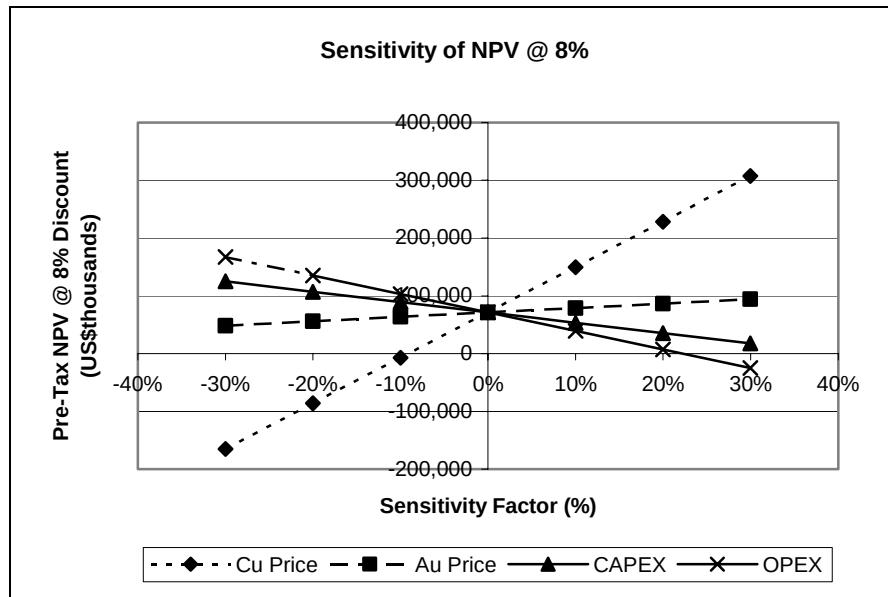


Figure 15-2: Base Case Pre-Tax IRR Sensitivity

