BATEMAN

AZTEC COPPER Inc.

Santo Tomas Project

Update Previous Prefeasibility Study

Project # TMMT02105

October 2003

BATEMAN ENGINEERING INC

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Figure 1

General Location Map of the Mine Area

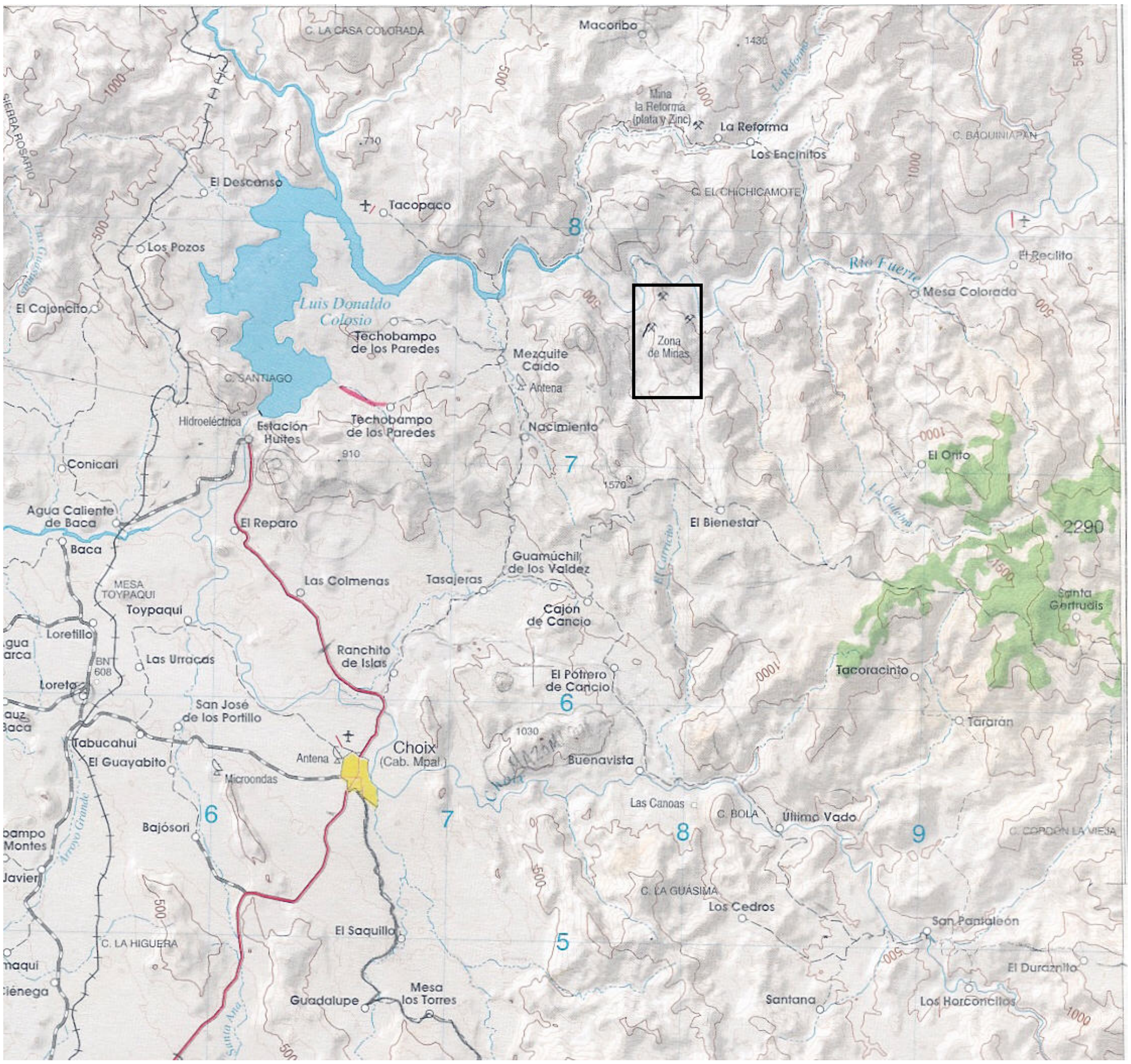
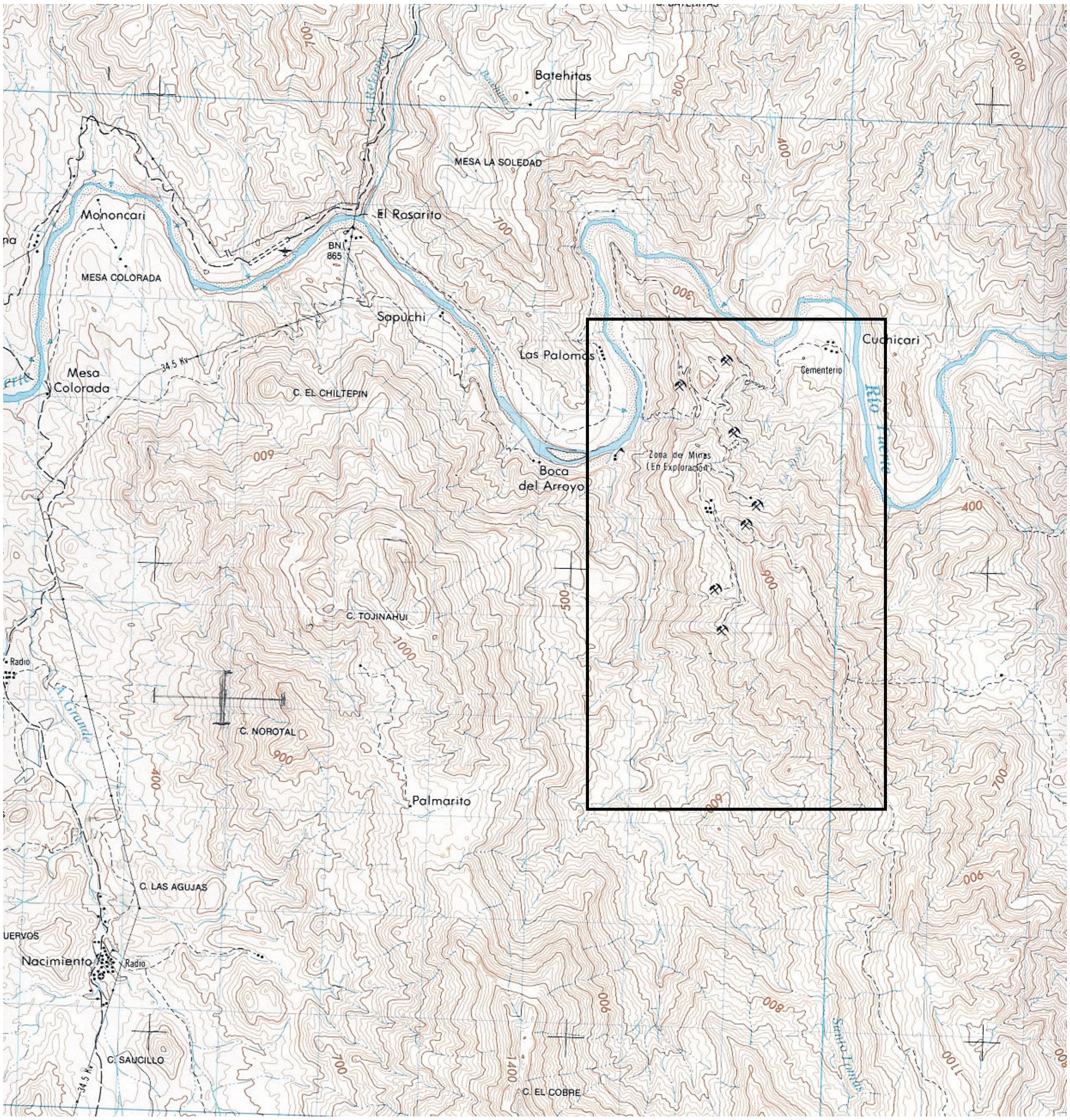


Figure 2

Detail Map of the Mine Area



SECTION 1

INTRODUCTION

1.0 GENERAL

On July 7, 2003, Aztec Copper of Edmonton, Canada, gave authorization for Bateman Engineering Inc., of Tucson, Arizona to proceed with an update of a previously completed pre-feasibility study of the Santo Tomas Copper Project. The original study was completed in 1994 for Exall Resources. Only specific sections of the study were to be updated i.e. those pertaining to the costs for the original project and the future feasibility study. The mining and resource section were not be addressed at this time as further drilling is planned to delineate the full potential of this ore body.

1.1 LOCATION AND SITE

The project is located in the northern portion of Sinaloa, Mexico about 150 km to the northeast of the city of Los Mochis. The ore body lies in the mountains on the south bank of the Fuerte River. Access to the site is by paved road from Los Mochis to Choix and about 22 km of unpaved road from Choix to the mine site. Please refer to attached map Figures 1 and 2.

A 34.5 kv powerline is located about four km from the site to the west. This power line passes through the small community of Nacimiento eight km to the southwest of the site. This power line once serviced the La Reforma Mine operated by Peñoles, located about seven km to the north of the site, on the other side of a mountain range.

The climate at the site is generally mild with temperature ranges from about -5°C (23°F) to +45°C (113°F). Rainfall occurs a little over forty days per year with heavy "monsoon" type rainfalls in July and August. The annual rainfall is about 580 mm (23 inches) with a maximum 24 hour rainfall of about 100 mm (4 inches). The elevation of the site is from 300 to 1100 meters above mean sea level.

A new hydroelectric power dam and water reservoir has been completed approximately 20 km west, downstream of the site. The water level near the site will rise from an elevation of 200 m to approximately 280 m. The maximum height of the reservoir is listed at 287.5 m at the spillway. This will interfere with the part of the north ore body and possible environmental concerns with the tailings containment.

1.2 HISTORY

Interest in the Santo Tomas copper deposit began with ASARCO Mexicana S.A. (ASARCO) in 1968. From October 1968 to May 1971, ASARCO completed a total of 16 vertical rotary and 43 vertical diamond drill holes. Results from this work identified an ore deposit of 86.9 million tonnes of copper ore at 0.52% Cu and 118.7 million tonnes of copper ore at 0.28% Cu.

In 1973, Tormex-Penoles (Tormex) took an option on the property and relogged 26 of the ASARCO core samples. Tormex drilled 6 additional holes to aid in defining the northern zone and refined the ore body to include delineation of the sulfide and oxide ore deposits. Tormex redefined the ore reserve as being 86.7 million tonnes of 0.5% copper with a 1.5:1 stripping ratio. The oxide reserve accounts for 22 million tonnes and the sulfide deposit 65 million tonnes.

One additional hole was drilled by Tormex in the southern zone. This reserve was redefined as being 140 million tonnes of 0.4% Cu. Indications of precious metals were also noted, however, assays were not available to define these resources.

Minera Real de Angeles S.A. de C.V. (MRA) became involved in 1991. MRA evaluated 12 of the ASARCO cores and reported findings that correlated closely with ASARCO.

Exall Resources hired Bateman Engineering Inc to complete a prefeasibility of the property in 1993 and 1994

Aztec Copper has recently become involved with the project by obtaining an option to acquire the property and performing additional drilling.

1.3 PRESENT WORK

The present work was centered on the 60,000 MTPD option presented in the original study. The earlier study determined at that time, that any operation less than 60,000 MTPD was not economical.

The following activities were undertaken:

- All equipment has been updated to August 2003 costs for a 60,000 MTPD concentrator.
- A review of current concentrate leaching processes was conducted.
- Verify that concentrate sales to third party smelter/refineries is required as most of the smelters identified in the 1994 study are closed down.
- Prepare an "order of magnitude" estimate for a SX/EW plant able to process 20 million tonnes of oxide ore.
- Comments of possible relocation of tailings, roads, plant site as a result of the recent site visit.
- Recommendations for further work necessary to prepare a full feasibility study for the project.

SECTION 2

EXECUTIVE SUMMARY

2.1 GENERAL

Aztec and Bateman representatives met to discuss the scope of work required for updating the previous study and presenting an estimated cost for a feasibility study. Certain sections were to be updated while others were not. Note particularly that ore reserves, mineralogy and environmental issues have not been address in this study. A site visit to the Santo Tomas property was set up to determine conditions, changes and activities since the original study in 1994.

2.2 STUDY OPTIONS AND CONSIDERATIONS

Based on the indicated size of the restricted north deposit from the 1994 report, the low average grade and the initial pre-feasibility study work investigations of various plant sizes and process options, the following key parameters were established for the study work:

- The plant size evaluated in this study was 60,000 MTPD.
- The capital cost of the plant updated to 2003 costs.
- The costs of "toll" smelting and refining the concentrate produced on a third party basis.
- The possible ramifications of the completed Huites reservoir on the ore reserve, tailings disposal location and location of the plant.
- Credits for gold and silver values should be considered in an economic evaluation, however credits are available and consistent with “toll” smelting and refining but not in concentrate leaching.
- The costs of leaching of concentrate followed by an SX-EW facility was not included in this study due to the lack of viable commercial processes being constructed during the past 9 years.
- Visit to the plant site to identify changes since the work done in 1994.
- Recommend order of magnitude cost estimates for a full feasibility study.

2.3 CURRENT SITE VISIT REVIEW – COMMENTS

The original Section 3 of the 1994 Study was titled **"GEOLOGY AND ORE RESERVES"** and was prepared by Mintec**.** No work has been done to up date that section at this time. The heading name has been changed to present items from the current site visit.

A Bateman senior process engineer, together with Aztec Copper's chief geologist, undertook a two-day visit to the project site and vicinity, to re-evaluate first the considerations and assumptions taken in the original study, and secondly to assess any changes that have occurred in the general area in the nine years since the original study, and their effect on the project. Specific focus was made on the effects of the completed Huites reservoir and electrical power to the site.

After the site visit, it is became apparent that diverting the river would be challenging because the reservoir reaches the project area at two different locations. But the diversion could be investigated at the tail end of the project to potentially recover the 138 million tones in the wall/berm.

The economic evaluation of the project and the review of mining reserves were not included as part of the scope of work of this update study. The impact of the potential increase in the minable reserves obtainable at the end of the project must be assessed on its own merits at that time.

The original study contemplated the deposition of the waste rock from the pits in the vicinity east of the pits next to the river, which makes sense considering the shortest haulage distances. But it is known, from other mining operations, that waste rock and low grade ore can produce acid mine drainage when exposed to the elements for long periods of time. This could create a future environmental problem for the project if not addressed. Consequently, it is suggested that a different location be looked at for handling the waste in a box canyon southwest of the mine called Bienestar. This area was originally considered for a tailings dam.

As a result of the site visit, a different location for the plant site was identified in the area called Palmarito This seems better suited for construction of the plant from the point that the site would allow gravity flow of tailings to a new location away from the Huites reservoir and would also allow better control of accidental spillages from the plant.

In the original study the tailings dam was to be located in the canyon southwest of the minesite. The toe of the dam was almost at the containment dam, which in turn was located almost at the high level of the Huites reservoir. From the regional maps provided by Aztec, that an area close to the town of Nacimiento appears to be more appropriate for a tailings dam rather than in the narrow canyon southwest of the minsite. This location, as well as others, would need to be studied for suitability by the appropriate disciple consultant.

The Choix substation is not big enough for the project's power requirements. Consequently the Santo Tomas project will be required to negotiate the power supply from El Fuerte substation, or probably from San Blas. Another possibility could be to take power from the new Huites plant.

See Section 3 for more details and photos.

2.4 METALLURGICAL TESTWORK

Previous test work for the Santo Tomas Project as provided by Exall in the original study indicated that flotation concentrate can be metallurgical achieved. However overall recovery of copper from the ore was determined by the ore grade with a significant differences occurring in ores with less than 0.6% copper as compared to ores with greater than 0.6% copper at a 0.02 cut off grade.

The results of the test work indicated that the ore responds favorably to flotation but is not amenable to direct leaching using sulfuric acid. Concentrate leaching appeared to be feasible under highly oxidizing conditions at controlled temperatures.

Future test work would include Bond Mill Index for crushing and grinding optimization, flotation testing to verify maximum concentrate grade and recovery from a low grade ore and oxide ore leaching characteristics if the oxide cap is to be processed.

2.5 PROCESS

Bateman reviewed the original work and found that many equipment suppliers that existed in 1994 are now consolidated into larger companies. Milling technology has developed and larger more efficient mills and grinding equipment are available. The process still indicates that conventional flotation concentrating facilities utilizing SAG milling should be used as the basis for this study.

A description of the selected process circuit is provided in Section 5 of this report.

The current Bateman review of concentrate leaching alternatives has been updated and indicates that there are three potential processes that appear economical. They are summarized in Section 5 and other leaching alternatives are presented in Appendix B.

2.6 SURFACE FACILITIES

Due to the remoteness of the site, all necessary surface facilities, plants, maintenance shops, warehouses, administration building and change house will be built on site. Descriptions of these facilities, the mine probable layout with alternate plant site locations are included in Section 6 of this report.

2.7 TAILINGS DISPOSAL

An alternate tailings dam location southwest of the proposed plant site is described. This location offers containment approximately 7 km away from the Huites reservoir and would not be an environmental concern.

Depending on the quality of the tailings to be produced by the Santo Tomas concentrator, it may be possible to construct the dam walls with tailings as is done at many of the mines in the Southwestern United States. The tailings from the plant site pass through a series of cyclones located on top of the starter dam. The cyclones separate the tailings into a sand fraction and a slimes fraction. The sands (underflow) form the tailings dam face and slimes (overflow) form the slimes pond upstream.

Water that is impounded within the tailings dam can be pumped back to the plant site for process use. Excess water, if any, will be evaporated within the pond. None of the recycled water will be discharged into the Huites reservoir.

A more complete description of the tailings dam location and operation is included in Section 7 of this report.

2.8 INFRASTRUCTURE

The infrastructure for the project will consist of the following major facilities:

- Access Road
- Powerline
- Water Supply

The access road to the plant/mine will consist of 15 km of new road, 9 km of reconditioned road, and 34 km of road to maintain. These distances were measured by car during the site visit and verified by topographical maps of the area provided by Aztec. This road will be used to haul workers and materials to and from the site and concentrate from the concentrator.

At the present time there are no power lines on or near the Santo Tomas property. The 34.5 kV line that passes 4 km west of the proposed plant site is inadequate for the plant size under consideration. The closest substation, at Choix, is rated at 1.5 MVA

The closest power substation large enough to handle the 53 MW power demand estimated for the plant in this study is in San Blas. The upgrading of the power line from San Blas to Choix, and to the project site, or the construction of a totally new powerline is a matter for future discussion and negotiation with the Federal Power Commission (CFE).

An overland power line from the town of Choix to the proposed plant site would have to be constructed. The power line would basically follow the plant/mine access road up through Nacimiento. The total poleline distance from Choix is approximately 27 km.

An alternative could be to take power from the new Huites plant, which at about 17 km closer to the project site than Choix. The powerline from this plant runs northwest, away from the project site, towards the State of Sonora. This alternative would require building a substation at Huites and a new powerline to the project site. Again, this is an option for future discussion with CFE.

Fresh water supply for the project will be from the Huites Reservoir (Fuerte River). This supply requires a federal permit. A pump station will be located near the plant site. The station will allow for water supply at varying levels of the reservoir.

2.9 ENVIRONMENTAL AND SOCIOECONOMIC

No work was done to update this section

2.10 MARKETING

The concentrate produced at Santo Tomas would be shipped to third party for smelting and refining. Of the three smelters sited, in Mexico, in the previous study, all but one have been shut down (Nacozari is operating). There is only one smelter operating in Arizona. However, existing smelters in Asia or Europe could be candidates for purchasing of concentrate.

Transport of concentrates will be by truck, railroad and ship. The trucks would transport the concentrate to an intermediate load out station for transfer to rail cars or ships. The exact method of transportation will depend on the end location of the smelter and refinery to be used. A detailed logistics study should be undertaken during the next phase of a study.

Marketing is more fully discussed in Section 10 of this report.

2.11 PROJECT IMPLEMENTATION

No work was done to update this section.

2.12 CAPITAL AND OPERATING COSTS

A preliminary order of magnitude (OOM) (±35%), new plant, capital cost estimate was prepared for the 60,000 MTPD concentrator. The methodology used to obtain the OOM estimate is summarized below:

- Update the costs of equipment from the previous equipment list by contacting appropriate vendors and suppliers.
- Adopted a traditional factorization method of calculating the commodity costs by using known percentiles or factors of the installed mechanical costs.
- The installed mechanical costs were derived from scale buildups.
- Commodity costs were derived from similar installations.
- All indirect costs i.e. EPCM, first fills, spares were determined by factors from industrial norms.

These capital cost estimates are based on new plant equipment and materials for the 60,000 MTPD option of the original 1994 study. The cost of this plant is $409.5 million dollars (US). No consideration for used plants was made in this study.

The concentrate production estimate was based on the information from the 1994 report and no further delineation of the ore body has been made since. The north pit calculation is based on the "restricted pit" as defined in the original report. The south pit was inadequately defined and no definition was made on the saddle between the north and south pits. The following table summaries the production estimates for concentrate and copper.

Production Summary

	75%		88%	
	Concentrate (MTPY)	Copper (MTPY)	Concentrate (MTPY)	Copper (MTPY)
North Pit	207,000	56,000	243,000	66,000
South Pit	174,000	47,000	204,000	55,000

The original geology and ore reserve section of the report prepared by Mintec, indicated that an additional 138 million tonnes of ore could be recovered from the berm between the north pit and the river if a river diversion project was initiated at the tail end of the mine life. The cost and feasibility of such a diversion should be investigated in the future.

There is an oxide deposit located on top of the main sulfide deposit that is not defined but estimated by Aztec as 20 million tonnes. Using current solvent extraction and electrowinning technology and assuming that the oxide is leachable with dilute sulfuric leaching solution the calculation indicates that the oxide could produce approximately 123,700 tons (112,200 tonnes) of LME grade copper over a 6 year period.

The estimated cost for an SXEW plant to process the ore from the oxide cap is approximately $80 to $90 million dollars (US) assuming that the mine and equipment are already in place.

2.13 ECONOMIC EVALUATION

No economical evaluation was made at this time.

2.14 CONCLUSIONS AND RECOMMENDATIONS

The Santo Tomas ore body is a low grade, high tonnage copper ore deposit with gold and silver values. The ore is amenable to conventional copper concentrating recovery. The project site will require that the construction and operation of the plant be conducted in a manner that is environmentally acceptable.

Based on the updated capital cost focused on a mining rate of 60,000 MTPD and that concentrate will be produced and shipped to a toll smelter/refinery and metallurgical testing performed previously by others the following conclusions may be reached:

- The estimates, provided from Mintec Inc., of minable reserves and assays indicate Santo Tomas is a low grade, high tonnage copper deposit containing modest gold and silver values. The north pit which was used as the basis for this study contains an estimated minable ore reserve at 0.2% cutoff of 428 million tonnes at an average copper grade of 0.368%, considering that a safety wall/berm is left unmined in the north side of the pit to avoid possible problems with the Huites reservoir at maximum water level.

- In the original work provided by Mintec, an ultimate pit with no mining limitations, the berm mined, was identified. The reserve would increase to 565 million tonnes in the north pit or an additional 138 million tonnes. The viability of this additional ore will be identified in the feasibility study.

- Past metallurgical testwork as listed in Section 4, indicated the ore is amenable to copper recovery by conventional concentrate processing consisting of crushing, grinding, and flotation to produce a saleable copper concentrate. The testwork also indicated that gold and silver contained in the ore will report with the copper concentrate.

- Based on the updated study work the "order of magnitude" capital cost estimate for a new grassroots conventional concentrator facility, utilizing SAG mills indicates that the Santo Tomas project would cost $410 million for the 60,000 MTPD plant.

- A more economical approach for the development of the Santo Tomas property would be to identify an existing concentrator plant and relocate the equipment to Santo Tomas.

- The geotechnical and hydrological feasibility of the preliminary mine design for the restricted north pit, which was used as the production basis for evaluation in this report, has not been determined.

- The recent site visit has presented optional locations for tailings containment, waste rock containment, and the plant site that seemingly would not impact the Huites reservoir.

RECOMMENDATIONS

In order to bring all the supporting data for the project up to equivalent levels of accuracy for support of more accurate feasibility level evaluation, we believe that the following activities will be required. These activities are broken down into two phases, one that will confirm the reserve and the second that will prepare the feasibility study for the processing facility. The costs associated with Phase I and Phase II are presented separately.

PHASE I – Confirmation Drilling and Preliminary Pit Design

- Additional exploration/confirmation drilling: As per the suggestion of Mintec Inc. an exploration/confirmation program that includes approximately 80 drill holes at 250 meter centers in the center saddle and south deposits are required to determine the size of the deposit. Mntec already has information of the 57 holes drilled in the north deposit. Based on estimates received from Aztec Copper for drilling the program cost, which includes sample assays and support the cost of drilling would be approximately $1.44 million dollars (US).

- Reserve evaluation - preliminary pit design: The information obtained from this drilling program and previous information would be used to perform a new reserve evaluation, preliminary pit design and production schedules in enough detail to provide the necessary information to review the optimum mine-plant capacity. Based on information from Mintec Inc. the estimated cost of this activity would be $40,000 (US) *(work to be done by Mintec)*

- Total cost of Phase I is approximately 1.5 million dollars (US)

PHASE II – Feasibility Study

- Additional feasibility confirmation drilling: To provide adequate mine reserve detail to prepare the feasibility study. It is estimated that 60 additional drill holes will be required at centers of less than 250 meters. Based on estimates received from Aztec Copper on drilling the program cost, which includes sample assays and support the cost of drilling would be approximately $1.1 million dollars (US).

- Reserve confirmation and mine design: The information obtained from this drilling program and previous information would be used to confirm the reserve and provide production schedule forecasts and pit design in enough detail to design the processing facility to ± 15% accuracy. Based on information from Mintec Inc. the estimated cost of this activity would be between $60,000 and $75,000 (US). *(work to be done by Mintec)*

- Metallurgical testing: From new drill core obtained from the program above, sample material could be collected for more detailed crushing, grinding and flotation studies which could provide and confirm data for equipment sizing and preliminary plant design. Test work is required to classify the Bond Work index to determine the hardness of the rock for specifying crushing and grinding parameters. Flotation tests need to be conducted to verify that a 28% concentrate grade can be achieved and what the actual recovery would be for the 0.368% ore. The order of magnitude cost for these studies would be $50,000 to $70,000 dollars. Approximately 450 to 500 pounds of new material would be required to conduct these tests. (work to be done by Mt. States R&D)

- Environment studies: Environmental studies undertaken by specialists are recommended for preliminary project permitting activities including Environment Impact Study (EIS) and others. Order of magnitude cost would be $25,000 to $30,000. *(work to be done by EPG – environmental planning group)*

- Geotechnical, hydrogeological and hydrological studies:

  - The mine pit design parameters, slope stability and plant soil conditions need to be fully investigated by specialists. Order of magnitude cost would be $230,000 to $350,000, including geotechnical-oriented diamond core drilling and laboratory testing. *(work to be done by Golder)*

  - Performance of a geotechnical study to confirm the feasibility of a berm at the north side of the pit, with respect to permeability, rock characterization, rock integrity. Order of magnitude cost would be $145,000 to $190,000, including hydrogeologic test hole diamond core drilling, field permeability testing and laboratory testing. *(work to be done by Golder)*

  - Performance of a geotechnical study to characterize the plant site foundation conditions and development of foundation design recommendations prepared by specialists. Order of magnitude cost would be $120,000 to $180,000, including geotechnical test hole drilling, backhoe test pits and laboratory testing. *(work to be done by Golder)* The plant throughput considered is 60,000 metric tons per day.

  - Performance of a geotechnical study to select site, perform detailed geotechnical investigation, complete engineering analtsis and prppare feasibility design drawings and report for tailings impoundment. Order of magnitude cost would be $190,000 to $260,000, including geotechnical test hole drilling, backhoe test pits and laboratory testing. *(work to be done by Golder)*.

- Socio-economic study of the area needs to be conducted. This work is dependent upon Aztec Copper's work and negotiations with the Mexican government as summarized below.

- Power Supply study to determine the viability and cost of getting 53 MW of power to the plant. This work is dependent upon Aztec Copper's work and negotiations with the Mexican government as summarized below.

- Feasibility study. When all the above information is available from Phase I and II, a full feasibility study could be performed which would develop the information required for plant design and capital cost and operating cost estimates to ± 15% accuracy. The estimated cost of such a study would be in the range of 3.1 to 3.5 million dollars over a 12 to 14 month period, excluding owners' costs and contingency. During the study discussions would take place with the various Mexican authorities regarding infrastructure development and power, water and fuel pricing structure. *(work to be done by Bateman)*

The estimated cost for Phase II, to produce a definitive estimate of ± 15% suitable for the raising of finance through appropriate institutions for the mine and concentrator would be in the range of 5.0 to 5.8 million dollars.

Once the oxide cap of the deposit is fully delineated and metallurgical testing completed, a solvent extraction electrowinning facility could be designed that could produce 21,700 tons (19,700 tonnes) per year of LME Grade cathode copper for approximately 6 years. An estimated cost to bring the facility to a viable level would be as follows:

- Oxide leaching: If recovery of the copper from the oxide ore is desired, column leach testing should be conducted to determine the optimum plant size, acid consumption and leachablilty of the oxide ore. Approximately 1,000 kg of newly drilled material would be required. The order of magnitude cost for this testing is approximately \$60,000 to \$75,000 dollars. This is secondary work compared to the main ore body.

- Feasibility study: When all the above information is available, a full feasibility study should be performed which would develop the information required for plant design and capital cost and operating cost estimates to a ± 15% accuracy. The estimated cost of such a study would be in the range of 0.9 to 1.2 million dollars over a 8 to 10 month period, excluding owners' costs and contingency.

The estimated cost to produce the feasibility study for the oxide portion of the project only would be the sum of the metallurgical testing and engineering and design study work. This assumes that the geotechnical and hydrological for the mine and associated sites are completed as listed above and that only minimal geotechnical and hydrological work would be required for identifying the stability of the leaching area and site location of the SXEW plant.

2.15 QUALIFICATIONS, OBLIGATIONS AND LIABILITY LIMITATIONS

These items are covered in the Terms and Conditions of the "Agreement for Pre-feasibility Review".

2.15.1 The Client acknowledges that any deliverables or results of the Services, including the Report prepared by Bateman in terms of this Agreement are prepared for the sole and exclusive use of the management of the Client. The Client agrees that the Report shall always be quoted in full, and not in part, summary or precis form and shall when reproduced contain the provisions of Clauses 10 and 11 in full and the disclaimer stipulated in clause 13.2.

2.15.2 Notwithstanding that reasonable skill, care and diligence have been exercised in the performance of the Services required for the preparation of the Report, shall neither Bateman nor, its principals, subcontractors, its or their officers, directors and employees accept any liability to the Client other than as specified in clause 10 of this Agreement for the execution of the Services and/or studies necessary for the Report.

Bateman specifically accepts no liability to any other organisation or person to whom the Report is presented for any loss or damage whether direct or indirect arising from statements made by Bateman, the interpretation by others of such Report, the use of or reliance upon information contained in such Report, or for any design and engineering or other work performed by others using such Report.

2.15.3 The Client hereby indemnifies and agrees to hold Bateman harmless against all claims from third parties which may arise directly or indirectly as a result of the failure by the Client to comply with the provisions of this Clause 13, or in relation to the Services or as a result of the third party's reliance, use or interpretation of the Report prepared by Bateman.

2.15.4 The Client undertakes that it will not use the Report or any part thereof for any purpose whatsoever until the costs of the Services have been paid to Bateman in full.

SECTION 3

CURRENT SITE VISIT REVIEW - COMMENTS

3.0 GENERAL

As part of this update study, a Bateman senior process engineer, together with Aztec Copper's chief geologist, undertook a two-day visit to the project site and vicinity, to re-evaluate, first the considerations and assumptions taken in the original study, and secondly to assess any changes that have occurred in the general area in the nine years since the original study, and their effect on the project.

3.1 Effects of the Huites Dam.

The upstream end of the Huites Reservoir borders the project site and for technical and environmental reasons this fact must be taken into consideration in any evaluation of the projectboth now and in the future.

- **Minable Reserves**

  Geographically the north side of the deposit extends into the Fuerte River, at 180 meters elevation. The design maximum level of the reservoir is 287.5 meters. In the original study, the pit design was arbitrarily restricted to 428 million tonnes to leave a non-mined wall, 50 meter wide at 300 meter elevation between the north pit and the river, which reduced the overall minable reserves as identified by Mintec by 138 million tonnes.

  After the site visit, it is became apparent that diverting the river would be challenging because the reservoir reaches the project area at two different locations. But the diversion could be investigated at the tail end of the project to potentially recover the 138 million tones in the wall/berm.

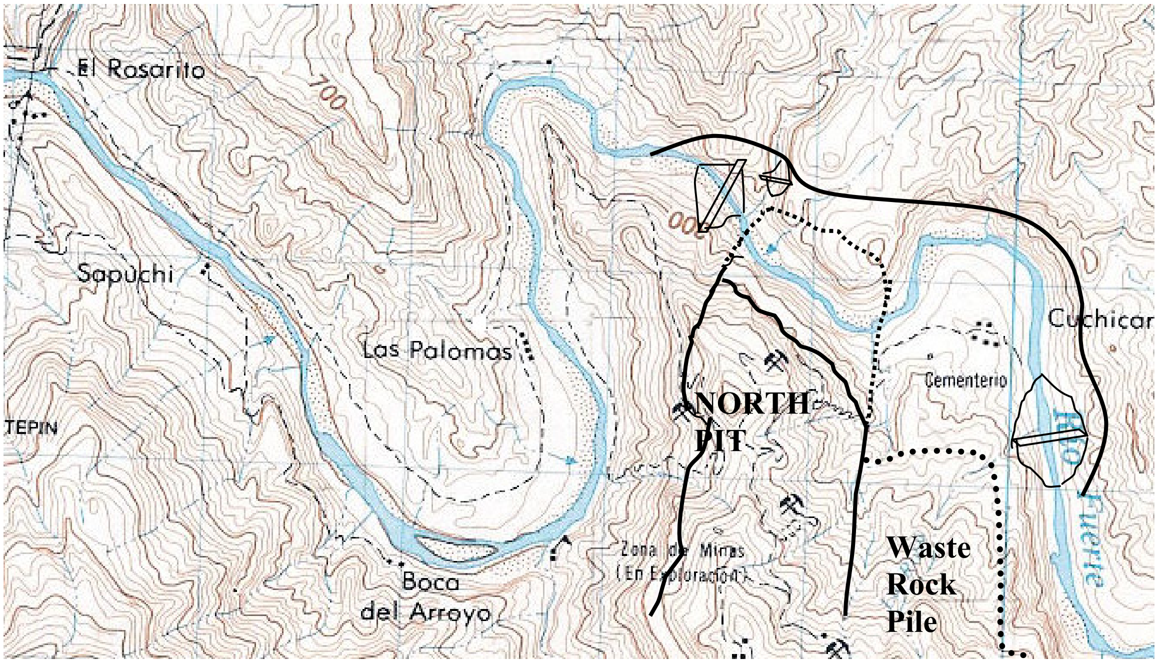
  The following picture, looking NE from Cerro Tojinahui, shows how the water level reaches the Santo Tomas mountainside and that the high level mark, apparently at 240 m elevation, can be seen on the riverbanks,. The river runs at about 180-200 meters elevation for more than 6 km downstream from the project. The Huites reservoir would have to reach a very low level before the riverbed, near the project area were accessible for construction of the downstream containment dam, after having built the upstream dam and diversion channel.

  The minesite could not be reached due to the bad condition of the existing road from the south and there was no transportation available via the reservoir. However, the ferry operators shuttling people across the river reported that the water level was reaching Cuchicari, at the east side (upstream) of the project area.

Bateman has prepared the following sketch as a visual example of where the dams and diversion channel might be located and can only be used for discussion and not construction. The sketch superimposed on the section of map below shows the north end of the orebody and the dams and channel/tunnels that might be required to be built in order to bypass the river from the project area.

Since the maximum level of the reservoir is at elevation 287.5 meters (dam spillway elevation) and the river bed is at approximately 180 meters (topo maps), the containment dams would need to be about 120 meters high, and capable of sustaining the full pressure of the lake. The upstream dam would create another lake, similar to the upstream end of Huites at maximum level. Even if the channel were built deep enough to divert the river at low level, eventually the upstream dam would have to support the high level of the Huites reservoir. The downstream dam also would have to sustain the Huites maximum level at site. The sketch also shows the possible expansion of the north pit, in broken lines, as estimated in the original study.

The economic evaluation of the project and the review of mining reserves were not included as part of the scope of work of this update study. The impact of the potential increase in the minable reserves obtainable at the end of the project must be assessed on its own merits at that time.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(En Exploración)
NORTH
PIT
Waste
Rock
Pile
Cementerio
Cuchicar
Rio
Fuerte

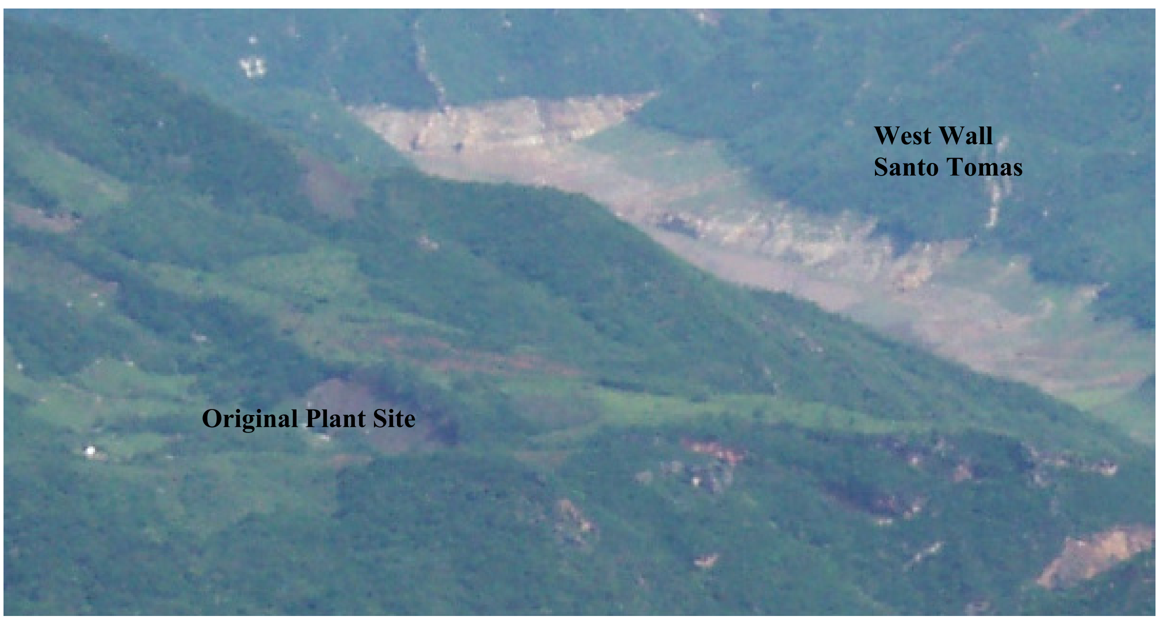
- **Waste Dumps**

The original study contemplated the deposition of the waste rock from the pits in the vicinity east of the pits next to the river, which makes sense considering the shortest haulage distances. But it is known, from other mining operations, that waste rock and low grade ore can produce acid mine drainage when exposed to the elements for long periods of time. This could create a future environmental problem for the project if not addressed. Consequently, it is suggested that a different location be looked at for handling the waste in a box canyon southwest of the mine called Bienestar. This area was originally considered for a tailings dam.

The advantages of this site for waste rock pile are similar to those for a tailings dam: Controllable watershed; possible to build a containment dam between the toe of the pile and the Huites reservoir, relatively close to the minesite. The average distance from this site to the north pit will be approximately 3 km. During the first years of operation, haulage from the north pit limit, elevation 480 meters, will be downslope. As calculated in the original study by Mintec, for tailings, at elevation 535 meters the waste site could hold about 250 million tonnes, while about 600 million tonnes of waste will be mined. However, if the north pit is developed first, it could be possible to deposit the overburden and waste rock from the south pit in the mined-out north pit, instead of hauling waste uphill.

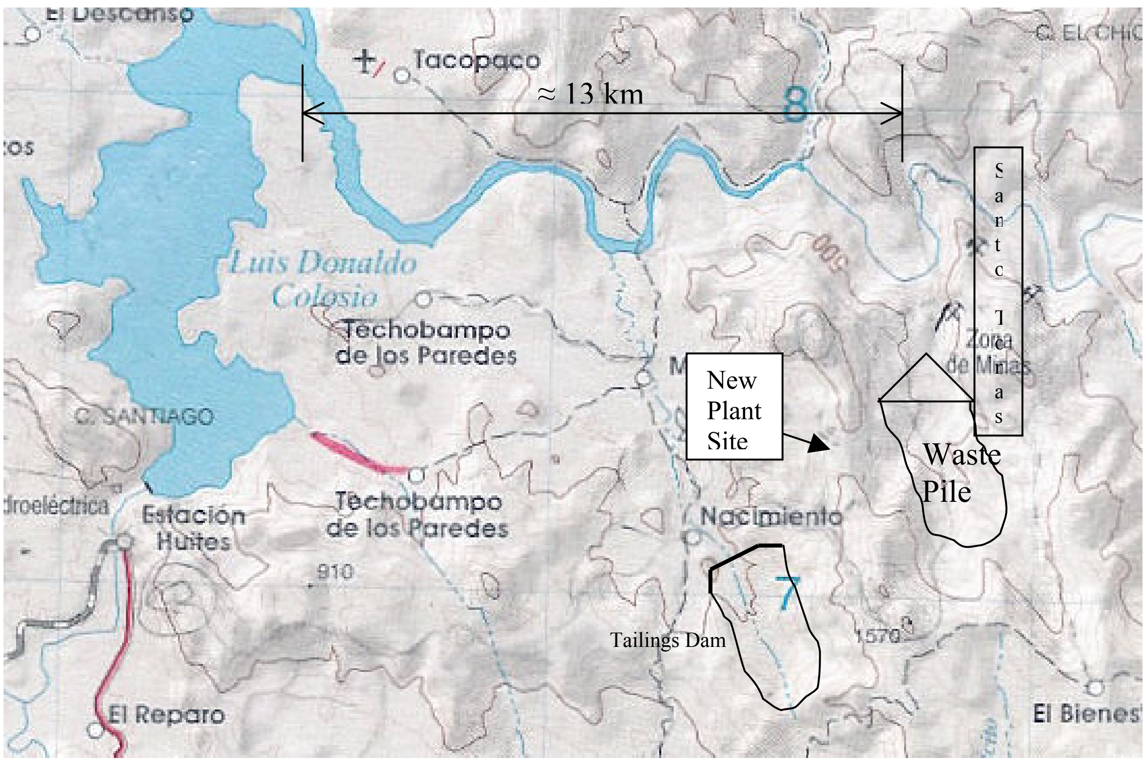
- **Plant location**

The original site proposed for the concentrator plant is at elevation 480 meters, well above the maximum level of the Huites reservoir. However, the horizontal distance to a branch of the eventual lake is only about 350 meters. Although the restriction for construction near a public-use body of water is only 10 meters, it could be difficult to obtain the necessary permits to build a large industrial installation so near the reservoir. The following picture offers a view of that site. As a result of the recent site visit, a different location for the plant site was identified in the area called Palmarito. This seems better suited for construction of the plant from the point that the site would allow gravity flow of tailings to a new location away from the Huites reservoir and would also allow better control of accidental spillages from the plant. This change would require a more complex system of conveyors from the primary crusher to the plant, but the decreased environmental risks, the easier plant construction and the elimination of pumping of tailings could offset the additional conveyors cost. The cost of these changes was not estimated and are not included in the updated capital cost estimate that includes the original items in the flowsheet. This is presented only for consideration.



- **Tailings Dam**

In the original study the tailings dam was to be located in the canyon southwest of the minesite. The toe of the dam was almost at the containment dam, which in turn was located almost at the high level of the Huites reservoir. From the regional maps provided by Aztec, it appears that an area close to the town of Nacimiento appears to be more appropriate for a tailings dam rather than in the narrow canyon southwest of the minsite. This location, as well as others, would need to be studied for suitability by the appropriate disciple consultant



3.2 Electric power

During the site visit it was learned that a project being developed in the area will take the power available in the Los Hornillos substation, close to El Fuerte. The Choix substation is not big enough for the project's requirements. Consequently the Santo Tomas project will be required to negotiate the power supply from El Fuerte substation, or probably from San Blas. Another possibility could be to take power from the new Huites plant. Further description in Section 6, Surface Facilities.

SECTION 4

METALLURGICAL TESTWORK

4.0 GENERAL

The Santo Tomas property has been explored for several years and several studies and metallurgical tests have been performed. The section below listed as "previous testwork" focuses on the early work done by Lakefield and others (See 1994 Study) for production of flotation concentrates of the ore.

In the second part of this section titled "present testwork", which was conducted in 1994 by Mountain States Research and Development, Inc., the tests provided similar results to those obtained by Lakefield. Focus must be made on the ability of the concentrator circuit to maximize copper recovery from the ore and produce a high grade copper concentrate for sale.

Bateman was not involved with any of the previous testwork and has only extracted that information from the work already presented to show the likely concentrate grade and recovery.

Leaching of concentrates is not addressed in this work per se, but a summary of current concentrate leaching technologies are identified. Only two of these technologies are in commercial production.

4.1 PREVIOUS TESTWORK

The information received from Exall (1994 Prefeasibility Report) has been used to identify the copper recovery associated with producing copper concentrates suitable to be sold on the open market.

- Recovery of Copper, by Lakefield Research of Canada, March 1975

Five tests of a sample assaying 1.0% copper, 2.48% iron and 1.51% sulfur were performed to determine the quality of concentrate that could be produced from the Santo Tomas ore and the results are summarized in Table 4.1, below. The feed sample was minus 10 mesh. For the first three tests the primary grind time at 60% solids was varied and the rougher concentrate was cleaned in three stages. In the last two tests the rougher concentrate was reground for 20 and 40 minutes before cleaning. The source location of the sample is not specified in the report. However, it is assumed to come from the north pit area.

Table 4.1
Grinding and Flotation Test Results, 1975
by Lakefield Research

Test No.	Grinding Time Min.	Grind % Passing -200 Mesh	Rougher Con. % Cu	Primary Recovery % Cu	Final Con. % Cu	Final Recovery % Cu
1	30	72.8	6.04	95.4	17.72	90.5
2	20	55.7	7.69	94.3	16.31	89.3
3	40	83.9	6.33	95.7	19.86	90.5
4	30 + 20		5.87	95.6	22.88	89.1
*5	30 + 40		6.90	93.9	28.45	88.2

*The final concentrate for test No. 5 contained 0.055 oz/t Gold and 2.71 oz/t Silver.

The Lakefield report suggests that a grind of 60% minus 200 mesh, could be sufficient to obtain a copper recovery of 95%, which is substantiated by the results above for the rougher flotation, however the final recovery would be about 90%. In general, the response of the ore to grinding and flotation was positive. The power consumption was 13.4 kwh/t for primary grinding and regrinding, test No. 5.

- Flotation Test, Quartz-Monzonite, by Comision de Fomento Minero, November 1991

This test was made with samples from 11 diamond drill holes, separated in high grade, >0.6% Cu, and low grade, < 0.6% Cu. The detailed drill hole information is included in the original report, Appendix II.

The **high grade sample averaged 0.87% Cu**, 2.12% Fe and 0.008% Mo. The reagents dosage was varied on 8 tests. With grinding time of 14 minutes, 65 to 75 passing 200 mesh, and rougher flotation of 5 minutes the rougher concentrate had a grade between 14.8% Cu with 84.8% recovery to 18.5% Cu with 79.4% recovery. Two tests included 5 minutes of regrind and one step of cleaner flotation producing a concentrate with 22.0% Cu and 81.1% recovery. Two steps of cleaning increased the concentrate grade to 23.2% Cu with 80.7% recovery.

The **low grade sample assayed 0.34% Cu**, 2.16% Fe and 0.008% Mo. The reagents dosage was varied in 6 tests. The rougher concentrate ranged from 3.92% Cu and 74.93% recovery with 14 minutes of grinding and 5 minutes of flotation to 7.5% Cu and 62.1% recovery with 10 minutes of grinding and 3 minutes of flotation. Three stages of cleaner flotation produced a concentrate with 23.8% Cu and 61.5% recovery. The recovery results for the low grade ore are significantly lower than that for the higher grade ore.

- Flotation Test, Andesite, by Comision de Fomento Minero, November, 1991

The Andesite zone samples from the same drill holes used in the tests for quartz-monzonite were used to test the flotation response of high (>0.6% Cu) and low (<0.6% Cu) grade andesite.

In the best of five tests, high grade andesite assaying 0.84% Cu produced a concentrate with 23.4% Cu and 84.6% recovery, using 30 minutes grind, 75% minus 200 mesh, 5 minutes of rougher flotation and two stages of cleaning, at 2 minutes each. The copper recovery in the rougher concentrate was 91.8%.

For the low grade material, that assayed 0.38% Cu, the best of nine tests produced a concentrate grade of 19.0% Cu with 70.0% recovery.

- Flotation Test, by Consejo de Recursos Minerales, December 1993

In this test 500 kilograms of ore assaying 0.68% Cu were milled for 15 minutes and floated for 2 minutes to produce 14 kilograms of rougher concentrate for subsequent bacterial leach testing, by others, with the following results:

Table 4.2
Chemical Analysis of Flotation Test

Element	Cu	Fe	Pb	Zn	Au	Ag
Grade, %	10.6	28.8	0.09	0.20	1.163 g/t	32.05 g/t
					0.041 oz/t	1.12 oz/t

4.2 PRESENT TESTWORK

- Flotation and Leaching, by Mountain States Research and Development, Inc. March 1994

Five core samples from diamond drill holes STD-02, 08, 20, 27 and 43 were provided by EXALL to MSRDI for mineralogical examination and metallurgical testing. All these drill holes are located in a North-South axis, 850 meters long, through the center of the proposed north pit, approximately at coordinates E778900 from N2976800 to N2977650. A weighted composite corresponding to the hole interval produced 39.5 kilograms of sample containing **0.56% copper** of which 92% was determined to be chalcopyrite, plus 5% acid soluble copper and 3% cyanide soluble copper. The composite contained approximately 0.003 oz/t gold and 0.10 oz/t silver. The sample was crushed to minus 1/2 inch.

The results of four flotation tests to produce a rougher concentrate are presented in Table 4.3.

Table 4.3
Flotation Test Results

Test No.	Time, mins.			Rougher Concentrate Grade, % Cu	Recovery %
	Grind	Condition	Float		
1	15	10	10	14.18	90.57
2	10	10	7	15.35	85.96
3	20	5	7	20.00	94.83
4	15	5	7	17.64	91.53

These tests show very similar results to those obtained previously by Lakefield.

4.3 FUTURE TEST WORK

Bond Mill Index – This test work is required to determine the hardness of the ore so that the proper grinding materials can be used in the mills and that the mills are designed for optimum power draw.

Flotation Tests – Further test work needs to be conducted on low grade ore, < 0.4 to determine actual recovery and concentrate grade. The current reported average mine grade is 0.368%. Precious metal recovery must also be addressed during the flotation test work.

Oxide Ore Leaching Characteristics – It was brought to Bateman's attention the there is a 15 to 20 million tonne oxide cap on top of the sulfide ore body. If this is to be processed, a series of column leach tests are required to determine the leachability and acid consumption of the oxide ore and identify the potential PLS (pregnant leach solution) grade obtainable for proper design of a solvent extraction / electrowinning facility.

Bateman can provide a comprehensive testwork program for primary and oxide ores should Aztec Copper require this assistance.

4.4 CONCLUSIONS

4.4.1 Grinding and Flotation

Although a relatively fine grind at 200 mesh is required, the Santo Tomas ore responded favorably to flotation using common reagents. The test results; although limited in nature, indicate that the Santo Tomas ore would be amenable to beneficiation in a conventional concentrator to produce copper concentrate for smelter treatment. All the testwork indicates that a 28% copper concentrate can be produced but the recovery of the copper can be expected to be 88 – 90% for ore grades greater that 0.6% copper and 70 – 75% for ore grades less than 0.6%. Further testwork needs to be conducted to maximize copper recovery from the ore and produce a high grade copper concentrate for sale.

4.4.2 Leaching of Ore

The Santo Tomas sulfide ore did not respond to direct leaching, because of the high chalcopyrite content and low oxide and chalcocite content.

4.4.3 Concentrate Leaching Test

Since the original test work done, which was done by BRISA and Mountain States Research and Development, Inc. on concentrate leaching, various concentrate leaching processes have been developed and some discontinued. A summary table reviewing the current process technologies is shown in Appendix B. There are only two commercial processing plants at present. One is a chalcocite leaching operation in Australia and the second is an autoclave process used to leach chalcopyrite concentrate in Baghdad, Arizona. All the others are pilot or demonstration size plants. Only two of the processes address precious metal recoveries from the concentrate as an integral part of the process. The others produce a residue of precious metals and matrix which must be sent out for additional processing. It is not advisable to continue concentrate leaching tests until after the quality of the concentrate has been determined from the "Future Test Work" referred to in sub section 4.3 has been completed.

SECTION 5

PROCESS

5.0 GENERAL

Based on preliminary discussions with Aztec personnel, a production rate for the Santo Tomas deposit was initially established at 60,000 tonnes per day through a conventional sulfide ore concentrator. The end product of the concentrator would be the production of saleable concentrate which is the focus of this study.

As an alternative to selling the concentrate directly, production of copper cathode via leaching of the concentrate, solvent extraction and electrowinning was not considered in this study. Leaching of the concentrate allows production of a lower grade concentrate which raises the copper recovery by flotation and eventually produces a more saleable product albeit at a higher capital cost. The additional capital cost required for the leaching-solvent extraction-electrowinning facilities may be partially offset by the elimination of the equipment necessary to clean the concentrate from a rougher grade to final concentrate. This could be done if future test work for chalcopyrite yields adequate recoveries. In the last few years copper leaching has gained favor as a processing means because of the reduced environmental impact that leaching generally creates and also because leaching in some cases is more cost effective than traditional crushing, grinding, concentrating, smelting, and refining. Chalcopyrite concentrate leaching in particular has been the subject of intense interest and some new technologies currently under demonstration are expected to be applied on a commercial scale in the near future. One difficulty with acceptance of leaching is that the overall recovery could be lower than via beneficiation. If the driving force behind a project is total tonnes produced, then beneficiation will have an advantage, but if the driving force is economics (return on investment) then more often than not some form of leaching may be the best alternative.

An alternative that has been proposed at this stage is leaching and processing of the oxide ore part of the overcap that will be mined in the early stages of the pit development, prior to, or in parallel to, the operation of the concentrator. The facilities required to produce copper cathode from oxide leaching could eventually be used to process leach solutions resulting from concentrate leaching. Most of the unit processes that could be applied to the Santo Tomas are discussed in the following sections. These sections are:

- Concentrating
- Leaching
- Solvent Extraction
- Electrowinning

5.1 CONCENTRATING

5.1.3 Primary Crusher

For primary crushing there are three distinct types of machines, the jaw crusher, the gyratory, and the impact crusher. All three have their own operating characteristics. However, only the gyratory crusher is available in a sufficient range of sizes to accommodate rates of 600 to 6,000 tonnes per hour. The gyratory crusher has the largest unrestricted feed opening when compared with other crushers, which is important when considering the size variations which can be encountered in run-of-mine ore from an open pit mine.

A 60" X 89" gyratory crusher would have the capacity of 3,238 tonnes/hour at an open size setting of 190 mm (7-1/2") which, with 19 operating hours, yields 61,552 tonnes/day, which is well within the capacity of a 60,000 tonnes/days concentrator.

Almost all of the gyratory crushers manufactured today can be installed as in-pit mobile or semi-mobile units. However, capital costs for these installations escalate rapidly and cannot be justified for the early stages of development of the pit. Consequently, in the original Pre-feasibility Study, Bateman had located the primary gyratory crusher near the pit perimeter.

It is customary to include a coarse ore stockpile at the terminus of the primary crusher discharge which acts as a surge pile ahead of any down stream processes. The purpose of this coarse ore stockpile is to allow for programmed maintenance or emergency repairs in the primary crushing area. These stockpiles are comprised of both live and dead storage, and ideally the greater the amount of live storage, the greater the amount of down-time which can be tolerated in the primary crusher. Stockpile sizes are site specific in terms of available area and topography, however, for a 60,000 tonnes per day facility a 40,000 - 50,000 live capacity stockpile would be adequate. For the SAG option the reclaim conveyors from the coarse ore stockpile would feed the SAG mills directly. If conventional three-stage crushing is required, the coarse ore reclaim conveyors would feed secondary crushers.

5.1.3 Secondary and Tertiary Crushers

The selected option for milling is semi-autogenous grinding (SAG), which does not require secondary and tertiary crushing. However, the amenability of the Santo Tomas ore for SAG milling should be confirmed by future test work. The following description of equipment, which is not included in current equipment list, would apply in case the ore proves to be not amenable to SAG.

A cone crusher is generally used in secondary and tertiary crushing applications, either in closed circuit or in open circuit with vibrating screens. Where the gyratory crusher reduction ratio is 8:1, the standard cone crusher is 6:1 and the short head cone 4:1. Depending on the ore-hardness, these machines can range in horsepower from 300 to 1,000.

An open circuit is the simplest configuration in secondary - tertiary crushing in which vibrating screens are used ahead of both the standard and short head cones in order to scalp finished material. A 60,000 tonnes per day crushing plant would include three 1,000 HP standard seven feet cone crushers and six 1,000 HP seven feet short head cone crushers.

The objective in using secondary and tertiary crushers in this application is to prepare feed for the grinding mills, either rod (90% - 3/4") or single stage ball mills (90% - 1/2").

As in the case with a primary crushing facility, the finished product from the secondary- tertiary circuit must be conveyed to a covered fine ore storage pile or silos located prior to the grinding mill bays in the concentrator. The end of the conveyor is generally equipped with a traveling tripper mechanism for ore distribution to the grinding sections, and reclaim systems are installed beneath the pile. Since the dead storage in an enclosed fine ore storage pile is not available for reclaim by bull-dozers, it behooves the designer to maximize the live storage capacity. Requirements for a 60,000 tonnes per day of ore are at least 50,000 tonnes, or 20 hours of operation. As in the case with the coarse ore stockpile, the fine ore storage building is site specific in that it should be located as close to the grinding mills as is practical in order to avoid excessive lengths in the reclaim conveyors feeding the grinding mills.

5.2 GRINDING

Grinding or milling of copper porphyry ores is a function of the ore hardness expressed as the bond work index, Wi. At a given work index the type of grinding mills can be selected based on the power required in grindability tests (expressed as W = KWH per short ton), to reduce a crushed ore to a size amenable to flotation.

Typical grinding scenarios in practice today are as follows:

- Rod-Mill/Ball Mill
  Three stage crushing followed by rod mills and ball mills.

- Single Stage Ball Mills
  Three stage crushing followed by single stage ball mills

- Semi-Autogenous Grinding

Primary crushed ore is used as grinding media, supplemented by a 5% - 10% ball charge, with the SAG mill being followed by ball mills with normal ball charges (40% - 45%).

Although the rod mill - ball mill configuration will insure a satisfactory grind to flotation, the limited availability in obtaining rods of adequate size and quality has directed the newer concentrators to single stage ball mill circuits. Some plants converted rod mills to ball mills after the introduction of fourth stage crushing (Vertical Impact Crushers).

The simplest operation is three stage crushing to -1/2", followed by single stage ball-milling when treating an ore of medium hardness (Wi. 9 - 11). This assumes that plant space requirements can be satisfied for a three stage crushing facility, coarse and fine ore storage, and multiple single stage ball mills.

Semi-autogenous grinding has its application with ores at a Wi. of 12 or higher, and particularly those deposits where the ore hardness is continuous or nearly so throughout the life of the mine. Extreme variations in the work index can play havoc with SAG mill operation resulting in erratic metallurgy. However, there are SAG plants operating satisfactorily where mine operations are able to maintain a uniform feed hardness by blending the ore from different zones of the mine. Assuming the data contained in the Lakefield Report regarding the Santo Tomas deposit (1975) is valid, then the 7 KWH/ton to grind thru 60% passing 200 mesh indicates a work index of 12.0, which is typical of the quartz monzonite poryphyries prevalent in the U.S. southwest.

With semi-autogenous grinding circuits there will be possible operating and capital cost savings over conventional crushing and grinding flowsheets. Since a SAG circuit eliminates secondary and tertiary crushing and any accompanying screening/conveying, a given throughput is possible in fewer pieces of equipment, reducing operating and maintenance cost. For 60,000 MTPD an arrangement of two SAG/two ball mill would be required. Power consumption with a SAG mill installation will always be higher as the variable speed drives adjust for changes in ore hardness which in turn varies the mill capacity. Prior to any decision regarding the applicability of semi autogenous grinding, extensive test work should be conducted, both in the laboratory and at pilot-plant level.

5.3 FLOTATION

Flotation as defined today includes the collection and distribution of the classified overflow from the primary or secondary grinding mills to a series of mechanical cells or columns. The objective is to produce a final concentrate of a suitable grade for subsequent by-product recovery or, in the absence of the latter, directly to smelting. The normal arrangement in the majority of copper concentrators is to initially produce a rougher concentrate which is generally reground, classified, cleaned and scavenged. The highest grade is obtained from the final cleaning stage, columns or mechanical cells, and the scavenger concentrate is combined with the rougher concentrate for additional regrinding. Dewatering of the rougher/scavenger concentrate is normally incorporated in order to produce a pulp high enough in solids to satisfy regrind mill feed requirements. Tailings from the rougher and scavenger rows normally report to thickeners for water recovery with the thickened slurry going to a tailings dam. Rougher tailings are directed to retreatment facilities when the rougher tailing still contains sufficient copper to justify the installation (Freeport, Indonesia and El Salvador, Chuqicamata, El Teniente, Chile).

Bateman's Santo Tomas Flow-sheet 00-FS-001 shows a typical bulk concentration circuit with a primary cyclone overflow slurry line between the grinding circuit and the flotation plant. The more common configuration is to have the grinding and flotation circuits under a common structure, however, where the topography does not permit this arrangement the only alternative is to separate the two facilities. Conveying primary cyclone overflow by gravity via a pipeline may not be a common practice, but in terms of quantities of solids it is very similar to tailings thickener underflow systems where the thickened tailing is conveyed by gravity via HDPE lines for considerable distances. Allowances must be made in the pipeline(s) sizing and slope since cyclone overflow solids will average 35% versus tailings thickener underflows at 50 - 55%. Regardless of the application, critical velocities must be considered in order to avoid the settling out of solids in the lines.

5.3.3 Rougher Flotation

For a 60,000 MTPD concentrator, the rougher flotation arrangement would be four rows of 10 each 1,500 ft.^{3} cells with a 2 x 2 x 3 x 3 arrangement. These flotation cells are the self-induced air type which eliminates the air blowers found in concentrators utilizing machines which require positive air displacement in the flotation process.

The performance of the 1,000, 1,500, and 3,000 ft.^{3} cells is well established in the industry. However, with bigger cells requiring fewer rows any major maintenance to any individual cell would require shutting down the entire row, as well as the grinding mills up-stream of this unit. Four rows of flotation cells, two rows of 1,500 cu. ft. machines per grinding line will allow for greater flexibility as well as equal metallurgy.

The rougher flotation criteria for a 60,000 tonnes per day concentrator at Santo Tomas is based on a 15 min. retention time at 30% solids in the feed and a weight recovery of 15% of mill feed in the rougher concentrate.

Rapid, rougher flotation unit cells are available for this application but the utilization of a fast flotation unit as a replacement for or in conjunction with conventional roughers in the Santo Tomas flotation circuit appears premature. Only when flotation testing has confirmed recovery should alternatives in rougher flotation be considered. The more conservative approach at this time is to consider conventional rougher flotation for any of the options under study.

5.3.2 Regrind Mill Circuit

The flotation circuit includes four vertical regrind mills to process the rougher concentrate from the primary flotation circuit and the scavenger concentrate from the concentrate cleaning circuit. The vertical mills will help produce a final concentrate for sale, according to the conventional flowsheet, while at the same time they provide the necessary flexibility to produce the fine grind feed required by most of the new bacterial oxidation and autoclave technologies to leach concentrates.

5.3.3 Cleaner, Recleaner, and Scavenger Cells

The 60,000 TPD capacity would require three rows of 7 - 1,000 $ft.^3$ cells as first cleaners and scavengers, with three rows of 3 - 500 $ft.^3$ cells ahead of the first cleaners as final or second cleaners. This configuration of in-line cleaning and scavenging facilities optimizes pumping as the regrind cyclone overflow can be fed by gravity to the first cleaner - scavenger, which only leaves pumping of the first cleaner concentrate to the head of the final cleaner. Final concentrate is pumped to the final concentrate thickener and the scavenger concentrate to the regrind mill circuit, together with the rougher concentrate.

As alternatives, column cells could replace the first and second cleaners. However, columns require a dedicated filtered high-pressure air supply for the column air injectors to function properly. The replacement of the self-induced mechanically agitated cells with columns is based mostly on economic factors. In new concentrators where columns are designed for cleaners, conventional scavenger cells are included for final tailing control.

5.3.4 Thickening, Filtering, and Drying

- Thickening

  Two concentrate thickeners have been included for the purpose of dewatering the final concentrate prior to filtering. The thickener sizes are a function of the surface area per ton of concentrate per day. In the absence of any settling test data, Bateman has used a conservative number of 9.0 sq. ft./ton/day, with 5 - 10 being the range for copper concentrates.

- Filtering

  Filtering of the final concentrate would be accomplished through the utilization of conventional disc filters.

  Provided the disc filter is properly sized and operated with an efficient vacuum system, the filter cake produced should be in the range of 8%

moisture. At this moisture content copper concentrates can be directly shipped to the smelters, via rail or by truck, thus avoiding the capital and operating costs and the subsequent copper losses associated with rotary driers.

Porcelin filters have gained much success in the past 5 years and should be evaluated for cost in the future.

- Tailings Disposal

  Thickening of flotation tailings at Santo Tomas would be accomplished using two 225 EIMCO Thickeners

  This thickener incorporates a specially designed baffled feed well with flocculent addition points. Flocculent or polymer addition is a "must" with these units, with which the required square feet of area per ton per day can be reduced from the normal 4 to 10 $ft.^2$/ton/day in the conventional tailings thickener to 0.4 to 1.0 $ft.^2$/ton/day.

  In the absence of any tailings settling rate data and to be conservative, a unit rate of 2 $ft.^2$/ton/day has been used in estimating tailings thickener sizes.

5.4 LEACHING

In the context of this study, leaching would apply to an initial process of oxide ore (see Section 4) and to a future process of concentrates. In the original study in 1994 several leaching methods and processes were discussed. Some of those have been proven to be un-economic and new processes have emerged. In this update study only the most currently feasible options are considered. Some methods or processes that would seem obvious, but that in reality are not applicable are mentioned only in order to justify their non-inclusion.

The various processes normally produce copper in a metallic form either as cathode, powder, or crystals. Processes producing copper sulfate have not been considered for Santo Tomas because the market for copper sulfate is limited in demand and very competitive. Some methods and processes are explained below. Short discussions of other processes and a summary table are presented as Appendix B.

5.4.1 Methods

Dump, Heap and Thin Layer Leaching of raw ore.

Dump leaching is probably the most common type, due primarily to the fact that most of the ore for leaching is produced as oxide and low grade secondary sulfide run-of-mine waste from open pit operations. The operating cost to produce copper in solution is relatively small. Heap leaching is differentiated from dump leaching by the method of construction. In more and more cases ore is specifically mined for leach. Typically, the ore is high grade material that has been crushed so as to yield fast recovery of the copper and the piles are also low in height because they are formed by loading with front end loaders or conveyors. Because of all the material handling, heap leaching has a higher capital and operation costs than dump leaching and is normally applied to high grade ores that can bear the added costs.

Thin Layer Leach was developed by Sociedad Minera Pudahuel (SMP) in Chile many years ago. The process is basically a heap leach of crushed and agglomerated ore. The agglomeration is normally carried out with high strength leaching solution so an acid cure step is the net result. As discussed under heap leaching, the use of such a process is limited to high grade ores that can pay for the extra material handling and crushing costs.

However, the ore from Santo Tomas will be mainly chalcopyrite, which is typically refractory to leaching and most probably leaching in dumps or heaps, with current bacterial oxidation techniques, but could be an economic proposition for the oxide ore that has to be mined as part of the stripping of the mine, and not for the main ore body.

Concentrate production is seen to be currently uneconomic, due to low head grades and the projected capital and operating costs for this process option, whereas bioheap leaching with a suitable bacterial strain adapted to chalcopyrite leaching may provide a viable alternative, as capital and operating costs are generally lower than the concentrator option.

Chalcopyrite heap leaching is of particular interest as it would allow for the initial treatment oxide or supergene copper deposits by conventional heap leaching methods, followed by the treatment of the primary sulphide ores, at a scale of operation not limited by the current low oxide reserves, which could potentially produce sufficiently low capital and operating costs as to present a viable project at current copper prices. This scenario is clearly subject to the leaching testwork primary ore providing acceptable copper extractions and a cash flow analysis with relevant capital and operating costs supporting this contention.

Agitated Leach.

Leaching in agitated tanks is a relatively high capital cost process and consequently is more applicable to bacterial leaching of concentrates, to be discussed later, than to leaching of raw ore.

<u>Autoclave Leaching of concentrates</u>

These type of processes are carried out in pressure vessels, normally with super-atmospheric oxygen pressure and moderately high temperatures.

- **Phelps-Dodge Bagdad**

  In April, 2003 Phelps Dodge, the world's second largest copper producer commissioned a pressure oxidation commercial/demonstration plant at Bagdad Arizona, which will treat about 136 tonnes/day of chalcopyrite concentrate to produce approximately 16,000 MTPY of copper cathode via conventional SX/EW. The project uses technology developed by Phelps Dodge and Placer Dome and it is the first to treat chalcopyrite concentrates. The process produces an excess of acid which requires the availability of a secondary leach system, such as oxide leach, to consume that acid. Otherwise, a green-field plant without that facility would require to neutralize the acid.

- **Mt Gordon Ferric Leach Process**

  This process is in successful commercial operation at Western Metals' Mt Gordon operation in Queensland, which produces about 50,000 tones/annum of high grade cathode copper from high grade chalcocite ore by ferric leaching and low pressure oxidation with oxygen addition. It incorporates a moderate feed grind of 75-106 microns, and copper recovery is by conventional SX/EW. Future plans include the treatment of chalcocite concentrates from lower grade ore, and the further development of the process to treat chalcopyrite concentrates.

5.4.2. Chloride Leaching

- **<u>Intec</u>**

  The Intec process was developed in Australia with the support of a consortium of major copper producers. The process utilizes a chloride/bromide solution to leach concentrates at 85C and atmospheric pressure. High purity copper is electrowon in a diaphragm cell to produce the copper without the necessity of a solvent extraction step. Impurities are precipitated with lime. Silver is precipitated with mercury, and then produced as a chloride for further processing. Gold is recovered on to carbon for further processing. The crystalline copper metal that is produced is immediately washed and dried in an inert atmosphere to prevent oxidation of the metal which is then fed into a "conform" machine which extrudes the copper into a marketable shape. After successful pilot plant and demonstration plant campaigns, Intec are assessing opportunities for commercial plants, including Ivanhoe's Turquoise Hill Copper/Gold Project in Mongolia. A training/demonstration plant was planned to be built in Japan.

According to the developers of the process, the advantages of using the Intec Process for base metals producers is its ability to recover high purity base and precious metals at a capital cost as low as $US1,400 per tonne of annual copper production capacity, compared with $US3,500 per tonne for a traditional concentrator/smelter, while also reducing the heavy environmental impacts associated with conventional smelters. The Intec process has the potential to improve the economics of new projects in remote areas, with high transportation costs to a smelter, and also could have an advantage over other leaching processes if the concentrate contains important precious metal values to be recovered, which could be lost with conventional or bacterial leaching methods.

5.4.3 Ferric-Bacterial Leaching

Ferric-bacterial processes are based on bacterial oxidation of sulfide minerals which renders the minerals soluble in mild sulfuric acid solutions which can in turn be treated by conventional copper solvent extraction and electro-winning methods.

- **Alliance Copper**

  Codelco, the world's largest copper producer have teamed up with major producer BHP Billiton to form Alliance Copper dedicated to developing and applying bio-technology for copper and molybdenum ores and concentrates. The first project is a prototype bio-leaching facility near Chuquicamata, Chile to treat 77,200 t/a concentrate and produce 20,000 tonnes/annum of cathode copper, with a view to building a commercial facility for the nearby Mansa Mina Project. Billiton had previously run a pilot plant at Chuquicamata utilizing their BioCOP technology.

- **BacTech/Mintek Process**

  BacTech and Mintek have jointly developed their agitated-tank bio-leaching process up to demonstration plant level at the Peñoles Research Center at Monterrey, Mexico. The Peñoles demonstration plant is designed to produce 500 kg/day of cathode copper via conventional SX/EW and operated for six months in 2000-2001. A 25,000 tonnes/annum copper/zinc commercial plant is under consideration.

5.5 Possible Alternatives

Leaching of Santo Tomas concentrate could be an option if by the time the project is mature the technologies to leach chalcopyrite have been commercially proven. At this time, the only hydrometallurgical alternatives considered feasible are:

a) Short/medium term Leaching-SX-EW of limited oxide ores reserves.
A study of heap leaching of the oxides should be done during the feasibility study, since even for a relatively small tonnage, compared to the total sulphide ore reserves, this process could provide additional benefits to the project, since the oxides and waste have to be mined in any instance. Additionally, the initial SX-EW plant could be expanded to process leach solutions produced by any future concentrate leaching process.

b) Autoclave leaching-SX-EW of chalcopyrite concentrates. This process seems the most likely to have commercial success by the time the Santo Tomas is ready to proceed to full-feasibility study.

c) Agitated, ferric-bacterial, Leaching-SX-EW of the concentrates.

   The developers of the existing and developing processes to leach secondary sulfide concentrates claim that the technology can be easily adapted to process chalcopyrite concentrates. As such, the process is worth reviewing during the feasibility study.

d) Chloride leaching, such as Intec, which could be attractive if the ore is confirmed to contain appreciable amounts of gold and silver.

e) Leaching of both the oxide and primary sulfide ores by dump/heap leach techniques using newly developed bacteria species for chalcopyrite may offer an approach that provides better economics than more conventional alternatives. This work has been developed for nickel sulfides in Australia. See appendix B

5.6 SOLVENT EXTRACTION

Solvent Extraction (SX) for copper has been around for over thirty six years and the technology has been improving steadily, with the biggest strides being made in the development of the reagents. Designs of the processing facilities have evolved over the years along similar basis in different regions of the world and therefore the differences between SX plants from one property to another are quite small. Construction methods and materials have changed slowly in efforts to reduce the capital costs but basic design details and plant layouts are remarkably similar.

The purpose of the solvent extraction (SX) plant is to selectively extract copper ions from a dilute, impure, pregnant leach solution (PLS) into an organic solution or phase, and then transfer those ions out of the organic phase and into a concentrated, purified, electrolyte suitable for electrowinning. The transfer of copper ions into and out of the organic phase is a reversible reaction which is controlled by the sulfuric acid content of the aqueous solutions.

A typical SX plant is divided into two process sections; extraction and stripping, which are distinguished by the direction of the chemical reaction. In the extraction section, the relatively low acid content of the PLS allows the copper ions to be "extracted" from the PLS into the organic phase. In the stripping section the reaction is reversed when the organic phase is contacted with high-acid electrolyte and the copper ions are "stripped" from the organic phase into the electrolyte.

The organic phase is a solution of an extractant reagent diluted in a high-flash point kerosene. The extractant is a specifically formulated chemical that selectively extracts copper ions and rejects other ions. The rejection of other ions is not 100% and is affected by the copper loading. Consequently, copper extraction is purposely held to about 90% to insure that an excess of copper ions are always available which in turn will minimizes the transfer of other ions.

Generally, after extraction of the copper from the PLS, the barren leach solution (raffinate) is discharged to a raffinate pond and is eventually returned to a leach heap or process as leaching solution. Therefore, the 10% copper purposely left in the raffinate will constitute a recycle that will be recovered at the end of the project life.

The SX plant consists of mixers and settlers. The actual copper exchange takes place in the mixing compartments where the aqueous and organic solutions are put in contact. In the primary mixing compartment, the pumping mixer does two jobs as the name implies, mixing and advancing solutions from one stage to the next. Secondary mixers provide additional mixing time. The settlers are tanks where the solutions are allowed to separate. A coalescence enhancement channel between the mixer and settler will begin the process of phase separation between the aqueous and organic. The aqueous and organic phases separate into two distinct layers before reaching the discharge launders located at the end of the settler adjacent to the mixers. The organic flows over a weir and into a discharge launder and the aqueous flows out of the settler from the bottom and into a level control column that is used to regulate the interface level in the settler.

5.7 ELECTROWINNING

The purpose of the electrowinning (EW) plant is to produce pure copper (LME Grade) cathodes for sale. The EW plant produces metallic copper from an electrolyte containing copper sulfate and sulfuric acid by passing a direct electric current from an insoluble anode through the electrolyte to a stainless steel cathode mother blank. Stainless steel mother blanks are used to provide full plate cathodes for two principle reasons: Cathode quality and current efficiency. Current efficiency relates not only to power savings, but also to labor and maintenance savings. When copper is deposited on the surface of the cathode, oxygen is liberated on the surface of the anode and a portion of the water is broken down into oxygen and hydrogen. The hydrogen combines with (SO_4) to make acid (H_2SO_4) and the oxygen bubbles out of the cell.

Theoretically, for every 843.3 ampere-hours of power consumed, 1 kg. of copper is produced in each cell. However, the current efficiency is less than 100% so somewhat less copper is produced.

The electrical power used in excess of the theoretical power required produces heat in the cells which is absorbed by the electrolyte. Therefore, the electrolyte leaving the tankhouse circuit is normally passed through a heat exchanger where the excess heat is transferred from the spent electrolyte leaving the recirculation tank to the rich electrolyte entering the recirculation tank.

The plant has been designed to operate at about 24 amps./ft^2(260 amps./m^2) which should make it possible to produce a smooth cathode with few impurities.

The finished cathodes are harvested by stripping the copper deposits from the cathode blanks. The harvest cycle is normally 7 days; however, the exact pulling and charging schedule can vary according to the production rate and cathode weight desired. This flexibility is another advantage of using permanent cathode blanks. After stripping the cathode deposits from the blanks, the cathodes are normally stacked, sampled, weighed, and put in storage for future sale, and the clean blanks are returned to the plating cells.

SECTION 6

SURFACE FACILITIES

6.0 SITE LAYOUT

The general site original plan is shown on drawing 00-GA-001 in Appendix C at the back of this report.

The ore body lies along a steep, narrow ridge that runs approximately north to south. The north end begins at the Fuerte River at elevation 180 meters. The south edge of the ore body at the highest point is over 1,100 meters in elevation. The ore body is about four km long and about one km wide. However, the maximum level of the Luis Donaldo Colosio (Huites) water reservoir built in 1995, at 287.5 meters, restricts the extent of the minable reserves. For the purposes of this study, 300 meters is considered a safe elevation for the possible location of facilities, access roads, etc.

It is likely, depending on the cut off grade, that there will be a north pit and a south pit. The north pit will probably be mined first and the south pit second. The initial waste pile for the north and south pits, containing absolutely no metal values, could be located to the east below the mine elevation and adjacent to the pit limit, open towards the river. Average haul distances from the pit limit to the initial waste rock pile would be about one half km. Chalcopyrite leaches very slowly, even under induced conditions. Nevertheless, over a long period of time the waste pile could produce acid mine drainage, a serious potential problem given the nearness of the Huites Reservoir. Consequently, once the initial stripping of the mine reaches the mineralized zone it will be necessary to place the waste and low grade ore in a long term pile to the south-east of the south pit, where the tailings dam was located in the original study. This location is a box valley, open only towards the river and reservoir and it will be possible to monitor and control any future acid drainage.

The primary crushing plant will be located in or at the edge of the pits and will be relocateable. The plant site will be located across the valley to the west of the mine, at an approximate elevation of 900 meters between the Cerro Tojinahui and Cerro El Cobre, around the area of Palmarito, a location that seems better with respect to available surface than the site proposed in the original study, which was very close to the river/reservoir. This would necessitate a long conveyor system from the primary crusher to a coarse ore pile near the plant, but the advantages of easier plant construction and the reduction of the permitting risks for the waste pile and the tailings dam associated with the Huites reservoir will offset the additional conveyors cost. Moreover, the government environmental and safety regulations could restrict the distance from the reservoir at which any industrial installation can be located.

It is proposed that the tailings will be deposited into a tailings dam located south-west of the proposed plant site, near the town of Nacimiento. The dam will have a final elevation of 600 meters, with the toe of the main wall at 460 meters, or an ultimate wall height of approximately 140 meters. At the above elevation the dam will be capable of containing approximately 400 million tonnes of tailings, sufficient for about half the years of operation of the project.

The availability of the selected tailings area needs to be investigated further.

6.1 UTILITIES

The main substation will be located at the southwest corner of the plant site where the high voltage powerlines will arrive. A lower voltage line will deliver power to the mine substation located at the edge of the mine pit near the crushing plant.

Water for operation will be obtained from several sources. Approximately 60% of the process water could be reclaimed from the tailings pond, thickeners and from mine dewatering, if necessary. Make up water will be pumped from a pumping station located near the plant site in the Huites Reservoir, if permitted.

Water storage tanks will be located above the plant site and mine for both fresh and reclaimed water. Fresh water will be taken from the top half of the tanks only. Fire water for the site will be taken from the bottom of the fresh water tanks.

A diesel fire water pump will also be located at the water supply station in Huites Reservoir. Hydrants will be located strategically around the site. Fire extinguishers will also be located in appropriate locations. In addition the water trucks for the mine and site roads will be equipped with hoses that can be used to fight fires.

Sewage disposal will be by a leach field located upstream of the tailings dam in an area where suitable soils exist.

6.2 BUILDINGS

Major buildings on site will include the following:

- Main Concentrator
- Primary Crushing Building
- Mill Building
- Mine Shop/Warehouse
- Plant Site Warehouse
- Administration
- Changehouse
- Security and First Aid
- Main Substation Electrical Room
- Mine Substation Electrical Room

The main concentrator building will include a laboratory. Electrical control rooms and equipment rooms will be located throughout as required.

6.3 SITE GRADING

Plant site roads, parking and storage areas will be constructed of gravels produced on site. These areas will require occasional grading and periodic water to control dust and reduce wear. Site drainage will be at a minimum of 2% slopes to collection ditches.

The plant site ditches will discharge downstream to the tailings pond.

6.4 SECURITY

Access roads to the site will be controlled by gates. The main access road gate will be opened and closed from the security/first aid building. Other less frequently used roads will remain locked most of the time.

The plant site and substations will be enclosed inside an 8 ft. high cyclone fence with a barbed wire top strand. Security guards will man the project 24 hours per day.

6.5 COMMUNICATIONS

It has been assumed that telephone to the site will be by microwave sending and repeater units. On-site communications will be by "hard wire" or by radio.

6.6 STORAGE AND LOADOUT

Fuel and reagents will be received at the warehouses and stored in appropriate containers or storage yards. Concentrate will be loaded into haul trucks for transportation to the Topolobampo port or it can be transferred into train cars if a rail spur was built for the project.

If cathode copper is produced by leaching oxides it will be loaded onto trucks by forklift at the electrowinning building for shipment to customers. Accommodations will also have to be available for unloading and storage of sulfuric acid for leaching of the oxide.

SECTION 7

TAILINGS DISPOSAL

7.0 TAILINGS DISPOSAL

A possible new site for the tailings dam is located southwest of the proposed plant site, within a small valley formed by the Hacienda creek, southeast of the town of Nacimiento. Please refer to Appendix C drawing no. 00-GA-002. This would place the tailings pond approximately 2.4 km southwest of the proposed plant site. It would also place the tailings dam about 7.0 km away from the Huites reservoir, compared with 300 meters in the original study. *The availability of this proposed area needs to be investigated.*

Cerro Saucillo would serve as center anchor for a two-section wall, to close off the valley at the 600 meter elevation, with the toe of wall at 460 meters, or an ultimate wall height of approximately 140 meters. At the above 600 meter elevation the dam would be capable of containing approximately 400 million metric tons of tailings, sufficient for about half the years of operation of the project. *Other tailings disposal areas will need to be explored in the future.*

The tailings will flow by gravity to the pond. A small containment/monitoring dam would be constructed downstream of the tailings dam.

Depending on the quality of the tailings to be produced by the Santo Tomas concentrator, it may be possible to construct the dam walls with tailings as is done at many of the mines in the Southwestern United States. The tailings from the plant site pass through a series of cyclones located on top of the starter dam. The cyclones separate the tailings into a sand fraction and a slimes fraction. The sands (underflow) form the tailings dam face and slimes (overflow) form the slimes pond upstream.

Using centerline construction, the sands (cyclone underflow) will be deposited over the starter dam, making up the mass of the tailings dam. The centerline construction will be designed to provide a 2 m freeboard above the tailings. The only restriction to the angle of deposition is that the toe of the dam at final height should not encroach upon the containment dam impoundment. The sands will be placed at a slope of approximately 3:1. This would allow the tailings dam to reach an elevation of 600 m and have plenty of free space for monitoring purposes between it and the containment dam.

The containment dam would be placed down stream from the starter dam. Throughout the life of the mine, it will serve as an emergency protection barrier between the tailings dam and the towns of Nacimiento, Mezquite Caido and the Huites reservoir. If for any reason there is a spill, leak, or overflow of tailings, the containment dam will temporarily stop the leak from reaching the reservoir. The released fluids can then be pumped back to the dam.

The proposed tailings dam site is flanked to the east by the Cerro El Cobre and Cerro El Chileno, at about 1400 meters elevation, which could pose a problem with excessive water being collected in the tailings dam during heavy rainfall. The hills to the south and west, at about 700 meters in elevation constitute a natural divide and would not contribute significantly to the problem.

In order to prevent large amounts of watershed from collecting in the tailings dam or the containment dam a diversion ditch will be constructed around the east side of the tailings area. This ditch will discharge downstream of the containment dam, to the Arroyo Grande and hence to the Huites reservoir. Water that is impounded within the tailings dam will be pumped back to the plant site for process use. Excess water, if any, will be evaporated within the pond. None of the recycled water will be discharged into the Huites reservoir.

SECTION 8

INFRASTRUCTURE

8.0 GENERAL

The infrastructure required for the Santo Tomas mine will basically consist of the following major facilities:

- Access Roads
- Powerline
- Water Supply

A project location map is shown on Figure 1.2 of this report.

8.1 ACCESS ROADS

The general access to the mine site is from the intersection of the International Highway (Mexico 15) at Los Mochis, 150 km by paved road to the town of Choix. The topography in the area of the project was evaluated to determine a suitable location for the access road from Choix to the plant site. The Chihuahua-Pacifico railroad from Chihuahua to the Topolobampo port in Sinaloa runs approximately 15 km west of Choix through the town of Loreto (see Section 10).

The access road to the plant/mine will consist of 15 km of new road, 9 km of reconditioned road, and 34 km of road to maintain. These distances were measured by car during the site visit and verified by topographical maps of the area provided by Aztec. This road will be used to haul workers and materials to and from the site and concentrate from the concentrator.

The access road will consist of 4 sections:

Section 1 : From Choix to Tasajeras (9 km)
Section 2 : From Tasajeras to Nacimiento (7 km)
Section 3 : From Nacimiento to the plant site (5 km)
Section 4 : From the plant to the mine (6 km)

Section 1

The existing road from Choix to Tasajeras is approximately 9 km of all weather dirt road. Some short sections require up-grading, and it will require routine maintenance during construction and operation of the plant.

Section 2

The road from Tasajeras to Nacimiento is roughly 7 km of fair weather dirt road. This section of road requires some grading and widening as well as continual maintenance.

Section 3

From Nacimiento, the plant access road will follow an existing footpath northward that will require major reconditioning. The path will be widened to 8 m. The road surface will consist of well graded gravel aggregate that is compacted to an 8" depth. A hillside drainage ditch and culvert system will be provided for means of water removal where necessary.

At a point around 1.5 km north of Nacimiento, the access road will leave the existing footpath and head east through the mountains. This section will be entirely new and require cutting, grading, and a gravel surface similar to the section preceding. Due to the mountainous terrain, the road will most likely require drainage and some guardrails, since it has to reach from an elevation of approximately 400 meters to 900 meters over a straight distance of 3.5 km.

The road should reach the area of Palmarito, where the new location for the plant is proposed.

Section 4

This section of the route is the mine access road. . From the plant site, the road will turn northeast and descend down the eastern face of Cerro Tojinahui at a grade of about 12%, to reach the site originally proposed as plant site, which is now being proposed for mine shop/warehouse. It will be an unpaved road 12 m wide with a drainage ditch on the hillward side and a 2 m high safety berm on the edge. The road to the mine will start at the mine shop/warehouse and ascend to the crushing plant at a slope of about 12%. Mine trucks and equipment may be moved between the shop and the mine on a one way basis. At other times smaller vehicles may use the road on a two-way traffic basis.

8.2 POWERLINE

At the present time there are no power lines on or near the Santo Tomas property. The 34.5 kV line that passes 4 km west from the proposed plant site is inadequate for the plant size under consideration. There is enough power generated in the region, as follows: Juan de Dios Batiz (Topolobampo) thermoelectric plant at 360 MW, Luis D. Colosio (Huites) hydroelectric plant at 422 MW and 27 de Septiembre (El Fuerte) at 59.4 MW. However, the main distribution lines do not reach close to the project site. The closest substation, at Choix, is rated at 1.5 MVA

The closest power substation large enough to handle the 53 MW power demand estimated in this update study is in San Blas. The power available in the Hornillos substation will be used by another project under development. The upgrading of the powerline from San Blas to Choix, and to the project site, or the construction of a totally new powerline is a matter for future discussion and negotiation with the Federal Power Commission (CFE).

An overland power line from the town of Choix to the proposed plant site will have to be constructed. It will basically follow the plant/mine access road up through Nacimiento. Only the structures themselves have to be within a close proximity to the road for maintenance accessibility. The total poleline distance from Choix is approximately 27 km.

An alternative could be to take power from the new Huites plant, which at about 17 km is closer to the project site than Choix. The powerline from this plant runs northwest, away from the project site, towards the State of Sonora. This alternative would require building a substation at Huites and a new powerline to the project site. Again, this is an option for future discussion with CFE.

The Federal Power Commission (Comision Federal de Electricidad, CFE) is in charge of operation and maintenance of power plants and powerlines. It is expected that an agreement could be reached where the initial investment on the main powerline would be made by the project and the actual cost would be recovered through lower power tariffs, since eventually ownership of the powerline would revert to CFE.

8.3 WATER SUPPLY

Fresh water supply for the project will be from the Huites Reservoir (Fuerte River). This supply requires a federal permit. A pump station will be located near the plant site. The station will allow for water supply at varying levels of the reservoir.

Storage and head tanks will be located at the mine and the plant site to allow gravity feed of water to the sites.

It is expected that the water from the reservoir will require treatment in order to produce potable water. Potable water tanks will also be located at the plant site and mine site.

It is conceivable that the Mexican government may provide assistance to the construction of the infrastructure under various Federal and State development programs that are intended to encourage employment and economic growth in Mexico generally, and specifically in rural areas similar to the area surrounding the Santo Tomas deposit. This needs to be examined.

SECTION 9

ENVIRONMENTAL AND SOCIECONOMIC

9.0 ENVIRONMENTAL AND SOCIECONOMIC

No work was done to update this section.

SECTION 10

MARKETING

10.0 CONCENTRATE

The Santo Tomas Project is located in a mountainous region where the cost of building a smelter/refinery complex is prohibitive. For this update study it was decided to consider only the production of concentrate for shipment to third party toll treatment facilities because the smelters in the original study have been closed down or have restricted input.

Since the ore grade is relative low 0.386% compared to other sulfide projects > 0.6%, the recovery, through concentration, can also be expected to be low ≈ 75% compared to >90%. Calculated concentrate quantities for 60,000 MTPD are estimated below:

60,000 MT of ore mined and milled per day

Mineable resources – North pit only – (restricted by the river) = 428,000,000 MT
Average copper grade of the north pit = 0.368%
Mine life north @ 60,000 MTPD = 20 years
Mineable resources (estimated) – South pit only = 283,887,000 MT
Average copper grade of the south pit = 0.309%
Mine life south @ 60,000 MTPD = 14 years

For copper grades above 0.6%, a copper recovery in concentrate = 88%
For copper grades less than 0.6%, a copper recovery in concentrate = 75%
All test work done previously indicated that the higher grade had better recovery.
Concentrate copper grade = 28%
Concentrate gold content (Lakefield test work) = 0.055 oz/ton (0.061 oz/tonne)
Concentrate silver content (Lakefield test work) = 2.71 oz/ton (2.99 oz/tonne)

Calculated concentrate production:

For copper grades above 0.6% = 243,000 tonnes per year (North Pit)
For copper grades above 0.6% = 204,000 tonnes per year (South Pit)

For copper grades less than 0.6% = 207,000 tonnes per year (North Pit)
For copper grades less than 0.6% = 174,000 tonnes per year (South Pit)

10.1 SMELTER LOCATIONS

The project is located at roughly the same distance of 700 to 800 km from the three copper smelters specified in the original study in Mexico. The one in Cananea, Mexico is closed down, Nacozari, located to the north, is operating has smelting capacity but how capicity is not known and San Luis Potosi, to the south, is small and has no capacity. Off shore concentrate shipments have been evaluated.

10.2 ELECTROWON CATHODE

If an SXEW plant were built to process the oxide ore overlying the sulfide ore body, electrowon LME Grade "A" copper cathodes would be the output. Estimate production is listed below:

Mining Rate = 10,000 MTPD (oxide)
Total Copper Content = 0.75%
Ore Reserve = 20 million tonnes
Heap Leach = ROM (run of mine) ore. No crushing required
Copper Recovery from Oxide = 75%
Oxide Ore Mine Life = 5.7 years
Cathode Production = 19,688 tonnes per year (21,700 tons per year)

10.3 TRANSPORTATION

The Chihuahua-Pacifico railroad from Chihuahua to the Topolobampo port in Sinaloa runs approximately 15 km west of Choix. A railroad spur could be built at Loreto or from Loreto to Choix, (See Figure 1), possibly with government support to service this project.

Railroad access to the west would allow overseas export through the Topolobampo port and also link with the main trunk line to Nogales for access to Nacozari.

Electrowon cathode would be loaded on to trucks and taken to the rail spur for transportation to the Topolobampo port or to Mexico City.

SECTION 11

PROJECT IMPLEMENTATION

11.0 PROJECT IMPLEMENTATION

No work was done to update this section.

SECTION 12

CAPITAL AND OPERATING COSTS

12.0 General

The capital and operating costs contained in this study were developed based on Bateman's experience and knowledge of other similar projects in northern Mexico, Arizona and Australia.

A preliminary order of magnitude (OOM) (±35%), new plant, capital cost estimate was prepared for the 60,000 MTPD concentrator. The methodology used to obtain the OOM estimate is summarized below:

- Update the costs of equipment from the previous equipment list by contacting appropriate vendors and suppliers.
- Adopted a traditional factorization method of calculating the commodity costs by using known percentiles or factors of the installed mechanical costs.
- The installed mechanical costs were derived from scale buildups.
- Commodity costs were derived from similar installations.
- All indirect costs i.e. EPCM, first fills, spares were determined by factors from industrial norms.

Infrastructure costs were determined by escalating previous.

Process plant operating costs were prepared based on confidential discussions with operators of large copper facilities in Mexico and other Bateman study work in the US and Mexico.

Mine operating costs were developed from Mexican mine operations costs and mining costs in Southern Arizona.

12.1 CAPITAL COSTS

Preliminary order of magnitude (± 35) capital cost estimates were developed for 60,000 MTPD by preparing equipment lists for mining and process equipment (see Appendix A). The updated prices for this equipment were determined by soliciting quotations from qualified vendors.

The capital cost estimate is shown in Table 12.1, for the 60,000 MTPD plant.

Order of Magnitude capital cost estimates for 90,000 MTPD and 120,000 MTPD were calculated from typical expansion size factors.

60,000 MTPD	=	$410 million dollars (US)
90,000 MTPD	=	$522 million dollars (US)
120,000 MTPD	=	$621 million dollars (US)

Table 12.1

60,000 MTPD CONCENTRATOR
CAPITAL COST ESTIMATE

		A	%	B	TOTAL	%
CODE	DESCRIPTION	SUPPLY COST	TOTAL A	ERECTION COST	COST A+B	TOTAL DFC
	Direct Field Costs					
001	Civil Works		0.00%	12,038,766	12,038,766	4.94%
002	Structural Steelwork	8,667,912	5.09%	1,300,187	9,968,098	4.09%
003	Platework & Liners	4,815,506	2.83%	963,101	5,778,608	2.37%
004	Conveyor Mechanicals included in Mechanical Equipment		0.00%		0	0.00%
005	Conveyor Steelwork		0.00%		0	0.00%
006	Mechanical Equipment	48,155,064	28.28%	7,223,260	55,378,324	22.74 %
007	Piping - IBL	2,889,304	1.70%	2,311,443	5,200,747	2.14%
008	Valves - IBL	433,396	0.25%	65,009	498,405	0.20%
009	Electrical	7,464,035	4.38%	1,492,807	8,956,842	3.68%
010	Instrumentation	6,019,383	3.54%	1,203,877	7,223,260	2.97%
011	Infrastructure		0.00%	5,297,057	5,297,057	2.17%
012	OBL Piping - Slurry Line	1,961,857	1.15%	490,464	2,452,321	1.01%
100	Civil Works P & G		0.00%	3,611,630	3,611,630	1.48%
110	Structural Steelwk Erection P & G		0.00%	390,056	390,056	0.16%
115	Platework & Liners Erection P & G		0.00%	288,930	288,930	0.12%
120	Mechanical Equipment Erection P & G		0.00%	2,166,978	2,166,978	0.89%
130	Piping & Valves Erection P & G		0.00%	712,936	712,936	0.29%
140	Electrical Erection P & G		0.00%	447,842	447,842	0.18%
150	Instrumentation Erection P & G		0.00%	361,163	361,163	0.15%
150	Infrastructure P & G		0.00%	1,589,117	1,589,117	0.65%
160	OBL Piping - Slurry Line P & G		0.00%	147,139	147,139	0.06%
170	Mining Equipment	77,170,825	45.33%	192,927	77,363,752	31.77 %
180	Allowance for Tailings Dam, Access Roads & Power Line incl P&G's		0.00%	31,000,000	31,000,000	12.73 %
200	Transportation of Equipment @ 8% of ex. works supply costs	10,026,071	5.89%		10,026,071	4.12%
210	Commissioning Spares @ 1.5% of mech eqt	722,326	0.42%		722,326	0.30%
220	First fill of lubricants @ 1.5% of mech eqt	722,326	0.42%		722,326	0.30%
230	Vendor assistance during constr & comm @ 2.5% of mech eqt	1,203,877	0.71%		1,203,877	0.49%

	TOTAL DIRECT FIELD COSTS	**170,251,881**	**100%**	**73,294,689**	**243,546,570**	**100%**
	Home Office & Indirect Field Costs					
300	Home Office Resources including Sundries	29,225,588	17.17%		29,225,588	12.00 %
310	Construction Management including Sundries & Site Estab	8,524,130	5.01%		8,524,130	3.50%
320	Commissioning Resources including Sundries	3,653,199	2.15%		3,653,199	1.50%
330	Home Office Sundries		0.00%		0	0.00%
340	Site Sundries		0.00%		0	0.00%
350	Audit - Allowance (Bateman Internal)	15,000	0.01%		15,000	0.01%
360	Consultants - Allowance (Mining, etc.)	2,435,466	1.43%		2,435,466	1.00%
	TOTAL H.O. & INDIRECT FIELD COSTS	**43,853,383**	**25.76%**		**43,853,383**	**18.01 %**
	TOTAL NET COST	**214,105,263**	**125.76%**	**73,294,689**	**287,399,952**	**118.01 %**
	Other Costs					
400	Interest & Surety Bond - allowance for Bond costs	431,100	0.25%		431,100	0.18%
410	Fee @ 10% of TNC	28,739,995	16.88%		28,739,995	11.80 %
420	Insurance - allowance @ 1.5% of DFC	3,653,199	2.15%		3,653,199	1.50%
430	Contingency @20% of TNC	57,479,990	33.76%		57,479,990	23.60 %
	Duties	15,627,276	9.18%		15,627,276	6.42%
	Spare Parts and Reagents	16,126,397	9.47%		16,126,397	6.62%
	TOTAL OTHER COSTS	**122,057,957**	**71.69%**		**122,057,957**	**50.12 %**
	TOTAL ESTIMATED CAPEX COST - USD	**336,163,220**	**197.45%**	**73,294,689**	**409,457,910**	**168.12 %**

NOTES - All costs are instantaneous at date with no allowance made for forward escalation

12.2 PRODUCTION ESTIMATES

The concentrate production estimate was based on the information from the 1994 report and no further delineation of the ore body has been made since. The north pit calculation is based on the "restricted pit" as defined in the original report. The south pit was inadequately defined and no definition was made on the saddle between the north and south pits.

Table 12.2 shows the calculated production from the north and south pits in light of the ore reserve in 1994 and the recovery of 75%. Table 12.3 shows the calculated production at 88% recovery. Table 12.4 summaries the production estimates for concentrate and copper.

Table 12.4
Production Summary

	75%		88%	
	Concentrate (MTPY)	Copper (MTPY)	Concentrate (MTPY)	Copper (MTPY)
North Pit	207,000	56,000	243,000	66,000
South Pit	174,000	47,000	204,000	55,000

The original geology and ore reserve section of the report prepared by Mintec, indicated that an additional 138 million tonnes of ore could be recovered from the berm between the north pit and the river if a river diversion project was initiated at the tail end of the mine life. The cost and feasibility of such a diversion should be investigated in the future.

There is an oxide deposit located on top of the main sulfide deposit that is not defined but estimated by Aztec as 20 million tonnes. Using current solvent extraction and electrowinning technology and assuming that the oxide is leachable with dilute sulfuric leaching solution the calculation indicates that the oxide could produce approximately 123,700 tons (112,200 tonnes) of LME grade copper over a 6 year period.

Table 12.2

Annual Production Based on 75% Recovery of Copper
Old Information with Updated Production Rate of 60,000 MTPD

	Assumed	Mined	Mined	Grade	Contained	Mine	Concentrate						Smelter	Au/Ag	Copper	Gold	Silver
	Mineable	Material	Ore		Copper	Life	Recovery	Grade	Produced	Gold	Silver		Recovery	Recovery			
Option	(tonnes)	(tonnes)	(tonnes)	%	(tonnes)	(years)	%	%	(tonnes)	(oz/tonne)	(oz/tonne)		%	%	(tonnes)	(ounces)	(ounces)
1	244,237	17,500	7,000	0.369%	26	35	90%	28%	83	0.05	2.62		97%	95%	23	4	207
1A	244,237	17,500	7,000	0.369%	26	35	92%	15%	158	0.05	2.62		-	95%	23	8	394
2	244,237	52,500	21,000	0.369%	77	12	90%	28%	249	0.05	2.62		97%	95%	68	12	620
3	244,237	8,750	3,500	0.369%	13	70	90%	28%	42	0.05	2.62		97%	95%	11	2	103
4	543,890	105,000	42,000	0.318%	134	13	90%	28%	430	0.05	2.62		97%	95%	117	20	1,070
5	244,237	26,250	10,500	0.369%	39	23	90%	28%	125	0.05	2.62		97%	95%	34	6	310
6	244,237	78,890	35,000	0.371%	130	7	90%	28%	417	0.05	2.62		97%	95%	113	20	1,039
NEW – North	**428,000**	**50,400**	**21,000**	**0.368%**	**77**	**20**	**75%**	**28%**	**207**	**0.061**	**2.99**		**97%**	**95%**	**56**	**12**	**588**
NEW - South	**283,887**	**50,400**	**21,000**	**0.309%**	**65**	**14**	**75%**	**28%**	**174**	**0.061**	**2.99**		**97%**	**95%**	**47**	**10**	**494**

(tonnes and ounces are shown in 1000's)

Original Information

1 Based on 350 day per year operation
2 Grades are based on average over mine life
Mine plan is not available
3 Assume concentrate for sale at 28% Cu and rougher concentrate for leach at 15%
4 Based on average grade in mine over ten years for copper and precious metals
5 Minable reserves: 244,237 tonnes in N. Pit @ 0.369% grade and
assumed minable: 299,653 tonnes in S. Pit @ 0.277% grade for
overall 543,890 tonnes mineable reserve @ 0.318% grade
6 Geological reserves: 369,408 tonnes in N. Pit @ 0.368% grade
at 0.15% cutoff 453,162 tonnes in S. Pit @ 0.277% grade
822,570 tonnes 0.318% grade
7 Assume strip ratio of 1.5:1 for all options

Updated Information

Based on 350 day per year operation
Grades Based on mine plan of 1994 MINTEC
Copper grade less than 0.6%
Assume concentrate for sale at 28% - No concentrate for leach

Minable reserves: 428,000 tonnes in N.Pit @ 0.368%
283,887 tonnes in S.Pit @ 0.309%
Overall 711,887 tonnes in N+S.Pit @ 0.344%

Assume strip ratio of 1.4:1 for 60,000 MTPD scenario

Table 12.3

Annual Production Based on 88% Recovery of Copper
Old Information with Updated Production Rate of 60,000 MTPD

Option	Assumed Mineable (tonnes)	Mined Material (tonnes)	Mined Ore (tonnes)	Grade %	Contained Copper (tonnes)	Mine Life (years)	Concentrate Recovery %	Concentrate Grade %	Concentrate Produced (tonnes)	Concentrate Gold (oz/tonne)	Concentrate Silver (oz/tonne)	Leach Recovery %	Smelter Recovery %	Au/Ag Recovery %	Copper (tonnes)	Gold (ounces)	Silver (ounces)
1	244,237	17,500	7,000	0.369%	26	35	90%	28%	83	0.05	2.62	-	97%	95%	23	4	207
1A	244,237	17,500	7,000	0.369%	26	35	92%	15%	158	0.05	2.62	95%	-	95%	23	8	394
2	244,237	52,500	21,000	0.369%	77	12	90%	28%	249	0.05	2.62	-	97%	95%	68	12	620
3	244,237	8,750	3,500	0.369%	13	70	90%	28%	42	0.05	2.62	-	97%	95%	11	2	103
4	543,890	105,000	42,000	0.318%	134	13	90%	28%	430	0.05	2.62	-	97%	95%	117	20	1,070
5	244,237	26,250	10,500	0.369%	39	23	90%	28%	125	0.05	2.62	-	97%	95%	34	6	310
6	244,237	78,890	35,000	0.371%	130	7	90%	28%	417	0.05	2.62	-	97%	95%	113	20	1,039
NEW – North	**428,000**	**50,400**	**21,000**	**0.368%**	**77**	**20**	**88%**	**28%**	**243**	**0.061**	**2.99**	**-**	**97%**	**95%**	**66**	**14**	**690**
NEW – South	**283,887**	**50,400**	**21,000**	**0.309%**	**65**	**14**	**88%**	**28%**	**204**	**0.061**	**2.99**	**-**	**97%**	**95%**	**55**	**12**	**579**

(tonnes and ounces are shown in 1000's)

Original Information	Updated Information
1 Based on 350 day per year operation	**Based on 350 day per year operation**
2 Grades are based on average over mine life Mine plan is not available	**Grades Based on mine plan of 1994 MINTEC** **Copper grade greater than 0.6%**
3 Assume concentrate for sale at 28% Cu and rougher concentrate for leach at 15%	**Assume concentrate for sale at 28% - No concentrate for leach**
4 Based on average grade in mine over ten years for copper and precious metals	
5 Minable reserves: 244,237 tonnes in N. Pit @ 0.369% grade and assumed minable: 299,653 tonnes in S. Pit @ 0.277% grade for overall 543,890 tonnes mineable reserve @ 0.318% grade	**Minable reserves: 428,000 tonnes in N.Pit @ 0.368%** **283,887 tonnes in S.Pit @ 0.309%** **Overall 711,887 tonnes in N+S.Pit @ 0.344%**
6 Geological reserves: 369,408 tonnes in N. Pit @ 0.368% grade at 0.15% cutoff 453,162 tonnes in S. Pit @ 0.277% grade 822,570 tonnes 0.318% grade	
7 Assume strip ratio of 1.5:1 for all options	**Assume strip ratio of 1.4:1 for 60,000 MTPD scenario**

12.3 SXEW CAPITAL COST ESTIMATE – (New Information)

In a meeting on July 3, 2003, Dave Hermiston of Aztec stated that there was a 15 to 20 million oxide cap on top of the sulfide ore body. It must be removed to get to the sulfide. Dave asked if there was a possibility to remove this oxide ore and move it to a dump for leaching and copper recovery.

A cursory design for an SXEW plant was evaluated to determine the viability of recovering the copper from the oxide ore. Without knowing anything about the metallurgical characterizes of the ore the following basis was used to estimate a capital and operating cost for this plant.

Known Factors -

1. Mining Rate = 10,000 MTPD (oxide)
2. Total Copper Content = 0.75%
3. Ore Reserve = 20 million tonnes

Assumed Factors –

1. Heap Leach = ROM (run of mine) ore. No crushing required
2. Copper Recovery from Oxide = 75%
3. PLS Grade = 2.50 gpl
4. Acid Consumption = 5.0 kg acid per kg copper
5. Labor Rates same as Nacozari and Cananea, Mexico
6. Power Cost = \$0.05 per kwh
7. Delivered Acid = \$50 per ton
8. Cathode Shipping Costs = 2.55¢ per lb. (to Mexico City by rail)

Calculated Results –

1. Oxide Ore Mine Life = 5.7 years
2. SX Aqueous Flow Rate = 4,600 gpm (1,042 m^3/hr)
3. Cathode Production = 62 tons per day (56.2 tonnes per day)
4. Cathode Production = 21,700 tons per year (19,688 tonnes per year)

5. CAPEX for Leach, SX and EW = \$80 to \$90 million US (±40%)

These costs are estimates made on the above mentioned assumptions from factors used to design other SXEW plants. Verification must be made to identify actual metallurgical properties of the ore as the costs will change if the ore requires more acid to leach the copper or has a different recovery, whether the ore must be crushed or not to maximize copper recovery, what the actual PLS composition would be and the future pricing of delivered reagents.

The CAPEX is only for the SXEW plant and based on similar built plants in Mexico, Arizona and Australia updated to 2003 costs. The cost assumes that the infrastructure and administration exists from the mine and concentration portion of the project.

In general the oxide ore would be hauled to a dump area which had previously been lined with an HDPE liner. The run of mine oxide ore would be placed on the liner and stacked to a height of 15 to 20 feet. Piping with sprinklers or drippers would be placed on top of the dump and a solution of sulfuric acid and water would be applied to the dump and allowed to percolate down through the ore to the HDPE liner. The copper is leached from the ore by the acid and remains in the leaching solution. The copper rich solution exits the bottom of the dump and is directed to a lined PLS (pregnant leach solution) collection pond.

The PLS solution is pumped to the SX (solvent extraction) plant where copper is removed from the PLS and concentrated in a rich electrolyte solution. The PLS, now depleted of copper, also know as raffinate, is pumped back to the dump as new leaching solution. The rich electrolyte solution is pumped into the EW (electrowinning) tankhouse where the copper is electrically plated on to SS (stainless steel) blanks known as cathodes. The SS blanks are removed from the plating cells once every 7 days so the accumulated copper can be removed from the plates and stacked as pure copper cathodes ready for shipment to the market.

SECTION 13

ECONOMIC EVALUATION

13.0 ECONOMIC EVALUATION

No work was done to update this section.

SECTION 14

CONCLUSIONS AND RECOMMENDATIONS

14.0 CONCLUSIONS

Based on the updated capital cost focused on a mining rate of 60,000 MTPD and that concentrate will be produced and shipped to a toll smelter/refinery and metallurgical testing performed previously by others the following conclusions may be reached:

- The estimates, provided from Mintec Inc., of minable reserves and assays indicate Santo Tomas is a low grade, high tonnage copper deposit containing modest gold and silver values. The north pit which was used as the basis for this study contains an estimated minable ore reserve at 0.2% cutoff of 428 million tonnes at an average copper grade of 0.368%, considering that a safety wall/berm is left unmined in the north side of the pit to avoid possible problems with the Huites reservoir at maximum water level.

- In the original work provided by Mintec, an ultimate pit with no mining limitations, the berm mined, was identified. The reserve would increase to 565 million tonnes in the north pit or an additional 138 million tonnes. The viability of this additional ore will be identified in the feasibility study.

- Past metallurgical testwork as listed in Section 4, indicated the ore is amenable to copper recovery by conventional concentrate processing consisting of crushing, grinding, and flotation to produce a saleable copper concentrate. The testwork also indicated that gold and silver contained in the ore will report with the copper concentrate.

- Based on the updated study work the "order of magnitude" capital cost estimate for a new grassroots conventional concentrator facility, utilizing SAG mills indicates that the Santo Tomas project would cost $410 million for the 60,000 MTPD plant.

- A more economical approach for the development of the Santo Tomas property would be to identify an existing concentrator plant and relocate the equipment to Santo Tomas.

- The geotechnical and hydrological feasibility of the preliminary mine design for the restricted north pit, which was used as the production basis for evaluation in this report, has not been determined.

- The recent site visit has presented optional locations for tailings containment, waste rock containment, and the plant site that seemingly would not impact the Huites reservoir.

14.1 RECOMMENDATIONS

In order to bring all the supporting data for the project up to equivalent levels of accuracy for support of more accurate feasibility level evaluation, we believe that the following activities will be required. These activities are broken down into two phases, one that will confirm the reserve and the second that will prepare the feasibility study for the processing facility. The costs associated with Phase I and Phase II are presented separately.

PHASE I – Confirmation Drilling and Preliminary Pit Design

- Additional exploration/confirmation drilling: As per the suggestion of Mintec Inc. an exploration/confirmation program that includes approximately 80 drill holes at 250 meter centers in the center saddle and south deposits are required to determine the size of the deposit. Mintec already has information of the 57 holes drilled in the north deposit. Based on estimates received from Aztec Copper for drilling the program cost, which includes sample assays and support the cost of drilling would be approximately $1.44 million dollars (US).

- Reserve evaluation - preliminary pit design: The information obtained from this drilling program and previous information would be used to perform a new reserve evaluation, preliminary pit design and production schedules in enough detail to provide the necessary information to review the optimum mine-plant capacity. Based on information from Mintec Inc. the estimated cost of this activity would be $40,000 (US) *(work to be done by Mintec)*

- Total cost of Phase I is approximately 1.5 million dollars (US)

PHASE II – Feasibility Study

- Additional feasibility confirmation drilling: To provide adequate mine reserve detail to prepare the feasibility study. It is estimated that 60 additional drill holes will be required at centers of less than 250 meters. Based on estimates received from Aztec Copper on drilling the program cost, which includes sample assays and support the cost of drilling would be approximately $1.1 million dollars (US).

- Reserve confirmation and mine design: The information obtained from this drilling program and previous information would be used to confirm the reserve and provide production schedule forecasts and pit design in enough detail to design the processing facility to ± 15% accuracy. Based on information from Mintec Inc. the estimated cost of this activity would be between $60,000 and $75,000 (US). *(work to be done by Mintec)*

- Metallurgical testing: From new drill core obtained from the program above, sample material could be collected for more detailed crushing, grinding and flotation studies which could provide and confirm data for equipment sizing and preliminary plant design. Test work is required to classify the Bond Work index to determine the hardness of the rock for specifying crushing and grinding parameters. Flotation tests need to be conducted to verify that a 28% concentrate grade can be achieved and what the actual recovery would be for the 0.368% ore. The order of magnitude cost for these studies would be $50,000 to $70,000 dollars. Approximately 450 to 500 pounds of new material would be required to conduct these tests. (work to be done by Mt. States R&D)

- Environment studies: Environmental studies undertaken by specialists are recommended for preliminary project permitting activities including Environment Impact Study (EIS) and others. Order of magnitude cost would be $25,000 to $30,000. *(work to be done byEPG – environmental planning group)*

- Geotechnical, hydrogeological and hydrological studies:

    - The mine pit design parameters, slope stability and plant soil conditions need to be fully investigated by specialists. Order of magnitude cost would be $230,000 to $350,000, including geotechnical-oriented diamond core drilling and laboratory testing. *(work to be done by Golder)*

    - Performance of a geotechnical study to confirm the feasibility of a berm at the north side of the pit, with respect to permeability, rock characterization, rock integrity. Order of magnitude cost would be $145,000 to $190,000, including hydrogeologic test hole diamond core drilling, field permeability testing and laboratory testing. *(work to be done by Golder)*

    - Performance of a geotechnical study to characterize the plant site foundation conditions and development of foundation design recommendations prepared by specialists. Order of magnitude cost would be $120,000 to $180,000, including geotechnical test hole drilling, backhoe test pits and laboratory testing. *(work to be done by Golder)* The plant throughput considered is 60,000 metric tons per day.

    - Performance of a geotechnical study to select site, perform detailed geotechnical investigation, complete engineering analtsis and prppare feasibility design drawings and report for tailings impoundment. Order of magnitude cost would be $190,000 to $260,000, including geotechnical test hole drilling, backhoe test pits and laboratory testing. *(work to be done by Golder)*.

- Socio-economic study of the area needs to be conducted. This work is dependent upon Aztec Copper's work and negotiations with the Mexican government as summarized below.

- Power Supply study to determine the viability and cost of getting 53 MW of power to the plant. This work is dependent upon Aztec Copper's work and negotiations with the Mexican government as summarized below.

- Feasibility study. When all the above information is available from Phase I and II, a full feasibility study could be performed which would develop the information required for plant design and capital cost and operating cost estimates to ± 15% accuracy. The estimated cost of such a study would be in the range of 3.1 to 3.5 million dollars over a 12 to 14 month period, excluding owners' costs and contingency. During the study discussions would take place with the various Mexican authorities regarding infrastructure development and power, water and fuel pricing structure. *(work to be done by Bateman)*

The estimated cost for Phase II, to produce a definitive estimate of ± 15% suitable for the raising of finance through appropriate institutions for the mine and concentrator would be in the range of $5.0 to 5.8 million dollars.

Once the oxide cap of the deposit is fully delineated and metallurgical testing completed, a solvent extraction electrowinning facility could be designed that could produce 21,700 tons (19,700 tonnes) per year of LME Grade cathode copper for approximately 6 years. An estimated cost to bring the facility to a viable level would be as follows:

- Oxide leaching: If recovery of the copper from the oxide ore is desired, column leach testing should be conducted to determine the optimum plant size, acid consumption and leachablilty of the oxide ore. Approximately 1,000 kg of newly drilled material would be required. The order of magnitude cost for this testing is approximately $60,000 to $75,000 dollars. This is secondary work compared to the main ore body.

- Feasibility study: When all the above information is available, a full feasibility study should be performed which would develop the information required for plant design and capital cost and operating cost estimates to a ± 15% accuracy. The estimated cost of such a study would be in the range of 0.9 to 1.2 million dollars over a 8 to 10 month period, excluding owners' costs and contingency.

The estimated cost to produce the feasibility study for the oxide portion of the project only would be the sum of the metallurgical testing and engineering and design study work. This assumes that the geotechnical and hydrological for the mine and associated sites are completed as listed above and that only minimal geotechnical and hydrological work would be required for identifying the stability of the leaching area and site location of the SXEW plant.

APPENDIX A

EQUIPMENT LIST

APPENDIX B

CONCENTRATE LEACHING

SUMMARY

APPENDIX C

DRAWINGS