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# THE FEASIBILITY OF MINING THE TIGER DEPOSIT BY OPEN PIT HEAP LEACH METHODS

## VOLUME I

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INTRODUCTION

## 1.0 Introduction

The purpose of this study was to determine the economic feasibility of mining precious metal ore of the Tiger deposit by open pit and heap leaching methods. Magma initiated this study after an incomplete exploration program by Cyprus Minerals that would have earned Cyprus an interest in the property.

This study encompasses investigation and analysis of the major aspects of profitably mining the Tiger deposit. An exploration drilling program to enhance the information base on the mineralization was performed. The resource model and pit design were developed using the MEDSYSTEM computer software from a geologic and assay data base. Exhaustive metallurgical testing was performed to determine the cyanide leaching characteristics and parameters of the mineralized rock. An evaluation of the potential effects of pit mining on the SX-EW building was performed by qualified consultants. Capital and operating cost estimates were completed and the economic results were calculated.

Mining the Tiger deposit by open pit methods, heap leaching the ore, requires a \$450 price to achieve a zero net present value using a 15% discount rate. The recommendation is not to mine the Tiger deposit at this time, but to re-evaluate the feasibility when a sustained \$450 gold price can be foreseen or significant changes, favorable to the project, occur.

This report does not contain or describe all the work performed for this study. Significant portions are found in the project record. This record contains Volume IV, the detailed information and files required to use the MEDSYSTEM data base. The reports by Cyprus Minerals, copies of internal correspondence, drill logs, assay reports, contractor cost estimates, and all other materials related to this study are also in the record. The computer files are stored, in duplicate, on magnetic tape or disk as described in Volume IV.

EVALUATION SUMMARY

## 2.0 Evaluation Summary

To determine the feasibility of mining the Tiger deposit, Magma investigated the size and tenor of the mineral reserve. A drilling program was designed to complete the program of Cyprus Minerals. The results of both programs were combined into a single data base for the reserve estimation. Geostatistical techniques were used to analyze the drilling results. The geologic resource was estimated to be 6.5 million tons of material with an average gold grade of 0.035 troy ounces per ton (OPT). The silver grade of that same material is 0.123 OPT.

The amenability of the mineral resource to cyanide heap leaching was investigated by exhaustive laboratory testing. These tests, direct agitated cyanidation tests, agglomeration tests, and column percolation leach tests, were performed at various feed sizes and indicated that a 3/8 inch crush would be required. These test results and the material mix in the minable reserve, indicate that 57% recovery of the gold can be expected from a heap leach treatment of the ore. To achieve this recovery, ore must be under leach in excess of 200 days on the operating heap.

The ultimate pit design was selected from several possible designs, based on the net present value of mining each pit. From the ultimate pit, a minable pit, with roads and triple benches, was designed. Two separate pits resulted with a total minable reserve of 2.4 million tons with a gold grade of 0.052 OPT and a silver grade of 0.18 OPT. The Mammoth and Mohawk pits are shown in Figure 2.1; a plot of the project site plan over an air photo. The close proximity of the SX-EW tankhouse to the Mammoth pit requires that extreme care be exercised to prevent blast damage.

The leach pad was designed to contain the ore from the pits and the gold bearing Tiger tails located just west of the proposed crusher site. These tails will be agglomerated with ore and will provide an additional 4000 recovered ounces of gold to the project. The leach pad is a zero discharge facility capable of containing runoff from a severe rainfall event in its solution and overflow ponds. The liner will be a synthetic geomembrane placed on compacted soils. Leach detection and recovery will be installed under collection ditches and solution ponds.

To minimize the capital required, a mining contractor will be utilized to strip, mine, and crush ore. Blasting will be done by Magma. Ore, crushed to -6 inches in size, will be delivered by the contractor to a preparation plant owned and operated by Magma. In this plant, ore will be crushed to -3/8 inch, blended with tails if available, agglomerated with portland cement, and sent by belt conveyor to the pad. The agglomerated material will be stacked 20 feet high and leached, with dilute cyanide solution, for a minimum of 130 days before fresh ore is placed on top. The gold bearing solutions will collect in a system of ditches and report to the pregnant solution pond. Activated carbon, fluidized in columns, will adsorb the gold as the pregnant solution is pumped through the column. The gold bearing carbon will be processed by Magma Nevada Mining Co. at its Ruth mill. Tiger dore' will be refined by the same refiner that Magma Nevada uses.

The costs to mine and crush to -6 inch were estimated by four potential mining contractors. Ore preparation, leaching, and metal extraction costs, capital and operating, were developed by the Metallurgical Department of Magma. The 1992 Business Plan of the San Manuel Mining Division provided additional cost information to this evaluation. Dan Turk, of Magma Nevada Mining Co. provided assistance in reviewing these costs. The capital cost of the leach pad was estimated from recent construction experience by the oxide department of the Phase 5 leach dump and the design of the Weary Flats Leach Project at Magma Nevada.

The revenue used in the evaluation was derived from the gold and silver prices suggested in the 1992 Business Plan guidelines and the expected metal recoveries indicated in the metallurgical test results. The project cash flow was estimated and the internal rate of return is 7.4%, excluding sunk costs. The net present value is a negative $1.4 million using a 15% discount rate. To achieve a zero net present value, at a 15% discount rate, a constant gold price of $450 is required.

It is recommended to re-evaluate the project when a sustained $450 gold price can be foreseen, or significant changes favorable to the project occur.

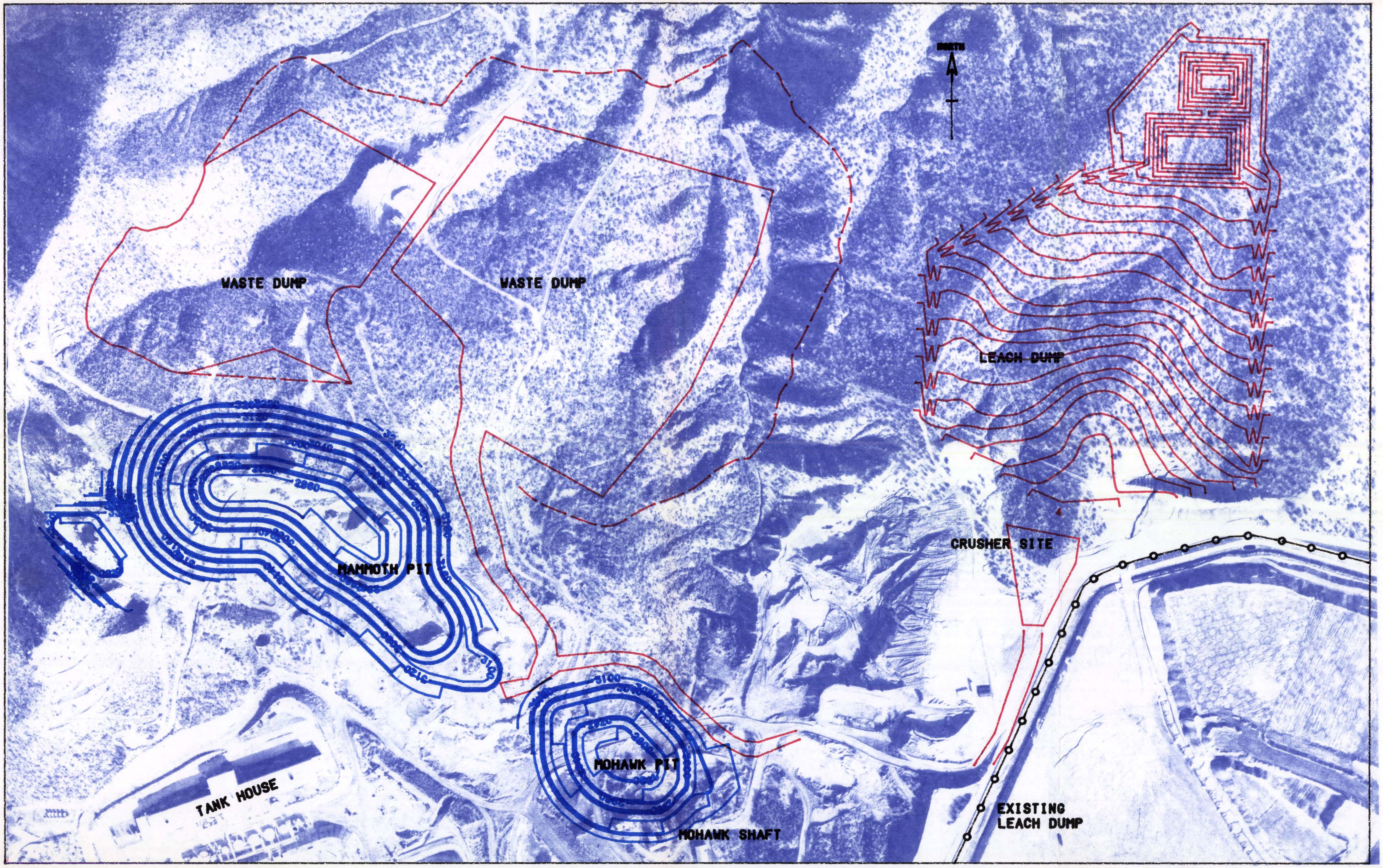



FIGURE 2.1 PROJECT AIR PHOTO

LOCATION AND HISTORY

## 3.0 Location and History

### 3.1 Project Location

The Tiger Project is located in Pinal County, Arizona, on patented and unpatented mining claims owned by Magma Gold Ltd. adjacent to the San Manuel Mining Division's underground block caving and open pit copper oxide dump leach facilities. The mineral reserves are located several hundred feet north and east of the SX-EW facilities and are part of a mineral occurrence discovered in 1879 and mined intermittently until 1953. Magma acquired the property in 1953 from Mammoth-St. Anthony Mining Company for living accommodations to support development of the San Manuel Mine. The reserves evaluated here are wholly contained on the patented mining claims shown on Figure 3.1, Tiger Project Area Claim Map.

### 3.2 Tiger History

The history of Tiger is eloquently recounted by Kim Howell and is the source of the following discussion.

The Bureau of Land Management records the earliest claim at Tiger as the Hackney, located by Charles Dyke and T. C. Weed on July 14, 1879. This claim is on what is known as the Collins vein. The vein was mined by open cut methods and the first recorded production came in 1881. In February 1882, Frank Schultz located the Mars claim on what he called a "mammoth lode gold vein." The vein and mine have since been called the Mammoth. Mr. Schulz is also credited with the name of the district, the Old Hat district. After failed attempts to mine or mill the Mammoth ores, Schulz sold his claims to George W. Fletcher in 1884. Captain Johnson, Fletcher's manager, sank a shaft 300 feet into the vein and built 30 stamp mills on the San Pedro River. Gold was extracted from the milled ore by amalgamation. The town that grew up around Fletcher's mill became Mammoth and a Post Office was established in 1887.

In 1889, Fletcher leased his property to a British syndicate which established the company known as Mammoth Gold Mines, Ltd. The mill was expanded to 50 stamps; revenues were reported to be $14 per ton and costs were $4 per ton. The community around the mines was called Schulz after Frank Schulz, who opened a store to serve the residents.

On New Years Day 1891, the Mohawk claim was located by Andrew Dannon and J. G. Fraser on the Mammoth vein southeast of the Mammoth mine. The claim was sold to and developed by the Mohawk Gold Mining Company. By 1895, the mine was down to the 300 level and 10 stamp mills were operating on the property.

The difficult haul from forests in the Santa Catalina Mountains made timber in the mines scarce. This contributed to the cave-in in 1893 of the Mammoth Mine between sections of the 200 to 400 levels and forced Mammoth Gold Mines, Ltd. to cease operations.

During 1894, the Collins mine went into production with the sinking of 300-foot shaft and the shipment of ore to the idle Mammoth mill. The caved upper levels of the Mammoth were also worked.

The Mohawk mill commenced production in May 1896 in the town of Schulz. The Schulz Post Office was established on July 27, 1896. Mammoth Gold Mines, Ltd., reorganized as Mammoth Gold Mining Company, began negotiations to acquire the Collins Mine located on the Collins vein. At the end of 1897, Mammoth Gold Mining Company acquired the Collins Mine and became the Mammoth Collins Gold Mines, Ltd. This company built the 2-3/4 mile "Bleichert wire rope transportation system," an aerial tram, to move ore from the mines to the mill in Mammoth.

As the "free-milling" gold content of the ore decreased, the recovery decreased. Mammoth Collins Gold Mines, Ltd. then contracted with the St. Louis Gold Saving Company to treat the stamp mill tailings by cyanidation and zinc precipitation.

By 1900 the Mammoth shaft was down to a depth of 800 feet. The water level was at 760 feet. With in-flows to the shaft of over 150 gpm and poor ground conditions below the water level, the 760 level was as deep as the mine could go. An engineering report of the time by T. J. Davey estimated reserves of 10,000 tons at $8.63 per ton remained between the 700 and 760 levels and none above the 700 level. The average costs per ton reported in 1900 and a comparison, by percentage, to today's estimated costs are found below.

| | 1900 | | 1991 |
|---|---|---|---|
| Mining | $1.15 | 52% | 52% |
| Milling | 0.64 | 29% | 40% |
| Tram | 0.13 | 6% | |
| Management | 0.28 | 13% | 8% |
| Total | $2.20 | | |

The Mammoth Collins Gold Mines Ltd. failed to pay on the bond used to lease the property. But the ground over the Collins vein did pay off, from the 760 level to the surface. As a contemporary report described it:

> "One night in April, 1901, an extensive caving occurred in the main [stopes], but without loss of life or injury to the shaft. It started suddenly without warning at the north end of the mine and extended from the surface to the bottom of the workings, 750 feet deep... The cave-in brought the company to realize that if they continued to work they would have to do extensive and expensive timbering."

The cave-in closed the first chapter of mining in the Old Hat district. With the ore above water practically mined out, ownership problems and caving workings, the mines closed. In 1902, the Post Office in Schulz closed; in 1903, the population of Mammoth was down to 300 from 700 in 1890. The production from Tiger amounted to 350,000 tons of ore from which 150,000 ounces of gold were recovered.

From 1901 to 1906 various new claims were staked in the district, but no real activity occurred until the Mohawk Gold Mining Company was reorganized. A new mill with the capacity of 30 tons per day was built. From 1907 to 1912, production from the Mohawk was valued at $88,945.

The Young brothers leased the Mammoth-Collins mine and Mammoth mill in 1913. They did not produce any ore, but completed several projects which benefitted the mines. Supported by capital from their Great Western Copper Company, they installed pumping capacity to de-water the mines, connected the Collins and Mammoth on the 700 level, repaired the Oracle and Mammoth road, deepened the Mammoth shaft, developed ore on the 760 level and drove new drift around the old caved area. In addition, a new headframe and hoist were installed and diamond drilling was done on the 600 level of the Collins Mine. After expending some $200,000 the Young brothers decided that further effort was not to be profitable. An additional $250,000 would be required to rebuild the burned out mill. In addition, the ore was thought to be too complex to be concentrated profitably.

The First World War increased the demand for molybdenum. Tailings from the mills at Mammoth contained wulfenite, a lead and molybdenum oxide. The Arizona Rare Metals Company converted the old cyanide mill to gravity separation and supplied the demand from the 250,000 tons of tailings at Mammoth. For the three years during the war, the operations in Mammoth supplied the only molybdenum marketed in the United States. Concentrate was hauled by freight team to Tucson for shipment by rail to Pittsburgh, Pa.

When the Young brothers relinquished control of the Mammoth-Collins property, it was immediately leased by the Mammoth Development Company. The prime mover of this company was Col. Epes Randolph, who also served as Chancellor of the University of Arizona and head of the Arizona operations of the Arizona Eastern railroad lines.

Arizona Rare Metals Company suspended operations in mid-1918 due to the unsettled molybdenum market. In the mean time, the tailings were re-worked again for molybdenum by the Hondo Oil Company by flotation methods. From late 1918 to sometime in 1919 the Arizona Rare Metals Company changed names twice, finally settling on the St. Anthony Mining and Development Company. The post-war depression caused the closure of the mines of the Old Hat district and by 1921 most of the mines in Arizona were shut down.

Interest in the mines of the Old Hat mining district resumed when the government raised the price of gold in 1933 and 1934 from $20 per ounce to $34.95 per ounce. The Molybdenum Corporation of America gained control of the New Year and Mohawk mines. The St. Anthony Mining and Development Company became affiliated with the Mammoth-St. Anthony Company, Ltd. and shared operation of the Mammoth-Collins mine. Production and employment peaked as the country came out of the Depression. By 1939, the mines were consolidated into one operation under the name Mammoth-St. Anthony, Ltd. The total payroll was 400 and on March 15, 1939 a Post Office was re-established at Schulz under the new name of Tiger. Exactly how the name came to be is unclear. One version is that the area became known as Tiger when Sam Houghton was active in the area. Houghton was a Princeton alum and the Princeton football team was called the "fighting Tigers." Another version is that a popular vote was taken. Two choices were offered. St. Anthony, for the company who owned the mines, or Tiger, after a famous tobacco pouch, made from the scrotum of a tiger, belonging to either the mine's owner or the mine manager. The vote was unanimous.

A small lead-zinc smelter was built by the Molybdenum Gold Mining Company in 1937 and was expanded in 1939 with the addition of a new 24-ton reverberatory furnace. Bullion was shipped to El Paso for refining.

Tiger again became a source of strategic metals at the start of World War II. Labor was short in the district, so the United States Army sent and then discharged from service 65 men to work the mines. The Mammoth-St. Anthony Mining Company struggled through the war and probably have failed were it not for the bonuses the government was paying for its metals. In 1943 churn drilling by the Bureau of Mines commenced into the San Manuel copper deposit. The following year, Magma Copper Company under W. P. Goss purchased the San Manuel claims.

In early September 1944, the Mohawk shaft caught fire, shutting off the water supply for the entire community as well as the fire fighting effort. The pumps were back on line in three days. The shaft was repaired quickly by using tailings as a work platform. The shaft was filled with tailings and then drawn out through the Mammoth mine from below as sets were replaced. The work was complete by mid-November.

After the war, lead and zinc ores were still being produced from the Collins vein. Foreign imports of lead and zinc were forcing many domestic mines to shutdown. The Collins was now at the 1125 level with very high water inflows and lower grade. St. Anthony Mining and Development suspended operations on December 1, 1952.

Production from the district then totalled:

| | |
|---|---|
| Gold | 400,000 ounces |
| Silver | 1,000,000 ounces |
| Copper | 3,500,000 pounds |
| Lead | 75,000,000 pounds |
| Zinc | 50,000,000 pounds |
| Molybdenum oxide | 6,000,000 pounds |
| Vanadium oxide | 2,500,000 pounds |

Magma purchased Tiger in February 1953 primarily for the facilities to support the development of the San Manuel Mine. The purchase price was 10,000 shares of stock, worth about $275,000 at the time.

The townsite at San Manuel was built and the people of Tiger moved to Mammoth or San Manuel. By June 1954 Tiger was vacant. The Post Office closed on November 26, 1954.

Production from Tiger continued when, in 1962, McFarland and Hullinger purchased the tails at the San Pedro River in Mammoth and sold them to the Hayden smelter of ASARCO. In 1963, tails at Tiger were shipped by McFarland and Hullinger to Hayden.

Magma produced flux ores for the San Manuel smelter from Tiger intermittently through 1987. Difficulties maintaining silica content and sporadic gold content plagued the operations. The ores of Tiger are not suitable to the new flash furnace of the smelter.

In 1987 Cyprus Minerals and Magma entered into an agreement to explore Tiger for precious metals. Cyprus failed to meet its earn-in requirements and that's were this begins.

GEOLOGIC RESERVES

## 4.0 Geologic Reserves

### 4.1 Summary

Magma and Cyprus, separately, performed precious metals exploration in the vicinity of Tiger intermittently from December 1987 to October 1991. These programs included 130 reverse circulation rotary drill holes and one diamond drill core hole. A total of 43,834 feet of drilling was completed. Cyprus completed 37,151 feet and Magma accounted for 6,683 feet. The results prove a geologic reserve, estimated by Magma, of 6.5 million tons of material with an average grade of 0.035 OPT of gold at a cut-off grade of 0.010 OPT. The silver grade of that same material is 0.123 OPT.

The Tiger reserve was estimated using data compiled from the exploration programs and methods accepted as reasonable and prudent practice for mineral reserve calculations. These methods include assay compositing, variography, discriminator Kriging of a block model, and three dimensional geologic rendering. The computer software system produced by Mintec, called MEDSYSTEM, facilitated the application of these methods.

### 4.2 Drilling Programs

As part of their earn-in to a Magma-Cyprus Joint Venture at Tiger, Cyprus executed an exploration program to define a reserve of millable, open pit minable ore. Stanton P. Dodd, a Consulting Geologist retained by Cyprus, directed the exploration program of surface and underground mapping, sampling, and 114 reverse circulation drill holes totalling 37,151 feet. His summary report is found in Appendix I. The data obtained in the Cyprus program provides the majority of the information used in the development of the ore reserve estimate made here.

A drilling program was initiated by Magma in July 1990 to test the continuity of mineral zones along strike and to fill in gaps left in the Cyprus program. Eleven reverse circulation holes, totalling 3969 feet, and one core hole, 399 feet deep, were completed by September. In addition 2970 feet of reverse circulation drilling was performed to condemn sites under consideration for waste rock disposal, ore leach pads, as well as to provide ground water monitor wells. No mineral or water was encountered by condemnation drilling.

In June 1991, based on preliminary pit design results, a second Magma drilling program commenced. The narrow objective of this program was to test a very high grade (+ 0.3 OPT) zone of mineralization at the bottom of the pit. Five reverse circulation holes, totalling 2315 feet, were drilled on four cross-sections perpendicular to the strike of the mineral zones. Coincident with this program, the recommendation by Call & Nicholas to place subsurface monitors in drill holes at the SX-EW facility was implemented. Three monitors are installed north of the tankhouse on either side of the Mammoth Fault.

4.2.1 Cyprus Results

The following will only summarize the results of the Cyprus exploration program. For a more detailed description the reader is referred to Appendix I, Stanton P. Dodd, Consulting Geologist, Summary Report, Tiger Project, Pinal County, Arizona, January 1989.

Cyprus began drilling in December 1987 and finished in October 1988. Initial results indicated significant mineralization along the main vein structure. Figure 4.1, a drill hole location map, displays the established standard set of cross-sections, locally perpendicular to the strike of the vein structure and 100 feet apart. In the main mineral zones, the Mammoth and Mohawk, at least two holes were drilled along each section.

In general, the drilling was difficult. Extensive underground workings, fractured ground, and backfill, caused difficult drill penetration and poor sample recovery. This was especially true of section 540. This section passes through the surface expression of a massive cave zone. Due to the lack of usable data Cyprus initially treated this section as waste. Later, the section was assigned an ore tonnage based on the average tonnage and grade of the two adjacent sections.

All drill holes were sampled at 5 foot intervals and assayed for gold and silver, except for the post mineral Gila conglomerate. Selected samples were analyzed for molybdenum, lead, zinc, copper and vanadium. These other metals were found in low grade, sporadic concentrations. Geologic cross-sections were constructed for most of the section lines. (Maps 15 through 45 of Dodd's report which are archived and available on request)

From the drilling data, Cyprus estimated reserves using the polygon method, both by digital computer and manually. The details of that estimation are reported by Howard Harlan, Pre-feasibility Study of the Tiger Joint Venture Project, Cyprus Minerals Company, Pinal County, Arizona, April 1989. In this document, using relatively high costs for carbon in pulp milling and open pit mining, Carl Gerity reports a minable reserve of 1,722,000 ton at 0.074 OPT gold with a 3.5:1 stripping ratio and 15% dilution. A manual check of these reserves was performed by Ken Bondurant which agreed within 6% of Gerity's report.

### 4.2.2 Magma Results

In July 1990, Cyprus failed to complete their earn-in requirements and Magma initiated a follow-up exploration program. The objectives were to test the projection of mineral values along strike of the vein and to fill in sparsely drilled areas on the established sections. Eleven rotary reverse circulation drill holes, totalling 3969 feet, were drilled and sampled. Six holes were targeted to test the mineral continuity along strike. Two holes were placed on either side of section 540, cited above as an area of very difficult drilling. (See Figure 4.1) Two holes were targeted along the Mammoth fault in an attempt to extend a very high grade zone found on section 647. One hole was drilled on the southern end of the Mammoth pit mineral zone to confirm the narrowing of the mineral zone.

The six holes designed to test continuity along strike were generally successful. Hole MM-209 was the one exception in that it did not reach its target depth due to the loss of drilling tools down the hole. One hole, MM-207, by chance, passed through a drift near the Mohawk shaft and was important to the underground bulk sampling program. The five holes that did reach full depth provided valuable data to variogram analysis.

The two holes on either side of section 540 addressed the questions about the mineralization near the massive caved zone. Unfortunately, the holes near the Mammoth fault did not prove an extension of a high grade zone along the fault. The hole MM-210 did confirm the narrow vein width between the two pits as designed by Cyprus.

Magma drilled one diamond core hole to determine rock strength data of the rock mass between the proposed pit and the SX-EW plant. That hole was collared at the pit edge and angled at $45^0$ on a bearing toward the north east corner of the plant. This hole provided valuable data to the analysis performed by Call and Nicholas. (Section 13.0) The core was assayed by the same procedures as the reverse circulation drill cuttings. No new mineralization was found.

In addition, drilling was performed, at sites under consideration at the time for waste dumps and leach pads, to confirm the absence of mineral on those sites. Two of these condemnation holes were to be used as groundwater monitoring wells. Neither one encountered groundwater or mineral and have been abandoned as monitor wells.

Over the next several months, a complete data base was compiled of Cyprus drill files and the new Magma drilling. Many models of the geologic resource were created using the MEDSYSTEM mine planning computer software. (The complexity of this issue will be dealt with in the following sections.) A model was generated and an ultimate pit was designed. This design relied heavily on the existence of a very high grade zone of ore centered around one drill hole at the pit bottom. Adding to the uncertainty of the situation, a very slight change in the interpolation parameters in the modelling routine caused a drastic reduction in the grade of the zone and consequently a reduction in size of the pit design. The need to test this high grade zone with additional drill holes was apparent.

Five additional reverse circulation holes were drilled to the pit bottom. Another possible high grade zone was not drilled due to time and budget constraints. The second zone was not believed to be as sensitive to interpolation errors. After the 1991 drilling, testing of interpolation methods in the second zone showed it to be affected within only a narrow range.

Only one of the five holes encountered sufficient mineralization to generate a bench composite above 0.10 OPT gold. The zone driving the large pit design was disproved.

Coincident with the drilling program, subsurface subsidence monitors were placed in the vicinity of the SX-EW tankhouse in locations recommended by Call & Nicholas. For a detailed discussion, refer to Section 11.0 and Appendix IX, Dave Nicholas and Ross Barkley, Potential Ground Movement Near SX-EW Plant due to Mining the Tiger Deposit, Call & Nicholas, April 1991.

## 4.3 Resource Model

### 4.3.1 Data Compilation

The majority of the drill data available to build a computer block model was compiled by Cyprus Minerals. The file was corrected and verified against the assay reports and drill logs. Survey information was checked by locating any visible hole collars. The omitted silver assays and all the rock type codes were added to the drill hole file. Magma's drilling results were added to the data base. The details of how the data was handled in the MEDSYSTEM are discussed in Appendix X, A. J. Fernandez, MEDSYSTEM Details to Plan and Design the Tiger Open Pit, Magma Copper, November 1991. Once the data base was established and verified the geologic resource modelling commenced.

At this point, the drill hole data consisted of gold and silver assay data, rock type information, collar location and elevation, hole bearing and inclination, and assay interval measured down the hole from the collar. The predominant rock types are listed below:

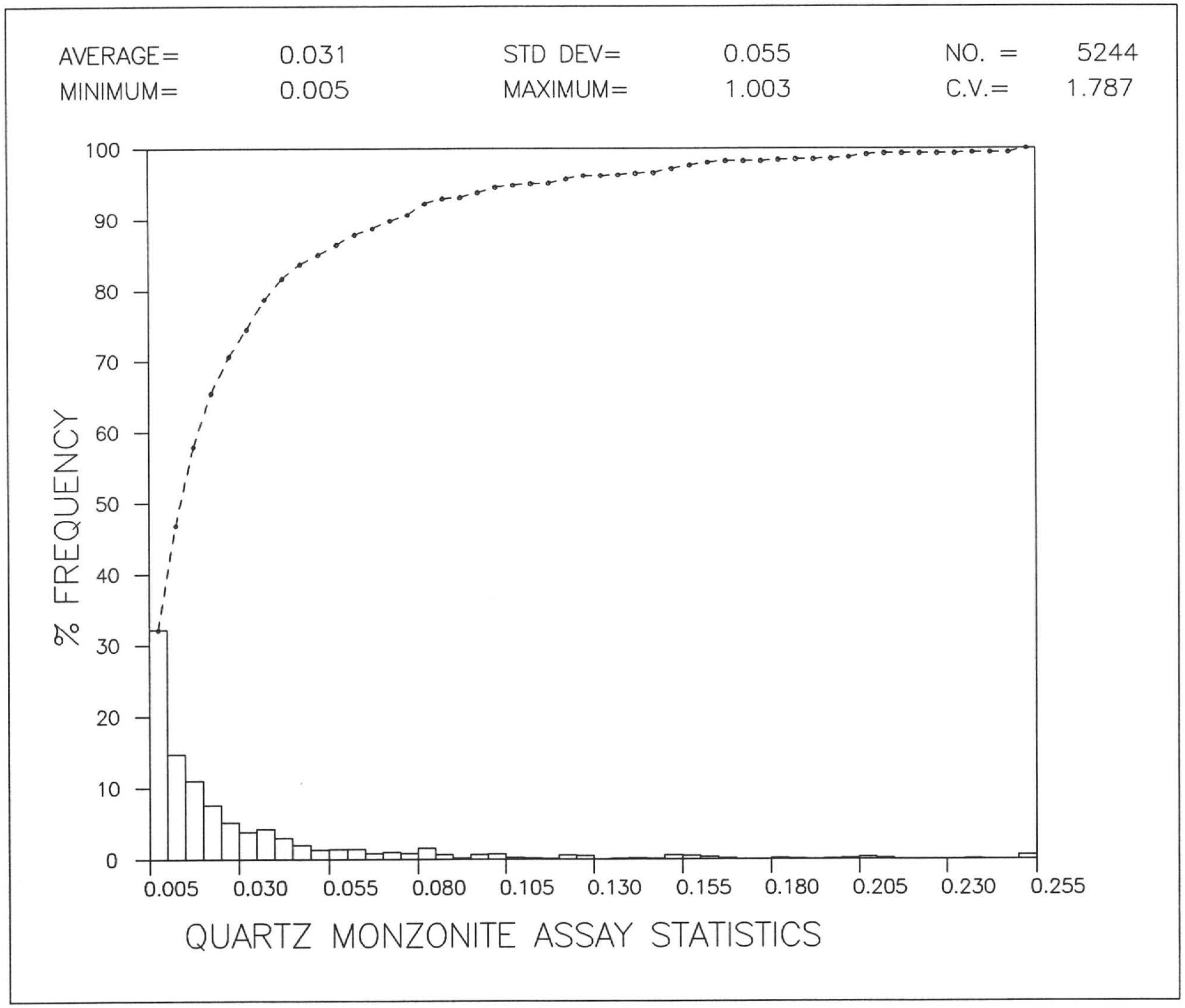
Quartz Monzonite-of Precambrian age
Rhyolite
Rhyolite Breccia
Cloudburst Formation-mostly andesites and intrusive breccia
Quartz Vein-the ore vein contained in all mineralized rocks
Gila Conglomerate
Arkose-derived from the Quartz Monzonite

The data was then plotted on section for further verification and to aid the geologic interpretation. Statistics of the gold assays hosted by each rock type were generated. Figures 4.2 through 4.7 are histograms from that analysis.

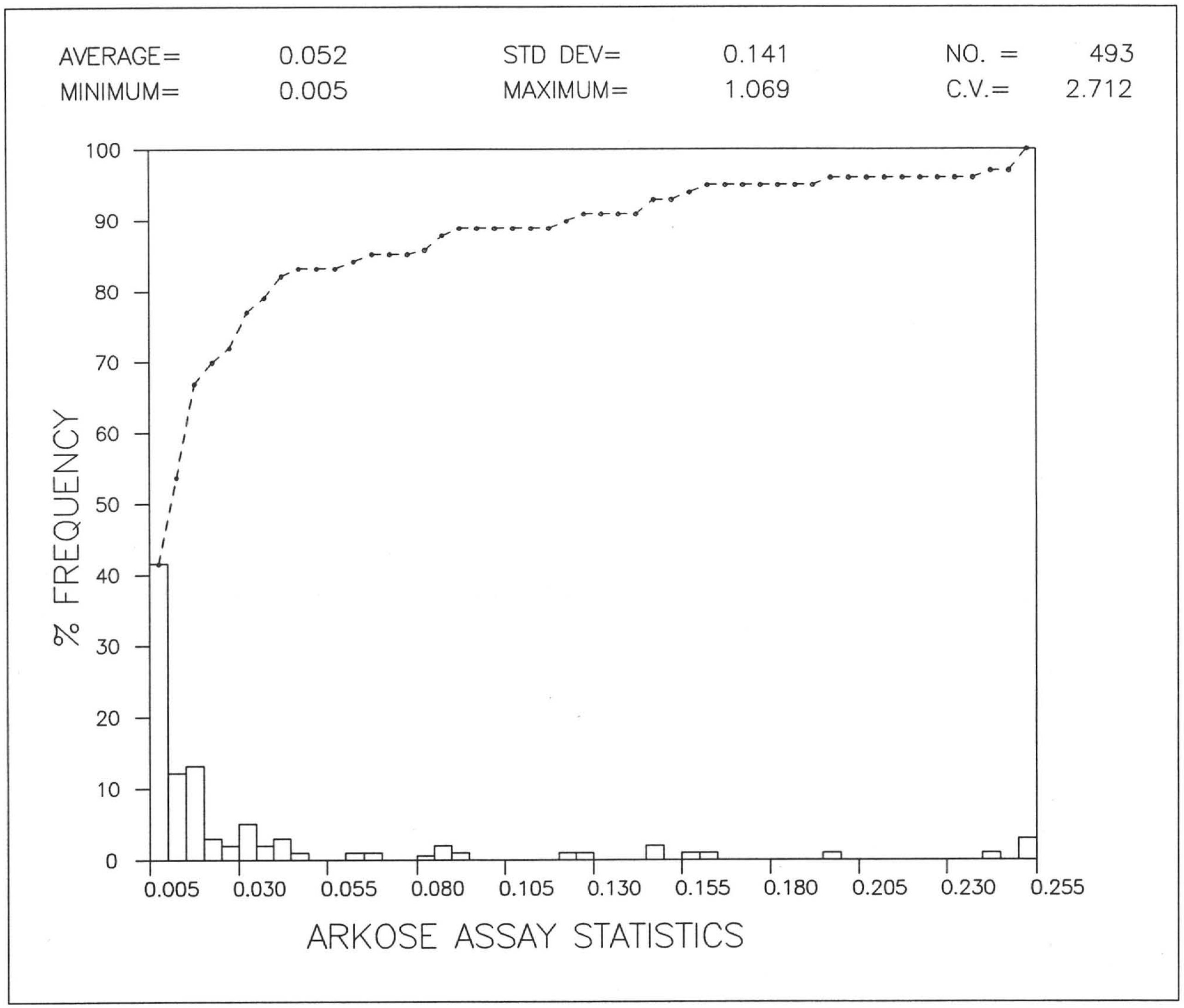
The next step in the analysis was to composite the drill hole data. To simplify the mine planning process, it is necessary that the elevations of the composites, the model block and the mining bench, coincide. Although one may not know the best mining bench height, a reasonable estimate can be made. A comparison of statistics of 15, 20, and 25 foot composites was performed and the indications were that the 20 foot composites provided the best compromise between maximum bench height and least dilution.

The block model extends from the 3500 foot elevation, a few feet above highest point on the property, down to 2600 foot elevation, about 100 feet below the limits of the drilling. The drill hole data was composited to 20 foot benches on even elevations. This file of composited drill hole data is the basis for the modelling procedures. Statistics of the composites, shown in Figures 4.8 through 4.13 were developed and compared to the drill hole statistics. These statistics indicate what type grade interpolations will be valid. Data sets with a high coefficient of variance, greater than 1.5, may not be amenable to Kriging. Relatively small data sets, as we have here if we take each rock type separately, can be difficult to interpolate with some of the advanced geostatistical techniques.

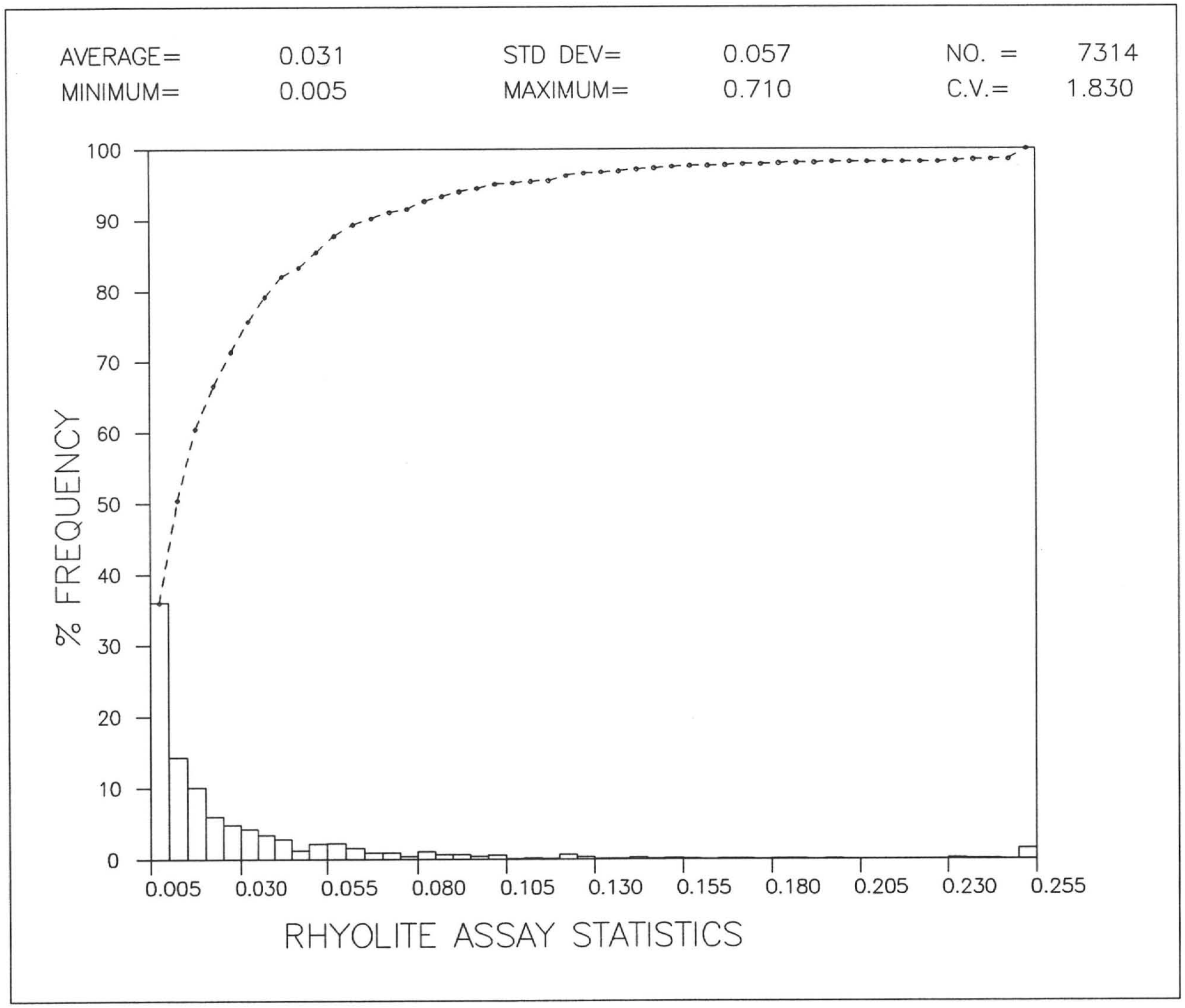
# FIGURE 4.2 HISTOGRAM OF QUARTZ MONZONITE ASSAYS



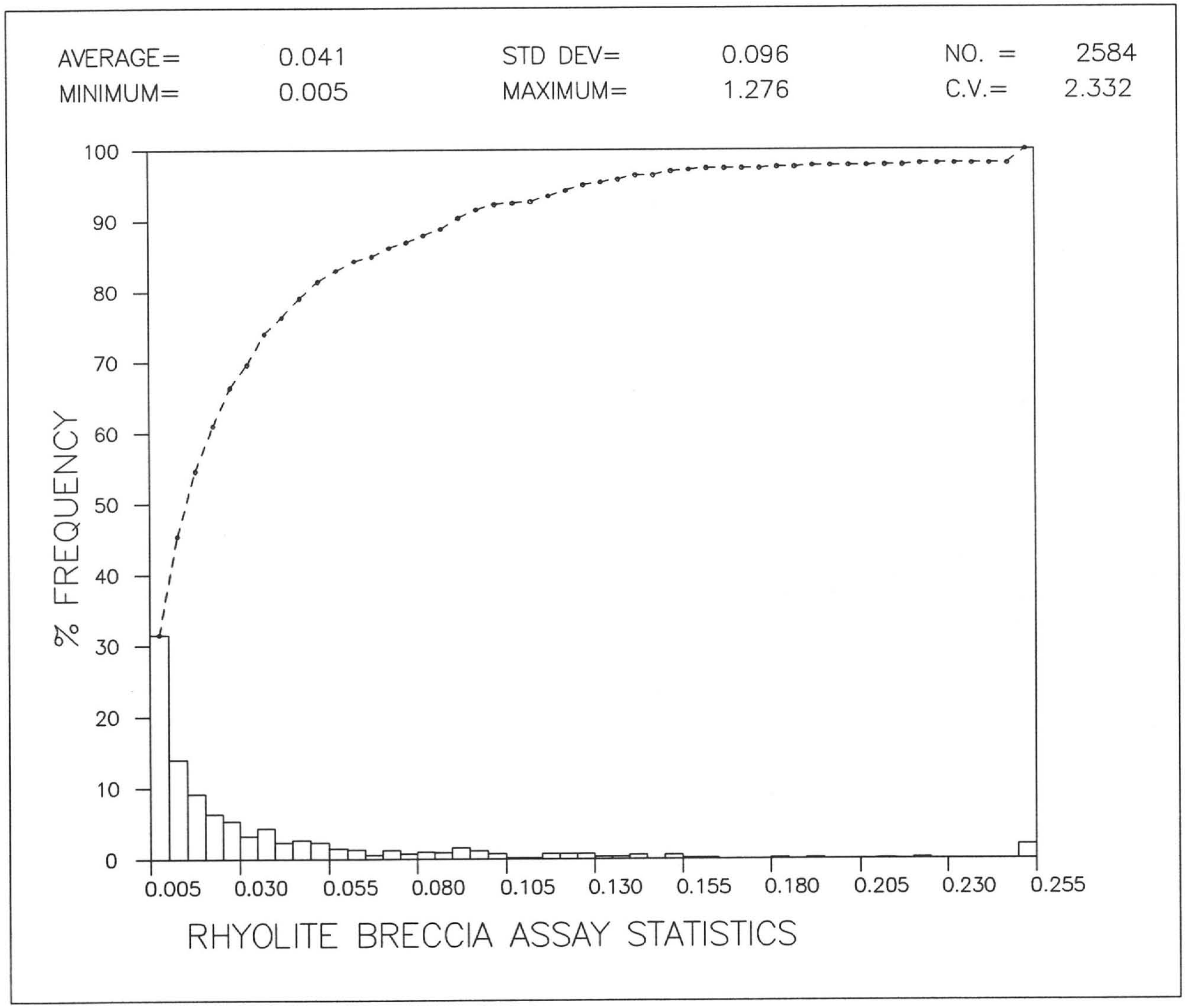
# FIGURE 4.3 HISTOGRAM OF ARKOSE ASSAYS



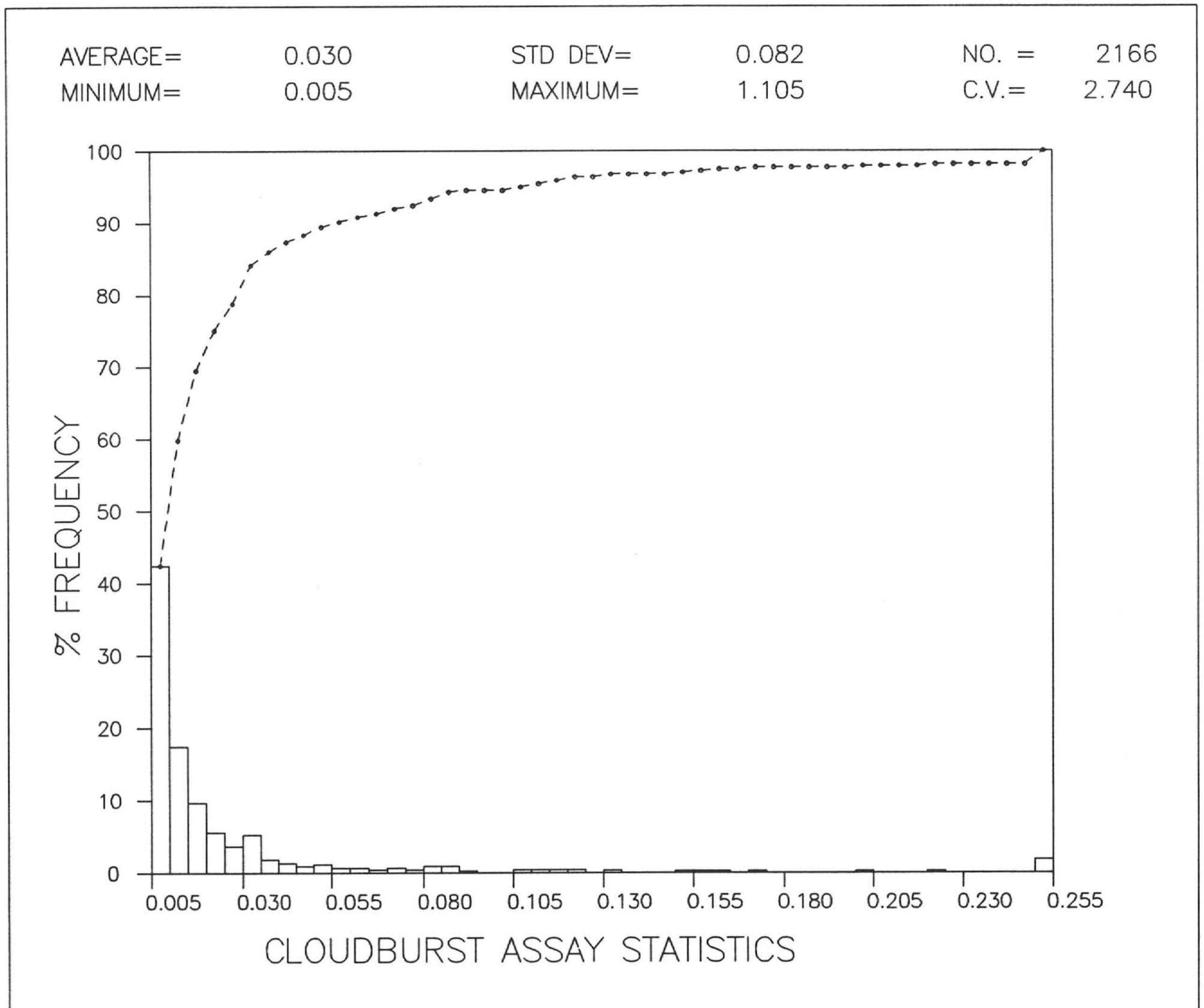
# FIGURE 4.4 HISTOGRAM OF RHYOLITE ASSAYS



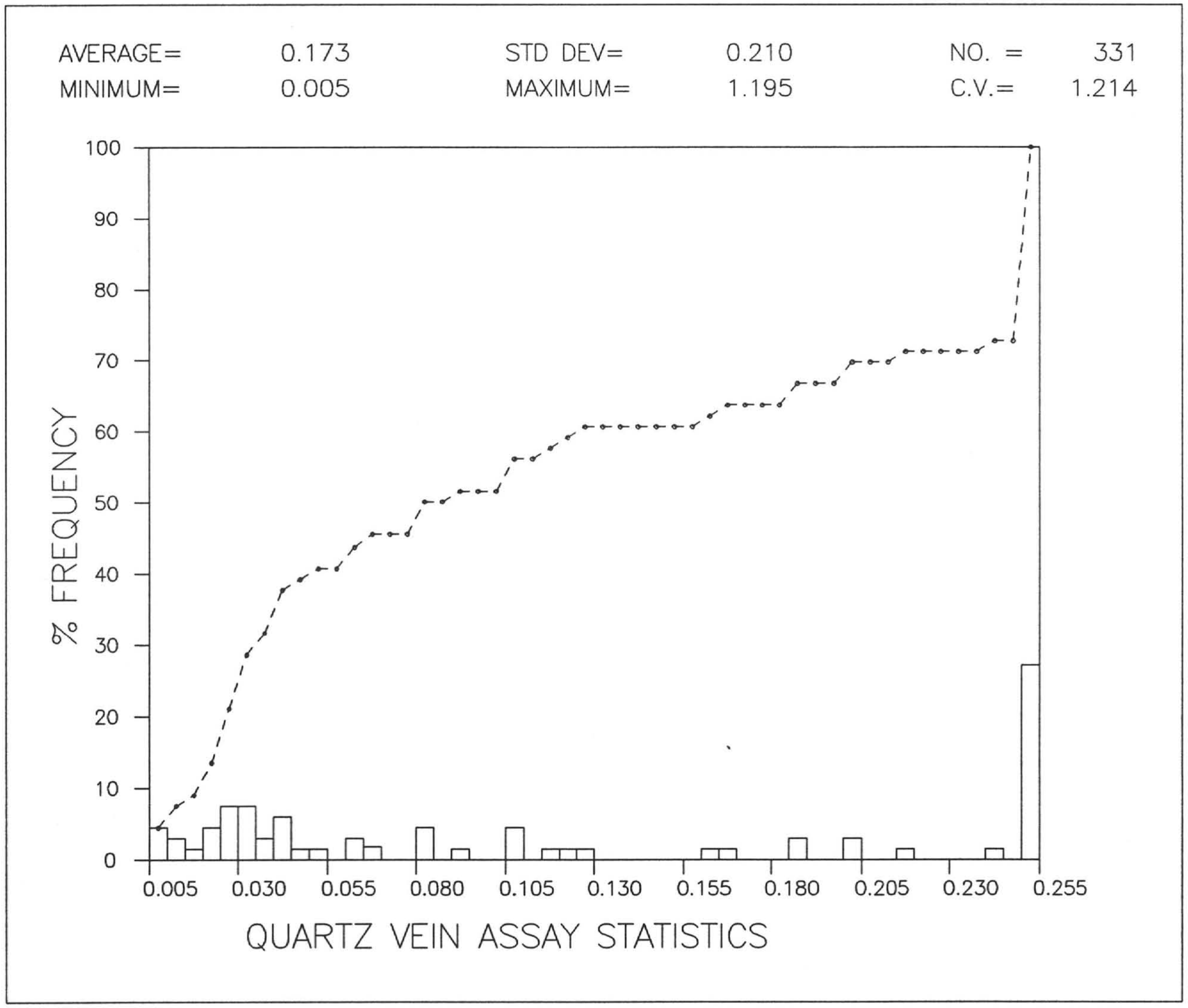
## FIGURE 4.5 HISTOGRAM OF RHYOLITE BRECCIA ASSAYS



## FIGURE 4.6 HISTOGRAM OF CLOUDBURST ASSAYS



# FIGURE 4.7 HISTOGRAM OF QUARTZ VEIN ASSAYS



# FIGURE 4.8 HISTOGRAM OF QUARTZ MONZONITE COMPOSITES

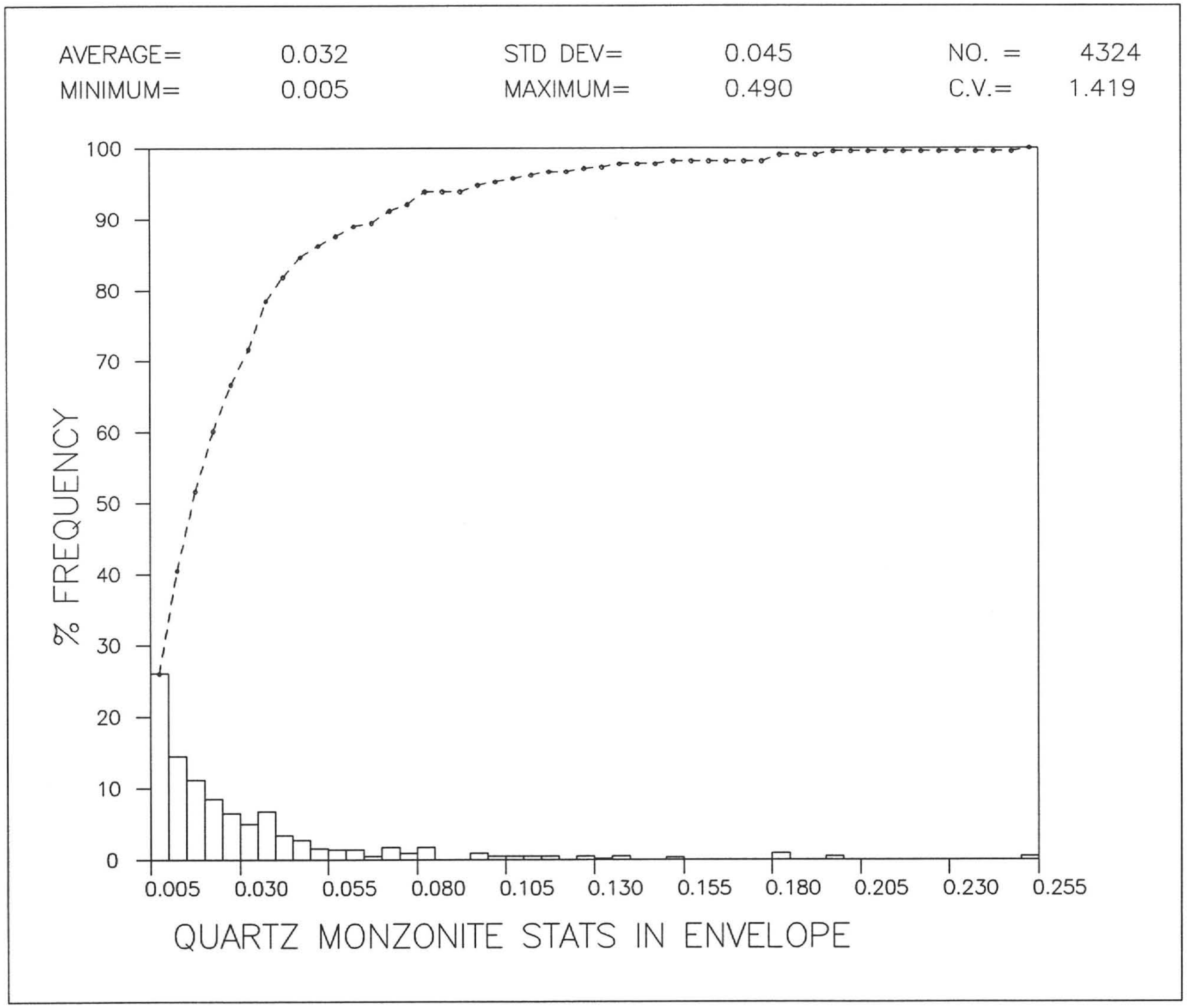
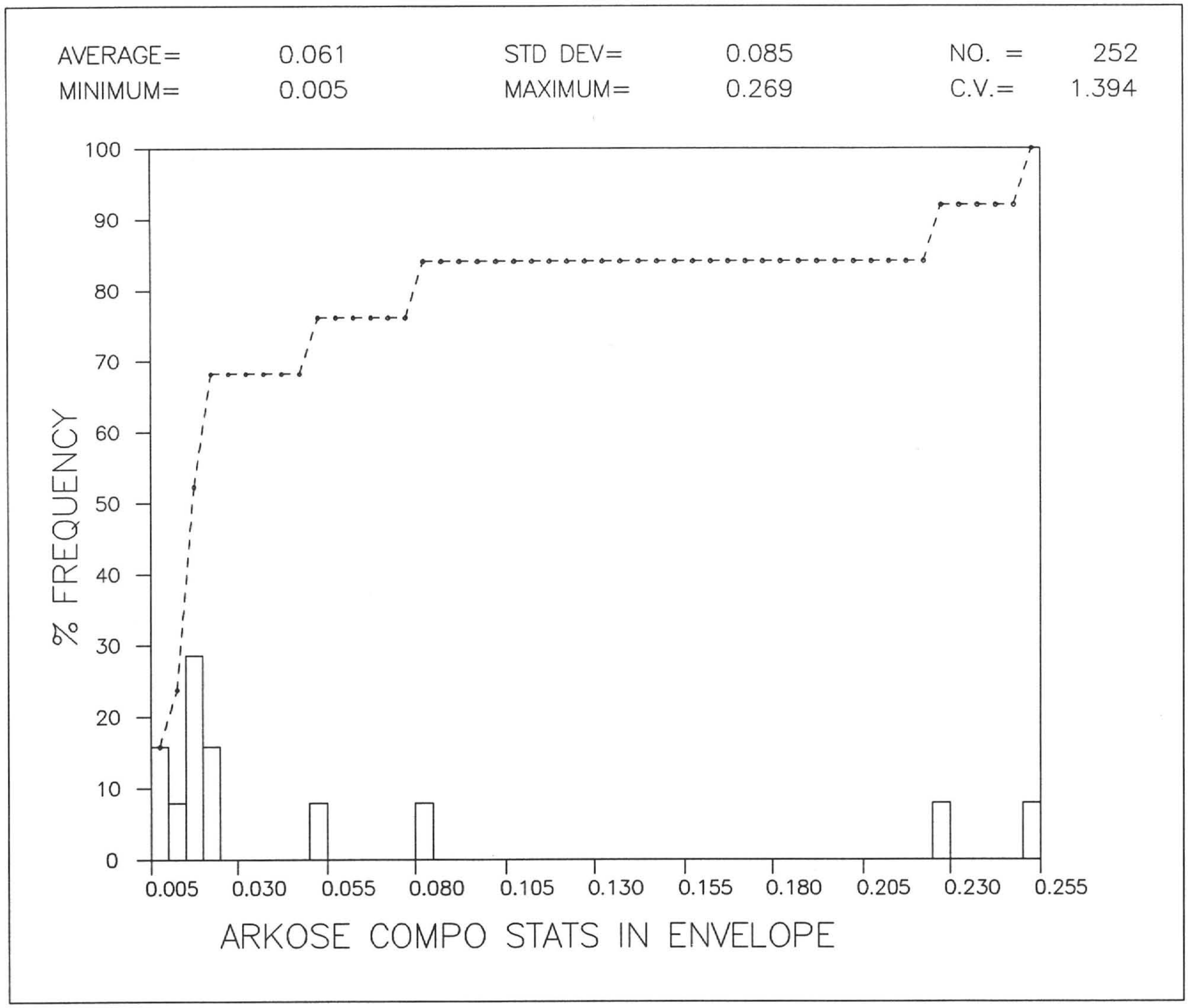
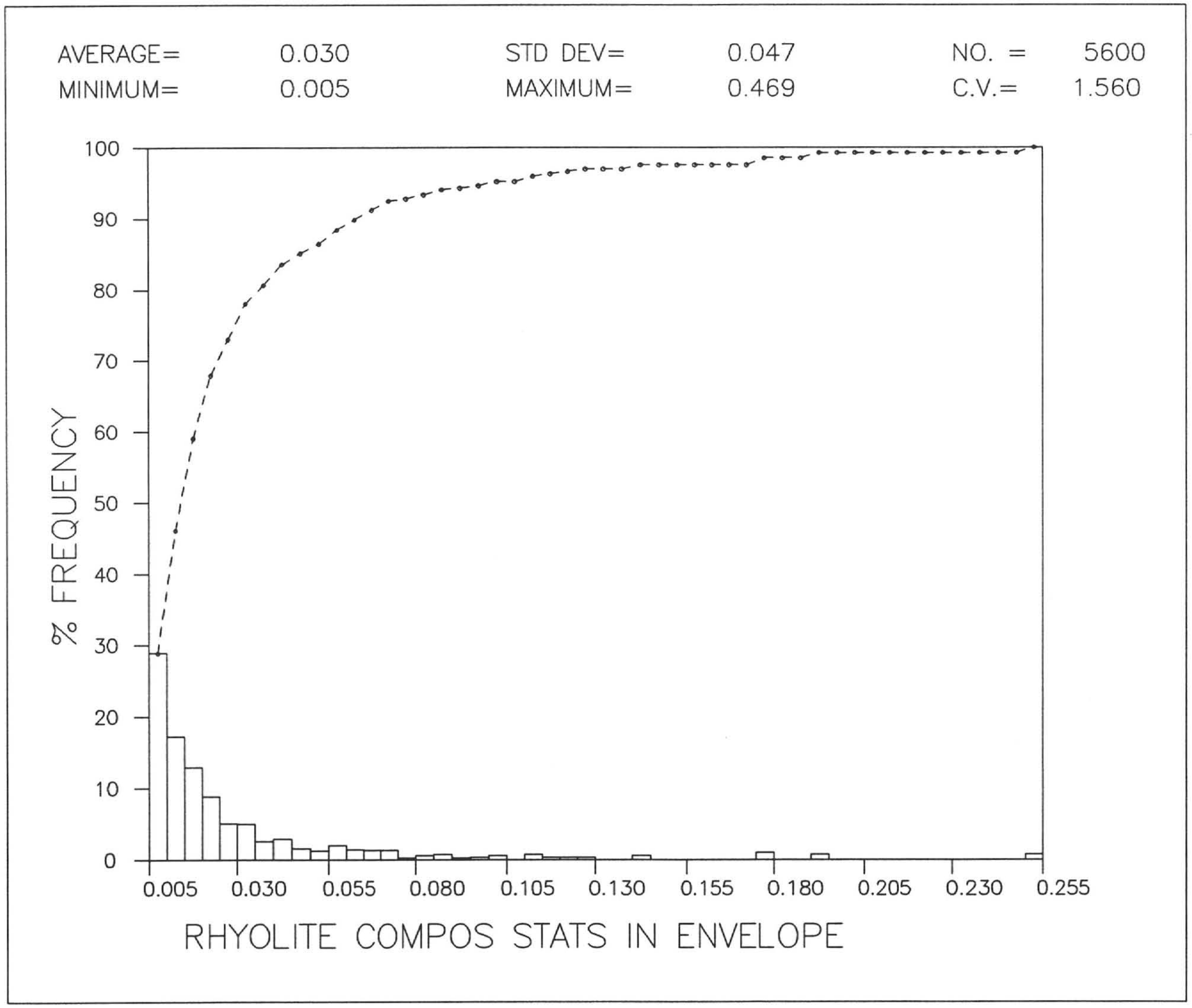


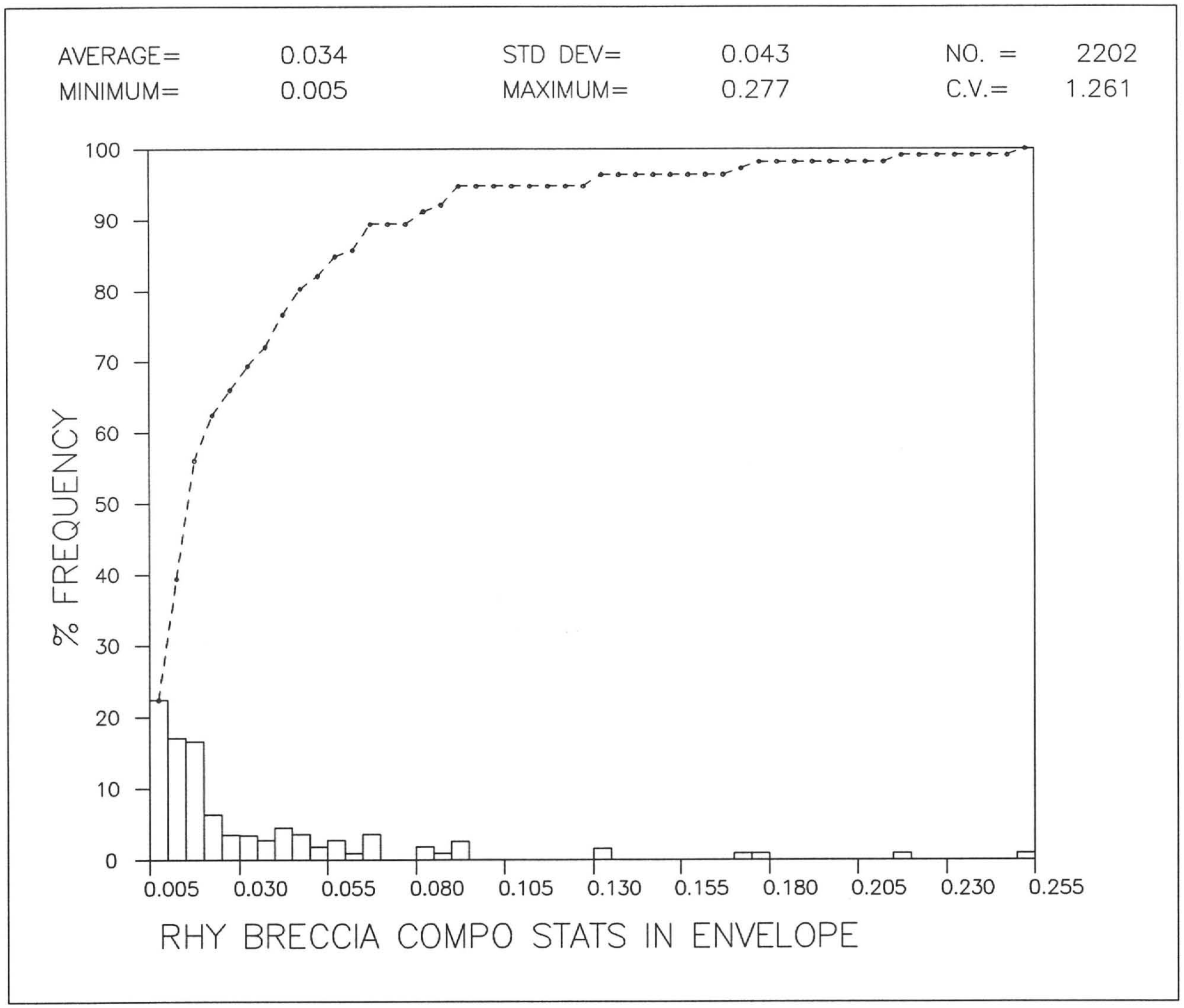
FIGURE 4.9 HISTOGRAM OF ARKOSE COMPOSITES



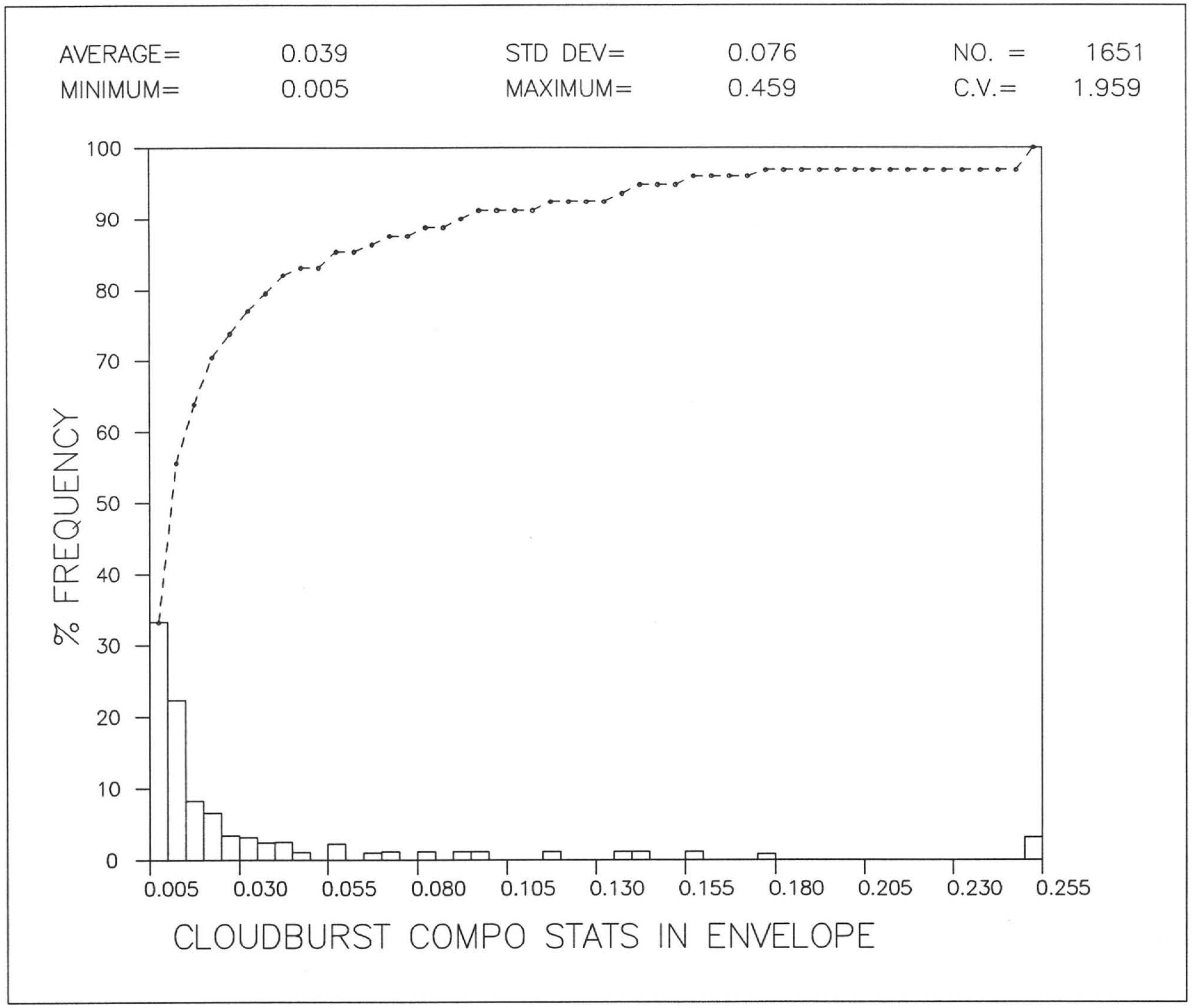
# FIGURE 4.10 HISTOGRAM OF RHYOLITE COMPOSITES



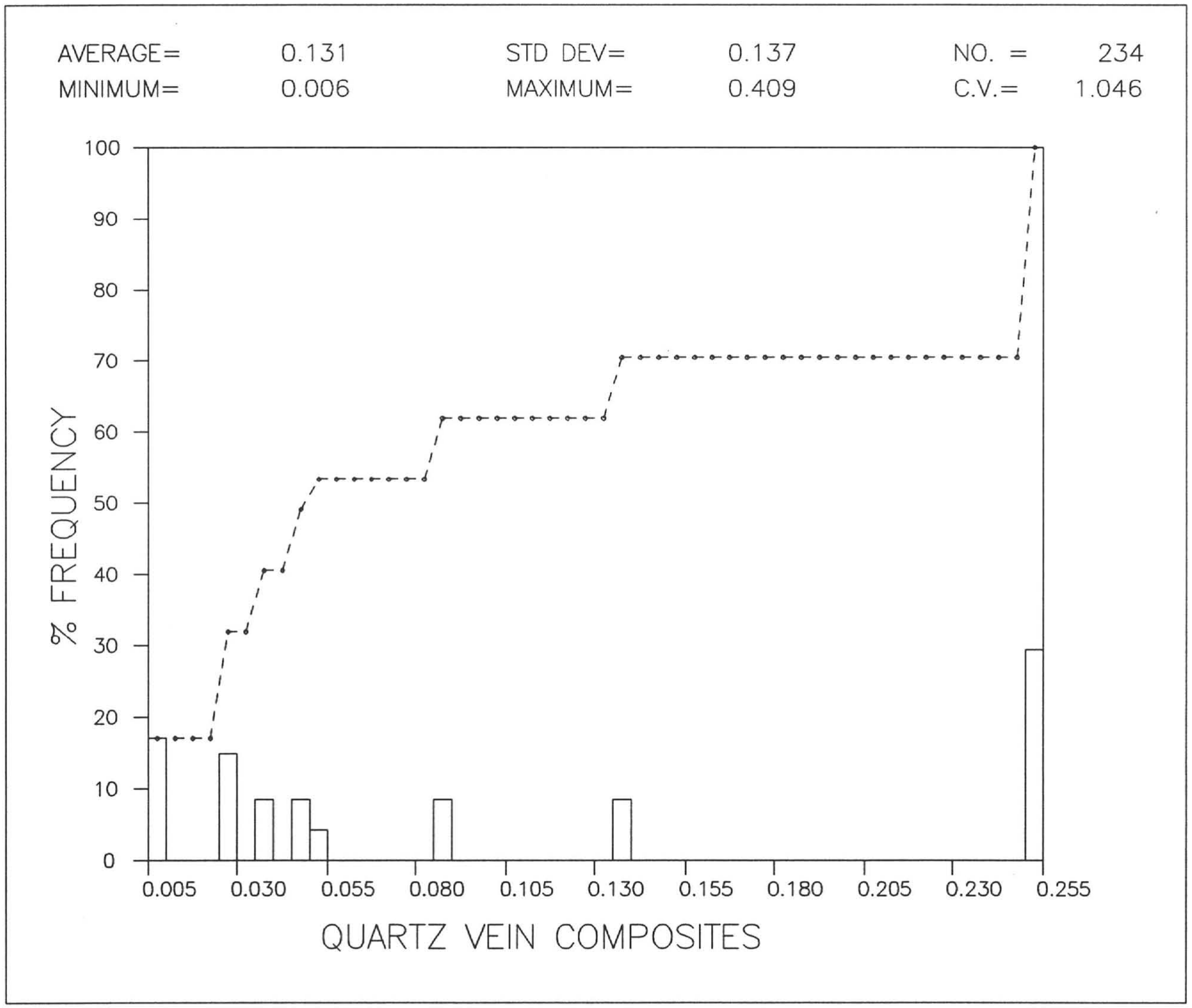
# FIGURE 4.11 HISTOGRAM OF RHYOLITE BRECCIA COMPOSITES



# FIGURE 4.12 HISTOGRAM OF CLOUDBURST COMPOSITES



# FIGURE 4.13 HISTOGRAM OF QUARTZ VEIN COMPOSITES



### 4.3.2 Mineral Envelope

Another difficulty in ore reserve estimation is the disproportionate amount of waste data. The data in this case is in a range near the detection limits of the assaying techniques employed. The statistical variations may be more related to assay error than the natural phenomenon of ore deposition. A large portion of these waste composites are distant from the known mineralized zone and are not influenced by the ore deposition processes.

To limit the analysis and interpolation to only mineralized material, the concept of a mineral envelope was employed to label composites and blocks. On section plots of the drill hole composites, gold grade contours were hand drawn based on various cutoffs. This exercise made clear the structurally controlled nature of this deposit. Contours above 0.040 OPT gold are very closely spaced, where as the lower grade contours are more widely spaced. The widest spacing appears between the 0.010 and 0.020 OPT contours which provide the boundaries of two mineral envelopes digitized into the computer.

The mineral envelopes were drawn with all geologic data in mind. For example, the known Gila conglomerate to bedrock contacts were incorporated into the mineral envelopes. The reconciliation of these envelopes was made using a three dimensional solid modeler included in the MEDSYSTEM. Level maps were generated from the solid model. These plan outlines were checked and then loaded to the block model. Each block has two codes; one indicating it was in or out of the +0.01 OPT envelope; the second indicating that it was in or out of the +0.02 OPT envelope. The prime objectives of the envelope are (1) not to allow projection of mineral into known or projected unmineralized areas, (2) to limit the influence of numerous very low grade composites, and (3) to limit the influence of the very high grade composites.

The computer was then used to assign the mineral envelope codes to the composites. Statistics and variograms were developed for only those composites within the delineated mineralized boundaries, eliminating extraneous statistical information or "noise".

### 4.3.3 Variography

The development of variograms was the next step to understanding the spatial variability of gold grades in the deposit and the selection of the best interpolation method. Variograms were developed for each rock type along the strike, down dip, and perpendicular to the strike of the vein. Data used to calculate these variograms were limited to the largest contiguous mineral zone located about the Mammoth shaft and flux pit. This was necessary to avoid complications of changing strike and dip and to exclude the influence of small insignificant parallel zones.

Another complication of the variogram analysis was the extremely high grade composites. As is typical with precious metal vein deposits, high grade zones constitute a small percentage of the deposit volume, but contain a large proportion of the metal. Great care must be exercised in interpolating block values from high grade composites, especially regarding their range of influence. Traditionally, the high grade values are set back to some "cut value" and used in the interpolation as any other value or simply ignored altogether. Journel and Arik[1] comment that, "The ideal would be to delineate all such high grade mineralizations and limit the extrapolation of high grade data to these zones." Detailed geologic mapping of these very high grade zones would be ideal and could facilitate the type of interpolation suggested. Drill cuttings lack such information and inaccessible underground workings make direct mapping impossible.

At Tiger, a combination of the traditional and the ideal suggested by Journel and Arik was derived. A cut value of 0.17 OPT was selected for several reasons. It is approximately 3 times the average grade of the expected minable reserve. Less than 3% of the composites are above this value. From the costs reported from about 1900, it is estimated that the cutoff grade was approximately 0.17 to 0.18 OPT. Also in 1900, T. J. Davey reported no remaining reserve above the 700 level of the Mammoth Mine. The 700 level corresponds to the 2540 elevation, 60 feet below the block model limits. An attempt at a lower cut value of 0.10 OPT, based on a probability distribution plot of gold values, produced an overly

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[1]A. G. Journel and A. Arik, Dealing With Outlier High Grade Data in Precious Metals Deposits, Computer Applications in the Mineral Industry, Rotterdam 1988

conservative model. The ideal was introduce by assigning a discriminator to the blocks. This strategy is discussed in the following section.

Figure 4.14, lists the results of the variograms generated on the Mammoth zone with composite values above the 0.17 OPT cut value excluded. The variogram plots are found as Figures 4.15 through 4.19

Figure 4.14 Variogram Results by Rock Type

| Rock Type | No. Sample Pairs | Mean Au Grade | Variance Au Grade | Nugget | Range | Leach Recovery % |
|---|---|---|---|---|---|---|
| Quartz Monzonite | 255 | 0.024 | 0.0007 | 0 | 130 | 60 |
| Rhyolite | 392 | 0.017 | 0.0005 | 0.0002 | 92 | 53 |
| Rhyolite Breccia | 130 | 0.023 | 0.0008 | 0 | 54 | 51 |
| Cloudburst | 133 | 0.016 | 0.0008 | 0 | 46 | 52 |
| Quartz Vein | No readable | variogram | | | | 53 |
| Arkose | No readable | variogram | | | | not sampled |
| All Rock Types | 1073 | 0.021 | 0.0007 | 0.0003 | 104 | N/A |

The statistics of the individual rock types produced variograms relatively close to each other and to the whole. All rock types were treated together in the remaining statistical analyses and metal grade interpolations. At the same time these analyses were being done, metallurgical test results (last column Figure 4.14) available, indicated that expected recovery of gold was dependent on rock type. Based on that information, blocks were classed into three categories; Quartz Monzonite or all rocks of pre-cambrian age (60% recovery), Tertiary or all rocks of tertiary age (51% to 53% recovery), and Gila or Gila conglomerate and alluvium. Quartz Monzonite and Tertiary age units are the potentially mineralized rock classes and Gila conglomerate was always treated as un-mineralized. These classifications are used throughout the design and evaluation process to determine gold recovery, mining costs, and in reporting reserves.

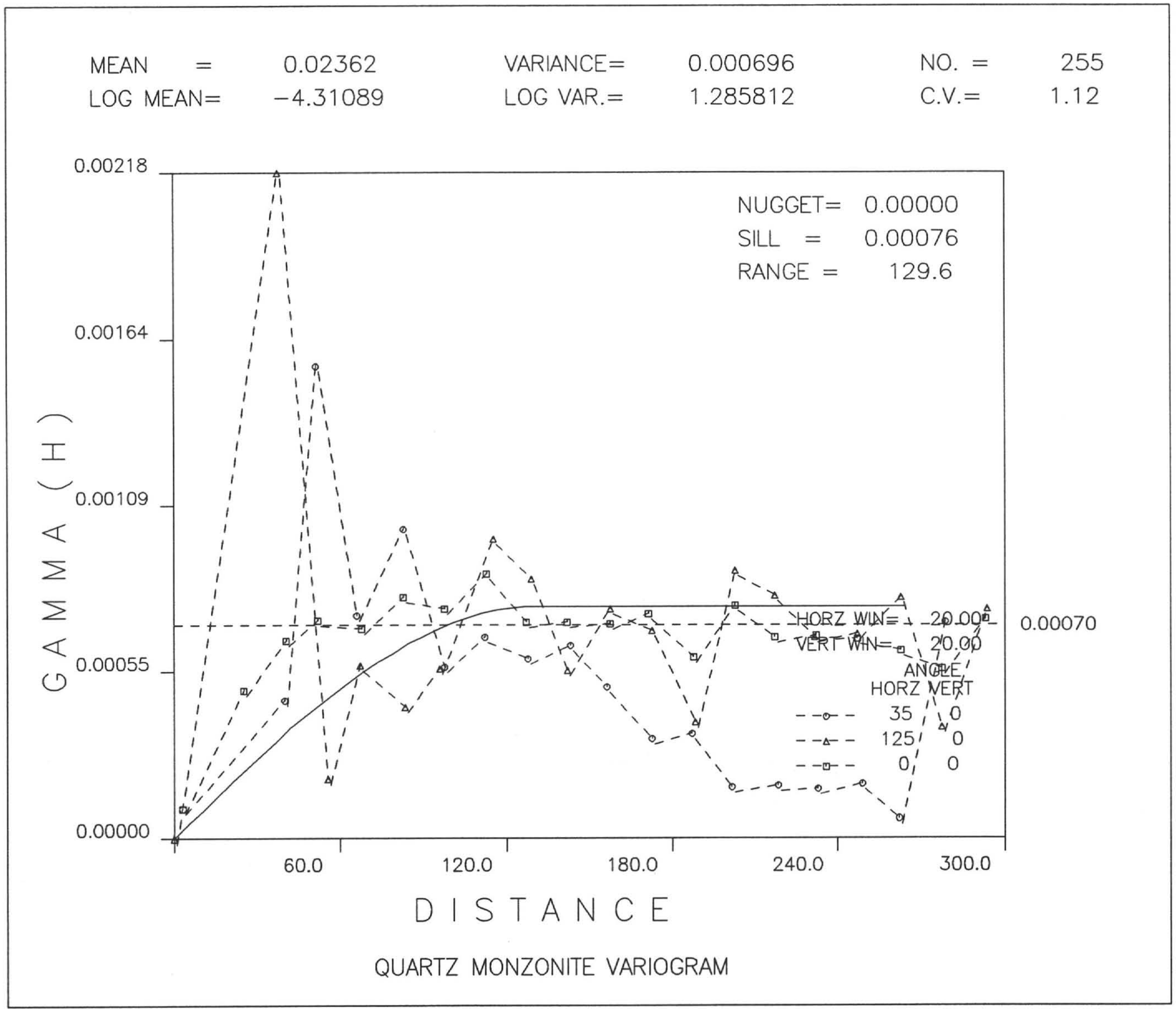



FIGURE 4.15 VARIOGRAM OF QTZ. MONZONITE IN MAMMOTH ZONE

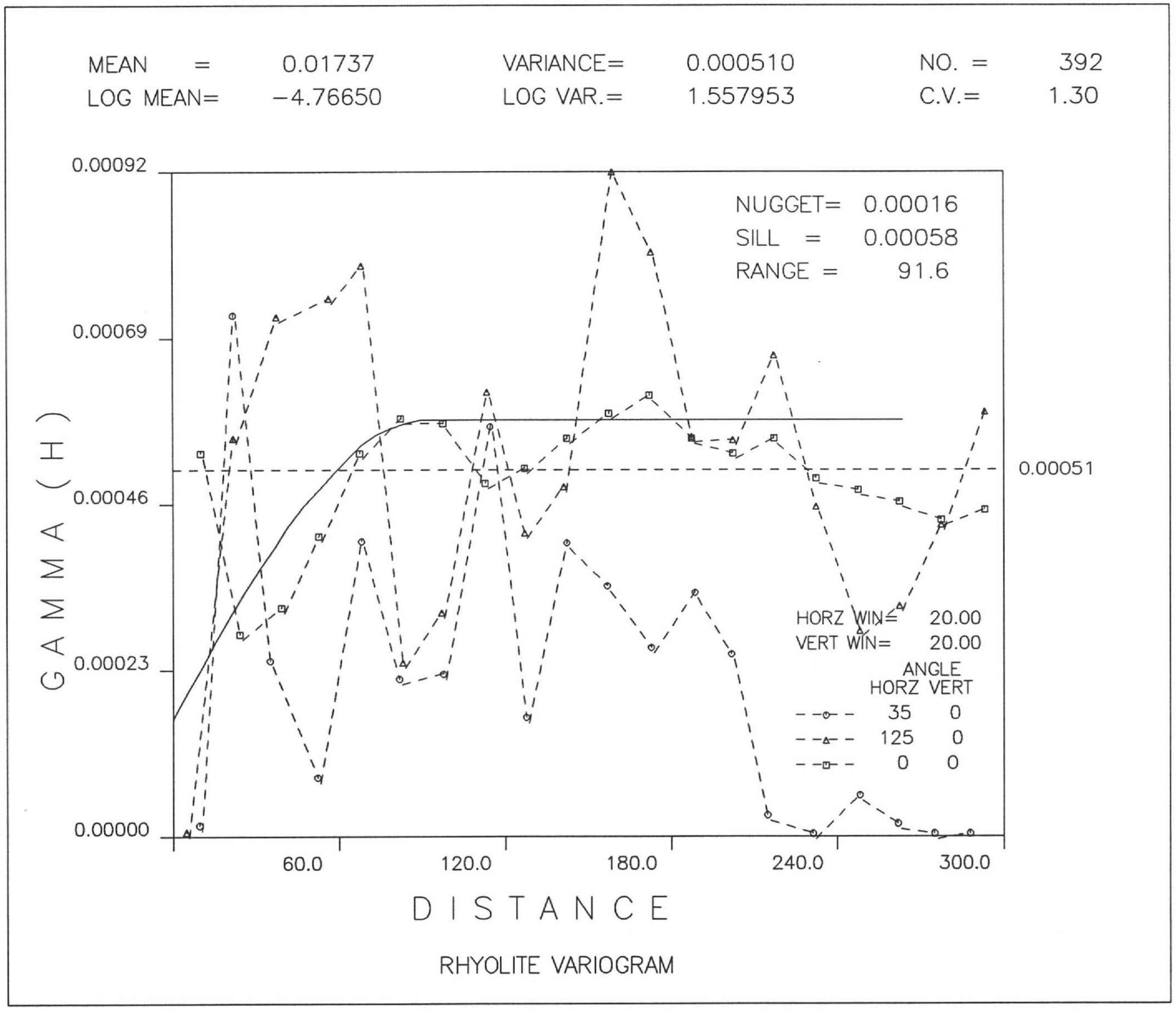



FIGURE 4.16 VARIOGRAM OF RHYOLITE IN MAMMOTH ZONE

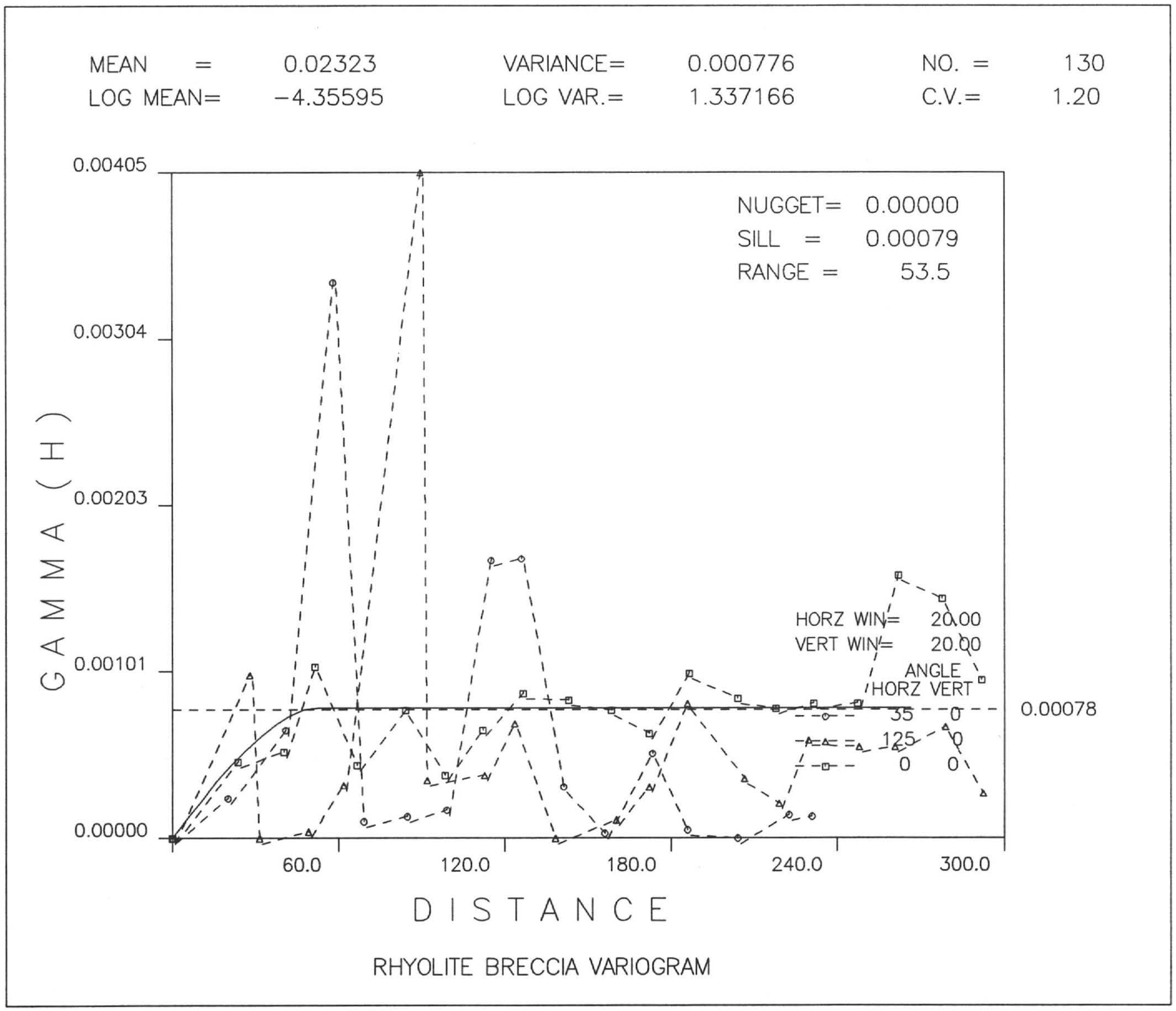



FIGURE 4.17 VARIOGRAM OF RHY. BRECCIA IN MAMMOTH ZONE

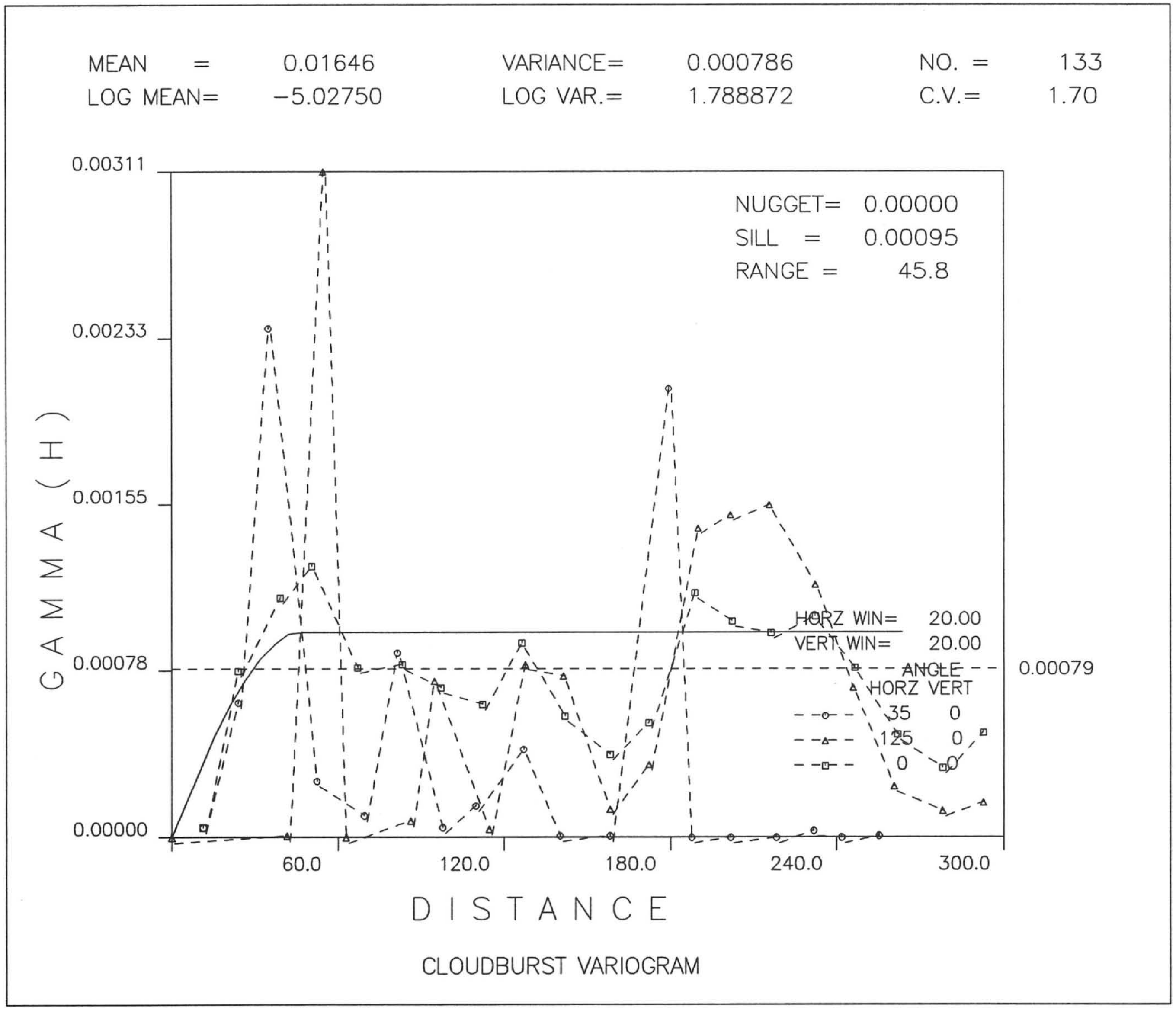



FIGURE 4.18 VARIOGRAM OF CLOUDBURST IN MAMMOTH ZONE

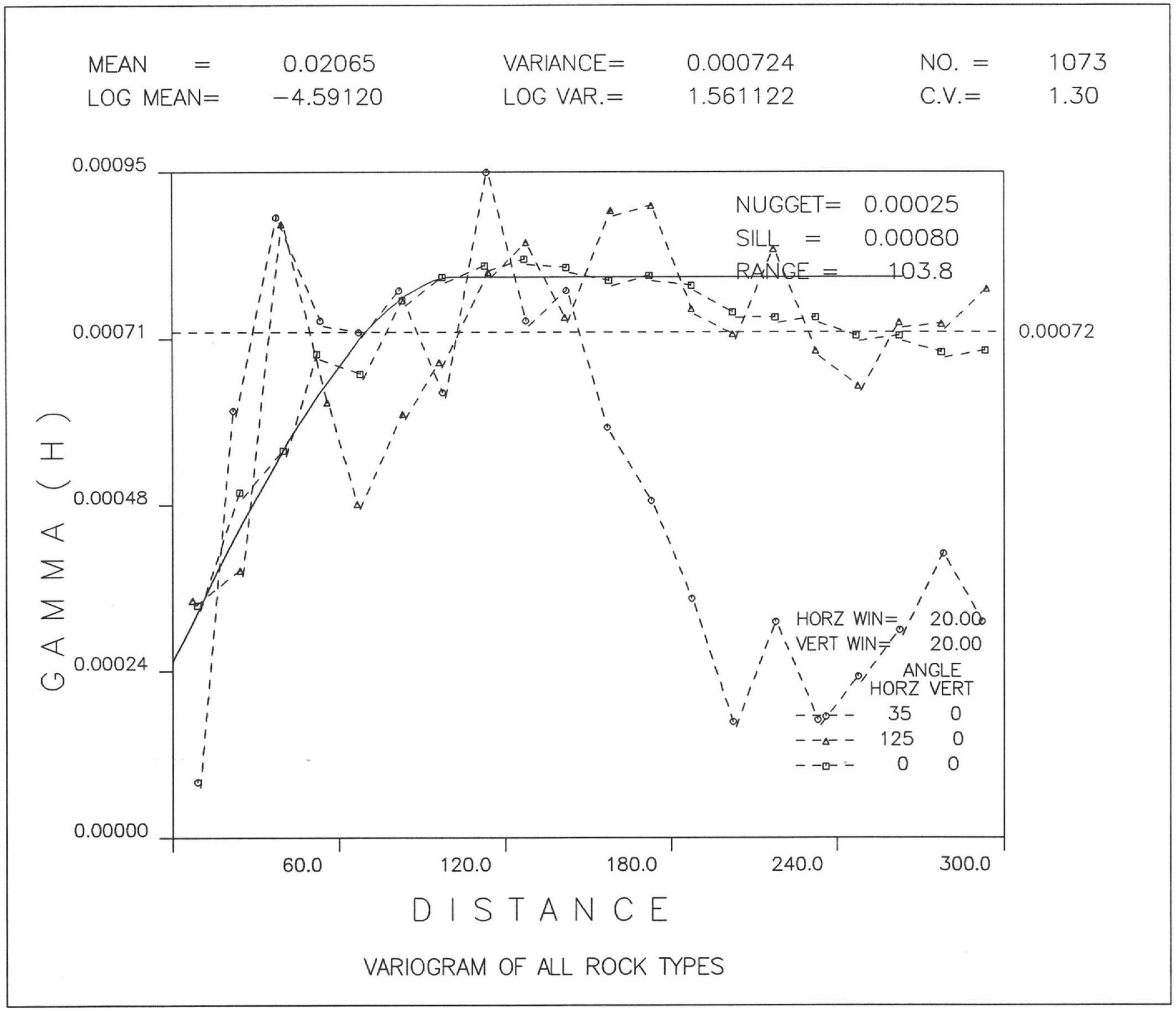



FIGURE 4.19 VARIOGRAM OF ALL ROCK TYPES

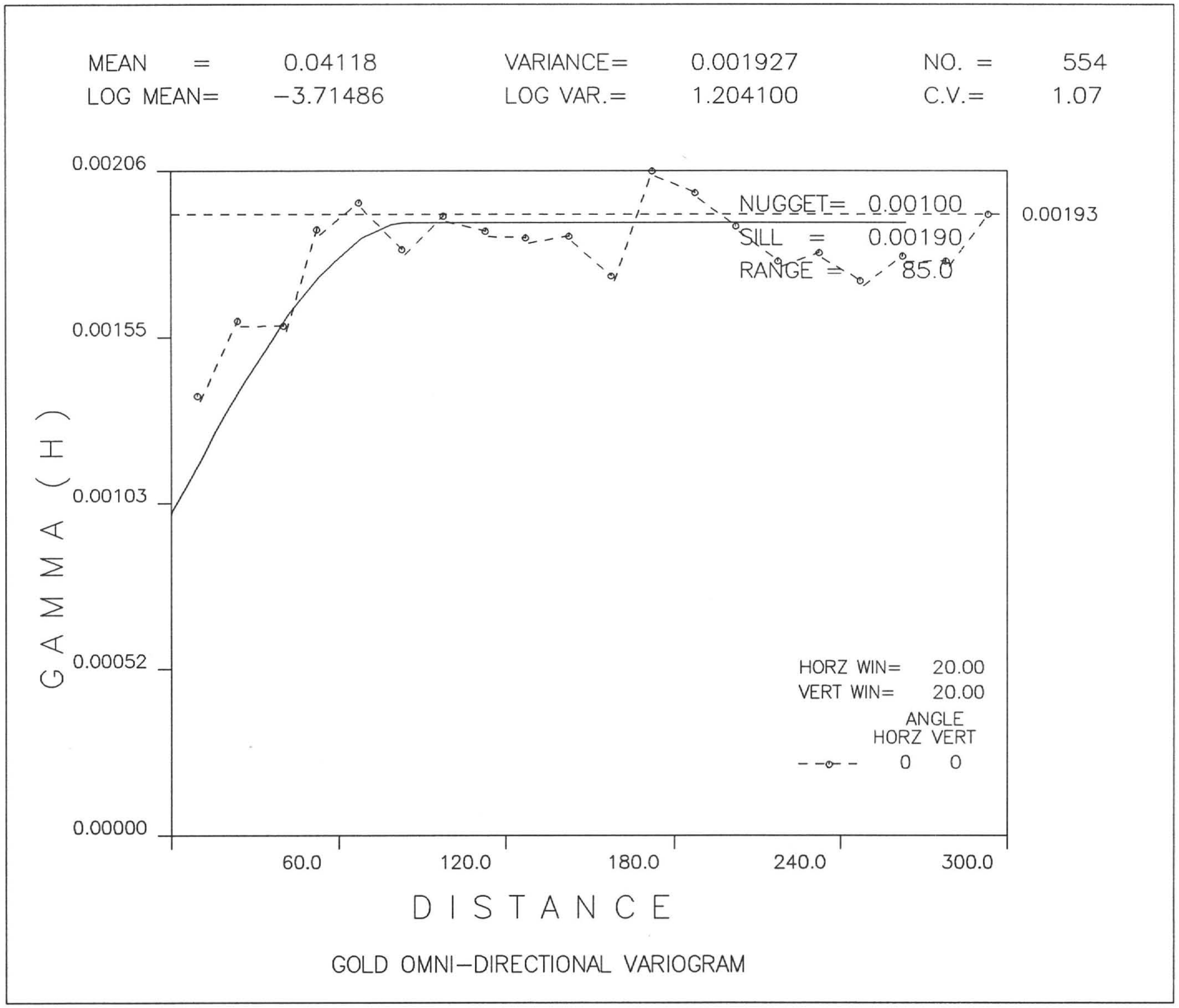



FIGURE 4.21 OMNI-DIRECTIONAL VARIOGRAM IN MAMMOTH ZONE

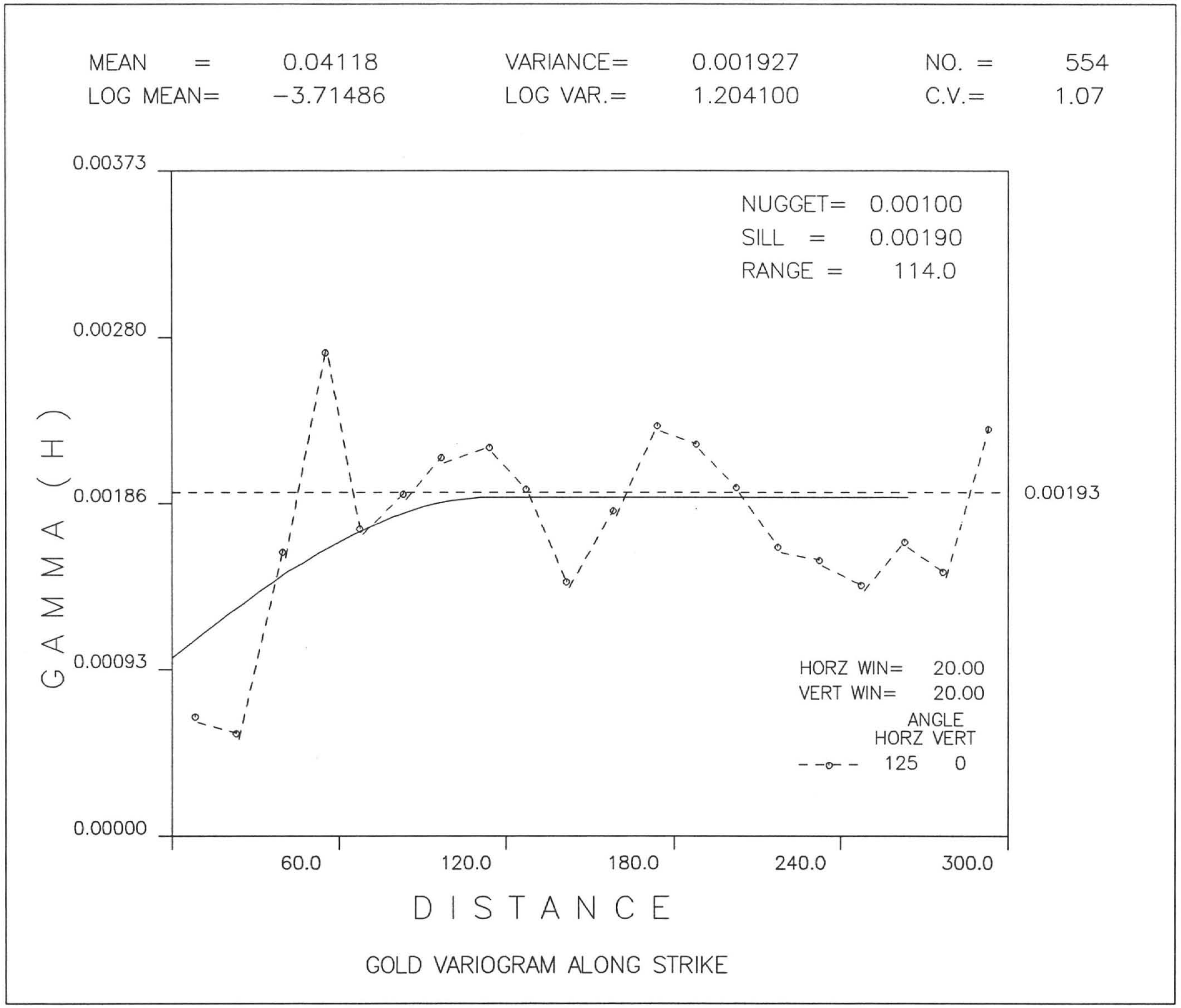



FIGURE 4.22 VARIOGRAM ALONG STRIKE IN MAMMOTH ZONE

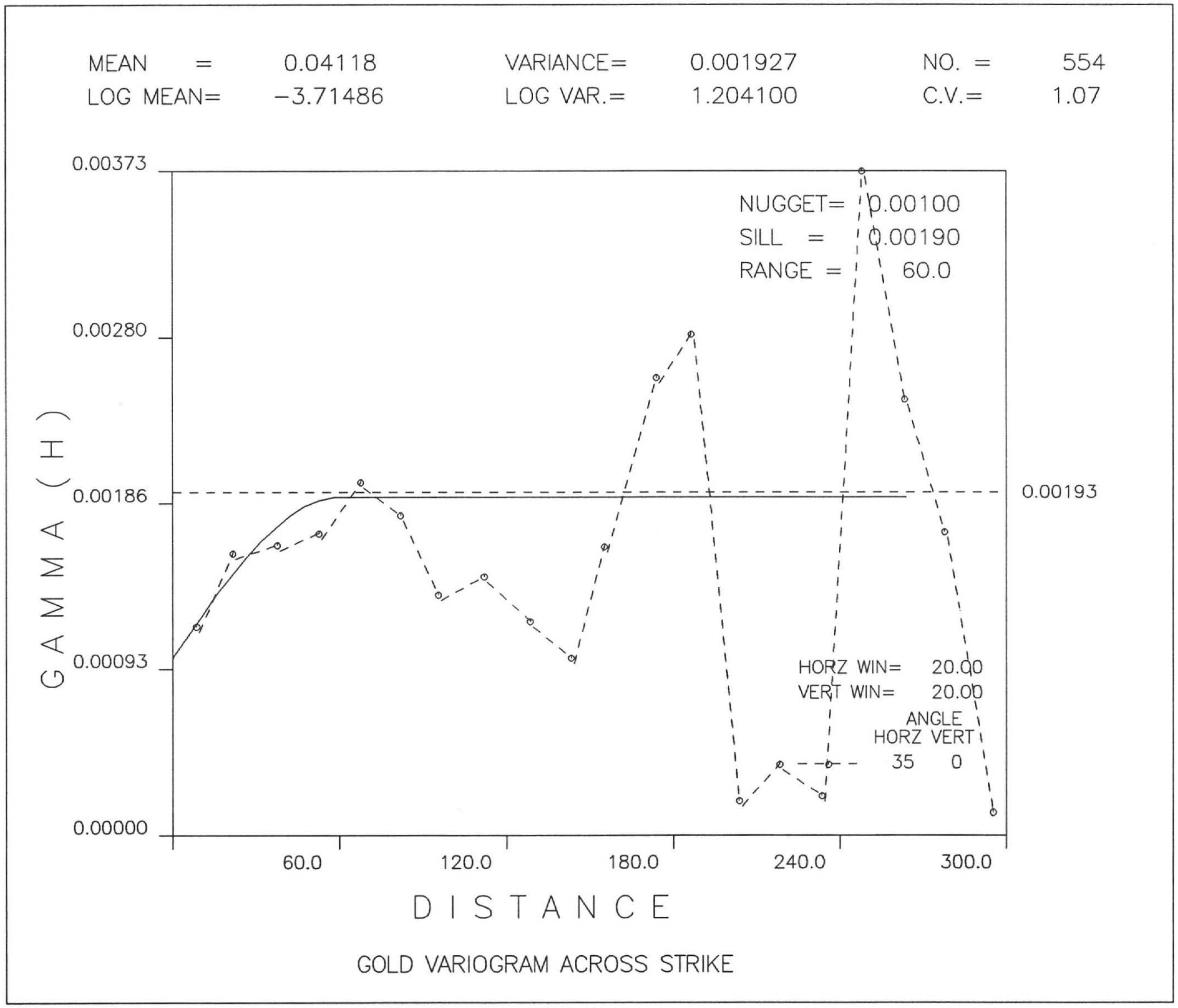



FIGURE 4.23 VARIOGRAM ACROSS STRIKE IN MAMMOTH ZONE

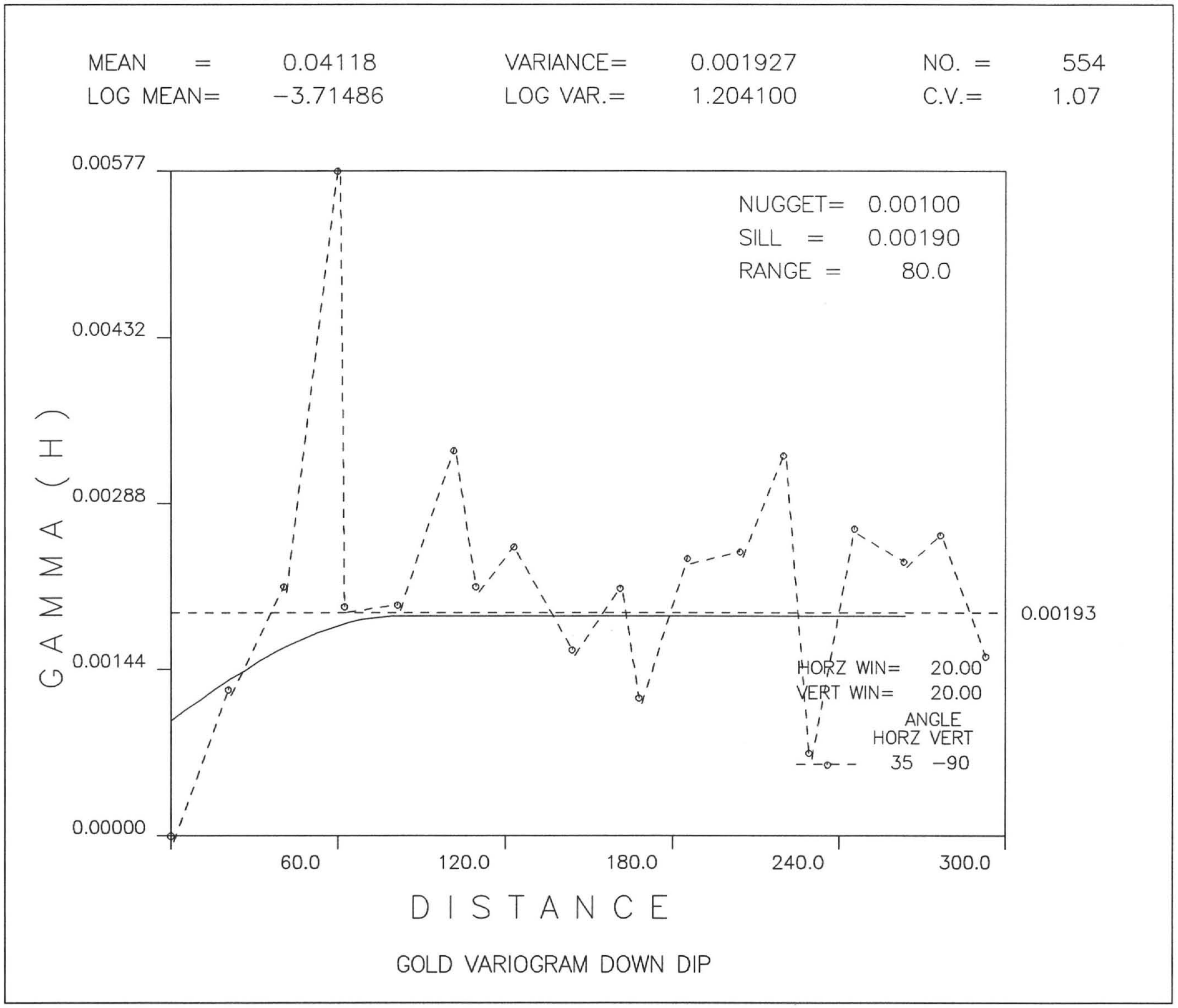



FIGURE 4.24 VARIOGRAM DOWN DIP IN MAMMOTH ZONE

The variograms were calculated of all rock types taken together to determine the interpolation parameters. These variograms used the cut grades of the composites. These grades would be used in the interpolation. Figure 4.20 lists the parameters derived from these variograms. (Figures 4.21 through 4.24)

Figure 4.20 Interpolation Parameters from Variography

| | |
|---|---|
| Nugget | 0.0010 |
| Sill (Variance) | 0.0019 |
| Range (omni-directional) | 85 |
| Range (parallel to strike) | 114 |
| Range (perpendicular to strike) | 60 |
| Range (down dip) | 80 |

The last three ranges listed define an ellipsoid within which the gold grades are correlated. Although they were developed in a limited area, these parameters are used to interpolate gold grades throughout the deposit. This is a valid extrapolation because the vein was formed by the same geological processes throughout and only distorted after mineral deposition.

### 4.3.4 Metal Grade Interpolation

The idea of interpolating the high grade zone separately, suggested by Journel and Arik, was introduced by assigning a discriminator to the composites. Using the actual gold values, composites greater than 0.17 OPT were assigned a discriminator of 1. Composites less than or equal to 0.17 OPT, were assigned zero. Then, the blocks in the model were assigned a discriminator value (from 0 to 1) using an inverse weighing interpolation of the composite discriminator values (0 or 1). A block could then be classified as high grade if its discriminator value was 0.5 or greater. The high grade data and the high grade blocks (the outlier population) were defined and labeled in the composite file and the block model.

At this point, the model was loaded with the two sets of mineral envelopes, the rock classifications, and the high/low grade discriminator. Each composite was given two gold grades, the actual and the cut values, a high/low discriminator, mineral envelope codes loaded from the model, and rock classifications. The search ellipsoid (ranges parallel to strike, perpendicular to strike, and down dip), the nugget (0.001), and the sill (0.0019), were determined by the variography.

The best interpolation method and parameters were now to be determined. For the blocks with a high/low discriminator value less than 0.5, well over 90% of the blocks inside either mineral envelope, the answer was straightforward. The search ellipsoid or the distance that a composite would be extrapolated, was limited to 114 feet along strike and 60 feet perpendicular to strike. Although the variograms indicated an 80 foot range down dip, it was decided that, since drill holes are biased vertically, a limit of two benches up or down (40 feet) would be imposed on the interpolation. This follows a traditional practice in ore reserve estimation. The statistics generated show a coefficient of variance of 1.07. This value is acceptable for Kriging to be valid.

The first interpolation estimated the gold grade of blocks between the + 0.01 OPT and + 0.02 OPT envelopes. This run used only the composites in the same envelope. No high grade composites or blocks occur in this zone. Blocks in the zone but not interpolated, due to the absence of data inside the search ellipsoid, were assigned zero grade.

Then gold grades were assigned to the blocks inside the + 0.02 OPT envelope. The cut value of the composites inside the zone were used to interpolate the grades of the blocks inside the zone regardless of the high/low discriminator values. No blocks were assigned values above 0.17 OPT. Later, the blocks with a high/low discriminator of 0.5 or greater were re-interpolated. If the high grade interpolation method does not estimate a grade for a high grade block, the previous interpolation prevails. It should be noted here that some blocks were assigned values less than 0.02 OPT in the + 0.02 OPT envelope. There are occasional composites in the envelopes of lower grade than the boundary cutoff would indicate. The envelopes do contain very low grade material. Similarly, the + 0.01 OPT envelope contains material below 0.01 OPT.

There are other very important interpolation parameters used in the resource modelling. The minimum number of composites required to interpolate the grade of a block can be varied. Generally, the more composites used to interpolate a block grade, the higher the confidence in that interpolation. However, the more composites required, the fewer the blocks that are assigned grades due to the limits of the data. The lower the number of required composites the more confident one has to be in other interpolation parameters. The Tiger model required only one composite for the low and average grade gold estimations. For the high grade estimates more composites were required. The use of a single composite to determine a block grade can also be limited. The maximum distance to project a single composite, where no others fall within the search ellipsoid, can be set. The maximum used for the low grade interpolations in Tiger model was 85 feet or the range of the omni-directional variogram. Another parameter related to the distance from a block to composites is the maximum distance to the closest composite. When there are multiple composites available to interpolate a block this parameter requires that at least one be no less than this distance away. For Tiger, the maximum distance to the closest composite was set to 85 feet, again from the variogram. For example, we have only 3 composites in the ellipsoid, about 95 feet away from a particular block. That block would not be assigned a grade since there are no composites within 85 feet. If one of our hypothetical composites was 60 feet away and in the ellipsoid, all three composites would be used.

It is also possible to limit the number of composites from a particular drill hole. This was not done in estimating the gold grades to this point. This option was used to test the interpolation of the high grade blocks. The objective of the second Magma drilling program was related to that test. The model, developed as described using the pre-1991 drilling data, still required an estimation of the high grade (discriminator greater than 0.5) blocks. The actual composite values and the same parameters used before were used to estimate the grade of the high grade blocks (Model #6). This resulted in a pit with 93,000 recoverable ounces of gold. A second interpolation of the high grade blocks (Model #7), requiring two composites and not more than one composite from an individual drill hole resulted in a pit with 49,000 recoverable ounces. The majority of this difference was the assignment of very high grades (+ 0.3 OPT gold) by the first method to a cluster of blocks near the bottom of the drilling limits around one hole. The second method did not assign high grades to these blocks due to the fact that two drill holes with high grade composites were not available to those blocks. Other areas within the model showed similar characteristics, but the cluster at the pit bottom became the focus of the second Magma drilling program.

As stated earlier, this high grade zone was disproved by the drilling. Intercepts, that were expected to be high grade or at least 0.10 OPT, were near a minable cutoff grade insufficient to support stripping. A block containing 0.30 OPT gold can support a significant volume of stripping. When a Model #6 type interpolation was done using the new drilling, fewer ounces were indicated than Model #7 in the test area. A better interpolation method for the high grade blocks was required.

The high/low discriminator was sound and the best way to delineate the high grade zones given the data. Adjustment of the interpolation parameters was required. There was an insufficient number of high grade composites to derive variograms for only the high grade so the statistical parameters used in the lower grade estimates were maintained. The remaining parameters that could be adjusted were the number of composites required and the maximum distances to project and accept data. At this point, several test models were generated. These parameters were adjusted and the results compared. Figure 4.25 lists the key parameters of this analysis.

Figure 4.25 Key High Grade Interpolation Parameters

| | Max. distance to closest composite | Required No. of composites | Max. Composites per Drillhole |
|---|---|---|---|
| Model #6 | 85 | 1 | no limit |
| Model #7 | 85 | 2 | 1 |
| Model #11 | 60 | 3 | no limit |

Model #11 performed the best in the testing. These parameters were selected more on the visual study of the composite section plots with block gold values than on direct statistical analysis. The results when plotted in plan give a reasonable interpolation of the high grade zones. This model falls between the two extremes of Model #6 and Model #7 type interpolations. The selection of Model #11 as the correct model, as related to the high grade interpolation, was based on familiarity with the available data and the experience of the engineer.

The mines of Tiger exploited the high grade vein by underground methods. The upper workings of those mines were worked late in the last century. The records that survive are from the 1930s, some 40 years after the mining. Access to the stopes of interest to this study are extremely hazardous or do not exist. To accurately estimate the open pit minable reserve remaining, the stoping must be accounted for. These old records and the drill intercepts of voids or backfill were plotted on the same standard sections as the drilling. The stope was then hand drawn from that information and digitized as section data. The MEDSYSTEM solid modeler was used to reconcile the sections in three dimensions and plan maps were drawn in a manner similar to the construction of the mineral envelopes. Since some of the stopes are backfilled presumably with mine waste or caved material, some of the voids may contain ore. Assays of fill material vary from less than 0.005 OPT to over 0.50 OPT. Backfill is not a naturally occurring phenomenon and one can not attempt to estimate its extent or grade without additional data. All voids and backfilled areas were treated as air. That is, no tons and no grade. As a result, it is presumed that backfill would pay its own way overall and that void has no impact on the reserve. The data available does not define all the stopes. Certainly some stopes will be encountered by mining that are not presently known.

The deposit contains a small amount of silver of which only 7% is expected to be recovered. The interpolation of the silver values was not as sophisticated as the gold estimation. The omni-directional variogram of the silver values was used to determine the interpolation parameters. That variogram is shown in Figure 4.26. Silver values were interpolated only for blocks within the 0.01 OPT mineral envelope. All blocks were treated the same regardless of the high/low grade discriminator and no cut value was applied to the composite silver grades.

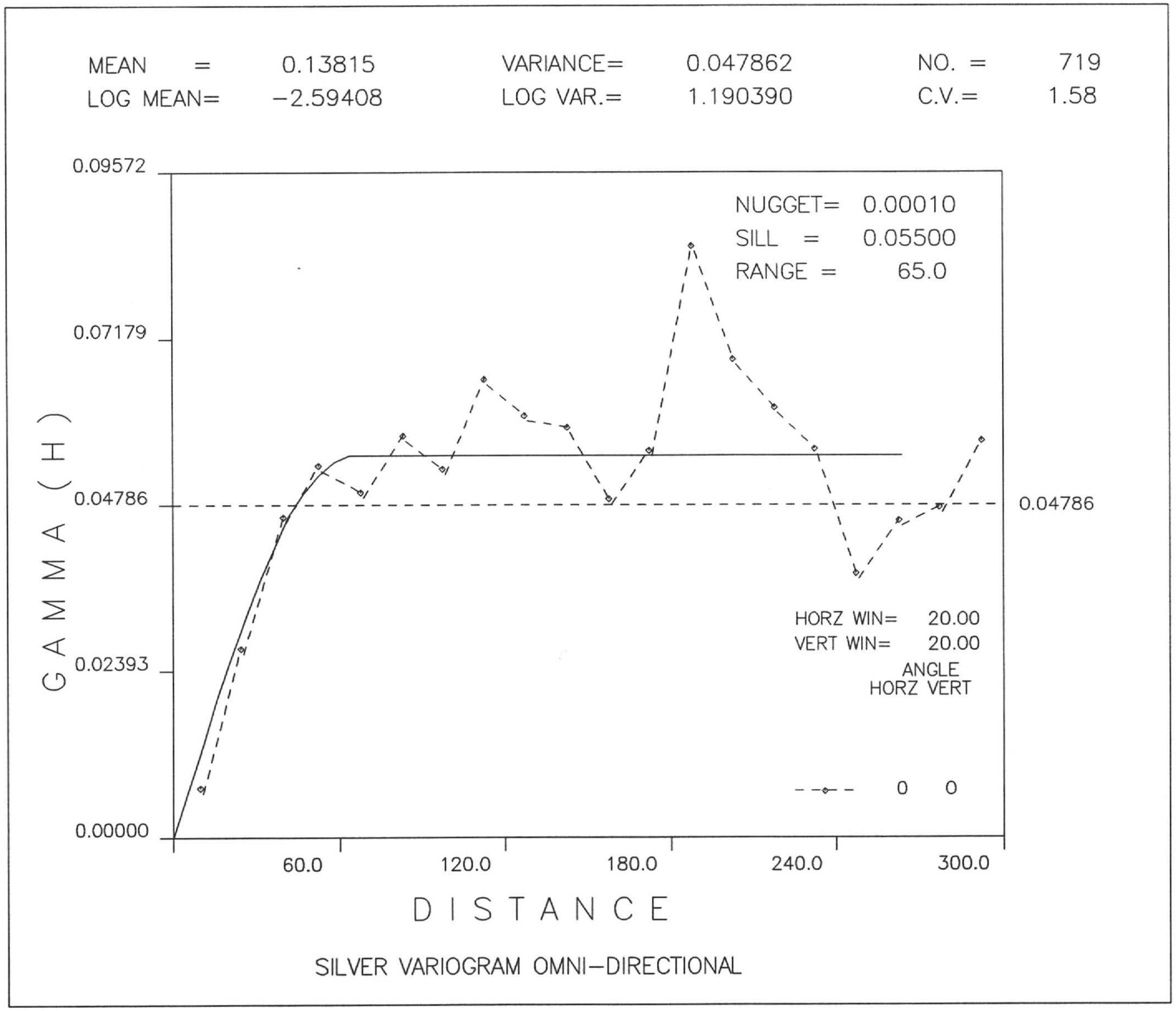



FIGURE 4.26 VARIOGRAM OF SILVER COMPOSITE GRADES

### 4.3.5 Geologic Reserves

The geologic reserves of the model used in this study (Model #11) are found in Figure 4.27. This reserve is calculated using the + 0.01 OPT mineral envelope. The model extends from the surface down to the 2600 elevation or approximately 250 feet below the Pit 004 bottom. These are in-place reserves which are not entirely minable by open pit methods. The minable reserve is reported in Section 6.3 Minable Pit Design.

Figure 4.27 Table of Tiger Geologic Reserves

| Cut-off Grade OPT Gold | 0.010 | 0.015 | 0.020 |
|---|---|---|---|
| Tons | 6,500,000 | 4,938,000 | 4,186,000 |
| Gold Grade OPT | 0.035 | 0.043 | 0.048 |
| Silver Grade OPT | 0.123 | 0.140 | 0.152 |
| Contained Gold Ounces | 227,500 | 212,300 | 201,000 |
| Contained Silver Ounces | 799,500 | 691,300 | 636,300 |

## 4.4 Potential Reserve Expansion

Exploration of the Mammoth vein below the limits of the current model would surely prove additional low grade gold mineralization down to the bottom of the oxidized zone. Further exploration of the Collins vein has the potential of adding significant metal to the reserve. These areas could be explored, but would not be minable until cessation of oxide copper production from SX-EW facility.

The possibility of a high grade underground minable reserve exists. Proving and developing such a reserve would be costly and very risky. This study does not address this issue, but investigation into a small underground operation, possibly operated by a lessor, may be warranted.

METALLURGY

# 5.0 Metallurgy

## 5.1 Summary

Extensive metallurgical testing was done on various samples, drill cuttings and bulk samples, of the significant mineralized rock types found in the Tiger deposit. The testing included direct agitated (bottle-roll) cyanidation, vat leach tests, column percolation leach tests (agglomerated and unagglomerated), gravity concentration and fine grinding followed by cyanidation. The expected metal recovery, the best ore sizing and agglomeration parameters, and the reagent consumption were determined for heap leaching.

The tests results predict an overall recovery of 57% of the gold and 7% of the silver from ore crushed to 80% passing 3/8 inch and agglomerated with 10 pounds portland cement per ton after 180 days under leach. Additional tests indicate that 0.010 ounces of gold per ton can be extracted from the Tiger tails, located just east of the proposed pit. Cyanide consumptions expected for the ore and tails are 0.3 and 0.7 pounds per ton respectively.

## 5.2 Bulk Sampling

A crucial aspect of any metallurgical test program is that sampling must be representative of the whole. Flux pit mining by Magma on the Mammoth vein provided excellent sites from which to retrieve bulk samples of nearly all the mineralized rock types of Tiger. The fact that flux mining occurred as late as 1988 means that the faces left in the bottom of the flux pit are "fresh" and unweathered compared to undisturbed vein outcrops which would otherwise be available. The pit bottom is also 100 feet below the Mammoth shaft collar minimizing further any near surface effects of weathering. Sample sites were identified based on rock type and assays. Five samples were taken from the flux pit bottom and packed in barrels. Samples consisted of Rhyolite, Rhyolite Breccia, Quartz Monzonite, Andesite of the Cloudburst formation and Quartz Vein material from the Rhyolite. The Rhyolite Breccia flux pit sample proved to be very low grade and was not used in the testing.

Access to underground workings is provided via the Mohawk shaft. Cyprus mapped and sampled extensively the 100, 200, 300, 400, and 550 foot levels of the Mohawk mine. Two locations, on the 200 and 300 levels, were identified as good sample sites. One, the 200 level site, had been intersected by a reverse circulation drill hole. This site was selected to test the relationship of the drill hole cuttings to the rock in place. This was done by "raising up" on the drill hole. The average gold grade of the bulk sample taken and the drill hole intercept were 0.075 and 0.071 OPT respectively. The second underground site was of rhyolite on the 300 level. Twenty-six barrels of breccia and ten barrels of rhyolite were hoisted to the surface. Griffith Exploration Company was contracted for the sampling program which included some shaft and shaft station rehabilitation on the 200 and 300 levels.

The barrels of sample from the surface (flux pit) and underground were first shipped to Magma's metallurgical lab in San Manuel for blending and splitting. The samples were then repacked for shipment to McClelland Laboratories in Reno.

## 5.3 Cyanidation Tests

The first series of direct agitated cyanidation (bottle roll) tests were done on drill hole cuttings of the mineralized intercepts from the Cyprus exploration program. Later, bottle-roll tests were conducted on samples of the surface, underground bulk samples, and Tiger tails. These tests were conducted at McClelland Laboratories under the direction of Frank Macy and Gene McClelland. Their detailed reports are included in Appendices V through VIII. Figures 5.1A and 5.1B summarize the bottle-roll test results for drill cutting composites and bulk samples, respectively.

Analysis of the leach residues from tests using bulk samples indicate that gold liberation occurs at -10 mesh.

Column percolation leach tests were performed on the bulk samples at three different feed sizes: nominal 2 inch, 3/8 inch and 1/4 inch. A vat leach test was also done on bulk sample material without any preparation. Recovery from the uncrushed sample and the 2 inch material was unacceptable. The difference between the 3/8 inch and 1/4 inch does not

# FIGURE 5.1A TABLE OF BOTTLE-ROLL RESULTS

Drill Cuttings Composites

| | | Extracted OPT | | Calc'd Head OPT | | Recovery % | | CN Cons. | Lime added |
|---|---|---|---|---|---|---|---|---|---|
| Hole No. | Ore Type | Au | Ag | Au | Ag | Au | Ag | lbs./ton | lbs./ton |
| MM-10 | QM | 0.049 | 0.02 | 0.072 | | 68.1 | | 0.74 | 6.3 |
| MM-14 | QM | 0.076 | 0.02 | 0.140 | | 54.3 | | 0.46 | 3.5 |
| MM-16 | QM | 0.013 | 0.01 | 0.022 | | 59.1 | | 0.31 | 5.1 |
| MM-27 | QM | 0.143 | 0.12 | 0.239 | 0.76 | 59.8 | 15.8 | 0.49 | 4.9 |
| MM-27 | RHY | 0.042 | 0.04 | 0.057 | 0.20 | 73.7 | 20.0 | 0.45 | 3.0 |
| MM-30 | RHY | 0.078 | 0.11 | 0.103 | 0.52 | 75.7 | 21.2 | 0.14 | 3.7 |
| MM-39 | QM | 0.034 | 0.06 | 0.055 | 0.20 | 61.8 | 30.0 | 0.58 | 6.4 |
| MM-41 | CB | 0.013 | 0.01 | 0.023 | | 56.5 | | 0.35 | 5.9 |
| MM-41 | RHY | 0.054 | 0.03 | 0.068 | 0.20 | 79.4 | 15.0 | 0.26 | 3.3 |
| MM-41 | CB | 0.011 | 0.04 | 0.019 | 0.28 | 57.9 | 14.3 | 0.42 | 6.9 |
| MM-41 | QM | 0.027 | 0.02 | 0.046 | 0.21 | 58.7 | 9.5 | 0.29 | 3.9 |
| MM-42 | RHY | 0.027 | 0.07 | 0.047 | 0.55 | 57.4 | 12.7 | 0.45 | 3.0 |
| MM-50 | QM | 0.030 | 0.04 | 0.049 | 0.23 | 61.2 | 17.4 | 0.59 | 6.1 |
| MM-50 | RHY | 0.071 | 0.04 | 0.111 | 0.22 | 64.0 | 18.2 | 0.48 | 4.7 |
| MM-53 | QM | 0.014 | 0.01 | 0.025 | | 56.0 | | 0.23 | 5.2 |
| MM-56 | RHY | 0.025 | 0.11 | 0.038 | 0.36 | 65.8 | 30.6 | 0.61 | 3.0 |
| MM-58 | CB | 0.016 | 0.01 | 0.032 | | 50.0 | | 0.31 | 5.9 |
| MM-60 | QM | 0.021 | 0.02 | 0.039 | | 53.8 | | 0.76 | 5.4 |
| MM-61 | QM | 0.033 | 0.06 | 0.057 | 0.28 | 57.9 | 21.4 | 0.59 | 5.2 |
| MM-67 | CB | 0.027 | 0.06 | 0.044 | 0.29 | 61.4 | 20.7 | 0.46 | 4.9 |
| MM-67 | QM | 0.013 | 0.03 | 0.019 | | 68.4 | | 0.43 | 5.3 |
| MM-70 | RHY | 0.142 | 0.03 | 0.199 | | 71.4 | | 0.29 | 3.4 |
| MM-76 | RHY | 0.075 | 0.40 | 0.105 | 1.64 | 71.4 | 24.4 | 0.79 | 3.2 |
| MM-76 | Qtz Vein | 0.109 | 0.37 | 0.137 | 1.27 | 79.6 | 29.1 | 1.21 | 6.6 |
| MM-77 | CB | 0.277 | 0.09 | 0.419 | 0.50 | 66.1 | 18.0 | 0.41 | 7.4 |
| MM-77 | QM | 0.036 | 0.04 | 0.068 | 0.31 | 52.9 | 12.9 | 0.39 | 8.6 |
| MM-81 | CB | 0.019 | 0.02 | 0.034 | 0.17 | 55.9 | 11.8 | 0.14 | 7.3 |
| MM-81 | QM | 0.023 | 0.02 | 0.045 | | 51.1 | | 0.15 | 5.2 |
| MM-92 | RHY BR | 0.013 | 0.04 | 0.029 | 0.21 | 44.8 | 19.0 | 0.32 | 3.4 |
| MM-99 | QM | 0.054 | 0.08 | 0.110 | 0.25 | 49.1 | 32.0 | 0.16 | 5.8 |
| MM-99 | RHY BR | 0.009 | 0.01 | 0.014 | | 64.3 | | 0.32 | 4.4 |

# FIGURE 5.1B TABLE OF BOTTLE-ROLL RESULTS

Bulk Samples

| | | | Extracted OPT | | Calc'd Head OPT | | Recovery % | | CN Cons. | Lime added |
|---|---|---|---|---|---|---|---|---|---|---|
| Sample | Type | Feed size | Au | Ag | Au | Ag | Au | Ag | lbs./ton | lbs./ton |
| Cloudburst | Surface | 1/4" | 0.016 | 0.01 | 0.032 | 0.14 | 50.0 | 7.1 | 0.43 | 5.0 |
| Cloudburst | Surface | 200M | 0.040 | 0.06 | 0.041 | 0.21 | 97.6 | 28.6 | 0.97 | 9.2 |
| QM | Surface | 1/4" | 0.042 | 0.02 | 0.061 | 0.32 | 68.9 | 6.3 | 0.24 | 3.2 |
| QM | Surface | 200M | 0.051 | 0.12 | 0.052 | 0.42 | 98.1 | 28.6 | 0.44 | 11.5 |
| Qtz Vein | Surface | 1/4" | 0.073 | 0.04 | 0.130 | 0.21 | 56.2 | 19.0 | 0.60 | 2.3 |
| Qtz Vein | Surface | 200M | 0.117 | 0.12 | 0.124 | 0.37 | 94.4 | 32.4 | 0.86 | 11.1 |
| RHY | Surface | 1/4" | 0.011 | 0.01 | 0.020 | 0.13 | 55.0 | 7.7 | 0.76 | 5.9 |
| RHY | Surface | 200M | 0.017 | 0.07 | 0.020 | 0.24 | 85.0 | 29.2 | 0.58 | 9.7 |
| RHY | U/G | 1/4" | 0.013 | 0.04 | 0.043 | 0.44 | 30.2 | 9.1 | 0.21 | 8.2 |
| RHY | U/G | 10M | 0.018 | 0.04 | 0.038 | 0.43 | 47.4 | 9.3 | 0.28 | 7.2 |
| RHY | U/G | 65M | 0.037 | 0.10 | 0.042 | 0.47 | 88.1 | 21.3 | 0.33 | 7.7 |
| RHY | U/G | 100M | 0.036 | 0.11 | 0.039 | 0.49 | 92.3 | 22.4 | 0.28 | 7.9 |
| RHY | U/G | 150M | 0.038 | 0.13 | 0.040 | 0.53 | 95.0 | 24.5 | 0.29 | 7.6 |
| RHY | U/G | 200M | 0.042 | 0.14 | 0.046 | 0.55 | 91.3 | 25.5 | 0.15 | 10.0 |
| RHY BR | U/G | 1/4" | 0.035 | 0.05 | 0.067 | 0.42 | 52.2 | 11.9 | 0.15 | 4.4 |
| RHY BR | U/G | 10M | 0.041 | 0.05 | 0.068 | 0.53 | 60.3 | 9.4 | 0.10 | 5.2 |
| RHY BR | U/G | 65M | 0.066 | 0.15 | 0.075 | 0.45 | 88.0 | 33.3 | 0.30 | 5.0 |
| RHY BR | U/G | 100M | 0.063 | 0.17 | 0.070 | 0.46 | 90.0 | 37.0 | 0.27 | 5.0 |
| RHY BR | U/G | 150M | 0.064 | 0.21 | 0.070 | 0.43 | 91.4 | 48.8 | 0.29 | 5.4 |
| RHY BR | U/G | 200M | 0.072 | 0.20 | 0.079 | 0.53 | 91.1 | 37.7 | 0.45 | 6.2 |

# FIGURE 5.2 TABLE OF COLUMN TEST RESULTS

| | | | Extracted OPT | | Calc'd Head OPT | | Recovery % | | CN Cons. | Leach Time |
|---|---|---|---|---|---|---|---|---|---|---|
| Sample | Type | Feed size | Au | Ag | Au | Ag | Au | Ag | lbs./ton | Days |
| Cloudburst | Surface | 2" | 0.01 | 0.000 | 0.04 | 0.2 | 31.4 | 0.00 | 1.0 | 90 |
| Cloudburst | Surface | 3/8" | 0.02 | 0.010 | 0.04 | 0.2 | 52.5 | 5.00 | 1.4 | 77 |
| Cloudburst | Surface | 1/4" | 0.021 | 0.010 | 0.039 | 0.2 | 53.8 | 7.10 | 1.6 | 70 |
| QM | Surface | 2" | 0.041 | 0.010 | 0.091 | 0.4 | 45.1 | 2.40 | 1.6 | 92 |
| QM | Surface | 3/8" | 0.042 | 0.030 | 0.07 | 0.5 | 60.0 | 6.50 | 1.8 | 96 |
| QM | Surface | 1/4" | 0.041 | 0.030 | 0.061 | 0.4 | 67.2 | 8.10 | 1.8 | 89 |
| Qtz Vein | Surface | 2" | 0.044 | 0.000 | 0.133 | 0.3 | 33.1 | 0.00 | 1.3 | 93 |
| Qtz Vein | Surface | 3/8" | 0.069 | 0.030 | 0.13 | 0.3 | 53.1 | 10.30 | 1.7 | 96 |
| Qtz Vein | Surface | 1/4" | 0.073 | 0.030 | 0.14 | 0.3 | 52.1 | 12.00 | 3.3 | 144 |
| RHY | Surface | 2" | 0.009 | 0.000 | 0.03 | 0.2 | 30.0 | 0.00 | 1.6 | 77 |
| RHY | Surface | 3/8" | 0.016 | 0.010 | 0.03 | 0.2 | 53.3 | 4.80 | 1.9 | 77 |
| RHY | Surface | 1/4" | 0.018 | 0.010 | 0.028 | 0.1 | 64.3 | 8.30 | 2.0 | 69 |
| RHY | U/G | 2" | 0.09 | 0.010 | 0.052 | 0.5 | 17.3 | 2.20 | 1.6 | 94 |
| RHY | U/G | 3/8" | 0.016 | 0.030 | 0.039 | 0.4 | 41.0 | 7.00 | 1.8 | 94 |
| RHY | U/G | 1/4" | 0.019 | 0.03 | 0.041 | 0.36 | 46.3 | 8.3 | 3.2 | 76 |
| RHY BR | U/G | 2" | 0.032 | 0.01 | 0.087 | 0.54 | 36.8 | 1.9 | 1.37 | 129 |
| RHY BR | U/G | 3/8" | 0.037 | 0.04 | 0.073 | 0.49 | 50.7 | 8.2 | 2.9 | 132 |
| RHY BR | U/G | 1/4" | 0.045 | 0.04 | 0.074 | 0.33 | 60.8 | 12.1 | 2.14 | 98 |

warrant the added cost of finer crushing. A 3/8 inch crushed size was selected as the optimum for this material. The column percolation tests on the 3/8 inch feed provide the bulk of the data used in estimating the heap performance. One column test was performed on a master composite of all rock types blended with Tiger tails. Figure 5.2 summarizes the column test results.

### 5.4 Gravity Concentration Tests

The metallurgical evaluation of milling Tiger ore, performed for Cyprus, indicated significant occurrences of coarse gold. The idea of treating fines (-35 mesh) by gravity concentration was tested. The source of the fines would be the ore crushing plant. The hope was to recover coarse gold from the ore before placing it on the heap. McClelland Laboratories reported very poor gold recovery from the fines, that the ore does not produce a large fine fraction and that the gold does not concentrate in the fines during crushing. Due to these factors the idea was abandoned.

### 5.5 Agglomeration Tests

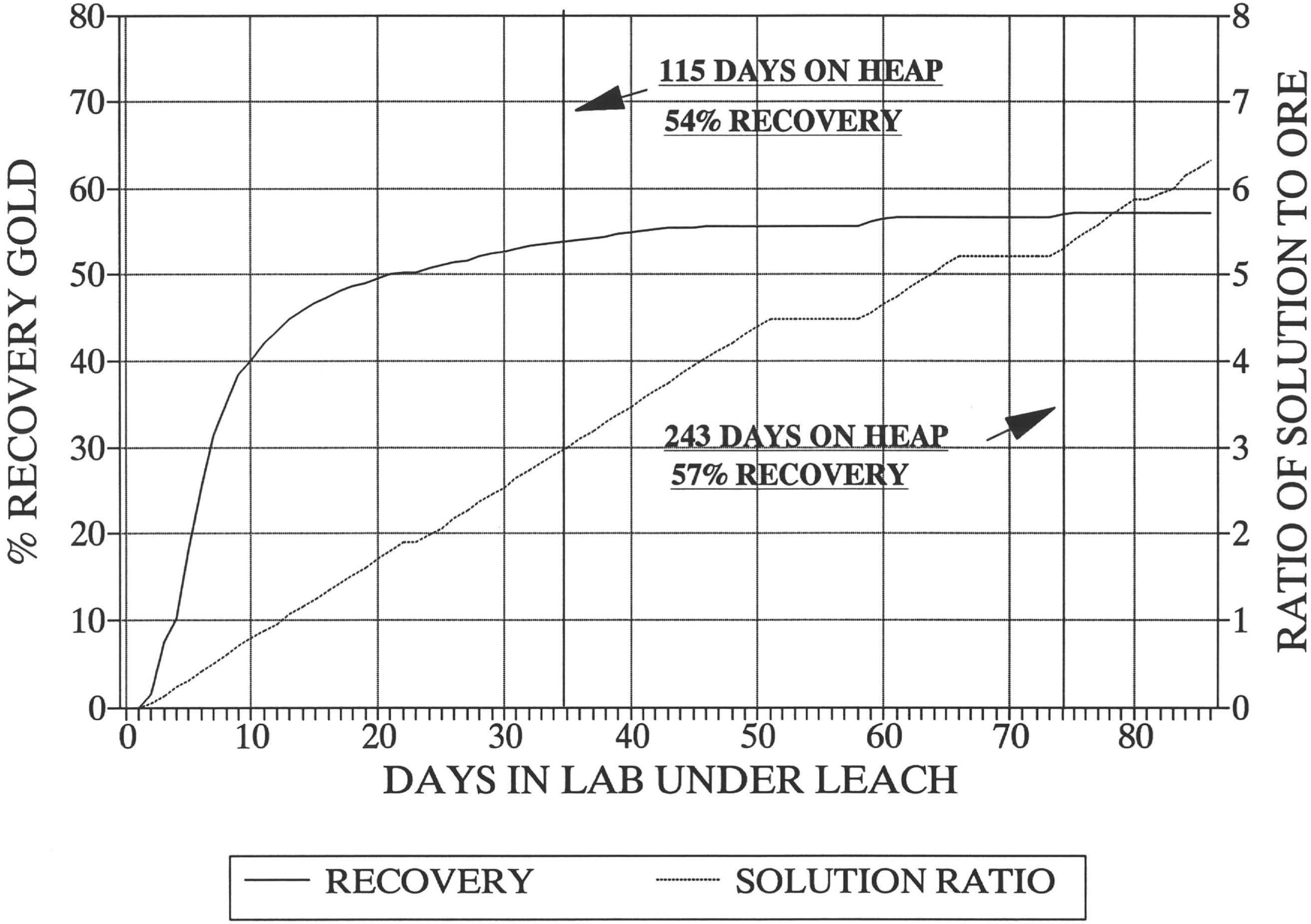
In preparing the Tiger ores for column percolation tests, Gene McClelland evaluated the qualities of the crushed ore and advised that the -3/8 inch material should be agglomerated prior to column loading. The -3/8 inch ore and the -1/4 inch ore were agglomerated with 10 pounds of portland cement per ton. The samples of Tiger tailings blended with a master composite of bulk sample material required 15 pounds of cement per ton to agglomerate.

### 5.6 Expected Heap Performance

There are several measurements of heap performance. Metal recovery, reagent consumption, and solution percolation rates are the critical attributes.

The recovery plot of Figure 5.3 relates recovered gold versus time under leach in the lab for the blended master composite and tails blend. The solution volume per ton of ore required to achieve a specific recovery and

**FIGURE 5.3**
**GOLD RECOVERY FROM MASTER COMPOSITE**



the rate solution is applied to the ore are the leach parameters that determine the time required on the heap. The points annotated on the graph show the estimated time on the heap to achieve the corresponding recovery at a solution delivery rate of 0.004 gallons per minute per square foot of dump surface.

The most significant reagents used on the heap are the cyanide and cement. The cement required to agglomerate was determined in the lab as described above and it is not expected to vary greatly in commercial operation. Cyanide consumption, on the other hand, will be substantially less on the commercial heap than the lab results indicate. Gene McClelland advises a factor of 25% as appropriate to convert lab consumption to expected commercial heap. In this study, 0.3 pounds per ton ore and 0.7 pounds per ton of tails was used.

The solution application rate used in the lab tests was 0.005 gallons per minute per square foot (gpm/sq. ft.) of surface area. It is not known whether the heap will accept solution at that rate or a higher rate. The optimum may only be found in operation. The number 0.004 gpm/sq. ft. is a common application rate for agglomerated precious metal ores.

PIT DESIGN

## 6.0 Pit Design

### 6.1 Summary

A floating cone algorithm was used to determine the ultimate pit shell from the resource model, operating cost estimates, pit wall design, and a range of gold prices. The floating cone based on a $350 selling price was selected as the minable pit design shell. The minable pit, with a reserve of 2.4 million tons of ore containing 0.052 OPT gold and 0.18 OPT silver, and 5.1 million cubic yards of waste, was designed. The stripping ratio is 2.1 cubic yards of waste per ton of ore or 4.4 tons of waste per ton of ore.

The final pit was designed with an inter-ramp pit wall slope of 60°. The pit wall will be triple-benched to a 60 foot 75° face, three 20 foot mining benches, for each 18.5 foot catchment. The final haul road will be 40 feet wide at a 12% grade. Standard open pit mining methods will be applied.

### 6.2 Ultimate Pit Design

The ultimate pit for Tiger was determined using the floating cone computer algorithm available with the MEDSYSTEM. This algorithm determines the break-even mining limits from the resource model and mining parameters for a given set of input costs and metal price. The resource model, the metal recovery, the cost estimates, the pit slope, and the metal price all come together to yield a pit shell to guide the minable pit design.

#### 6.2.1 Floating Cone Parameters

To prepare the model for the floating cone, a recovered gold grade was calculated for each block and stored. This recovered grade was the estimated gold grade of that block multiplied by the metallurgical recovery for the block rock classification plus the gold equivalent grade of the block's recoverable silver grade. A sample calculation is shown below:

Recovered Au = (Au grade X 60% recovery) + ((Ag grade X 7% recovery) / (Au price / Ag price))

The recovered grade and the topography data were extracted from the model to input the floating cone algorithm.

The cost data required for the floating cone was generated from various sources and is listed in Figure 6.1. The mining costs were based on estimates submitted by four contractors. (Section 10.0) Ore processing costs were based on estimates made by Magma's Metallurgical Department at San Manuel and by Dan Turk of Magma Nevada Mining Co. The leach pad cost was estimated from the Oxide Pit's recent construction experience of the Phase 5 leach pad. The 1992 Budget for the San Manuel Mining Division provided additional data for estimating the costs. The Net Value to Mine is the gold price minus the charges to ship and strip loaded carbon at Magma Nevada Mining and refine dore' at Handy & Harman.

These costs and revenues determine two cut off grades for the floating cone. The Mine cut off grade in Figure 6.1 refers to the recovered gold grade required to provide revenue equal to the costs to mine and process a ton of ore. The Heap cut off grade was calculated using only the ore processing costs. The frustrum of each cone evaluated must be in a block meeting the Mine cutoff. The Heap cut off was used to calculate the dollar value of the blocks contained in an evaluated cone.

The last parameter to input to the floating cone was the pit slope angle. The recommendations provided in a report to Cyprus, Dave Nicholas and T. M. Ryan, Tiger Project Preliminary Slope Design, Call & Nicholas, April 1989, [Appendix II] were followed in designing the Tiger pit slope, with the exception of using wire mesh and rock bolts. Given that a final pit would have a haul road 40 feet wide and a 60° inter-ramp slope, the overall pit slope was estimated to be 53°.

### 6.2.2 Sensitivity of Floating Cone Result to Price

Multiple floating cone designs were generated varying the price from $250 to $500 in $25 increments with all other parameters fixed. Maps of the resulting $250, $350, and $450 cone designs are included here as examples in Figures 6.2, 6.3, and 6.4. Using the 1992 Business Plan gold and silver price profiles and appropriate capital costs, the net present value of each pit was calculated and compared. The table of Figure 6.5 lists the results and other statistics of the floating cone pits. Figures 6.6 and 6.7 graph the net present values of these pits with the recovered gold ounces and the average revenue per ounce.

It should be noted here that the pits designed at costs above $350 were not becoming significantly deeper in the Mammoth pit area, but were mining near surface low grade material. The $350 pit mines down to the reasonable limits of the data in this area. As yet undrilled deep reserves might be minable at a higher price if it were not for the location of the SX-EW plant.

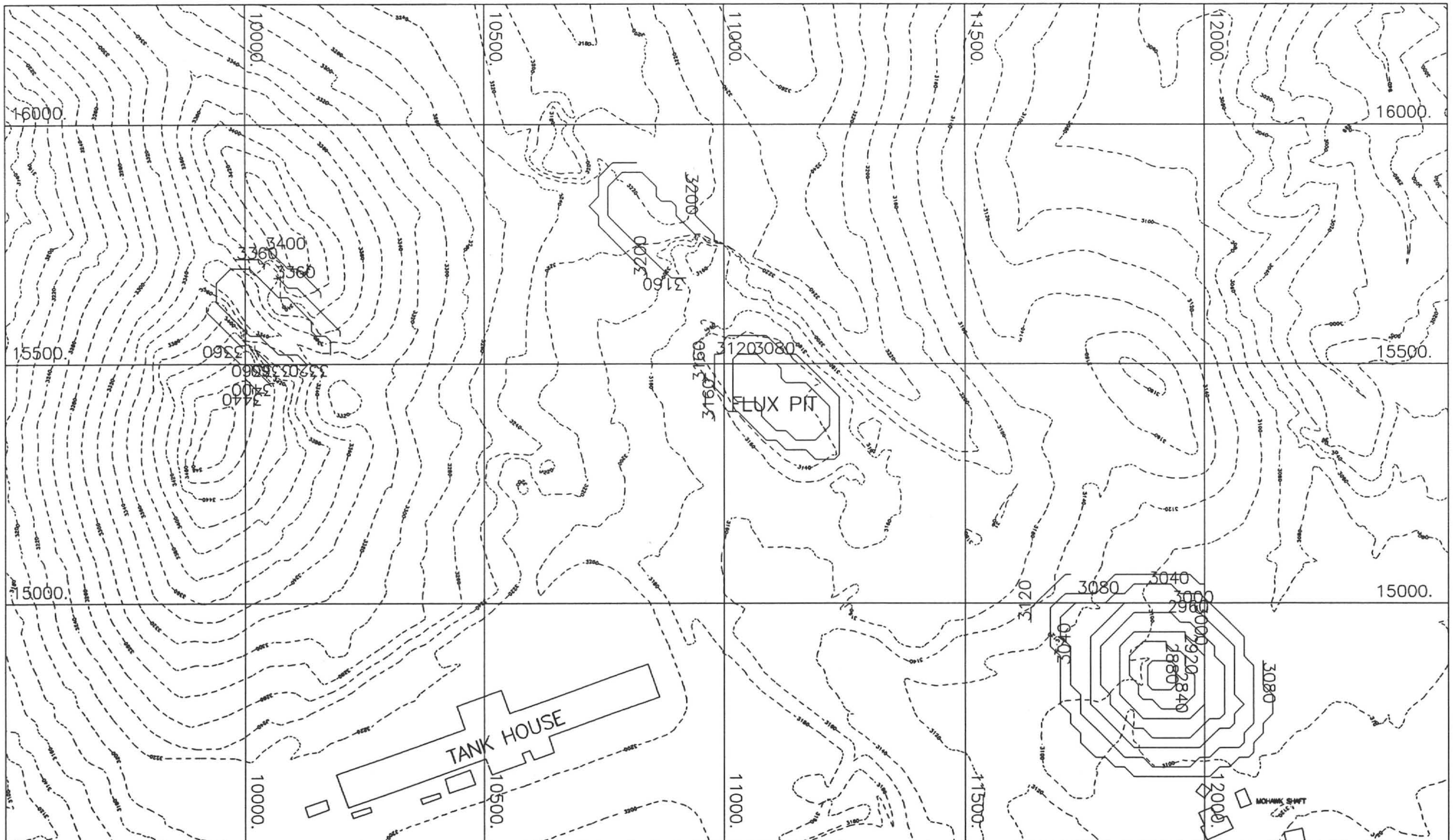
# FIGURE 6.1 FLOATING CONE COST INPUTS

| | PER TON | PER YARD |
|---|---|---|
| WASTE MINING COST | $0.844 | $1.783 |
| ORE COSTS | | |
| ORE MINING COST TO -6 INCH | $1.733 | |
| ORE PREPARATION COST | $0.687 | |
| ORE LEACHING COST | $1.022 | |
| GOLD ADSORPTION COST | $0.220 | |
| G & A COSTS | $0.650 | |
| AMORTIZED PAD COST | $0.544 | |
| SUBTOTAL ORE COSTS | $4.856 | |

NET VALUE TO MINE

| | |
|---|---|
| GOLD PRICE per ounce | $350.00 |
| OFFSITE DEDUCTIONS per ounce | |
| LOADED CARBON SHIPPING | $2.00 |
| CARBON DESORPTION | $8.50 |
| GOLD REFINING | $1.82 |
| SUBTOTAL | $12.32 |
| NET VALUE TO MINE per ounce | $337.68 |

| | RECOVERED OPT |
|---|---|
| HEAP CUT OFF GRADE | 0.012 |
| MINE CUT OFF GRADE | 0.014 |




| No | DATE | MADE BY | DESCRIPTION |
|---|---|---|---|
| 1 | | | |
| 2 | | | |
| 3 | | | |
| 4 | | | |
| 5 | | | |

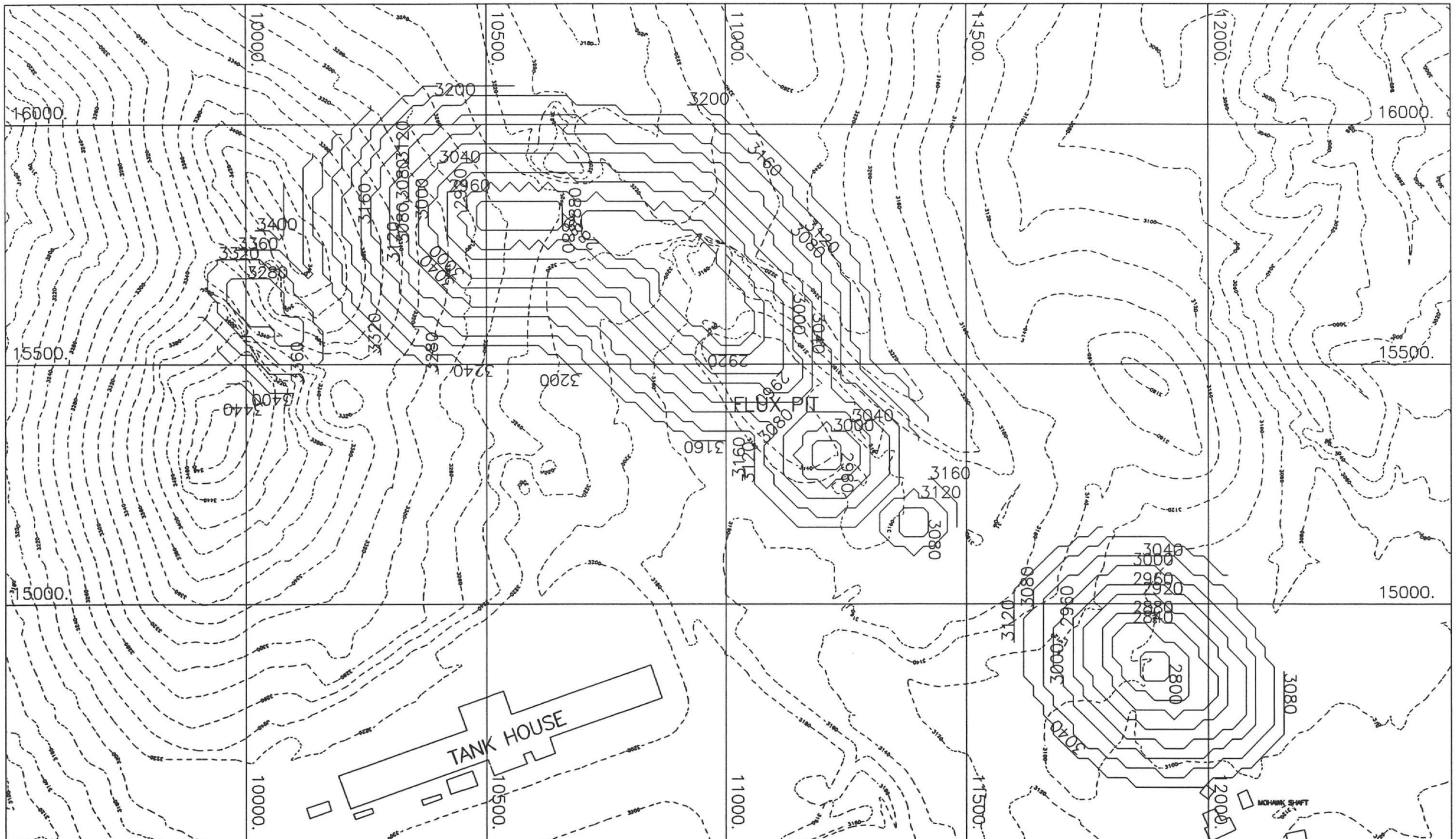
| DATE | DRAWN BY | CHECKED | APPROVED |
|---|---|---|---|
| 11-09-91 | | | |

FIGURE 6.2
$250 FLOATING CONE PIT

OFFICE | DEPARTMENT

TIGER FEASIBILITY STUDY
MAGMA COPPER COMPANY

| MAP INDEX NUMBER | SCALE | DRAWING NUMBER |
|---|---|---|
| | 1"=100.FT | |




MAP SCALE 0 100 200 300

| No | DATE | MADE BY | DESCRIPTION |
|---|---|---|---|
| 1 | | | |
| 2 | | | |
| 3 | | | |
| 4 | | | |
| 5 | | | |

REVISIONS

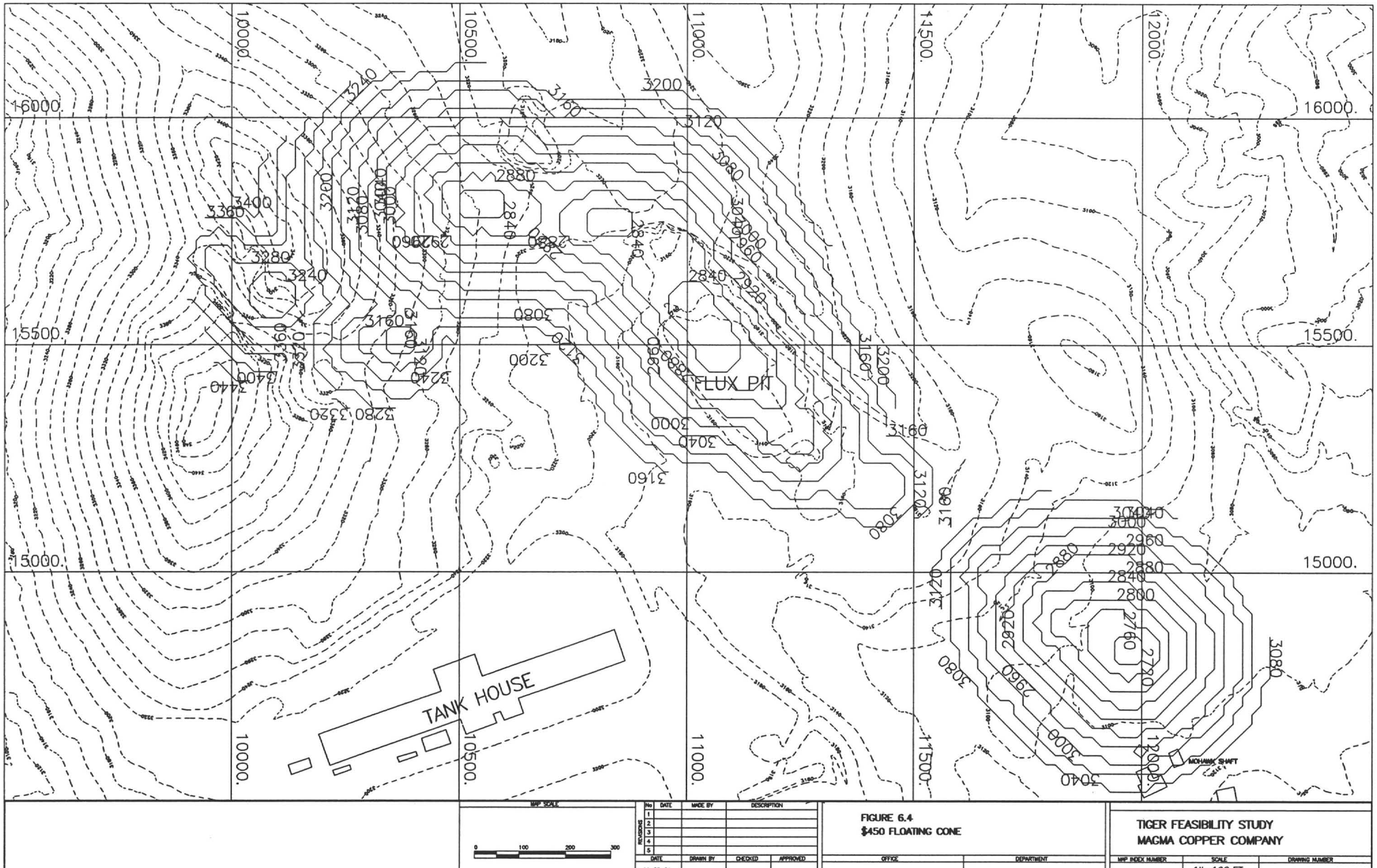
| DATE | DRAWN BY | CHECKED | APPROVED |
|---|---|---|---|
| 11-09-91 | | | |

FIGURE 6.3
$350 FLOATING CONE

OFFICE | DEPARTMENT

TIGER FEASIBILITY STUDY
MAGMA COPPER COMPANY

| MAP INDEX NUMBER | SCALE | DRAWING NUMBER |
|---|---|---|
| | 1"=100.FT | |

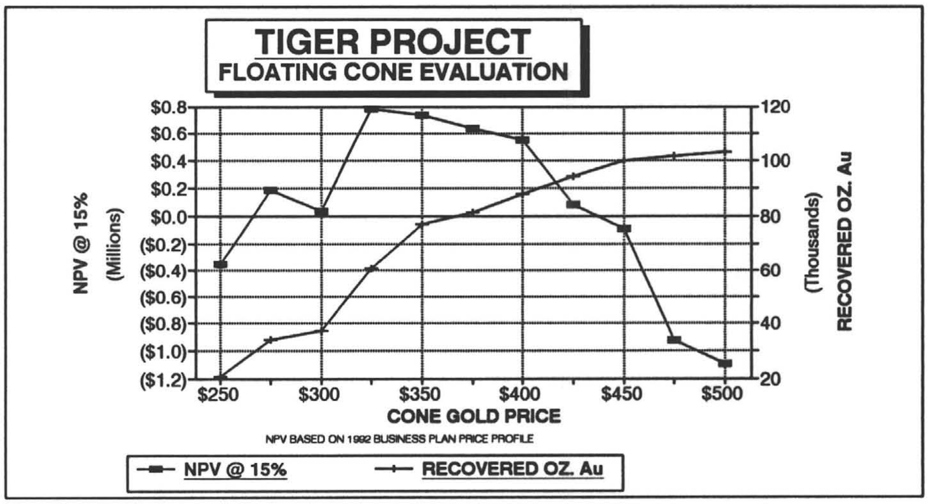


FLUX PIT
TANK HOUSE
MOHAWK SHAFT
MAP SCALE
0
100
200
300
DATE
11-09-91
FIGURE 6.4
$450 FLOATING CONE
TIGER FEASIBILITY STUDY
MAGMA COPPER COMPANY
SCALE
1"=100.FT

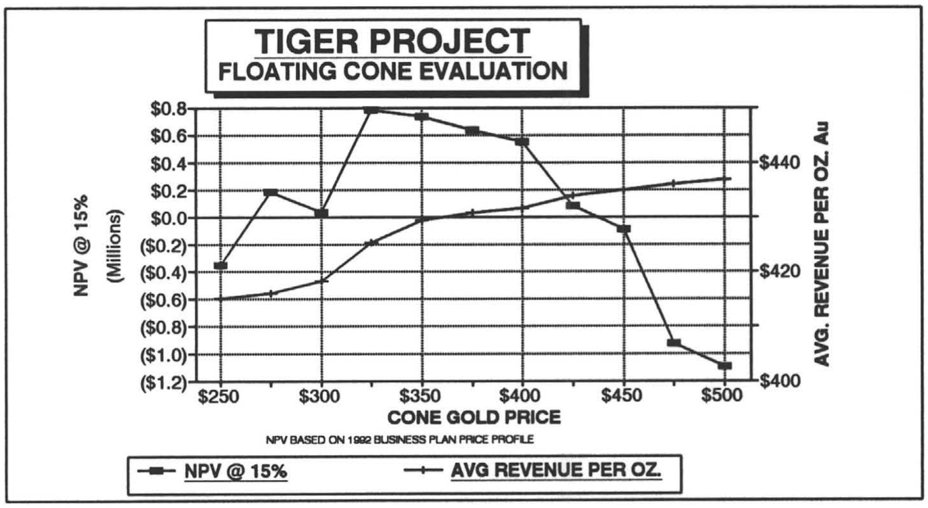
## FIGURE 6.5 RESULTS FROM FLOATING CONE PITS

| CONE PRICE | NPV @ 15% | RECOVERED OZ. AU | AVG REVENUE PER OZ. | GOLD REVENUE |
|---|---|---|---|---|
| $250 | ($350,966) | 20,817 | $415 | $8,640,535 |
| $275 | $192,408 | 33,874 | $416 | $14,093,342 |
| $300 | $40,743 | 37,439 | $418 | $15,661,862 |
| $325 | $783,084 | 60,762 | $425 | $25,840,804 |
| $350 | $735,275 | 77,232 | $430 | $33,175,427 |
| $375 | $635,500 | 81,553 | $431 | $35,138,105 |
| $400 | $550,924 | 88,086 | $432 | $38,025,902 |
| $425 | $87,116 | 94,478 | $434 | $41,014,256 |
| $450 | ($91,522) | 99,875 | $435 | $43,454,363 |
| $475 | ($928,716) | 101,650 | $436 | $44,334,601 |
| $500 | ($1,096,412) | 103,170 | $437 | $45,089,311 |

## FIGURE 6.6 FLOATING CONE NPV AND RECOVERED GOLD




## FIGURE 6.7 FLOATING CONE NPV AND AVERAGE REVENUE PER OUNCE OF GOLD



## 6.3 Minable Pit Design

The $325 and $350 pit shells have essentially the same net present value. The larger pit yields substantially more ounces and on that basis it was selected as the design shell for the minable pit design.

The pit shell is actually three separate pits as shown in Figure 6.3 The Mohawk pit is near the Mohawk shaft. The Mammoth pit includes the flux pit and the east flank of the hill to the north of the SX-EW. The Collins pit is a very small expansion of the original open cut on the Collins vein.

Figure 6.8 is a map, at 1"=100' scale, the mid-line design of the final pits, designated Pit004. This design exhibits the features discussed following.

### 6.3.1 Haul Road Design

The final pit haul roads, for the Mohawk and Mammoth pits, were designed using the specifications of a Caterpillar 773 off-highway truck. A very narrow road width of 40 feet was selected. The very small (7 the first year, 2 the last year) number of operating units required to meet the production schedule reduces the number of times trucks must pass one another on the main haul roads. Should a contractor wish to run larger trucks, one-way haulage would be necessary without modifying the ramp design. Whether or not lower haulage costs, resulting from a larger haulage unit, would offset added stripping costs, to build a wider road, was not addressed.

The maximum ramp grade used in the design is 12%. Most mechanical trucks of the 50 ton size, including the 773, can efficiently carry a full load up a ramp at that grade.

The final haul road into both pits circle down to the bottom in a clockwise direction. No switch-backs are required. The entrances to both pits were selected for easy access to the waste dump and ore preparation plant.

The road into the very small Collins pit is only 20 feet wide and is designed at a 15% grade. This is clearly a one-way road. High production rates are not required of this pit.

### 6.3.2 Pit Wall Design

The pit wall at Tiger is designed based on parameters from Dave Nicholas and T. M. Ryan, Tiger Project Preliminary Slope Design, Call & Nicholas, April 1989. A triple benched wall with a 60° inter-ramp slope and 75° face angle is diagramed in Figure 6.9.

The height of the pit wall standing at 60° varies with the location of the haul road. The overall pit slope varies with the number of ramps in a section of the wall. Figure 6.10 lists the extremes in the pit slopes. The Collins pit is very small and shallow and not included.

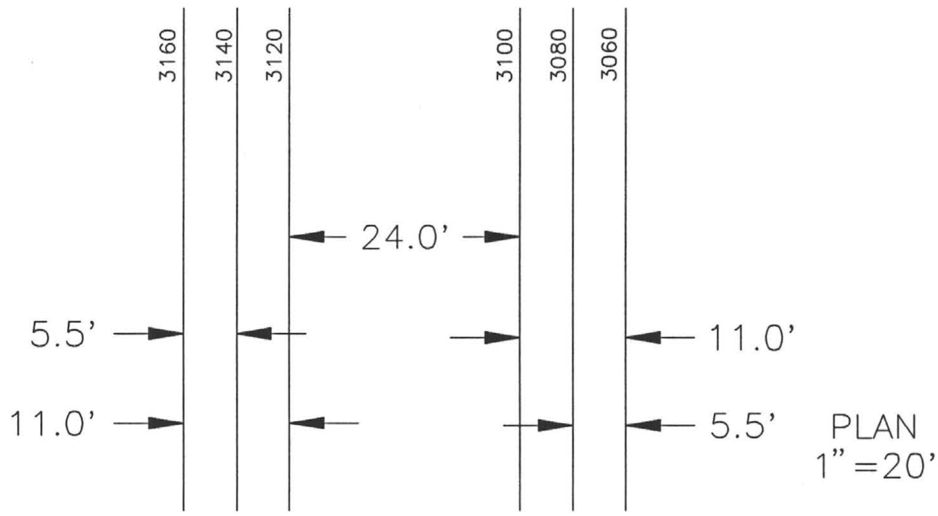
Figure 6.10 Tiger Pit Wall Statistics

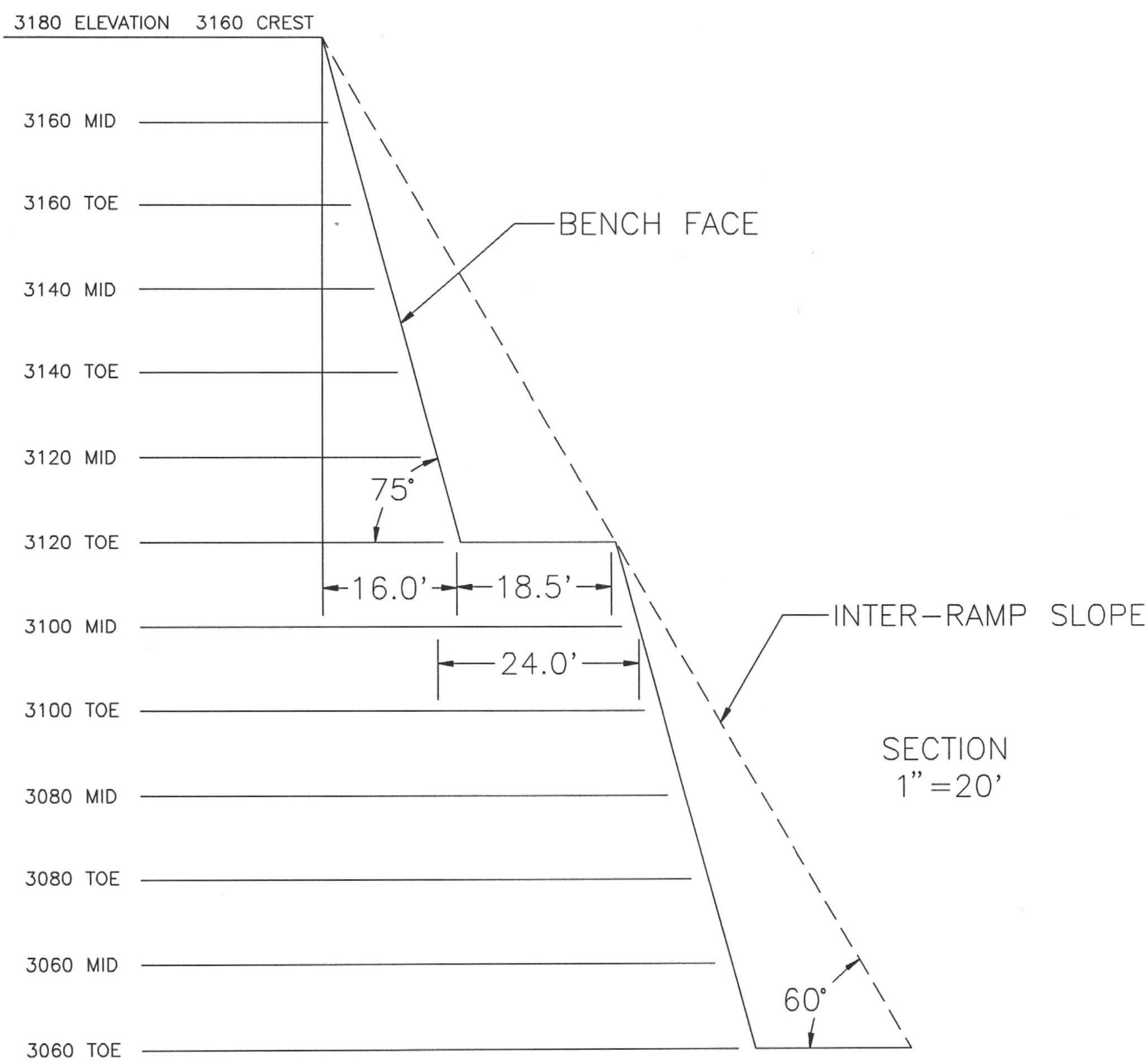
| | Mohawk Pit | Mammoth Pit |
|---|---|---|
| Maximum height at 60° | 140 feet | 420 feet |
| Steepest overall (height at angle) | 220 feet at 49° | 220 feet at 60° |
| Flattest continuous overall | 46° | 48.5° |

As discussed by Call and Nicholas in their reports, blasting practice will have a major impact on overall pit wall stability and the prevention of any blast damage to the SX-EW plant or the workings below the plant. Attention to both goals, minimal pit wall damage and no impact on the SX-EW, will be a prime operational concern.

# TIGER PIT WALL SECTION FIGURE 6.9

MID-LINE DESIGN TRIPLE BENCHES



### 6.3.3 Minable Pit Reserves

The minable reserves of the Pit004 designs are shown in Figure 6.11. The Collins pit is included in the Mammoth reserves. The two mineralized rock types are reported at separate cutoffs based on the expected gold recoveries. An example cutoff calculation is found below:

$$\text{Mining cutoff grade} = \frac{\text{Ore Processing Cost}}{\text{Net to Mine X Recovery}}$$

Figure 6.12 shows the cutoffs calculated on a range of prices from \$350 to \$450 per ounce gold.

Figure 6.12 Cutoff Grade Sensitivity to Price

| PRICE | NET VALUE TO MINE | CUTOFF QM OPT | CUTOFF TERTIARY OPT |
|---|---|---|---|
| \$350 | \$337.68 | 0.017 | 0.019 |
| \$375 | \$362.64 | 0.015 | 0.017 |
| \$400 | \$387.60 | 0.014 | 0.016 |
| \$425 | \$412.57 | 0.014 | 0.015 |
| \$450 | \$437.53 | 0.013 | 0.014 |

The ore processing costs include crushing, agglomeration, stacking, leaching, and leach pad cost. The cutoffs for a \$350 selling price is used in this study.

# FIGURE 6.11 TABLE OF TIGER MINABLE RESERVE

| | | Ore tons | Grade OPT | | Waste yards |
|---|---|---|---|---|---|
| | | | Gold | Silver | |
| Mammoth pit | Quartz Monzonite | 1,202,452 | 0.049 | 0.159 | 1,080,305 |
| | Tertiary | 750,085 | 0.049 | 0.202 | 2,111,976 |
| | Gila | 0 | 0.000 | 0.000 | 1,016,070 |
| Mohawk pit | Quartz Monzonite | 0 | 0.000 | 0.000 | 0 |
| | Tertiary | 446,874 | 0.066 | 0.197 | 648,645 |
| | Gila | 0 | 0.000 | 0.000 | 226,472 |
| TOTAL | | 2,399,411 | 0.052 | 0.180 | 5,083,468 |

Note: Cutoff for Quartz monzonite = 0.017
Cutoff for Tertiary = 0.019

## 6.4 Open Pit Mining Practice

### 6.4.1 Drilling and Blasting

A typical blast round at Tiger will have 42 holes, each 5 inches in diameter, drilled to a depth of 26 feet. They will be spaced 15 feet apart, in 3 rows of 14 holes, in a regular square pattern. Figure 6.13 is a schematic of part of a blast round. Each hole will be loaded with 170 pounds of heavy ANFO, a 3/4 pound cast booster and an in-hole delay primer. The initiation of the holes will be such that no two holes detonate simultaneously or at the resonant frequency of the SX-EW building. Each round is designed to break 7000 cubic yards of material. In the first year of full production, an average of 6 shots per week will be required. The stripping requirements decline as production proceeds and the number of shots per week will decline then also.

Due to the extremely close attention to blasting practice required of the pit operation, it is assumed that Magma will design, supervise the loading, and detonate each blast round. The mining contractor will drill the holes to the depth and in the location designed. Each hole will be surveyed and sampled for ore control purposes. After Magma accepts a pattern as complete and to specifications, Magma's current blasting contractor will charge the drill holes per the pattern design. Magma will then detonate the shot.

### 6.4.2 Loading and Hauling

Magma surveyors will stake ore and waste boundaries on muck piles prior to any loading activity by the mining contractor. These boundaries will be determined by the ore control engineer/geologist from blast hole sample assays and the best geologic information available at the time.

The contractors who responded to Magma's request for mining costs, intend to load 50 ton Caterpillar trucks with either a 992 or 988 Caterpillar front-end loader. One contractor, Bentson Contracting, suggests using an 85 ton truck. The contractors were at liberty to select what they thought would work best on this project. Given the production schedule, no more than two loading units will be required to operate at a time.

Magma surveyors will provide control staking for all pit toes, crests and ramps. Bench elevation control will be provided, but the contractor will be responsible for maintaining loader face elevation control.

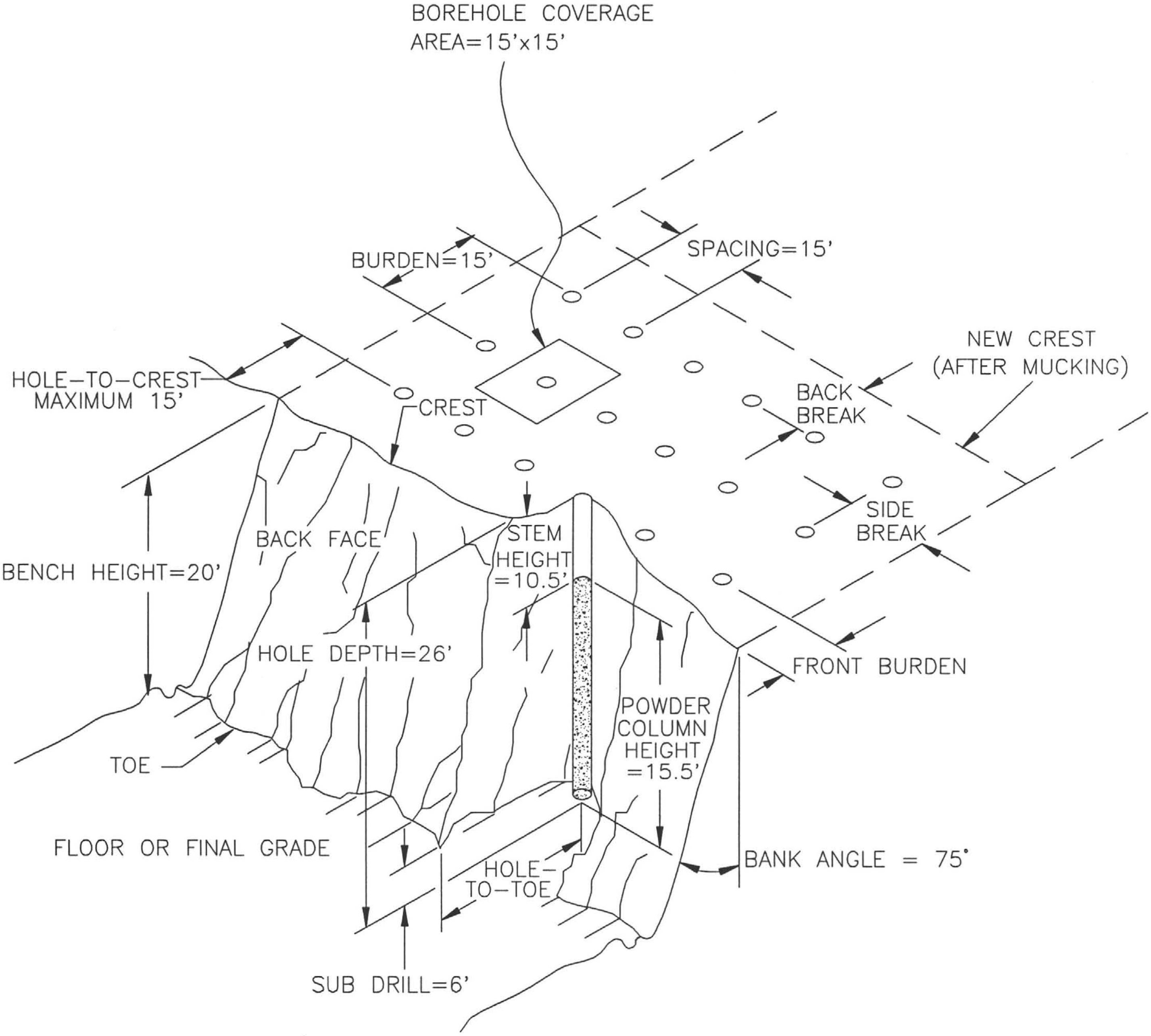
Two areas are designated as waste rock dumps. The dump at the 3240 elevation, is primarily for the upper waste material from the west end of the Mammoth pit and the Collins pit. This dump will be accessed by a series of temporary haul roads which will be abandoned as mining proceeds deeper. The dump at the 3200 elevation will be continuously accessible and will contain the balance of the waste rock of the Mammoth pit and all waste from the Mohawk pit.

### 6.4.3 Coarse Ore Crushing

Ore will be crushed by the mining contractor to 100% passing 6 inch. This facility will be provided by the contractor and located such that material can be fed directly to Magma's ore preparation plant. The contractor will crush as needed, either to clear mine run inventory or to replenish crushed ore feed to Magma's plant.

# SCHEMATIC OF TIGER BLAST ROUND

## FIGURE 6.13




VOLUME PER DRILLHOLE = 166.7 $yd^3$ OR 360 TONS

ORE PREPARATION

## 7.0 Ore Preparation

### 7.1 Summary

Crushed coarse ore received from the mine will be prepared for leaching by crushing it to -3/8 inch size, blending with Tiger tails and portland cement, agglomerating with cyanide solution, and stacking on the leach pad in 20 foot lifts. This will be accomplished by a plant, to be acquired and operated by Magma, placed adjacent to the leach pad. This plant will have two cone crushers discharging to an agglomerator (pugmill).

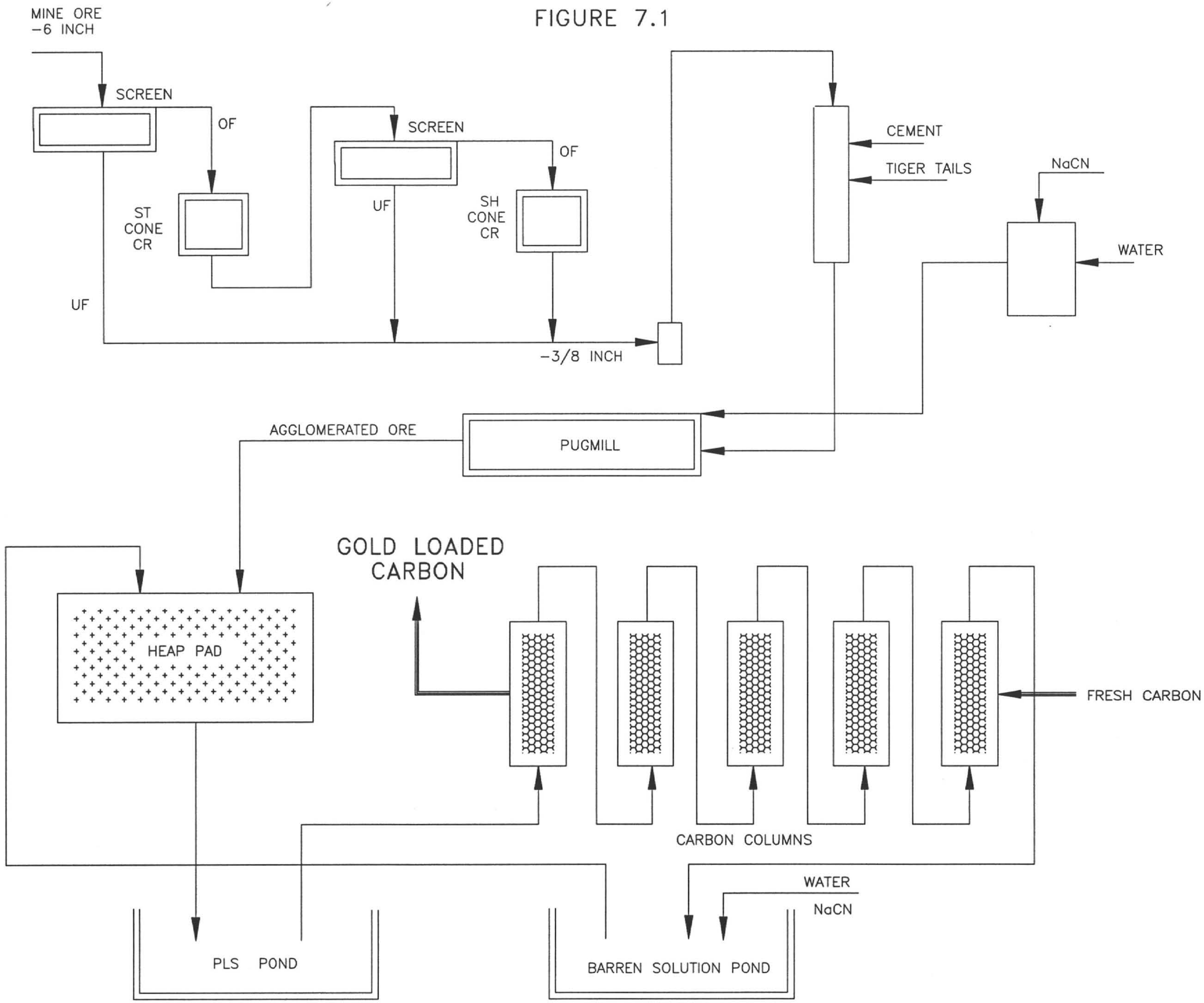
The agglomerated ore and tails will then be transported by belt conveyor to the leach pad and stacked to a height of 20 feet. A leach solution distribution system is then installed.

### 7.2 Process Flow Sheet

Mine ore (-6 inch) is conveyed to an ore crushing plant, where two screens and two cone crushers are located. The ore is crushed to - 3/8 inch, then conveyed to an agglomerator. Cement and, in the first year of operation, tails are fed to the pug mill agglomerator and blended. Moisture added in the agglomerator will contain sodium cyanide. The agglomerator discharge is then conveyed to the leach pad. A process flow sheet appears as Figure 7.1.

# PROCESS FLOW SHEET

FIGURE 7.1



## 7.3 Fine Crushing

Fine crushing will be accomplished by a 5 foot standard cone crusher and a 5 foot short head cone crusher. These units together are rated at 300 horsepower. A description of this facility if found in the report by Dick Um, Feasibility Study of Tiger Ore Gold Heap Leaching, ML-1567, Magma Copper, September, 1991.

## 7.4 Tails Blending and Agglomeration

Agglomeration tests predict that 10 pounds of portland cement per ton of crushed ore is required. This ore blended with Tiger tails at a 79% crushed rock to 21% tails ratio requires 15 pounds per ton. The plant described above will accomplish this in the pugmill. A lower ratio of rock to tails will likely require additional cement. No test work has been done at other ratios.

The addition of sodium cyanide solution to the ore during agglomeration is a common practice. This starts the precious metal dissolution as soon as possible. During the cure time significant gold leaching will occur.

## 7.5 Conveying and Stacking

Crushed and agglomerated ore and tails are conveyed to the leach dump by belt conveyors and stacked, by conveyor on the leach pad in 20 foot lifts. This method of leach pad loading insures that agglomerates remain intact and that no compaction of the ore occurs. Fines production from handling the ore is minimized.

LEACH DUMP DESIGN

# 8.0 Leach Dump Design

## 8.1 Summary

The project production plan calls for 2.4 million tons of ore, and another 400,000 tons of Tiger tailings to be mined, agglomerated, and placed on the heap leach pad. This quantity of material will require a pad area capable of containing 2.25 million cu. yds. of agglomerated ore. This ore will be stacked up to 100 feet in five successive 20 foot lifts. The leach cycle for each lift is approximately 130 days. With a production rate of 2500 tons/day delivered to the heap, sufficient liner area is available to enable each dump to reach the planned leach cycle.

The leach pad area, including ditches, covers 27 acres. Twenty-two acres will be covered by agglomerated ore. The ponds and solution processing area, covers an additional 6 acres. A total of 33 acres will be under liner.

A large pregnant leach solution (PLS) pond and a smaller barren solution pond will be constructed at the lower end of the leach pad. These ponds have the capacity to hold nearly 15 million gallons of solution. The PLS pond has a capacity of 10.25 million gallons and will overflow to the barren pond if such an occurrence becomes necessary. A lined surge area is provided in the event all pond capacity has been exhausted. The surge area has the ability of holding an additional 10.25 million gallons of solution without discharge. The processing area is located adjacent to both ponds and is contained within the lined surge area.

## 8.2 Leach Pad and Solution Ponds

The leach pad and ponds were designed to leach 2.8 million tons of material. Figure 8.1 is a 1"=50' scale map of the site plan of the Tiger leach pad.

Besides having the capacity to hold the 2.8 million tons of agglomerated ore, the leach pad area must be of sufficient size to enable each individual lift to remain undisturbed during a leach cycle of at least 130 days. The leach cycle was determined by estimating the time required to apply a mass of solution that is 3 times the ore tonnage of each heap. A factor of three is approximately the point on the gold recovery curves where the rate of gold dissolution is changing very slowly. In other words, the curve has "flattened out." The ore will be stacked to 100 feet in five successive 20 foot lifts. Given the lift height and an application rate of 0.004 gallons per minute per square foot of dump surface, 115 days are required to apply the needed amount of solution. A 130 day cycle time includes this 115 days plus rest and construction time. A trial heap construction schedule with heap surface areas of 75,000 to 100,000 sq. ft. and 85,000 tons of ore placed at 2500 tons/day, allow each lift to remain under leach for the required cycle time.

A high density polyethylene (HDPE) geomembrane, 0.080 inches (80 mil) thick, was chosen for the liner. This type of liner will provide the strength and durability needed while ore is being loaded onto the pad. The HDPE liner is non-reactive to cyanide solutions and can withstand exposure to the elements, including sun, wind, and temperature variations. Installation of HDPE liner is not difficult and the welding of the individual panels does not detract from the overall liner specifications. A geotextile, a porous woven fabric of HDPE fiber, will be required where rock drains exist to protect the liner from possible puncture.

### 8.2.2 Solution Containment Requirements and Pond Sizing

A key aspect in determining the size of the ponds is the overall solution balance in the system. The two main components involved are the process circuit and the natural water circuit. The process circuit is relatively constant and is made up of make-up water, reagent addition, and bleed off water. A 4 foot deep solution level or 1.1 million gallons must be maintained to facilitate the pump systems. Another 20,000 gallons will be stored in process. The natural water cycle is superimposed on the system and includes rainwater and evaporation. The ponds are sized to contain the three wettest months' average rainfall and the process volumes.

To estimate the components of the solution balance many factors must be considered. Average annual precipitation at the site rate of 14.2" (taken from San Manuel mine records) will add 34.9 gpm to the system over a 130 day leach cycle. Evaporation from the dump and ponds takes 120 gpm from the system while the ore absorbs another 23.6 gpm at the stacking rate of 2500 tons/day. These factors add up to a negative flow balance of 108 gpm. In other words, 108 gpm must be added to the system on an annual basis. The balancing flow will vary throughout the year.

Another aspect in pond size determination is the capacity to contain major fluctuations in the solution circuit. The major factors in extreme pond fluctuations are heap draindown and maximum probable rainfall occurrence. Using draindown data from the McClelland labs reports on Tiger ore, an average draindown of 6.54 gal/ton of ore would occur in five days. At the end of the mine life, when the leach dump is at maximum capacity, a draindown of 12 million gallons could occur over a 5 day period. The other major factor, rainfall, could add an additional 18 million gallons to the system during a maximum probable event. This maximum event was found to be the 72 hour general storm (Sergent, Hauskins & Beckwith, leach dump report Tiger Mine site January, 1986). The maximum storm event is predicted to dump 19.7 inches of rain over a 72 hour period. The 18 million gallons is the amount of rainfall that would fall onto the lined pad area.

The pond sizing was based on the capacity to contain solution fluctuations. In the event of extreme variances, the PLS pond is designed to contain 10.25 million gallons of solution and will overflow to the barren pond which has a capacity of 4.1 million gallons. In the event of an overflow an additional 10.25 million gallons would be contained in the pond and process area should it be needed. This gives the system a capacity of 24.6 million gallons without discharge. As a note, the capacity of the surge area increases nearly 1 million gallons for every foot the dike is raised.

A leak detection system will be placed under the ponds and in the ditch areas where solution is continuously present with some hydraulic head. This detection system is comprised of wrapped geonet placed beneath the liner. This detection system carries any escaped solution to a sump for recovery.

A roadway will be constructed encircling the leach pad. This road will serve as access to the dump areas for ditch diversion, inspection, and solution testing. This roadway will be accessible from various locations including the processing area.

### 8.2.3 Solution Management

The leach pad drains into a perimeter channel which is divided into two separate ditches. These ditches will facilitate the segregation of lean solution from rich solution so that the lean solution can be reapplied to the heap. The purpose of reapplying the lean solution is to increase the overall grade of the feed solution to the carbon columns resulting in a higher carbon loading of precious metal. This is important, as discussed Section 9.0, to minimize carbon shipping and stripping costs.

### 8.2.4 Site Selection

The selection of the leach pad location was determined by several factors. The natural terrain is well suited for rapid drainage of solution from the heap. The strategy of stacking solutions to increase the grade requires that low grade solutions report quickly to the collection ditch. No other nearby locations on Magma property are easily adaptable to these requirements. A site closer to the pit was considered and would be acceptable except for the solution management requirements. Sites close to the pit have a high value as waste dumps as 4 times as much waste as ore will be mined. The proximity of the oxide copper leach dumps, the SX-EW plant, the local terrain, and privately owned property greatly restrict the available area.

Although the site selected is the best for the project, there are some serious concerns. The close proximity to the oxide leach dump presents possible hazards from cyanide and sulfuric acid solutions coming in contact with one another. The potential for airborne sulfuric acid droplets to fall on the cyanide heap or ponds is a concern both from a safety perspective and a reagent cost to the heap. Contamination of adjacent private land by either airborne reagents or catastrophic failure of a containment structure could result in severe financial penalties.

PRECIOUS METAL
EXTRACTION

## 9.0 Precious Metal Extraction

### 9.1 Summary

The availability to the Tiger project of processing facilities at Magma Nevada is the primary reason to select carbon in column leach solution treatment. Activated carbon contained in five columns will adsorb the precious metal effectively over the range of solution grades expected from the Tiger heap. A flow rate of 500 gallons per minute will be maintained through the columns.

Carbon, bearing the precious metal, will be shipped in containers, designed to transport dry sodium cyanide, to Magma Nevada's mill at Ruth, Nevada. The extraction of the precious metal to dore' will be performed by Magma personnel at that facility. The Tiger project dore' will be refined by the same arrangements as Magma Nevada enjoys with Handy and Harman.

### 9.2 Selection of Carbon Technology

The selection of carbon in column metal adsorption as the extraction process to be employed at Tiger is based on several factors. The availability of carbon elution and regeneration capacity at facilities of Magma Nevada Mining reduces the capital requirements for Tiger significantly. The very slow leaching characteristics of the ore will cause highly variable and very lean solution grades reporting to the plant. There is a possibility of slimy solutions as a result of leaching the Tiger tails. These last two factors suggest carbon over zinc precipitation. Carbon very efficiently recovers precious metals irrespective of solution concentration. Solution clarification is not required prior to carbon treatment as with zinc precipitation.

Although carbon is effective on solutions with very low precious metal content, the concentration of precious metal to which the carbon will load is dependent on the grade of the incoming solution and other factors. Solution management on the heap must have as a goal the enhancement precious metal grade.

## 9.3 Description of Carbon Columns

Loading precious metal onto activated carbon occurs as the pregnant solution passes through a tank or column of carbon fluidized by the flowing solution. Five carbon columns with solution flowing in series make up the extraction plant for Tiger. Carbon flows through the system counter-current to the solution flow. (Figure 7.1) The design of the columns is dependent on the solution flow rate and the size of the carbon particles. Fine carbon requires 30 gpm/sq. ft. (recommendation of Dan Turk) of column cross-sectional area to maintain a fluidized state. At 500 gpm inflow to the plant a column 4.6 feet in diameter is required. The height of the column determines the mass of carbon in the system. The rate that carbon is transferred is dependent on the metal in-flow and the mass of carbon. Columns 6 feet high and 4.6 feet in diameter containing one-third carbon by volume will have 5000 pounds of carbon in process. At heap equilibrium, it is estimated that the columns will require carbon movement every 12 hours.

## 9.4 Handling and Shipment of Loaded Carbon

Loaded carbon is pumped in a slurry from the last column into a modified used cyanide flow bin. These are designed to withstand the rigors of transport as safe and secure containers. The solution is drained from the bin and returned to the process stream. Once sealed the bin, containing 1000 pounds of carbon and 50 ounces of precious metal, will be transported to a secure site to await shipment by truck to Ruth, Nevada.

## 9.5 Toll Treatment Arrangements

The precious metal elution will be done by Magma Nevada Mining in the plant at Ruth, Nevada. The Tiger project will be charged $595 per ton of carbon processed regardless of metal content. Carbon elution is essentially a batch process and no mixing of carbon or metal with Magma Nevada material is expected.

Dore' produced from Tiger carbon will be shipped to Handy and Harman in Salt Lake City, Utah and refined under the same arrangements as Magma Nevada. The charges by Handy and Harman are detailed in

Section 10.0. There is fixed charge by the refiner to receive and assay the shipment, a per ounce refining charge and a payment schedule. Settlement on shipments normally occur in 14 days.

PRODUCTION PLAN

# 10.0 Production Plan

## 10.1 Summary

A contract miner will be employed to drill, load, haul, and crush ore. Magma will supervise the blasting, prepare and stack ore for leaching, and extract gold from leach solutions. For the purposes of this study the scheduled start-up of pre-production stripping and construction is January 1, 1992. Sustained ore production of 2500 TPD is expected by July 1, 1992 and would continue into the 1st quarter of 1995. The waste stripping requirements are high early in the project life and decline dramatically in the last year.

Heap leach dumps of 80,000 tons will be placed under leach at a rate of 1 per 32 days and will remain for 150 days before fresh ore is placed on top. Due to the very slow leach characteristics of the Tiger ores, metal production is expected to lag the placement of ore on the dump by 180 days.

Sixteen salaried, exempt and non-exempt, Magma personnel are required to staff the project.

The key production statistics, costs and revenues are outlined in Figures 10.1 through 10.5. The Tiger Project will require $7 million to build and start-up, will produce 75,000 oz. of gold and 46,000 oz. of silver, generate $31.8 million in total revenue at an operating cost of $20.8 million.

## 10.2 Production Schedule

The Pit004 pit design, based on the $350 floating cone shell, yielded the minable reserve tabulated in Figure 6.9 as described in Section 6.3.3. A long range schedule was derived using the MEDSYSTEM based on a 2500 tons per day (TPD) ore requirement. This production rate fits best with the leach pad design and solution application rate. Given the small size of the two pits and the waste stripping requirements efficient mining can be achieved at this relatively low rate.

# FIGURE 10.1 COSTS, PRICES AND STATISTICS

| | FEASIBILITY 1 1991 | START-UP 2 1992 | PRODUCE 3 1993 | PRODUCE 4 1994 | PRODUCE 5 1995 | RECLAIM 6 1996 | TOTAL |
|---|---|---|---|---|---|---|---|
| PRICES | | | | | | | |
| Silver price | $4.00 | $4.92 | $5.13 | $5.43 | $5.60 | $5.70 | |
| Gold price | $350.00 | $395.00 | $407.50 | $427.50 | $440.00 | $450.00 | |
| COSTS | | | | | | | |
| OPEN PIT ORE MINING per ton | $1.73 | (contract) | | | | | |
| OPEN PIT WASTE MINING per yard | $1.78 | (contract) | | | | | |
| PROCESS per ton ore | $1.93 | | | | | | |
| PROCESS per ton tails | $3.25 | | | | | | |
| OPEN PIT G&A per ton ore | $0.65 | | | | | | |
| CARBON SHIPPING per ton | $77.50 | | | | | | |
| CARBON STRIP per ton | $595.00 | | | | | | |
| OPERATIONS | | | | | | | |
| OPERATING DAYS | 0 | 360 | 360 | 360 | 360 | 180 | |
| ORE MINING RATE tpd | 0 | 1,573 | 2,500 | 2,500 | 92 | | |
| TAILINGS RECLAIMED tons | | 400,000 | | | | | |
| WASTE MINING RATE yds./day | 0 | 5,445 | 4,206 | 1,211 | 47 | | |
| OPEN PIT STRIP RATIO | | 3.46 | 1.68 | 0.48 | 0.51 | | |
| OPEN PIT Au GRADE | 0.052 | | | | | | |
| OPEN PIT Ag GRADE | 0.180 | | | | | | |
| Au RECOVERY | 56.50% | | | | | | |
| Ag RECOVERY | 7.00% | | | | | | |
| TAXES AND ROYALTIES | | | | | | | |
| FEDERAL ALT. MIN. TAX | 20.00% | | | | | | |
| STATE ALT. MIN. TAX | 3.00% | | | | | | |
| ROYALTY | 0.00% | | | | | | |
| SEVERANCE TAX | 1.50% | | | | | | |
| PROJECT CAPITAL | | | | | | | |
| FINAL FEASIBILITY | $450,000 | $0 | $0 | $0 | $0 | $0 | $450,000 |
| BUILD | | $4,609,462 | $0 | $0 | $0 | $0 | $4,609,462 |
| PRE-PRODUCTION STRIPPING | | $2,436,680 | | | | | $2,436,680 |
| SUSTAINING | | | $50,000 | $50,000 | $50,000 | $0 | $150,000 |
| RECLAMATION | | | | $0 | $0 | $250,000 | $250,000 |
| RESALE | | | | | | | $0 |
| TOTAL | $450,000 | $7,046,142 | $50,000 | $50,000 | $50,000 | $250,000 | $7,896,142 |

# FIGURE 10.2 ANNUAL PRODUCTION SCHEDULE

| | | FEASIBILITY 1991 | START-UP 1992 | PRODUCE 1993 | PRODUCE 1994 | PRODUCE 1995 | RECLAIM 1996 | TOTAL |
|---|---|---|---|---|---|---|---|---|
| PRESTRIP | | | | | | | | |
| | Waste cubic yards | | 1,173,672 | | | | | 1,173,672 |
| | Tons ore stockpiled | | 116,355 | | | | | 116,355 |
| | Au grade | | 0.055 | | | | | |
| | Ag grade | | 0.139 | | | | | |
| WASTE | | | | | | | | |
| | Waste cubic yards | 0 | 1,960,343 | 1,514,314 | 435,953 | 16,876 | 0 | 3,927,486 |
| ORE | | | | | | | | |
| | Tons ore including stockpile | 0 | 566,355 | 900,000 | 900,000 | 33,055 | 0 | 2,399,410 |
| | Au grade | 0.000 | 0.047 | 0.051 | 0.057 | 0.069 | | |
| | Au contained | 0 | 26,577 | 46,152 | 51,072 | 2,291 | 0 | 126,091 |
| | Ag grade | 0.000 | 0.197 | 0.177 | 0.167 | 0.271 | | |
| | Ag contained | 0 | 111,827 | 159,153 | 149,910 | 8,953 | 0 | 429,843 |
| TAILINGS | | | | | | | | |
| | Tons tailings | 0 | 400,000 | 0 | 0 | 0 | 0 | 400,000 |
| | Au grade | 0.000 | 0.017 | 0.000 | 0.000 | 0.000 | 0.000 | |
| | Au recovered | 0 | 4,000 | 0 | 0 | 0 | 0 | 4,000 |
| | Ag grade | 0.000 | 0.200 | 0.000 | 0.000 | 0.000 | 0.000 | |
| | Ag recovered | 0 | 16,000 | 0 | 0 | 0 | 0 | 16,000 |
| PROCESS | | | | | | | | |
| | Au recovery | 56.5% | 56.5% | 56.5% | 56.5% | 56.5% | 56.5% | |
| | Au recovered | 0 | 11,508 | 20,546 | 27,466 | 15,075 | 647 | 75,242 |
| | Ag recovery | 7.0% | 7.0% | 7.0% | 7.0% | 7.0% | 7.0% | |
| | Ag recovered | 0 | 19,914 | 9,484 | 10,817 | 5,560 | 313 | 46,089 |

# FIGURE 10.3 ANNUAL REVENUES

| | FEASIBILITY 1991 | START-UP 1992 | PRODUCE 1993 | PRODUCE 1994 | PRODUCE 1995 | RECLAIM 1996 | TOTAL |
|---|---|---|---|---|---|---|---|
| GOLD | | | | | | | 0 |
| Production | 0 | 11,508 | 20,546 | 27,466 | 15,075 | 647 | 75,242 |
| Price per ounce | $350.00 | $395.00 | $407.50 | $427.50 | $440.00 | $450.00 | |
| Revenue | $0 | $4,545,660 | $8,372,495 | $11,741,715 | $6,633,000 | $291,150 | $31,584,020 |
| | | | | | | AVG. $/oz. | $419.77 |
| SILVER | | | | | | | |
| Production | 0 | 19,914 | 9,484 | 10,817 | 5,560 | 313 | 46,089 |
| Price per ounce | $4.00 | $4.92 | $5.13 | $5.43 | $5.60 | $5.70 | |
| Revenue | $0 | $97,977 | $48,654 | $58,737 | $31,137 | $1,786 | $238,292 |
| TOTAL REVENUE | $0 | $4,643,637 | $8,421,149 | $11,800,452 | $6,664,137 | $292,936 | $31,822,312 |

# FIGURE 10.4 ANNUAL OPERATING COSTS

| | FEASIBILITY 1991 | START-UP 1992 | PRODUCE 1993 | PRODUCE 1994 | PRODUCE 1995 | RECLAIM 1996 | TOTAL |
|---|---|---|---|---|---|---|---|
| MINE AND MILL | | | | | | | |
| Waste mining | | $3,489,411 | $2,695,480 | $775,995 | $30,039 | $0 | $6,990,924 |
| Ore mining | | $979,794 | $1,557,000 | $1,557,000 | $57,185 | $0 | $4,150,979 |
| Processing | | $1,093,065 | $1,737,000 | $1,737,000 | $63,796 | $0 | $4,630,861 |
| Tails Processing | | $1,299,600 | | | | | $1,299,600 |
| G&A | | $292,500 | $585,000 | $585,000 | $585,000 | $292,500 | $2,340,000 |
| Total | | $7,154,370 | $6,574,480 | $4,654,995 | $736,020 | $292,500 | $19,412,365 |
| OTHER | | | | | | | |
| Property tax | | $0 | $0 | $0 | $0 | $0 | $0 |
| Royalties | | $0 | $0 | $0 | $0 | $0 | $0 |
| Severance tax | | $69,655 | $126,317 | $177,007 | $99,962 | $4,394 | $477,335 |
| Total | | $69,655 | $126,317 | $177,007 | $99,962 | $4,394 | $477,335 |
| TOTAL ON-SITE | | $7,224,024 | $6,700,797 | $4,832,002 | $835,982 | $296,894 | $19,889,700 |
| SHIPPING AND REFINING | | | | | | | |
| Carbon shipping | | $24,352 | $23,273 | $29,669 | $15,992 | $744 | $94,031 |
| Carbon stripping | | $186,960 | $178,679 | $227,784 | $122,779 | $5,715 | $721,917 |
| Dore shipping and refining | | $35,588 | $39,829 | $52,361 | $28,677 | $1,310 | $157,765 |
| TOTAL OPERATING COSTS | | $7,470,924 | $6,942,579 | $5,141,816 | $1,003,431 | $304,664 | $20,863,413 |

# FIGURE 10.5 PROJECT CAPITAL SCHEDULE

| | FEASIBILITY 1991 | START-UP 1992 | PRODUCE 1993 | PRODUCE 1994 | PRODUCE 1995 | RECLAIM 1996 | TOTAL |
|---|---|---|---|---|---|---|---|
| **FEASIBILITY STREAM** | **$450,000** | **$0** | **$0** | **$0** | **$0** | **$0** | $450,000 |
| CAPITAL | | | | | | | |
| **Preproduction** | **$0** | **$7,046,142** | **$0** | **$0** | **$0** | **$0** | $7,046,142 |
| **Sustaining** | **$0** | **$0** | **$50,000** | **$50,000** | **$50,000** | **$0** | $150,000 |
| **Reclamation** | **$0** | **$0** | **$0** | **$0** | **$0** | **$250,000** | $250,000 |
| **Resale** | **$0** | **$0** | **$0** | **$0** | **$0** | **$0** | $0 |
| **Total** | **$0** | **$7,046,142** | **$50,000** | **$50,000** | **$50,000** | **$250,000** | $7,446,142 |
| **TOTAL CAPITAL** | **$450,000 (sunk)** | **$7,046,142** | **$50,000** | **$50,000** | **$50,000** | **$250,000** | $7,896,142 |

This study assumes start-up of the project January 1, 1992. Were that to occur the scheduled production of ore would continue into the 1st quarter of 1995, with metal recovery continuing for at least another 180 days. Reclamation work would continue into 1996.

### 10.2.1 Mining Schedule

The long range scheduling routine of the MEDSYSTEM produces a schedule from the reserve summary and parameters defined by the user. The Pit004 schedule is shown in Figure 10.6. This schedule requires that 1.2 million cubic yards of waste removal be complete before sustained ore production of 2500 TPD is achieved. The schedule splits production between the Mohawk and Mammoth pits based on the total reserve in each pit. The mining contractors who provided costs estimates to this study based their costs on a schedule very similar to the Pit004 schedule.

### 10.2.2 Leaching Schedule

Ore delivered to the leach pad at 2500 TPD will be built in cells of approximately 80,000 tons each. This requires that a new cell be brought on line every 32 days. The surface area of these cells will vary with the lift height and average 90,000 square feet. Dumps will remain under leach for a minimum of 150 days before a new lift is placed on top. Planned rest periods will reduce the actual solution application time to 120 days during the first leach cycle.

### 10.2.3 Metal Production Schedule

The long leach cycle of the Tiger ore and the off-site processing causes a significant delay in the production of and payment for contained metal. For the purposes of this study, it is estimated that metal, contained in ore placed on the dump, will not generate revenue to the project for 180 days. This schedule is found in Figure 10.2. Metal recovery from the Tiger tails is immediate and so is the revenue.

# FIGURE 10.6 MINING SCHEDULE FOR PIT004

UNITS = TONS GRADE = OPT

## MOHAWK PIT

| QUARTER | ORE | GOLD | SILVER | WASTE | S.R. |
|---|---|---|---|---|---|
| 1 | 23,525 | 0.064 | 0.27 | 500,000 | 21.3 |
| 2 | 42,000 | 0.090 | 0.33 | 341,760 | 8.1 |
| 3 | 42,000 | 0.063 | 0.27 | 260,745 | 6.2 |
| 4 | 42,000 | 0.062 | 0.18 | 188,788 | 4.5 |
| 5 | 42,000 | 0.071 | 0.18 | 144,657 | 3.4 |
| 6 | 42,000 | 0.072 | 0.20 | 122,539 | 2.9 |
| 7 | 42,000 | 0.066 | 0.19 | 89,478 | 2.1 |
| 8 | 42,000 | 0.061 | 0.16 | 60,881 | 1.4 |
| 9 | 42,000 | 0.057 | 0.13 | 51,518 | 1.2 |
| 10 | 42,000 | 0.056 | 0.11 | 44,583 | 1.1 |
| 11 | 42,000 | 0.066 | 0.18 | 25,885 | 0.6 |
| 12 | 3,348 | 0.072 | 0.23 | 1,412 | 0.4 |
| SUBTOTAL | 446,873 | 0.066 | 0.20 | 1,832,246 | 4.1 |

## MAMMOTH PIT

| | | | | | |
|---|---|---|---|---|---|
| 1 | 92,830 | 0.053 | 0.11 | 2,000,000 | 21.5 |
| 2 | 183,000 | 0.039 | 0.20 | 2,512,508 | 13.7 |
| 3 | 183,000 | 0.036 | 0.19 | 867,220 | 4.7 |
| 4 | 183,000 | 0.042 | 0.22 | 1,595,326 | 8.7 |
| 5 | 183,000 | 0.036 | 0.11 | 434,968 | 2.4 |
| 6 | 183,000 | 0.051 | 0.16 | 346,080 | 1.9 |
| 7 | 183,000 | 0.061 | 0.21 | 305,365 | 1.7 |
| 8 | 183,000 | 0.065 | 0.12 | 237,688 | 1.3 |
| 9 | 183,000 | 0.050 | 0.12 | 213,132 | 1.2 |
| 10 | 183,000 | 0.049 | 0.21 | 163,724 | 0.9 |
| 11 | 183,000 | 0.060 | 0.24 | 125,134 | 0.7 |
| 12 | 29,707 | 0.069 | 0.28 | 15,927 | 0.5 |
| SUBTOTAL | 1,952,537 | 0.049 | 0.18 | 8,817,072 | 4.5 |

| | | | | | |
|---|---|---|---|---|---|
| GRAND TOTAL | 2,399,410 | 0.053 | 0.18 | 10,649,318 | 4.4 |

## 10.3 Staff

The assumption made in determining staffing requirements were that only the minimum of salary exempt and salary non-exempt positions would exist. Also, some of the positions may combine functions not traditionally the responsibility of one individual. Figure 10.7 lists the positions and a brief description of each.

Figure 10.7 Staff Position Descriptions

| POSITION TITLE | DESCRIPTION |
| --- | --- |
| Superintendent/Chief Engineer (1) | exempt-manager of all aspects of project |
| Junior Engineer/Surveyor (1) | exempt-responsible for pit production engineering and surveying |
| Geologist/Ore Control Technician (1) | exempt-samples blastholes and determines ore/waste boundaries-maps geology |
| Rodman/Draftsman (1) | non-exempt-works for Junior Engineer |
| Ore Preparation Supervisor (1) | exempt-supervises technicians operating ore preparation plant and leach dumps |
| Operating Technician (9) | non-exempt-operate ore preparation plant, leach dump distribution and recovery system, and carbon columns |
| Maintenance Technician (2) | non-exempt-perform preventative maintenance and routine repairs to all plant facilities |

## 10.4 Operating Costs

### 10.4.1 Mining

Mining cost estimates were solicited from five mining contractors. All five responded, but only four provided an estimate. Figure 10.8 lists the Cost Assumptions on which the contractors based their estimates.

Figure 10.8 Mining Contractor Cost Assumptions

DAILY WORK SCHEDULE: No restrictions. Prefer 3 shifts/day-7 days/week.

ELECTRIC POWER: Available line power at crusher site suitable for crushing/screening equipment. Include estimate of power consumption only (kw-hrs) with cost estimate. (Magma pays electric bill.)

WATER: Process water for dust control available at crusher site. Potable water available at nearby facilities.

FUEL: Diesel fuel available on site for off-road equipment only.

OFFICE SPACE: No on site offices available; contractor to provide their own.

CHANGE ROOM FACILITIES: No on site facilities available; contractor to provide their own.

EQUIPMENT MAINTENANCE FACILITIES: No on site facilities available; contractor to provide their own.

PROJECT ACCESS: Paved access from town of Mammoth to east side of Magma property (gate #5). Well maintained dirt road to mine site (about 1.5 miles) around north side of existing copper oxide leach dump.

The mining cost estimates were completed by adding the cost of fuel, electricity, and blasting to the contractor estimates. Figure 10.9 compares the cost estimates submitted by the four contractors. The names are not included to protect the contractors. The costs estimated by Contractor #1 are the costs used in this study.

Drilling and blasting costs were estimated using the blast round design described in Section 6.4.1. That is, a 42 hole round, breaking 7000 cu. yds., and a hole to hole delay timing. The supply costs are based on charges presently incurred for the required materials. The contract with Southwest Energy for blasting services to Magma provide the hole loading costs. Samples from nearly every blast hole will be assayed so that cost can be directly related to each unit of production and is included in the blasting cost estimate.

The cost of mining ore includes coarse crushing by the contract miner.

10.4.2 Ore Preparation

The estimated cost to operate the Ore preparation plant, crushing, agglomeration, and stacking, was developed from the report, Appendix III, Dick Um, Feasibility Study of Tiger Ore Gold Heap Leaching, Report ML-1567, Magma Copper Company, September 1991. The electric power cost was revised based on the Energy Cost Savings Breakthrough Project, Cost Allocation Methodology for: Power Distribution Report, Power Cost Invoicing, Magma Copper Company, August 15, 1991. The staff is to be of salaried non-exempt employees and those salaries were estimated at approximately pay grade 8, 1/3 of salary range. Figure 10.10 tabulates the results of these revisions.

# FIGURE 10.9 MINING COST COMPARISON

| ITEM | CONTRACTOR #1 | CONTRACTOR #2 | CONTRACTOR #3 | CONTRACTOR #4 |
|---|---|---|---|---|
| MOBILIZATION | | | | |
| LUMP SUM | $122,000 | $0 | $0 | $405,000 |
| FUEL REQUIRED | N/A | N/A | N/A | N/A |
| UNIT PRICE | N/A | N/A | N/A | N/A |
| SUBTOTAL PER YARD | $0.017 | $0.000 | $0.000 | $0.058 |
| PIONEERING | | | | |
| LUMP SUM | $66,000 | $0 | $0 | $0 |
| FUEL REQUIRED | 10216 | 0 | 0 | 0 |
| UNIT PRICE | N/A | N/A | N/A | N/A |
| SUBTOTAL PER YARD | $0.011 | $0.000 | $0.000 | $0.000 |
| DRILLING | | | | |
| PRICE PER FOOT | $1.60 | $0.60 | $1.72 | $1.73 |
| FUEL PER FOOT | 0.112 | 0.221 | 0.240 | 0.200 |
| SUBTOTAL PER YARD | $0.270 | $0.120 | $0.286 | $0.292 |
| BLASTING | | | | |
| AN PER TON | $168.60 | $168.60 | $168.60 | $168.60 |
| FUEL COST | $0.690 | $0.690 | $0.690 | $0.690 |
| CONTRACTOR PER CWT LOADED | $3.44 | $3.44 | $3.44 | $3.44 |
| ASSAYING | $10.00 | $10.00 | $10.00 | $10.00 |
| SUPPLIES PER HOLE | $7.43 | $7.43 | $7.43 | $7.43 |
| SUBTOTAL PER YARD | $0.228 | $0.228 | $0.228 | $0.228 |
| LOAD, HAUL & SUPPORT | | | | |
| UNIT PRICE | $1.120 | $1.260 | $1.490 | $2.450 |
| FUEL REQUIRED | $0.189 | $0.246 | $0.160 | $0.210 |
| SUBTOTAL PER YARD | $1.250 | $1.429 | $1.600 | $2.595 |
| DUST PALLIATIVE | | | | |
| VOLUME PER DAY | 96,000 | 80,000 | 80,000 | 120,000 |
| UNIT PRICE | $0.0004 | $0.0004 | $0.0004 | $0.0004 |
| SUBTOTAL PER YARD | $0.0063 | $0.0053 | $0.0053 | $0.0079 |
| WASTE COST PER YAR | $1.78 | $1.78 | $2.12 | $3.18 |
| CRUSH TO ORE -6" | | | | |
| UNIT PRICE | $0.780 | $1.260 | $0.470 | $1.010 |
| FUEL REQUIRED | 0.059 | 0.000 | 0.000 | 0.000 |
| POWER | 0.900 | 1.308 | 0.710 | 0.600 |
| SUBTOTAL PER TON | $0.888 | $1.376 | $0.520 | $1.057 |
| SUBTOTAL PER YARD | $1.919 | $2.973 | $1.124 | $2.282 |
| ORE COST PER YARD | $3.70 | $4.76 | $3.24 | $5.46 |

### 10.4.3 Ore Leaching

Costs from the Met Lab report, ML-1567, were revised as described in Section 10.5.2. Figure 10.11 tabulates these results. In addition, an estimate of the cost to transport liquid cyanide from the San Manuel mill to the Tiger leach plant was included. The descalant solution requirements were revised upward at the advice of Dan Turk of Magma Nevada Mining Co. The cost to perform solution assays for process control were also added.

### 10.4.4 Gold Recovery Plant Costs

Costs from the Met Lab report, ML-1567, were revised as described in Section 10.5.2. Figure 10.12 tabulates these results. In addition, the carbon make-up cost was revised to reflect a 10% loss of the in process material annually.

# FIGURE 10.10 ORE PREPARATION OPERATING COST

| | | | ANNUALLY | PER TON |
|---|---|---|---|---|
| UTILITIES | | | | |
| | POWER | | $118,814 | $0.132 |
| | WATER | | $1,200 | $0.001 |
| | SUBTOTAL UTILITIES | | $120,014 | $0.133 |
| STAFF | | | | |
| | OPERATING TECH. | 3 | $112,500 | $0.125 |
| | MAINT. TECH. | 2 | $75,000 | $0.083 |
| | SUBTOTAL | | $187,500 | $0.208 |
| MAINTENANCE | | | $250,000 | $0.278 |
| OPERATING SUPPLIES | | | $61,000 | $0.068 |
| TOTAL OPERATING COST | | | $618,514 | $0.687 |

## FIGURE 10.11 ORE LEACHING COSTS

| RAW MATERIAL | | $/POUND | LB/TON | ANNUALLY | PER TON |
|---|---|---|---|---|---|
| | SODIUM CYANIDE | $0.790 | 0.3 | $213,300 | $0.237 |
| | CYANIDE DELIVERY | | | $36,000 | $0.040 |
| | PORTLAND CEMENT | $0.038 | 10 | $340,200 | $0.378 |
| | SODIUM HYDROXID | $0.351 | 0.075 | $23,692 | $0.026 |
| | DESCALE SOLUTION | $1.000 | 5 PPM | $32,426 | $0.036 |
| | SUBTOTAL RAW MATERIALS | | | $645,618 | $0.717 |
| UTILITIES | | | | | |
| | POWER | | | $17,398 | $0.019 |
| | WATER | | | $22,706 | $0.025 |
| | SUBTOTAL UTILITIES | | | $40,104 | $0.045 |
| STAFF | | | | | |
| | OPERATING TECH. 4 | | | $150,000 | $0.167 |
| | SUBTOTAL | | | $150,000 | $0.167 |
| MAINTENANCE | | | | $30,000 | $0.033 |
| OPERATING SUPPLIES | | | | $54,000 | $0.060 |
| SOLUTION ASSAYS | | | | $45,000 | $0.050 |
| TOTAL OPERATING COST | | | | $919,722 | $1.022 |

# FIGURE 10.12 GOLD RECOVERY PLANT COSTS

| | | | ANNUALLY | PER TON |
|---|---|---|---|---|
| RAW MATERIAL | | $/POUND | | |
| | CARBON MAKEUP | $1.130 | $43,586 | $0.048 |
| | SUBTOTAL RAW MATERIALS | | $43,586 | $0.048 |
| UTILITIES | | | | |
| | POWER | | $19,048 | $0.021 |
| | WATER | | $400 | $0.000 |
| | SUBTOTAL UTILITIES | | $19,448 | $0.022 |
| STAFF | | | | |
| | OPERATING TECH. | 2 | $75,000 | $0.083 |
| | SUBTOTAL | | $75,000 | $0.083 |
| MAINTENANCE | | | $40,000 | $0.044 |
| OPERATING SUPPLIES | | | $20,000 | $0.022 |
| **TOTAL OPERATING COST** | | | **$198,034** | **$0.220** |

# FIGURE 10.13 GENERAL MINE EXPENSES

ANNUAL BASIS

| | | |
|---|---|---|
| **SALARY COSTS INCLUDING FRINGES** | | |
| SUPERINTENDENT/CHIEF ENGINEER | | $56,000 |
| JUNIOR ENGINEER/SURVEYOR | | $50,000 |
| GEOLOGIST/ORE CONTROL | | $48,000 |
| RODMAN/DRAFTSMAN (NON-EXEMPT) | | $34,375 |
| ORE PREPARATION SUPERVISOR | | $50,000 |
| | SUBTOTAL | $238,375 |
| **EXPENSED COSTS** | | |
| NON-EXEMPT HOLIDAY PAY | | $18,750 |
| NON-EXEMPT VACATION PAY | | $16,875 |
| MSHA COSTS | | $1,500 |
| TECHNICAL STAFF TRAINING | | $1,500 |
| SAFETY COSTS | | $6,188 |
| BUILDING MAINTENANCE COSTS | | $12,000 |
| SUPPORT VEHICLES | | $60,000 |
| SERVICE CONTRACTS | | $75,000 |
| OTHER COSTS AND SUPPLIES | | $12,000 |
| | SUBTOTAL | $203,813 |
| **ALLOCATED OVERHEAD** | | |
| AVERAGE OVERHEAD PER EMPLOYEE | | $8,931 |
| NUMBER OF EMPLOYEES | | 16 |
| | SUBTOTAL | $142,904 |
| **GRAND TOTAL** | | $585,091 |

### 10.4.5 General Mine Expenses

General Mine Expenses were estimated from the staff requirements of the project and the 1992 San Manuel Mining Division Budget. Figure 10.13 tabulates these expenses on an annual basis. The expense costs, holiday pay through safety costs, were calculated using the cost factors of the 1992 Budget. The service contracts include a quarterly aerial survey of the pit areas for reconciliation of pay quantities to the mining contractor. The allocated overhead is based on the San Manuel Mining Division's average overhead per employee.

### 10.4.6 Carbon Stripping and Dore' Refining

Activated carbon bearing precious metal will be shipped to Magma Nevada Mining Co. for stripping. The resulting dore' product will be shipped to Handy and Harman for refining. These three costs are tabulated in Figure 10.14 Off-site Costs.

Figure 10.14 Off-site Costs

| | |
|---|---|
| Carbon Shipping | \$1550 per round trip, 20 ton load, 100 oz. metal per ton |
| Carbon Stripping | \$595 per ton |
| Dore' Shipping and Refining | \$350 for transportation per shipment, \$50 assay charge, \$0.70 per oz. dore' to refine, pay 99.85% gold and 99.50% silver value |

Settlement with the refiner generally occurs in 14 days.

## 10.5 Capital Costs

The capital investment required to bring the Tiger Project into production is \$7 million. A summary of the capital budget is tabulated in

Figure 10.15. Pre-production stripping includes the contract mining cost and General Mine Expenses for the pre-production period. The leach pad construction cost was developed from the recent cost experience of the oxide pit's construction of Phase 5 leach dump applied to the design described in Section 8.0. The gold adsorption plant cost was reviewed by Dan Turk of Magma Nevada Mining Co. The carbon purchase is sufficient to have three 20 ton shipments available; one in process in Ruth, one in transit, and one at Tiger. The ore preparation plant cost was derived from recent sales of similar used facilities. The market for used plants of this type is somewhat volatile. The price and availability of what Tiger needs is highly dependent on timing. This is the primary reason for the 15% contingency added to the capital estimate. The power line estimate comes from a current price quote, to the oxide pit, for re-construction of a power line, of the same length and design, required for the Phase 6 leach dump.

## FIGURE 10.15 CAPITAL BUDGET

| | | |
|---|---|---|
| PRE-PRODUCTION STRIPPING | | |
| | 1.2 million yards | $2,436,680 |
| LEACH PAD | | |
| | Earthworks | $1,333,200 |
| | Liner | $697,800 |
| | Supplies/extras | $139,400 |
| | Construction management | $65,000 |
| TOTAL LEACH PAD CONTRUCTION | | $2,235,400 |
| GOLD ADSORPTION PLANT | | |
| | Plant | $300,000 |
| | Carbon | $140,000 |
| TOTAL PLANT | | $440,000 |
| ORE PREPARATION PLANT | | |
| | Purchase | $700,000 |
| | Installation | $100,000 |
| UTILITIES | | |
| | Power line | $200,000 |
| | Water line | $15,000 |
| SUBTOTAL | | $6,127,080 |
| | Contingency @ 15% | $919,062 |
| TOTAL | | $7,046,142 |

ECONOMICS

# 11.0 Economics

## 11.1 Summary

The cash flow results from the production plan are listed in Figure 11.1. The high pre-production stripping requirements and slow metal recovery combine to give a negative $9.9 million After Tax Cash Flow at the end of the first year. This negative cash flow is not paid back until the first quarter of the fourth year of the project. This results in 7.44% internal rate of return for the life of the project, excluding sunk costs. A total cost of $373 per ounce, excluding sunk costs, is projected against an average revenue per ounce of $420.

At a 15% discount rate, the Net Present Value of the project, with the given price profile, is a negative $1.4 million. The constant price of gold of $450 per ounce would yield a zero Net Present Value at a 15% discount rate. Figure 11.2 summarizes the cash flows at a $450 constant gold price.

# FIGURE 11.1 CASH FLOW RESULTS

| | FEASIBILITY 1991 | START-UP 1992 | PRODUCE 1993 | PRODUCE 1994 | PRODUCE 1995 | RECLAIM 1996 | TOTAL |
|---|---|---|---|---|---|---|---|
| GOLD PRODUCTION | 0 | 11,508 | 20,546 | 27,466 | 15,075 | 647 | 75,242 |
| GOLD REVENUE | $0 | $4,545,660 | $8,372,495 | $11,741,715 | $6,633,000 | $291,150 | $31,584,020 |
| SILVER REVENUE | $0 | $97,977 | $48,654 | $58,737 | $31,137 | $1,786 | $238,292 |
| CC&A COST/OZ Au | $0 | $1,253 | $338 | $187 | $68 | $855 | $379 |
| OPERATING CASH FLOW | $0 | ($2,827,288) | $1,478,571 | $6,658,636 | $5,660,707 | ($11,728) | $10,958,898 |
| DEPRECIATION | $0 | $1,207,681 | $2,156,156 | $2,882,362 | $1,582,027 | $67,916 | $7,896,142 |
| NET OPERATING INCOME | $0 | ($4,034,969) | ($677,585) | $3,776,274 | $4,078,679 | ($79,643) | $3,062,756 |
| Federal alt. min. tax | $0 | $0 | $0 | $755,255 | $815,736 | $0 | $1,570,991 |
| State alt. min. tax | $0 | $0 | $0 | $113,288 | $122,360 | $0 | $235,649 |
| AFTER TAX INCOME | $0 | ($4,034,969) | ($677,585) | $2,907,731 | $3,140,583 | ($79,643) | $1,256,117 |
| PROJECT CASH FLOW | ($450,000) | ($9,873,430) | $1,428,571 | $6,608,636 | $5,610,707 | ($261,728) | $3,062,756 |
| AFTER TAX CASH FLOW | ($450,000) | ($9,873,430) | $1,428,571 | $5,740,093 | $4,672,610 | ($261,728) | $1,256,117 |
| CUMULATIVE CASH FLOW | ($450,000) | ($10,323,430) | ($8,894,859) | ($3,154,766) | $1,517,844 | $1,256,117 | |

| RESULTS | FEAS | BUILD | PROD |
|---|---|---|---|
| PROJECT CASH COST/OZ | $277 | | |
| CC&A/OZ AU | $379 | $373 | $242 |
| IRR | 5.19% | 7.44% | |

| NET PRESENT VALUE | | |
|---|---|---|
| | 0.00% | $1,256,117 |
| | 5.00% | $38,186 |
| | 10.00% | ($821,498) |
| | 15.00% | ($1,425,849) |
| | 20.00% | ($1,846,486) |
| | 25.00% | ($2,133,914) |
| | 30.00% | ($2,324,171) |

# FIGURE 11.2 CASH FLOW RESULTS AT GOLD PRICE OF $450

| | FEASIBILITY 1991 | START-UP 1992 | PRODUCE 1993 | PRODUCE 1994 | PRODUCE 1995 | RECLAIM 1996 | TOTAL |
|---|---|---|---|---|---|---|---|
| GOLD PRODUCTION | 0 | 11,508 | 20,546 | 27,466 | 15,075 | 647 | 75,242 |
| GOLD REVENUE | $0 | $5,178,600 | $9,245,700 | $12,359,700 | $6,783,750 | $291,150 | $33,858,900 |
| SILVER REVENUE | $0 | $97,977 | $48,654 | $58,737 | $31,137 | $1,786 | $238,292 |
| CC&A COST/OZ Au | $0 | $1,254 | $339 | $187 | $68 | $855 | $380 |
| OPERATING CASH FLOW | $0 | ($2,204,791) | $2,337,368 | $7,266,425 | $5,808,969 | ($11,728) | $13,196,243 |
| DEPRECIATION | $0 | $1,207,681 | $2,156,156 | $2,882,362 | $1,582,027 | $67,916 | $7,896,142 |
| NET OPERATING INCOME | $0 | ($3,412,472) | $181,212 | $4,384,062 | $4,226,942 | ($79,643) | $5,300,101 |
| Federal alt. min. tax | $0 | $0 | $36,242 | $876,812 | $845,388 | $0 | $1,758,443 |
| State alt. min. tax | $0 | $0 | $5,436 | $131,522 | $126,808 | $0 | $263,766 |
| AFTER TAX INCOME | $0 | ($3,412,472) | $139,533 | $3,375,728 | $3,254,745 | ($79,643) | $3,277,891 |
| PROJECT CASH FLOW | ($450,000) | ($9,250,934) | $2,287,368 | $7,216,425 | $5,758,969 | ($261,728) | $5,300,101 |
| AFTER TAX CASH FLOW | ($450,000) | ($9,250,934) | $2,245,689 | $6,208,090 | $4,786,773 | ($261,728) | $3,277,891 |
| CUMULATIVE CASH FLOW | ($450,000) | ($9,700,934) | ($7,455,244) | ($1,247,154) | $3,539,619 | $3,277,891 | |

| RESULTS | FEAS | BUILD | PROD |
|---|---|---|---|
| PROJECT CASH COST/OZ | $278 | | |
| CC&A/OZ AU | $380 | $374 | $251 |
| IRR | 14.32% | 17.32% | |

| NET PRESENT VALUE | | |
|---|---|---|
| | 0.00% | $3,277,891 |
| | 5.00% | $1,783,138 |
| | 10.00% | $697,409 |
| | 15.00% | ($93,546) |
| | 20.00% | ($669,756) |
| | 25.00% | ($1,088,051) |
| | 30.00% | ($1,389,299) |

## 11.2 Recommendation

The Tiger Project should not be put into production as an open pit, heap leach operation at this time. A re-evaluation should be done when a sustained gold price of $450 per ounce is foreseen, or when other significant and favorable changes occur related to the project.

ENVIRONMENTAL

# 12.0 Environmental

## 12.1 Summary

The primary environmental permit required is an Aquifer Protection Permit (APP) issued by the Arizona Department of Environmental Quality (ADEQ). The facilities for this project are designed to meet all present regulatory requirements. The permitting process for ADEQ to issue an APP is approximately one year. Project construction can proceed concurrent with the process, but actual leaching cannot commence until a permit is issued. Other required permits can be processed in less than six months.

## 12.2 Permit Requirements

Magma's Environmental Affairs Department has identified several permits that must be issued to the Tiger project. A detailed reveiw of these requirement appears in Appendix IV, Tiger Project Environmental Permits, Dale Deming, Magma Copper Company, Inter-office Correspondence, November 1991. The following outline summarizes the major permits required:

> Aquifer Protection Permit (APP) An APP is issued by the ADEQ. Requires facility to meet "Best Available Demonstrated Control Technology". Approximately one year required to secure.
>
> Air Quality Installation Permit and Operating Permit A permit may be required of ADEQ. At the least an operating permit required from Pinal County Air Quality Control District.
>
> Well Construction Permits Issued by Arizona Department of Water Resources. Required for exploration drilling, water supply wells and monitor wells.

No other permit requirements are foreseen under current regulations.

SX-EW PLANT
FOUNDATION STABILITY

## 13.0 SX-EW Plant Foundation Stability

### 13.1 Summary

The close proximity of the SX-EW plant (300 feet) to the Mammoth pit and the abandoned workings of the Collins mine (below the tankhouse), are of critical concern to the project. A consultant, Call & Nicholas, was retained to review Magma's evaluation and make recommendations. The conclusion is that blast vibration velocities less than 0.5 inches per second, measured at the tank house, will not affect the structure or the workings beneath. Careful blasting design, practice, and seismic monitoring will be a high priority of the operating staff. Sub-surface monitors have been installed near the tank house to determine present ground conditions and to provide early warning of blast induced ground movement into the abandoned workings.

### 13.2 Discussion

The east third of the SX-EW tankhouse is located over the Mammoth fault zone. Past mining of the Collins Vein in the foot wall of the fault has caused fault gouge material to fall into the stopes. These failures have propagated to the surface in several locations. The most recent occurred about 100 feet north of the tankhouse. The issue of blast vibrations from Tiger mining inducing or accelerating these type of failures is of concern.

The shortest distance between the Mammoth pit and the SX-EW tankhouse is 300 feet. Any blast vibration induced damage to the tankhouse from Tiger mining is unacceptable.

Both of these issues are addressed in the report by Dave Nicholas and Ross Barkley, Potential Ground Movement Near SX-EW Plant due to Mining the Tiger Deposit,Call & Nicholas, Inc., April 1991. [Appendix IX]. The conclusion and recommendation of that report is that mining can occur at Tiger and not affect the building stability with careful blasting practice to maintain the peak particle velocity of vibrations below 0.5 inches per second. Further recommendations are that monitoring systems be installed, before and even without Tiger mining activity, to determine if natural stoping is occurring beneath the building and if movement is induced or changes due to mining.

## 13.3 Blast Monitoring Plan

At the start of mining at Tiger, further testing and monitoring of blasting will be done to enhance the understanding of the seismic characteristics of the site. The necessary equipment and procedures to perform blast monitoring will be acquired and established. A primary responsibility of a staff engineer of the project will be blast design, loading supervision and blast monitoring. It will be necessary and recommended to involve the SX-EW operating personnel in the monitoring system at the tankhouse.

## 13.4 Ground Monitoring Plan

The most significant part of ground monitoring recommendations of Call & Nicholas, the subsurface monitors are in place. These monitors are the responsibility of the SX-EW plant personnel and will remain so during mining activity at Tiger. A program to read the monitors is underway. This data and data from blast monitoring will be reviewed periodically by Call & Nicholas.

APPENDIX I

# SUMMARY REPORT

# TIGER PROJECT

# PINAL COUNTY, ARIZONA

Stanton P. Dodd, Consulting Geologist

January 1989

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SUMMARY REPORT
TIGER PROJECT
PINAL COUNTY, ARIZONA

## SUMMARY


The Tiger project area is located in sections 26 and 27, T.8S., R.16E. Pinal County, Arizona on a block of patented and unpatented mining claims owned by Magma Copper Company. In December, 1987, Cyprus Minerals Company entered into a joint venture agreement with Magma to explore for precious metal deposits on this claim block.

Extensive underground workings including the Mammoth Mohawk and Collins mines exist throughout the property. Past production totals approximately 400,000 ounces of gold and approximately 1,000,000 ounces of silver along with commercial quantities of lead, zinc, copper, molybdenum and vanadium.

A major northwest-trending vein system controls the bulk of the gold mineralization in the district. This zone cuts most of the main rock units in the area. These units include the Precambrian Oracle granite, the Tertiary Cloudburst Formation, consisting mainly of intermediate volcanic units, and Tertiary rhyolite dikes. One major post-mineral unit, the Gila Formation, is also present. Within the property, a major post-mineral, northwest trending normal fault (Mammoth Fault) offsets the mineralized zone. The Mammoth and Mohawk mines occur northeast of the fault whereas the Collins mine occurs along the deeper expression of the gold-bearing structure southwest of the fault.

Gold mineralization is typically associated with zones of quartz veining within the main structural zone. Extensive oxidation has occurred on the property to depths exceeding 900 feet at some of the mines. In the oxidized zones gold tends to occur in the native state. Reduced ore, which is typically low in gold occurs only at the Collins mine at depths greater than 700 feet. This indicates that oxidation occurred prior to displacement of the main mineralized structure.

The Tiger exploration program was designed to test for the presence of a bulk tonnage, open-pit gold deposit occurring along the major northwest-trending zone of mineralization. Initially, drill targets were determined through a combination of geologic mapping and sampling in conjunction with a review of existing data. During the program 114 rotary, reverse circulation holes were drilled totalling 37,151 ft. Three main zones of gold mineralization were delineated. The Mohawk zone, located in the South, the central Mammoth zone and the Collins zone to the northwest.

Preliminary, computer generated mineable open pit ore reserves to a depth of 300 feet for the Mammoth and Mohawk zones total approximately 1.7 million tons of ore grading 0.079 opt gold. The waste/ore ratio is approximately 11.9/1. Metallurgical tests indicate a recovery of approximately 80% using a combination of gravity and flotation or approximately 90% using gravity and CIL. In the defined ore zones some additional drilling is needed in order to confirm the higher intervals.

## INTRODUCTION

In December, 1987 Cyprus Minerals Company and Magma Copper Company entered into a joint venture agreement whereby Cyprus would explore for precious metal mineralizations on a block of patented and unpatented mining claims owned by Magma in and around the abandoned townsite of Tiger, Pinal County, Arizona (Map 1). Presented in this report is a summary to date of the work performed, data collected and conclusions reached regarding the Tiger Project.

## LOCATION AND ACCESS

The Tiger project area is located on the north slope of the Santa Catalina Mountain approximately 40 miles northeast of Tucson, Arizona (Fig. 1). The optional claim block, located in Pinal County, lies in sections 26 and 27, T8S, R16E. The property lies approximately 3 miles west of Mammoth, Arizona, 10 miles northwest of San Manuel, Arizona and just east of Magma Copper Company's San Manuel copper mine.

Access from Tucson by 40 miles of paved state highway to Magma Copper company's main mine entrance with an additional 2 miles of paved and gravelled mine roads.

## HISTORY AND PREVIOUS WORK

Much of the following discussion on the mining history at Tiger is taken from Howell (1988).

The project area is located in the Old Hat mining district where gold was initially discovered in 1879 on what eventually became known as the Collins vein. In 1882 the Mammoth claim, among others, was located east of the Collins and from 1882-1887 development work proceeded. The ore produced from these and other claims was hauled to the town of Mammoth to be milled. The gold was free milling and collected by amalgamation.

In 1889 the community centered around the mines was given the name Schultz after the locator of the Mammoth claim. In 1891 the Mohawk claim, bordering the Mammoth to the Southeast, was staked and between 1892-1894 a 300 foot shaft was sunk. In 1893 a major cave-in at the Mammoth caused the mine to close until 1897. Mining continued at the Mohawk as well as at the Collins.

During 1896 a 10-stamp mill was constructed to process the ore from the Mohawk. A year later an aerial tramway was built to haul ore from the Mammoth mine to the mill at the town of Mammoth, a distance of approximately 2 3/4 miles. At the same time, the mill at Mammoth began the process of cyanidation of the tailings as well as new ore, thereby greatly improving gold.

During 1898 the Mohawk mine closed. Production continued at the Mammoth and Collins until 1901. Two major factors were instrumental in their closure. First, the development of deep oxide ore at the Mammoth was hindered due tot he large amount of water being generated and second, a major cave-in occurred from the 760 foot level to the surface at the Mammoth.

From 1906-1916 the only mine production was from the Mohawk where a new 500 foot shaft was sunk. From 1915-1916 the bulk of production came from the reworking of the tailings at Mammoth with the primary intent to recover molybdenum. A concentrate rich in heavy oxide minerals including wulfenite, vanadinite, cerrasite and anglesite was produced through a gravity process. This concentrate was enriched in molybdenum, vanadium and lead. Since the outbreak of World War I the price of molybdenum had continued to rise. During the war years the tailings and ore mined from the district produced all of the molybdenum marketed in the U.S. Underground mining of gold-molybdenum ore continued from 1917-1919. During that time the Mammoth shaft was destroyed by fire but was re-timbered and enlarged. The end of World War I in 1919 precipitated a drop in the price of molybdenum causing all the mines in the district to close.

From 1919-1934 no ore was produced from the district. IN 1926 the New Year claim was located east of the Mohawk and a 140 foot exploratory shaft was sunk, encountering ore-grade material.

With the rise in the price of gold in 1934 mining activity was again renewed in all the major mines in the district. In 1935 a new gravity cyanidation mill was built at the Mohawk. In 1939 Mammoth - St. Anthony, Ltd. acquired the Mohawk and New Year mines giving them control of the district. At the same time the residents of Schultz decided to change the name of the town to Tiger.

Significant gold production continued from the mines through 1944. During World War II the mines were operated mainly for the production of molybdenum and base metals with with gold recovered as a by-product. During 1944 the Mohawk shaft was destroyed by fire, re-timbered and deepened. It was then used as a haulage shaft for the mining of deep ore from the Collins vein. Extensive workings now connected all the major mines in the district.

From 1945-1953 production was restricted to the 700 foot - 900 foot levels at the Collins mine. At these depths the Collins vein, being below the oxidation boundary, contains mainly sulfide ore typically low in gold. During this period the primary commodities produced were lead, zinc, copper and silver.

In 1948 Magma Copper Company, then a subsidiary of Newmont, began underground exploration on what became the San Manuel porphyry copper deposit located west of Tiger. Surface drilling began on this property as early as 1917.

In 1953 all mining operations at Tiger ceased. Later that year Magma acquired all of St. Anthony's properties, mainly for the living accommodations at Tiger to be used for their expanding copper operation. Through 1953 the mines at Tiger produced approximately 400,000 ounces of gold, 1 million ounces of silver, 3.5 million pounds of copper, 75 million pounds of lead, 50 million pounds of zinc, 6 million pounds of molybdenum oxide and 2.5 million pounds of vanadium oxide.

In 1963 the McFarland and Hollinger Company leased the tailings at Tiger from Magma. The tailings are still being hauled to the ASARCO smelter at Hayden where they are used as silica flux.

Intermittently from 1978-1987 Magma developed a small open-pit mine over part of the Mammoth claim primarily to acquire silica flux material for their won smelting operations. In 1987 Magma finished upgrading their smelter which now required a higher grade silica material than was available at Tiger. This caused an end to the pit development. Magma's silica flux pit produced approximately 300,000 tons of flux material averaging approximately 0.04 opt gold.

From 1978 - 1987 Newmont Mining Company examined the district several times with regard to precious metal potential. Their exploration programs included mapping and sampling along with limited drilling.

In 1987 Magma Copper became an independent company. In December of that year Cyprus Minerals Company, entered into a joint venture agreement with Magma, to explore for precious metals in and around the vicinity of Tiger.

## CYPRUS EXPLORATION PROGRAM

The exploration program at Tiger was designed to test for the presence of a bulk tonnage, open-pit gold deposit occurring along a major northwest-trending zone of mineralization. Initial drill targets were determined through a combination of geologic mapping and geochemical sampling in conjunction with a review of previous underground and drill data.

### Geologic Mapping

Geologic mapping at 1" = 200' was conducted over the entire property (Map 2). In areas of interest, where sufficient outcrop occurs, mapping was done at a scale of 1" = 40' (Maps 5 and 7).

### Geochemical Sampling

Rock-chip sampling was done on a selected basis throughout the property (Map 4). Where available, continuous sampling across the major zones of mineralization was done (Maps 6 and 8). An underground sampling program was instigated at the Mohawk mine, the only workings on the property still accessible. The 100 foot through 550 foot levels were sampled and mapped (Maps 9 through 13). Caving on all levels restricted access to the main ore zones.

### Drilling

A total of 114 rotary, reverse circulation holes totalling 37,151' were drilled on the property. Drilling began in December, 1988 and continued intermittently through October, 1989. Five different drilling contractors were used during this period.

The initial phase of drilling indicated significant zones of ore grade mineralization occurring along the main structure. Drilling then commenced along section lines spaced 100' apart (Map 14). At least two holes were drilled on each section in the Mammoth and Mohawk areas whereas one or two holes, depending on access, were drilled along section lines in the Collins area.

Additional exploration holes were drilled west of the Collins vein as well as several hundred feet north of the Mammoth flux pit. In addition, one deep hole (MM-3) was drilled to intersect the southeast extension of the Mohawk mineralized structure under Magma's copper oxide leach operation. A zone of limonite stained, quartz veined rhyolite containing disseminated chrysocolla was encountered at depth, however assays from this zone were nil in gold of the exploration holes in the Collins ridge area only one (CR-4) encountered any significant mineralization intersecting 60' of material averaging 0.073 opt gold.

Several deep holes were drilled in an attempt to test the potential for a possible underground mining operation. Due to drilling difficulties it was not possible to reach to goal depth in the majority of those holes.

Overall, the drilling was very difficult due to the highly fractured nature of the rock in and adjacent to the ore zones accompanied by the extensive amount of underground workings encountered. In several areas these workings would contain either mineralized or unmineralized back fill material. In some locations, primarily where caving had occurred, sample recovery was very poor to nonexistent. This was especially true in section 540 (Map 14) where eight holes were drilled with only local poor recovery in the projected ore zone.

All the drill holes were sampled at 5-foot intervals and subsequently logged at a scale of 1" = 10' (Appendix I). All drill samples, excluding those of post mineral material, were assayed for both gold and silver. Selected intervals were also assayed for molybdenum, vanadium, lead, copper, and zinc. The logs include all of the geochemical results as well as survey data such as collar coordinates and elevations.

Geologic cross sections at 1" = 40' were constructed for the majority of section lines (Maps 15 through 45). The sections also include drill hole gold geochemistry and, where applicable, surface gold geochemistry.

### Geophysics

An IP/resistivity survey was performed in an attempt to define buried rhyolite bodies in the Mammoth Mohawk area. This proved unsuccessful due to the electrical interference produced form the mine operation at San Manuel.

## GEOLOGY

### Regional Geology

The geologic units that comprise the Santa Catalina Mountains, the dominant physiographic feature in the area, represent a suite of rocks typical of a metamorphic core complex (Dickenson, 1988). Precambrian intrusive rock, occurring in the central portion of the range, is over lain by a sequence of upper Precambrian metamorphic and sedimentary rocks that in turn are over lain by a series of Paleozoic sedimentary units. Cretaceous age, felsic igneous bodies locally intrude all of the older units.

North of the Santa Catalina Range Precambrian intrusive rock is still common in outcrop whereas the Precambrian and Paleozoic sedimentary units are only locally exposed. Thick sections of Tertiary intermediate volcanic and sedimentary units along with small Tertiary age, felsic intrusive bodies increase in occurrence both north and east of the range.

Extensive northwest trending and northeast trending normal faults are present in the region. Thrust faults and detachment faults associated with the formation of the Catalina metamorphic core complex are also present.

## Geology of the Project Area

Previous geologic work in the Tiger area includes studies by Creasey (1950, 1967) and Peterson (1938).

### Lithologies

The oldest rock unit found in the project area is the Precambrian Oracle Granite. This intrusive body consists mainly of gray to green, medium - to coarse-grained, porphyritic biotite quartz monzonite. Aplite dikes are locally common. Widespread exposures of this intrusive unit occur west of the project area (Map 1) where it acts as host for the San Manuel and Kalamazoo porphyry copper ore bodies. Occurrences of granite rock around Tiger are mainly found in the area of the Collins ridge and in the Mammoth flux pit. underground maps indicate that exposures of this unit increase with depth in both the Collins and Mammoth areas. At the Mohawk mine exposures of this unit are rare.

Late Cretaceous early Tertiary, gray grandiorite to quartz monzonite prophyry dikes infrequently occur in the project area. The only exposures found to date are in the Collins ridge and north of the Mammoth mine near Tucson Wash. These intrusives are similar to those associated with porphyry copper mineralization found to the west.

Rocks of the mid-Tertiary cloud burst formation outcrop throughout the project area with the most extensive exposures occurring in the north. The formation consists mainly of interbedded andesite flows and volcanic agglomerate/breccia. The agglomerate/breccia is composed of angular to subrounded intermediate volcanic clasts with lesser subrounded granite and rhyolite clasts in an intermediate volcanic matrix. This unit is typically gray-green to dark-green in color. Other units mapped in the cloud burst include 1) an orange-brown fanglomerate member consisting of subangular to subrounded fragments of various lithologies in a sandy matrix and 2) blue-gray rhyolite to rhyodacite flows.

Tertiary rhyolite dikes are abundant in the area and can be found at all the major mines. The rock is light colored and typically fine grained. Local porphyritic zones and zones of brecciation are also present. In general, the intrusives tend to be elongate in a northwest direction.

In many areas rhyolite intrusive breccia occurs associated with the rhyolite dikes. This breccia is similar in color to the rhyolite dike rock as it is composed of altered rhyolite fragments in a fine grained rhyolitic ground mass. This unit was probably formed as a result of autobrecciation of the rhyolite dike as the margins cooled and intrusion of magma continued.

An andesitic intrusive breccia was also mapped in the project area. This unit typically contains abundant rhyolite fragments with minor granitic and intermediate volcanic fragments. The matrix ranges in color from green to maroon and appears to be andesitic in composition. This unit is commonly silicified. In many respects this unit is similar to breccia units found in the Cloudburst Formation yet due to the large volume of rhyolite fragments this unit was mapped as an intrusive.

The youngest rocks mapped in the area belong to the upper Tertiary Gila Formation. This past mineral unit consists of poorly to moderately consolidated, tan conglomerate and sandstone.

### Structure

Faulting in the area is extremely complex with at least four episodes represented. An early northeast episode associated with emplacement of the San Manuel porphyry copper system is present just west of the project area. This was followed by a northwest-trending system that appears to have controlled the mineralization at Tiger. These structures include the Dream fault and the main structural zone as seen in the Mammoth flux pit 9Plate 1). A series of northeast to east trending post mineral faults was observed in the Mammoth flux pit as well as in the northern part of the project area. The fourth and youngest episode of faulting consists of an extensive series of northwest-trending faults of post-Gila Formation age. These include the Mammoth fault.

## ORE DEPOSITS

### Structure

A main northwest trending zone of veining and mineralization encompasses both the Mammoth and Mohawk mines. The shear zone has been traced by drilling, at least 1,500 feet southeast of the Mohawk shaft.

Northwest of the Mohawk mine the main structure apparently. Splits to the Southeast forming two separate faults as indicated by Peterson (1938). Due to limited access to the underground workings, this was difficult to confirm. In the area of the Mohawk mine the structural zone tends to dip to the Northeast. The zone continues along a northwest strike until just past the Mammoth flux pit. The structure now tends to dip to the southwest. Past the pit the zone bends to the west and is eventually truncated by the post mineral Mammoth Fault. This northwest trending major fault dips to the northeast at an average of 60°. Data from underground level maps indicate that the dip on this structure tends to flatten with depth thereby appearing listric in nature. The Collins vein and Collins east vein located west of the Mammoth Fault, occur along the up thrown portion of the mineralized structure. The structure cuts all the major rock units except for the post mineral Gila Formation.

### Mineralogy

Due to extensive oxidation in the region the mineralization of the deposit is extremely complex. Several episodes of silica deposition ranging from crystalline, amethystine quartz to white coloform microcrystalline quartz are present. Other gangue minerals include minor fluorite, adularia, barite and calcite. Primary sulfide minerals, including galena, chalcophyrite, tetrahedrite, pyrite and sphalerite are present only in the deeper unoxidized levels of the Collins mine.

Numerous, secondary oxide minerals occur throughout all levels of the Mammoth and Mohawk mines and in the upper 700 feet of the Collins mine. The mines are known for producing museum quality samples of wulfenite, vanadinite, and descloizite among other rare oxide minerals (Bideaux, 1980). Strong hematite and limonite along with local chrysocolla are commonly found associated with the vein zones.

## Alteration

Hydrothermal alteration is confined to the main structural zone. Silicification is limited to those areas just adjacent to the veins whereas secondary clay, sericite, epidote and chlorite are more extensive. Other than silification the remaining alteration products tend to be more developed in the quartz monzonite than the rhyolite, the two main host units. Tourmaline is also locally present as an alteration product.

## Gold Mineralization

Gold mineralization is confined to several distinct areas along the main structure. In the area of the Mohawk mine an ore zone occurs at depth just southeast of the Mohawk shaft and continues northwest for approximately 700 feet. A low grade to barren zone, approximately 200 feet along strike, separates the Mohawk and mammoth ore bodies. At the Mammoth, ore grade gold mineralization occurs along strike for approximately 1,000 feet in northwest direction. At this point the structure turns to the west with gold mineralization occurring for an additional 500 feet until the intersection with mammoth Fault. Gold mineralization associated with the Collins system begins where the structure intersects the Mammoth Fault and can be traced for approximately 600 feet northwest along strike. At this point gold mineralization appears to become more sporadic.

Gold values also tend to decrease somewhat with depth as indicated by production records. Grades are still relatively high as evidenced by recorded assays in excess of 0.2 opt gold in oxidized ore as deep as the 800 foot level at the mammoth mine. It does appear that oxidation may have been important in gold concentration. Production from the Collins mine indicates a significant decrease in gold values associated with reduced ore. It has been reported that some select samples of galena have assayed as high as 0.375 opt gold (Bideaux, 1980).

In general, gold values tend to increase with increasing quartz veining and silicification but this is not always the case. There is some indication that near-surface, secondary enrichment of gold has taken place. North, of the mammoth flux pit drill hole MM-66 intersected and ore zone containing gold values as high as 0.375 opt. Drill cuttings of this zone show weakly altered quartz monzonite containing very little veining. Some silicification was noted and the mafic minerals were typically altered to chlorite. Bideaux (1980) also reports native gold associated with chlorite-rich zones in the upper portion of the Mammoth vein.

## Additional Metals

Of the metals analyzed for gold tends to occur in economic grades more consistently than the rest. Though silver values as high as 4 opt over 5 feet were obtained it is very uncommon to get assays in excess of 1 opt. Values in molybdenum, vanadium, lead and copper, though present in anomalous concentrations, are typically sporadic and low grade. Of the base metals, zinc tends to occur in higher concentrations. Drilling in the area of the Mammoth flux pit indicates an average zinc grade of approximately 0.7% for this area.

## Width of Mineralized Zones

The width of mineralization within the main structural zone varies throughout the property. Though individual veins as wide as 15 feet are present, a typical ore zone contains numerous subparallel veins ranging in thickness from less than ½ inch to about 2 feet. Local stockwork quartz vein zones and quartz breccia zones are also present. At the Mammoth the area of mineralization is up to 150 feet wide. The width of the ore zones is dependent on the density of veining and/or post mineral faulting which can juxtapose ore zones thereby increasing width.

In some areas post mineral faulting has affected the geometry of the ore zones. Several faulted segments of high-grade vein ore were encountered during drilling. An ore zone north of the Mammoth flux pit appears to be offset from the main trend by a possible splay of the Mammoth Fault.

## Other Mineralized Zones

Several additional mineralized structures occur throughout the property. These include the Dream vein (Map 1). Portions of this vein/fault are still accessible from the underground workings at the Mohawk mine. This mineralized structure strikes northwest and clips 45° - 60° northeast and typically has granite in the footwall and rhyolite in the hanging wall. Ore grade gold values were obtained along this structure both in underground samples (up to 3,000 opt on the 400 foot level) and in drill cuttings (up to 0.572 opt in MM-4). Underground sampling has shown that the distribution of this ore is erratic. However, the grades obtained do indicate some potential for the development of high grade ore. The projected near surface expression of this structure occurs under Magma's SX-EQ plant and copper leach pads. Thereby limiting any potential mining to underground. Other mineralized veins can be found north of the Collins area. Erratic gold values also occur along these structures which are typically narrow.

## ORE RESERVES AND METALLURGY

A computer generated ore reserve performed by Behre Dolbear Riverside utilizing a polygonal reserve estimate and a cone miner to determine a preliminary pit design was done on the Tiger ore body (Collins area excluded). Parameters included 1) a calculated internal cutoff of 0.02 opt Au, 2) all ore values greater than 0.5 opt Au were cut to 0.5 opt, 3) an estimated density of 14 cu. ft. per ton for ore and 15 cu. ft per ton for waste and 4) 50 foot ore projection on drill sections spaced 100 feet apart. The density for ore was estimated as high as 14 because of the highly fractured nature of the ore and the local presence of mineralized backfill. A higher waste density factor was used because of the amount of low density, post mineral Gila Formation rock present. No usable data was obtained from drill holes on section 540 within the projected ore zone. Therefore, the area of influence defined by this section was treated as waste.

Open pit, mineable ore reserves to a depth of 300 feet total approximately 1.7 million tons of ore grading 0.079 opt Au. The waste/ore ratio is approximately 11.9/1. A manual calculation using cross sections and ore pods was done by Ken Bondurant. his results show 1.75 million tons of ore grading 0.076 opt Au with a waste/ore ratio for 11.2/1.

Preliminary metallurgical studies indicate a total recovery of approximately 80% using a combination gravity and flotation process. Recovery increases to approximately 90% using gravity in conjunction with a CIL system.

Economic studies and optional mine plans are still being reviewed. A more detailed report on the ore reserves, metallurgy and the economics of the deposit will follow.

## EXPLORATION POTENTIAL

A few areas containing limited exploration potential still exist on the property. One area, located west of the Collins mine, is centered around drill hole CR-4 (Map 14). This hole intersected 60 feet of material averaging 0.073 opt Au. The extent of this mineralization is not known.

Another such area includes the northwest extension of the main Collins vein. No holes were drilled to test this part of the structure. Data obtained through geochemical sampling, mapping and from previous drilling was not very encouraging.

Along the main mineralized structure and along the Dream vein there still exists some potential for the discovery of sufficient quantities of deep, high-grade ore capable of supporting an underground mining operation. This potential is difficult to assess due to the variation in grade along strike and down dip, the extensive post-mineral faulting, and the large amount of high-grade ore already removed.

Several reconnaissance rock chip samples containing anomalous gold and silver values were collected west of the project area in Section 28, T8S, R16E. Some follow up reconnaissance work in this area may be warranted.

## EXPLORATION COSTS

Expenditures totalling approximately $960,00 were accrued on the Tiger project as of December 31, 1988.

## CONCLUSIONS AND RECOMMENDATIONS

Cyprus Minerals Company has defined a bulk tonnage gold deposit at Tiger, Arizona. A decision regarding feasibility has yet to be made. If this decision is favorable, additional drilling is recommended. Supplemental drilling should be done in selected areas along the main mineralized zone to confirm high grade continuity.

In addition, the area around drill hole CR-4, the area northwest of the Collins open cut, and the projection of the Dream vein could use some limited supplemental drilling.

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APPENDIX II

# TIGER PROJECT

# PRELIMINARY SLOPE DESIGN

*PREPARED FOR*

CYPRUS MINERALS INC.

*PREPARED BY*

D. E. Nicholas, P.E.
T. M. Ryan

*APRIL 1989*

CALL & NICHOLAS, INC.

## 1.0 INTRODUCTION

Call and Nicholas Inc., (CNI) was contracted by Cyprus Minerals Inc., to perform a feasibility study in regard to slope design for the Tiger Project. The scope of the study included the following:

1. a review of the geologic information available on the the two proposed pits;
2. geologic cell mapping of existing rock exposures;
3. determination of preliminary slope design recommendations.

The goal of the study was to assess the feasibility of mining the two deposits at very steep overall slope angles of 50 to 60 degrees (or steeper). It is our understanding that the mine plan and sections are currently based on an interramp slope angle of 60 degrees.

All conclusions presented herein are based on the work performed to date which included one day of geologic data review, two days of cell mapping at the site, and two days of slope design review. All of the geologic information used in the study was provided by Cyprus, and was very helpful in formulating the preliminary slope design recommendations.

## 2.0 SUMMARY

The preliminary slope angle recommendations for the Tiger Project are presented in Table 1. The angles presented are for the steepest, best estimate, and flattest slope angles achievable for the eight preliminary slope design sectors within the two pits. The design sectors (Figures 1 and 2) are based on the orientation of the walls in conjunction with the geologic information currently available.

Triple benching of twenty foot mining increments is recommended at this time. Twenty-four foot wide catch benches are recommended at each sixty foot interval to catch rockfalls from the bench faces. Since no major daylighted fault structures appear to be in the walls of the proposed pits, the design recommendations are primarily based on bench scale rock fabric, and estimated intact rock strengths.

The best estimate slope angles are for conventional drill and blast excavation, with some buffer row blasting along the final pit walls. If blasting is poorly implemented, and the rock fabric has dips that are flatter than currently expected, then the result will be narrower catch benches for the design slope angles.

To achieve the steepest recommended angles, both controlled blasting and some slope reinforcement would be necessary. The slope reinforcement recommended would include 10 to 12 gage wire mesh draped over the pit walls, held in place by spot bolting on a four foot spacing. With the mesh in place, and berms emplaced along the catch benches, the catch bench widths could be reduced to twenty feet. This would enable a steepening of the interramp slope angles up to 63 degrees in some sectors. Anticipated cost of the materials involved is approximately $0.23 per square foot, assuming a bolt spacing of four feet.

The water table in the area should be significantly below both pit bottoms. However, during our site visit we did notice that some water from the SXEW plant was being discharged into the north end of the Mammoth Pit. This practice would have to be discontinued.

Table 1

Preliminary Design Angles for the Tiger Project

| SECTOR | FLATTEST Interramp Slope Angle | FLATTEST Bench Face Angle | BEST ESTIMATE Interramp Slope Angle | BEST ESTIMATE Bench Face Angle | STEEPEST Interramp Slope Angle | STEEPEST Bench Face Angle |
|---|---|---|---|---|---|---|
| MAMMOTH PIT | | | | | | |
| 176 | 44.0 | 55.0 | 57.0 | 73.0 | 61.0 | 78.0 |
| 132 | 45.5 | 57.0 | 51.5 | 65.0 | 61.0 | 78.0 |
| 293 | 40.5 | 50.0 | 50.5 | 64.0 | 61.0 | 78.0 |
| 348 | 40.5 | 50.0 | 55.0 | 70.0 | 63.0 | 80.0 |
| Gila | 44.0 | 55.0 | 53.5 | 68.0 | 63.0 | 80.0 |
| MOHAWK PIT | | | | | | |
| 25 | 44.0 | 55.0 | 55.0 | 70.0 | 63.0 | 80.0 |
| 105 | 40.5 | 50.0 | 50.5 | 64.0 | 61.0 | 78.0 |
| 180 | 44.0 | 55.0 | 57.0 | 73.0 | 61.0 | 78.0 |
| 285 | 40.5 | 50.0 | 50.5 | 64.0 | 63.0 | 80.0 |

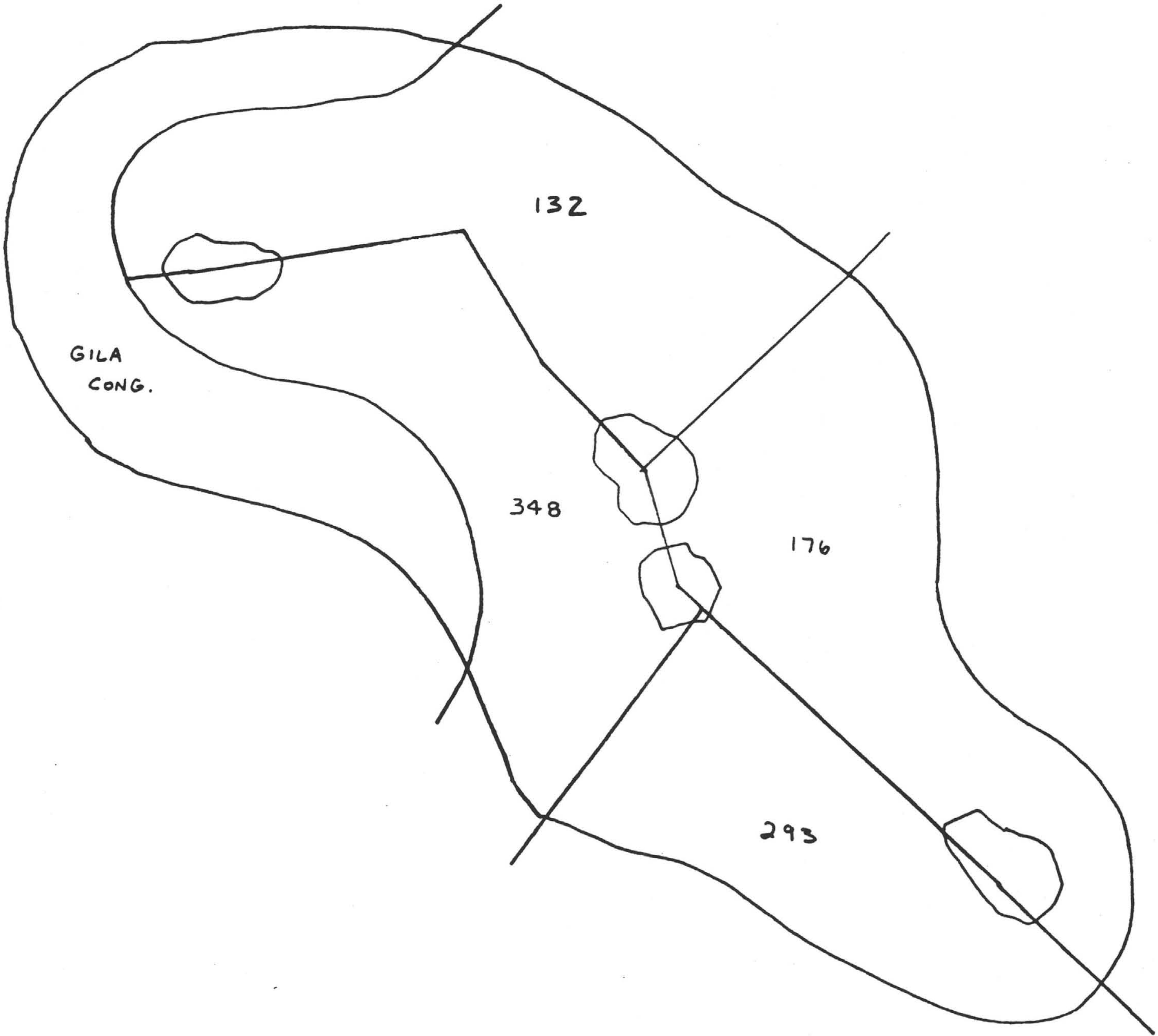



Figure 1. Design sectors for the Mammoth Pit.

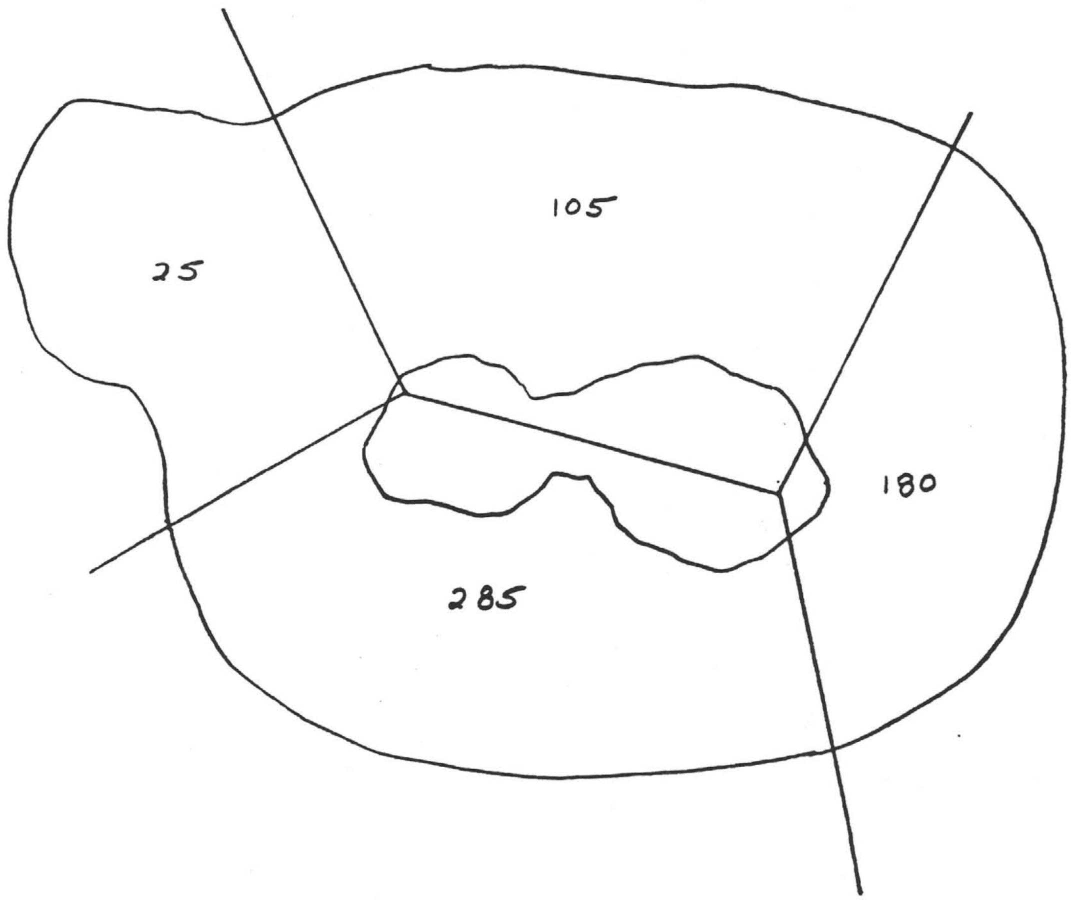



Figure 2. Design sectors for the Mohawk Pit.

## 3.0 SLOPE DESIGN APPROACH

Slope design involves analysis of the three major components of a mine slope: bench configuration, interramp slope angle, and overall slope angle (Figure 3). Bench configuration is defined by bench height, width, and face angle. The interramp slope is a series of benches, while the overall slope is a series of interramp slopes separated by haulroads.

The first portion of the slope that is designed is the bench geometry, enabling us to determine what the steepest interramp slope can be while still maintaining adequate catch bench widths (Figure 4). Bench configuration is governed by safety considerations, where minimum catch bench widths are maintained. The purpose of a catch bench is to stop rocks from rolling from the upper portion of the pit to the lower areas where men and equipment are working.

The second portion of the slope analysis is the interramp angle, where multiple bench failures are considered. Pit slopes are then checked for overall slope failure, which would extend from the crest of the slope to the toe of the slope.

The angle that is finally recommended for a sector is the lowest angle produced by bench, interramp, or overall stability analyses.

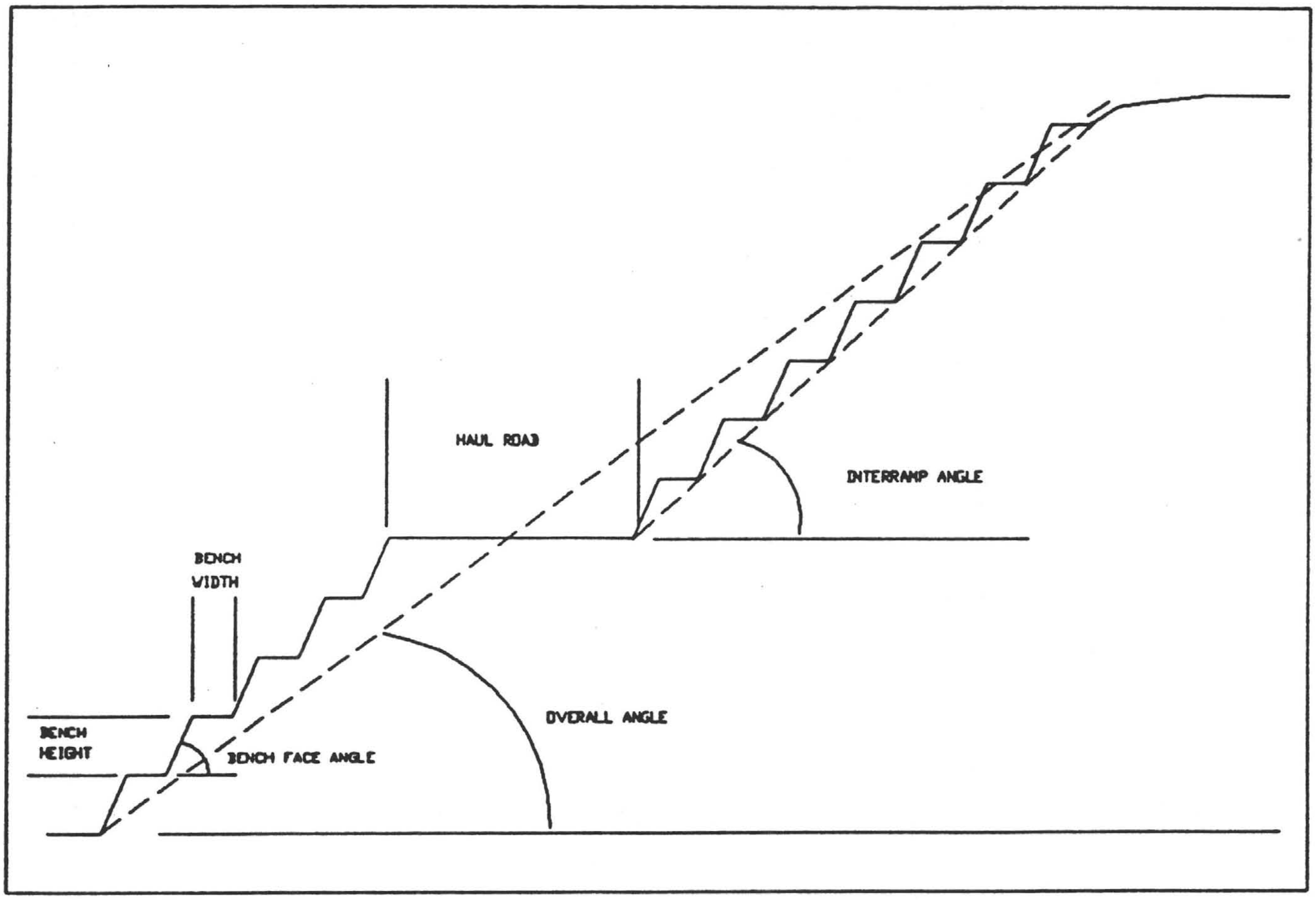



Figure 3.  General slope geometry.

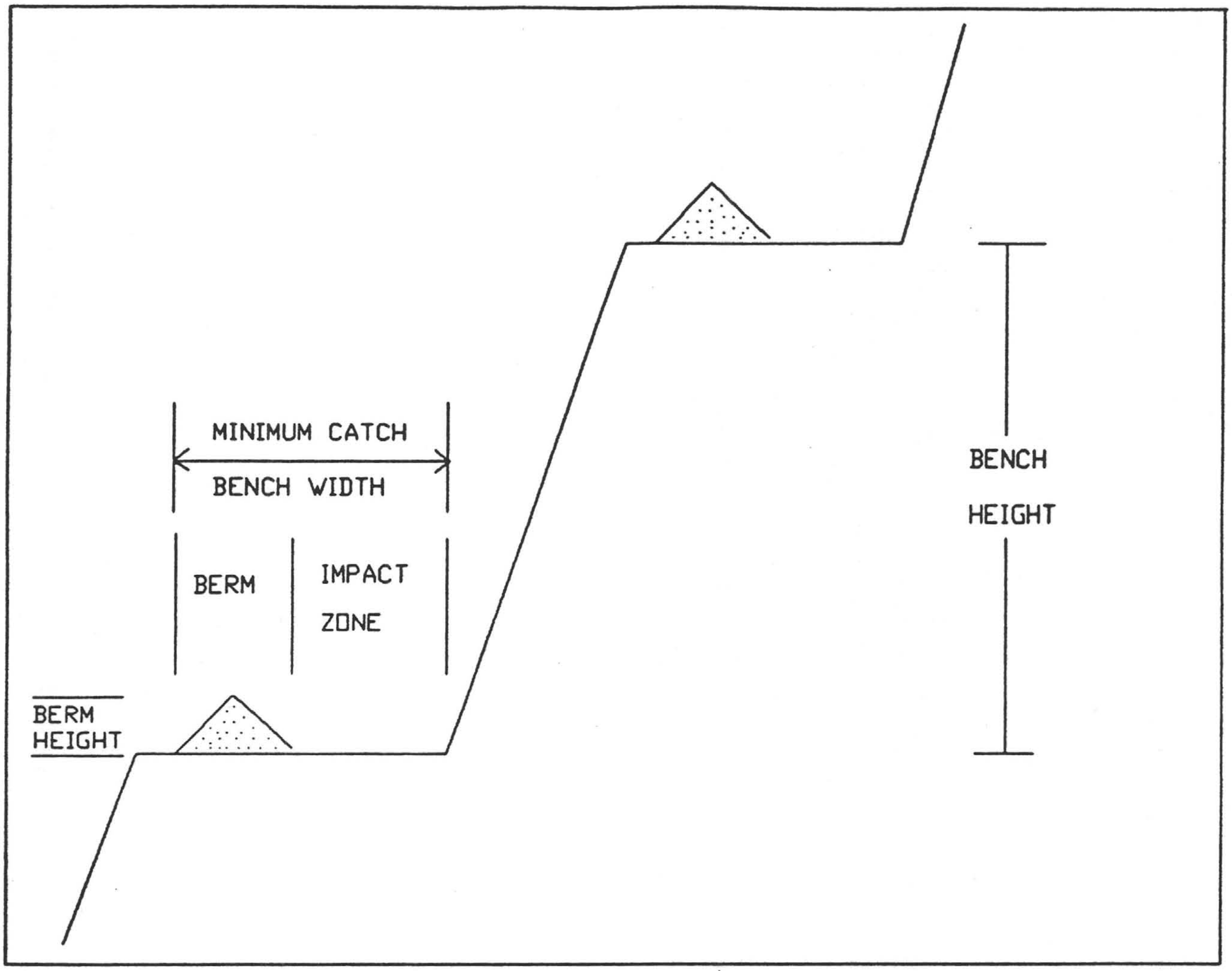



Figure 4. Catch bench design.

Slope angles within an open pit mine are influenced by rock strength, geologic structure, hydrology, pit wall orientation, ore distribution, and operational considerations. Design sectors are areas where these parameters are either the same or will have similar effects on slope design. For preliminary design, the primary factors used to define the limits of a design sector are (1) structural domain borders, (2) rock strength differences, and (3) pit wall orientation. Within each design sector, design structure sets are selected. Since structure orientation has more effect on slope design than does any other characteristic, design sets denote those structures which have a range of orientations that are expected to have a similar effect on the proposed design.

Bench configuration is a function of bench height, width, and face angle. The bench height is primarily a function of mining equipment; the bench width is a function of bench height and safety considerations; and the bench face angle is controlled by the orientation of geologic structures and by excavation methods used at the mine, particularly blasting.

Bench faces are normally mined as steeply as possible; as a result, rock fall and raveling are inevitable. Thus, it is customary, and in many cases mandated by mining regulations, that catch benches be left in the pit wall to retain rock falls and raveling. For a given bench height and corresponding bench width, the upper limit of the interramp slope angle becomes a function of the bench face angle.

Under ideal conditions (controlled blasting with vertical drill holes in unfractured rock), the bench face angle would be vertical. In actual conditions, however, the bench face breaks back to a flatter angle along jointing and other geologic structures. Obviously, uncontrolled blasting reduces the rock integrity, resulting in further backbreak.

# 4.0 PRELIMINARY PIT SLOPE DESIGN

## 4.1 Mammoth Pit

Five design sectors were defined for the Mammoth Pit (Figure 1). These sectors were delineated based on wall orientation and geology. Four major rock types will be exposed in the Mammoth Pit: Tertiary Gila Conglomerate, Tertiary Rhyolitic dikes, andesitic breccias of the Tertiary Cloud Burst Formation, and Pre-Cambrian Oracle Granite.

Based on past experience in similar alluvial gravels, and from examination of the Gila Conglomerate in the area of the pit, it is recommended that the Gila Conglomerate be excavated as steeply as possible with a combination of blasting and ripping. Benching will be desirable, and slope angles of 42 to 60 degrees should be achievable.

A Schmidt plot (equal area stereo projection) of the collected rock fabric data from cell mapping is presented in Figure 4. For the preliminary slope design, one structural domain was assumed for the analysis, with no distinction made between rock types. The preliminary slope angles were therefore dependent primarily on wall orientation.

Four wall orientations were examined with average strikes of 176, 132, 293, and 348 degrees. The amount of structure daylighting into the wall from both plane shear and wedge failure modes was then examined. From this analysis, estimates of achievable bench face angles for the four sectors were then made. Bench widths were estimated from experience and from preliminary rockfall model simulations of one, two, and three foot diameter falling rock blocks.

### 4.2 Mohawk Pit

The amount of rock exposure in the area of the Mohawk Pit is very limited. Since at this time one structural domain is assumed, the rock fabric data collected from the Mammoth Pit area was also used in the slope analysis of the Mohawk Pit. Since the predominant rock type in the Mohawk Pit will be rhyolite, the four preliminary design sectors for the Mohawk Pit were therefore based on wall orientation.

### 4.3 Results

The slope angle determinations for the Mammoth and Mohawk Pits are presented in Table 1. Schmidt plots of structure used in the analysis of design sectors for the two pits is presented in Appendix B. Measured bench face angles of existing benches in the flux pit are presented in Appendix C. Schmidt hammer data which can be used for preliminary estimates of unconfined compressive strength are presented in Appendix D.

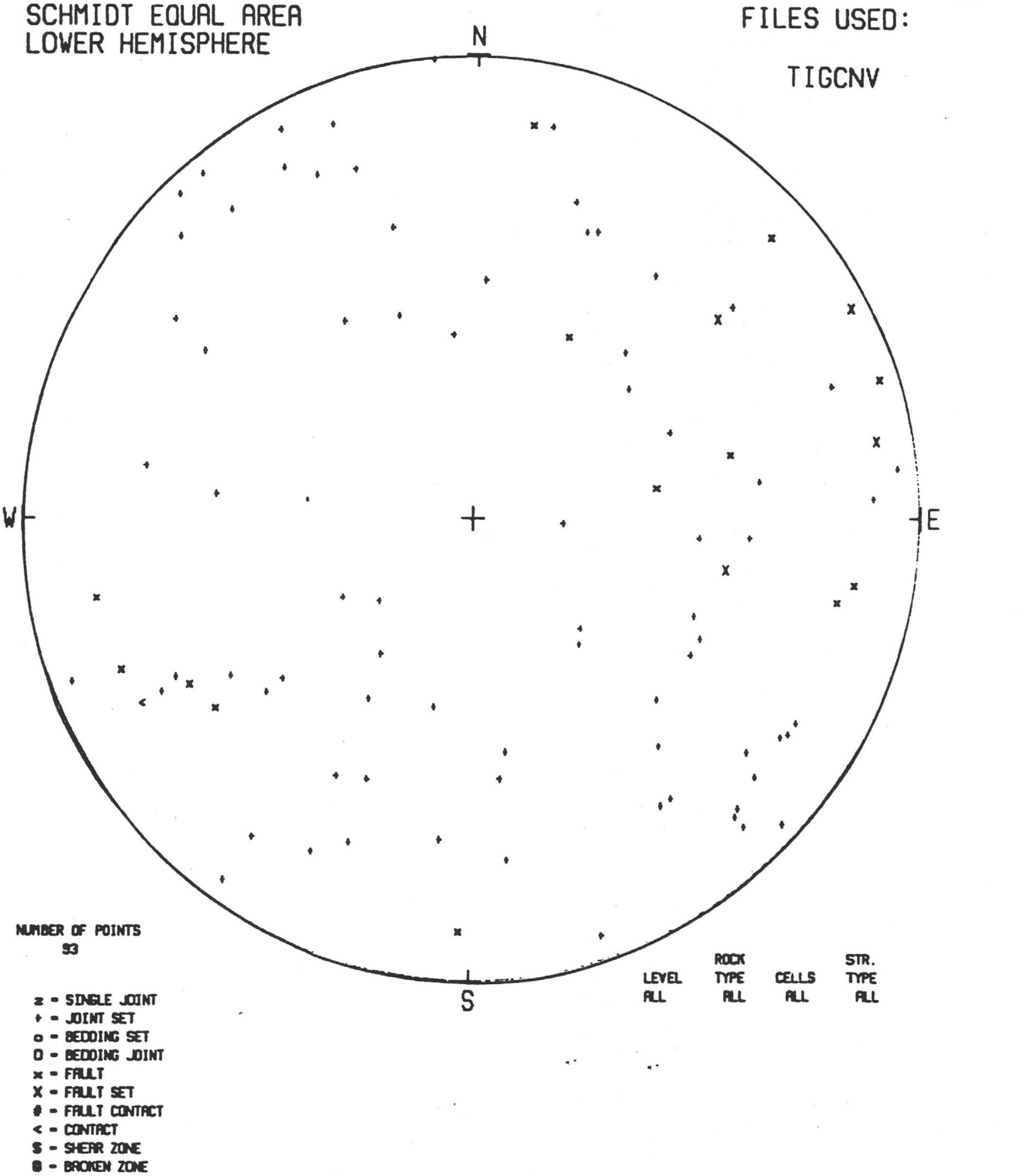



Figure 4. Rock fabric data collected for the Tiger Project.

## 5.0 RECOMMENDATIONS FOR FUTURE WORK

This study was of necessity limited in scope to a feasibility stage. If the financial analysis is favorable, and the decision is made to develop the property, a more detailed slope stability study is warranted. Specifically the following would be advisable:

1. several drill holes of oriented core;
2. rock testing of core samples;
3. further stability analyses;
4. a modified cost-benefit analysis;
5. design of an appropriate monitoring system.

*APPENDIX A*

*CELL MAPPING DATA*

TIGER PIT CELL MAPPING DATA AS OF 4-20-1989--CONVERTED DATA

| | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1. | 1. | 2003. | 1988. | 1. | 25. | 25. | 15. | 50. | RH | JS150 | 75 | 0. | 25.0 | 5.0025.00 | 7. | 0.000 | 0.0 | H |
| 1. | 1. | 2003. | 1988. | 1. | 25. | 25. | 15. | 50. | RH | JS140. | 85. | 0. | 3.0 | 1.0012.50 | 15. | 0.000 | 0.0 | H |
| 1. | 1. | 2003. | 1988. | 1. | 25. | 25. | 15. | 50. | RH | JS320. | 30. | 0. | 5.0 | 1.0020.00 | 11. | 0.000 | 0.0 | V |
| 1. | 1. | 2003. | 1988. | 1. | 25. | 25. | 15. | 50. | RH | FT 60 | 62 | 2. | 20.0 | 0.00 0.00 | 0. | 30.000 | 0.0 | |
| 1. | 2. | 2012. | 1962. | 1. | 30. | 25. | 20. | 44. | RH | JS152. | 83. | 0. | 6.0 | 1.00 8.50 | 11. | 0.000 | 0.0 | H |
| 1. | 2. | 2012. | 1962. | 1. | 30. | 25. | 20. | 44. | RH | JS316. | 28. | 0. | 1.5 | 1.0015.00 | 3. | 0.000 | 0.0 | V |
| 1. | 2. | 2012. | 1962. | 1. | 30. | 25. | 20. | 44. | RH | JS 58. | 28 | 0. | 1.0 | 1.0015.00 | 3. | 0.000 | 0.0 | V |
| 1. | 2. | 2012. | 1962. | 1. | 30. | 25. | 20. | 44. | RH | FT 78. | 75. | 0. | 18.0 | 0.00 0.00 | 0. | 10.000 | 0.0 | |
| 1. | 2. | 2012. | 1962. | 1. | 30. | 25. | 20. | 44. | RH | FT 67. | 74. | 0. | 25.0 | 0.00 0.00 | 0. | 6.000 | 0.0 | |
| 1. | 3. | 2035. | 1952. | 1. | 38. | 4. | 104. | 90. | RHRS | FT 2. | 79. | 0. | 4.0 | 0.00 0.00 | 0. | 18.000 | 0.0 | |
| 1. | 3. | 2035. | 1952. | 1. | 38. | 4. | 104. | 90. | RHRS | JS145. | 42 | 0 | 8.0 | 1.00 4.00 | 1. | 0.000 | 0.0 | V |
| 1. | 3. | 2035. | 1952. | 1. | 38. | 4. | 104. | 90. | RHRS | JS 61. | 68 | 0. | 2.5 | 1.0025.00 | 7. | 0.000 | 0.0 | H |
| 1. | 3. | 2035. | 1952. | 1. | 38. | 4. | 104. | 90. | RHRS | JS326. | 65. | 0. | 4.0 | 1.0030.00 | 9. | 0.000 | 0.0 | H |
| 2. | 4. | -10. | 7. | 2. | 25. | 15. | 236. | 56. | GR | JS191. | 74. | 0. | 15.0 | 4.0019.00 | 13. | 0.500G | 0.0 | H |
| 2. | 4. | -10. | 7. | 2. | 25. | 15. | 236. | 56. | GR | JS 22. | 52 | 0. | 15.0 | 3.0021.00 | 3. | 0.500G | 0.0 | H |
| 2. | 4. | -10. | 7. | 2. | 25. | 15. | 236. | 56. | GR | JS312. | 74. | 0. | 15.0 | 5.00 8.00 | 5. | 0.000 | 0.0 | P |
| 2. | 4. | -10. | 7. | 2. | 25. | 15. | 236. | 56. | GR | JS122. | 68. | 0. | 3.5 | 1.0015.00 | 4. | 0.000 | 0.0 | V |
| 2. | 5. | 1988. | 1971. | 2. | 25. | 15. | 220. | 69. | GR | JS 26. | 70. | 0. | 15.0 | 2.0020.00 | 6. | 0.300G | 0.0 | H |
| 2. | 5. | 1988. | 1971. | 2. | 25. | 15. | 220. | 69. | GR | JS310. | 69. | 0. | 15.0 | 2.00 3.00 | 2. | 0.000 | 0.0 | H |
| 2. | 5. | 1988. | 1971. | 2. | 25. | 15. | 220. | 69. | GR | JS268. | 78. | 0 | 15.0 | 2.0020.00 | 8. | 0.300G | 0.0 | H |
| 2. | 5. | 1988. | 1971. | 2. | 25. | 15. | 220. | 69. | GR | JS276 | 16. | 0. | 4.0 | 1.0015.00 | 3. | 0.000 | 0.0 | V |
| 2. | 5. | 1988. | 1971. | 2. | 25. | 15. | 220. | 69. | GR | JS172. | 32. | 0. | 2.0 | 1.0012.00 | 2. | 0.000 | 0.0 | V |
| 1. | 6. | 2012. | 1993. | 1. | 28. | 25. | 62. | 68. | CB | JS132. | 78. | 0. | 12.0 | 3.0028.00 | 7. | 0.000 | 0.0 | H |
| 1. | 6. | 2012. | 1993. | 1. | 28. | 25. | 62. | 68. | CB | JS318. | 77. | 0. | 12.0 | 3.0028.00 | 6. | 0.000 | 0.0 | H |
| 1. | 6. | 2012. | 1993. | 1. | 28. | 25. | 62. | 68. | CB | JS 48. | 23. | 0. | 8.0 | 1.0020.00 | 11. | 0.000 | 0.0 | V |
| 1. | 6. | 2012. | 1993. | 1. | 28. | 25. | 62. | 68. | CB | JS 35. | 74. | 0 | 10.0 | 1.0021.00 | 3. | 0.200G | 0.0 | H |
| 1. | 6. | 2012. | 1993. | 1. | 28. | 25. | 62. | 68. | CB | JS 94. | 48. | 0 | 4.0 | 1.00 7.00 | 3. | 0.000 | 0.0 | V |
| 1. | 7. | 2046. | 1976. | 1. | 48. | 20. | 62. | 68. | CB | JS 62 | 64. | 0. | 20.0 | 1.00 5.00 | 3. | 0.100G | 0.0 | P |
| 1. | 7. | 2046. | 1976. | 1. | 48. | 20. | 62. | 68. | CB | JS159. | 79. | 0. | 20.0 | 1.0040.00 | 5. | 0.000 | 0.0 | H |
| 1. | 7. | 2046. | 1976. | 1. | 48. | 20. | 62. | 68. | CB | JS 28. | 54. | 0. | 20.0 | 1.0040.00 | 8. | 0.200G | 0.0 | H |
| 1. | 7. | 2046. | 1976. | 1. | 48. | 20. | 62. | 68. | CB | JS 98. | 63. | 0. | 15.0 | 1.0040.00 | 5. | 0.100G | 0.0 | H |
| 1. | 7. | 2046. | 1976. | 1. | 48. | 20. | 62. | 68. | CB | JS218 | 55. | 0. | 3.0 | 2.0015.00 | 6. | 0.200G | 0.0 | V |
| 1. | 10. | 1992. | 1988. | 1. | 30. | 20. | 326. | 78. | RHCB | FS242. | 84. | 4. | 15.0 | 3.0015.00 | 5. | 0.500CS | 0.0 | H |
| 1. | 10. | 1992. | 1988. | 1. | 30. | 20. | 326. | 78. | RHCB | JS232 | 62. | 2. | 15.0 | 1.0015.00 | 5. | 0.100S | 0.0 | H |
| 1. | 10. | 1992. | 1988. | 1. | 30. | 20. | 326. | 78. | RHCB | JS 50. | 50. | 4. | 4.0 | 2.0020.00 | 5. | 0.100S | 0.0 | V |
| 1. | 10. | 1992. | 1988. | 1. | 30. | 20. | 326. | 78. | RHCB | JS314. | 48. | 4. | 15.0 | 1.0015.00 | 3. | 0.200S | 0.0 | V |
| 1. | 10. | 1992. | 1988. | 1. | 30. | 20. | 326. | 78. | RHCB | JS304. | 74. | 4. | 10.0 | 1.0010.00 | 3. | 0.100S | 0.0 | H |
| 1. | 11. | 1976. | 1968. | 1. | 20. | 20. | 314. | 78. | CB | FT 54. | 60. | 10. | 27.0 | 0.00 0.00 | 0. | 1.000G | 0.0 | |
| 1. | 11. | 1976. | 1968. | 1. | 20. | 20. | 314. | 78. | CB | JS 57. | 54. | 0. | 10.0 | 3.0016.00 | 11. | 0.000 | 0.0 | H |
| 1. | 11. | 1976. | 1968. | 1. | 20. | 20. | 314. | 78. | CB | JS251 | 73. | 0. | 10.0 | 1.0018.00 | 6. | 0.000 | 0.0 | H |
| 1. | 11. | 1976. | 1968. | 1. | 20. | 20. | 314. | 78. | CB | JS224. | 40. | 0 | 12.0 | 1.00 6.00 | 7. | 0.000 | 0.0 | T |
| 1. | 11. | 1976. | 1968. | 1. | 20. | 20. | 314. | 78. | CB | JS302. | 74. | 0. | 26.0 | 1.00 6.00 | 7. | 0.000 | 0.0 | T |
| 1. | 11. | 1976. | 1968. | 1. | 20. | 20. | 314. | 78. | CB | JS158. | 38. | 0. | 12.0 | 2.0025.00 | 6. | 0.000 | 0.0 | T |
| 1. | 11. | 1976. | 1968. | 1. | 20. | 20. | 314. | 78. | CB | FT262. | 34. | 0. | 35.0 | 0.00 0.00 | 0. | 0.000 | 0.0 | |
| 1. | 12. | 1965. | 1949. | 1. | 25. | 15. | 342. | 70. | RH | FT252. | 84. | 0. | 10.0 | 1.0025.00 | 1. | 0.500CS | 0.0 | H |
| 1. | 12. | 1965. | 1949. | 1. | 25. | 15. | 342. | 70. | RH | JS354. | 48. | 4. | 5.0 | 1.0015.00 | 3. | 0.200S | 0.0 | V |
| 1. | 12. | 1965. | 1949. | 1. | 25. | 15. | 342. | 70. | RH | JS318. | 80. | 0. | 12.0 | 2.0015.00 | 4. | 0.200S | 0.0 | H |
| 1. | 12. | 1965. | 1949. | 1. | 25. | 15. | 342. | 70. | RH | JS314. | 86 | 2. | 4.0 | 2.00 8.00 | 5. | 0.100S | 0.0 | H |
| 1. | 12. | 1965. | 1949. | 1. | 25. | 15. | 342. | 70. | RH | JS182 | 42. | 0 | 5.0 | 1.0015.00 | 3. | 0.100S | 0.0 | V |
| 1. | 12. | 1965. | 1949. | 1. | 25. | 15. | 342. | 70. | RH | JS 68 | 86. | 22. | 10.0 | 3.0012.00 | 10. | 0.000 | 0.0 | T |
| 1. | 12. | 1965. | 1949. | 1. | 25. | 15. | 342. | 70. | RH | JS 34. | 30 | 0. | 20.0 | 1.00 4.00 | 8. | 0.000 | 0.0 | T |
| 1. | 12. | 1965. | 1949. | 1. | 25. | 15. | 342. | 70. | RH | JS 50 | 46. | 0. | 15.0 | 1.0020.00 | 7. | 0.000 | 0.0 | H |
| 1. | 13. | 1990. | 2020. | 1. | 45. | 30. | 206. | 54. | CB | JS232 | 36 | 0 | 10.0 | 1.0025.00 | 12. | 0.000 | 0.0 | H |
| 1. | 13. | 1990. | 2020. | 1. | 45. | 30. | 206. | 54. | CB | JS276. | 42 | 0. | 25.0 | 1.0020.00 | 6. | 0.000 | 0.0 | H |
| 1. | 13. | 1990. | 2020. | 1. | 45. | 30. | 206. | 54. | CB | JS136. | 85. | 0. | 12.0 | 1.00 5.00 | 10. | 0.000 | 0.0 | H |
| 1. | 13. | 1990. | 2020. | 1. | 45. | 30. | 206. | 54. | CB | JS140 | 74 | 0. | 20.0 | 1.0010.00 | 4. | 0.000 | 0.0 | T |
| 1. | 13. | 1990. | 2020. | 1. | 45. | 30. | 206. | 54. | CB | JS 12. | 35. | 0. | 10.0 | 1.0023.00 | 5. | 0.000 | 0.0 | V |
| 1. | 13. | 1990. | 2020. | 1. | 45. | 30. | 206. | 54. | CB | JS317 | 76 | 0 | 8.0 | 1.0015.00 | 4. | 0.000 | 0.0 | V |
| 1. | 13. | 1990. | 2020. | 1. | 45. | 30. | 206. | 54. | CB | FS232 | 58 | 0. | 40.0 | 2.0010.00 | 5. | 1.000G | 0.0 | T |

CALL & NICHOLAS, INC.

| | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 2. | 14. | 1975. | 2043. | 2. | 100. | 65. 210. 88. GR | JS352 | 43 | 0 | 25. 0 | 1. 00 6. 00 | 3. | 0 000 | 0. 0 | T |
| 2 | 14. | 1975. | 2043. | 2. | 100. | 65. 210. 88. GR | JS 35. | 86 | 0. | 20. 0 | 1. 0025. 00 | 4. | 0. 000 | 0. 0 | V |
| 2. | 14. | 1975. | 2043. | 2. | 100. | 65. 210. 88. GR | FS282. | 48. | 0. | 60. 0 | 1. 0014. 00 | 6. | 6. 000 | 0. 0 | T |
| 2. | 14. | 1975. | 2043. | 2. | 100. | 65. 210. 88. GR | FS260. | 80. | 0. | 99. 9 | 1. 0060. 00 | 6. | 0. 000 | 0. 0 | H |
| 4. | 15. | 1992. | 2010. | 4. | 25. | 12. 219. 67. GR | JS ·6. | 60. | 0. | 10. 0 | 1. 0010. 00 | 3. | 0. 000 | 0. 0 | T |
| 4. | 15. | 1992. | 2010. | 4. | 25. | 12. 219. 67. GR | JS275. | 52. | 0. | 12. 0 | 2. 0015. 00 | 14. | 0. 000 | 0. 0 | H |
| 4. | 15. | 1992. | 2010. | 4. | 25. | 12. 219. 67. GR | JS154. | 70. | 0. | 10. 0 | 4. 0020. 00 | 7. | 0. 000 | 0. 0 | H |
| 4. | 15. | 1992. | 2010. | 4. | 25. | 12. 219. 67. GR | JS120. | 59. | 0. | 4. 0 | 1. 0010. 00 | 8. | 0. 000 | 0. 0 | T |
| 4. | 16. | 1974. | 2026. | 4. | 25. | 12. 236. 65. GR | JS264. | 84. | 0. | 12. 0 | 2. 0018. 00 | 2. | 0. 100S | 0. 0 | H |
| 4. | 16. | 1974. | 2026. | 4. | 25. | 12. 236. 65. GR | JS302. | 48. | 0. | 8. 0 | 2. 00 8. 00 | 3. | 0. 100CS | 0. 0 | H |
| 4. | 16. | 1974. | 2026. | 4. | 25. | 12. 236. 65. GR | JS248 | 39 | 2. | 12. 0 | 3. 0020. 00 | 3. | 0. 100 | 0. 0 | P |
| 4. | 16. | 1974. | 2026. | 4. | 25. | 12. 236. 65. GR | JS174. | 88 | 4. | 10. 0 | 4. 00 9. 00 | 5. | 0. 100S | 0. 0 | H |
| 4. | 16. | 1974. | 2026. | 4. | 25. | 12. 236. 65. GR | JS202. | 55. | 0. | 5. 0 | 2. 0025. 00 | 5. | 0. 100S | 0. 0 | H |
| 4. | 17. | 1955. | 2039. | 4. | 20. | 12. 239. 58. GR | JS264. | 54 | 0. | 10. 0 | 1. 0020. 00 | 4. | 0. 100S | 0. 0 | H |
| 4. | 17. | 1955. | 2039. | 4. | 20. | 12. 239. 58. GR | JS298. | 48. | 0. | 10. 0 | 1. 0020. 00 | 2. | 0. 100S | 0. 0 | H |
| 4. | 17. | 1955. | 2039. | 4. | 20. | 12. 239. 58. GR | JS354. | 64 | 0. | 10. 0 | 3. 0010. 00 | 6. | 0. 100 | 0. 0 | H |
| 4. | 17. | 1955. | 2039. | 4. | 20. | 12. 239. 58. GR | JS342. | 85 | 0. | 8. 0 | 2. 00 9. 00 | 3. | 0. 100CS | 0. 0 | H |
| 4. | 17. | 1955. | 2039. | 4. | 20. | 12. 239. 58. GR | JS198. | 60. | 0. | 10. 0 | 3. 0015. 00 | 4. | 0. 100CS | 0. 0 | H |
| 4. | 18. | 1941. | 2052. | 4. | 20. | 20. 214. 70. CB | JS320. | 55. | 0 | 15. 0 | 1. 0015. 00 | 14. | 0. 000 | 0. 0 | H |
| 4. | 18. | 1941. | 2052. | 4. | 20. | 20. 214. 70. CB | JS 21 | 65 | 0 | 12. 0 | 1. 00 4. 00 | 3. | 0. 000 | 0. 0 | T |
| 4. | 18. | 1941. | 2052. | 4. | 20. | 20. 214. 70. CB | FT257. | 49. | 0 | 30. 0 | 0. 00 0. 00 | 0. | 3. 000G | 0. 0 | |
| 4. | 18. | 1941. | 2052. | 4. | 20. | 20. 214. 70. CB | JS160 | 68. | 0. | 9. 0 | 1. 00 8. 00 | 4. | 0. 000 | 0. 0 | T |
| 4. | 18. | 1941. | 2052. | 4. | 20. | 20. 214. 70. CB | CT 61. | 73. | 0. | 30. 0 | 0. 00 0. 00 | 0. | 0. 000 | 0. 0 | |
| 1. | 19. | 1990. | 2008. | 1. | 25. | 20. 232. 71. GR | JS305. | 73 | 0. | 17. 0 | 1 0012. 00 | 7. | 0. 000 | 0. 0 | H |
| 1. | 19. | 1990. | 2008. | 1. | 25. | 20. 232. 71. GR | FT280. | 75. | 0. | 30. 0 | 0. 00 0. 00 | 0. | 12. 000G | 0. 0 | |
| 1. | 19. | 1990. | 2008. | 1. | 25. | 20. 232. 71. GR | JS294. | 45. | 0. | 12. 0 | 1. 0015. 00 | 7. | 0. 000 | 0. 0 | H |
| 1. | 19. | 1990. | 2008. | 1. | 25. | 20. 232. 71. GR | JS163 | 54. | 0. | 15. 0 | 1. 0017. 00 | 5. | 0. 000 | 0. 0 | H |
| 1. | 19. | 1990. | 2008. | 1. | 25. | 20. 232. 71. GR | JS204. | 56. | 0. | 11. 0 | 1. 00 7. 00 | 5. | 0. 000 | 0. 0 | T |
| 1. | 20. | 1973. | 2022. | 1. | 20. | 30. 230. 74. GR | FT208. | 36 | 0. | 40. 0 | 0. 00 0. 00 | 0. | 0. 300S | 0. 0 | |
| 1. | 20. | 1973. | 2022. | 1. | 20. | 30. 230. 74. GR | FT283. | 72. | 0. | 20. 0 | 0. 00 0. 00 | 0. | 0. 200S | 0. 0 | |
| 1. | 20. | 1973. | 2022. | 1. | 20. | 30. 230. 74. GR | FT188. | 74. | 0. | 20. 0 | 0. 00 0. 00 | 0. | 0. 700S | 0. 0 | |
| 1. | 20 | 1973. | 2022. | 1. | 20. | 30. 230. 74. GR | FT228. | 78. | 0 | 20. 0 | 0. 00 0. 00 | 0. | 0. 200S | 0. 0 | |
| 1. | 20. | 1973. | 2022. | 1. | 20. | 30. 230. 74. GR | JS324. | 65. | 0 | 5. 0 | 3. 0020. 00 | 6. | 0. 100 | 0. 0 | H |
| 1. | 20. | 1973. | 2022. | 1. | 20. | 30. 230. 74. GR | JS 30. | 38. | 0 | 10. 0 | 2. 0015. 00 | 5. | 0. 100 | 0. 0 | H |

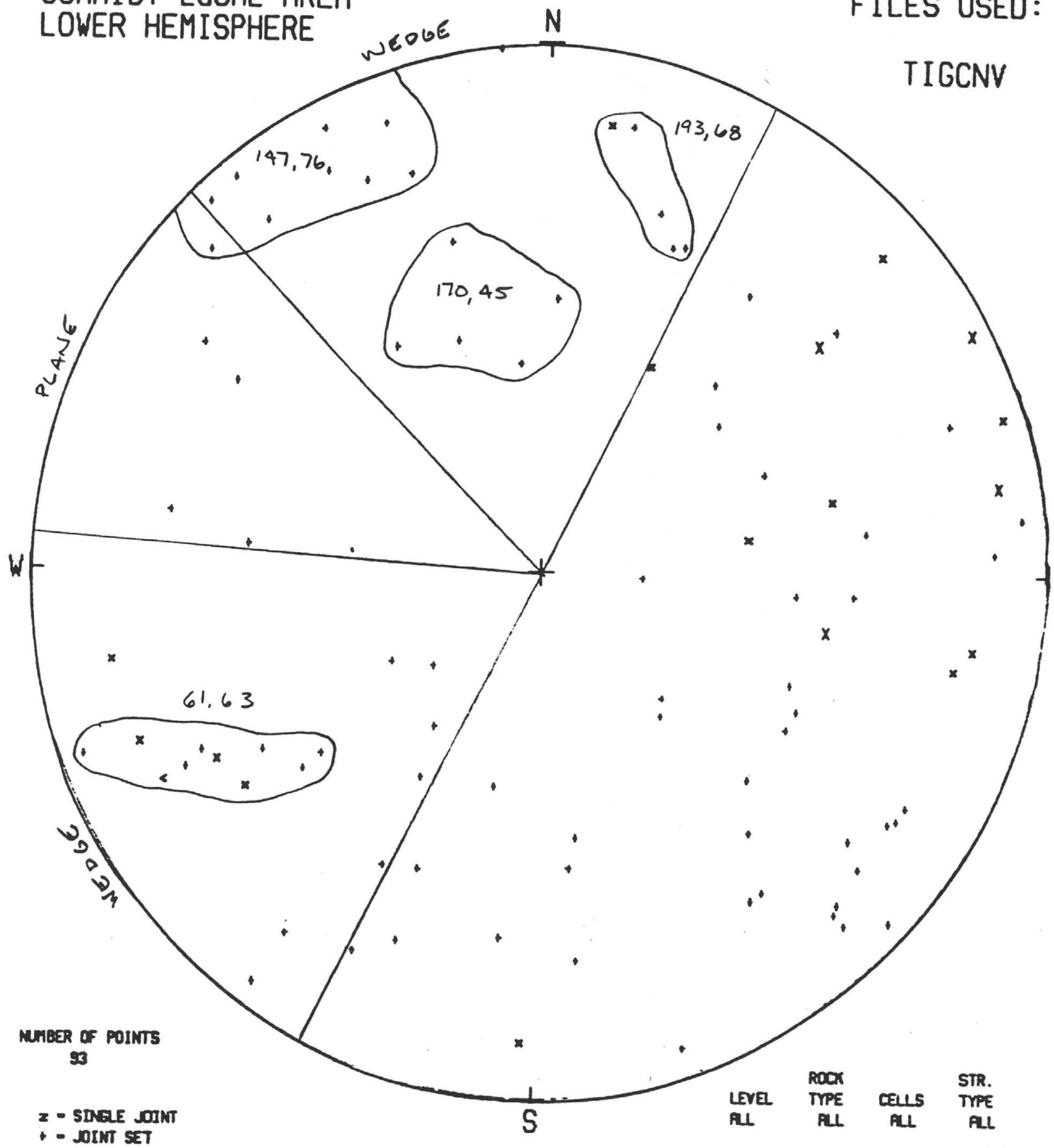
## *APPENDIX B*

## *SCHMIDT PLOTS FOR DESIGN SECTORS*

SCHMIDT EQUAL AREA
LOWER HEMISPHERE

FILES USED:

TIGCNV




NUMBER OF POINTS
93

| LEVEL | ROCK TYPE | CELLS | STR. TYPE |
|---|---|---|---|
| ALL | ALL | ALL | ALL |

z - SINGLE JOINT
+ - JOINT SET
o - BEDDING SET
O - BEDDING JOINT
x - FAULT
X - FAULT SET
# - FAULT CONTACT
< - CONTACT
S - SHEAR ZONE
B - BROKEN ZONE

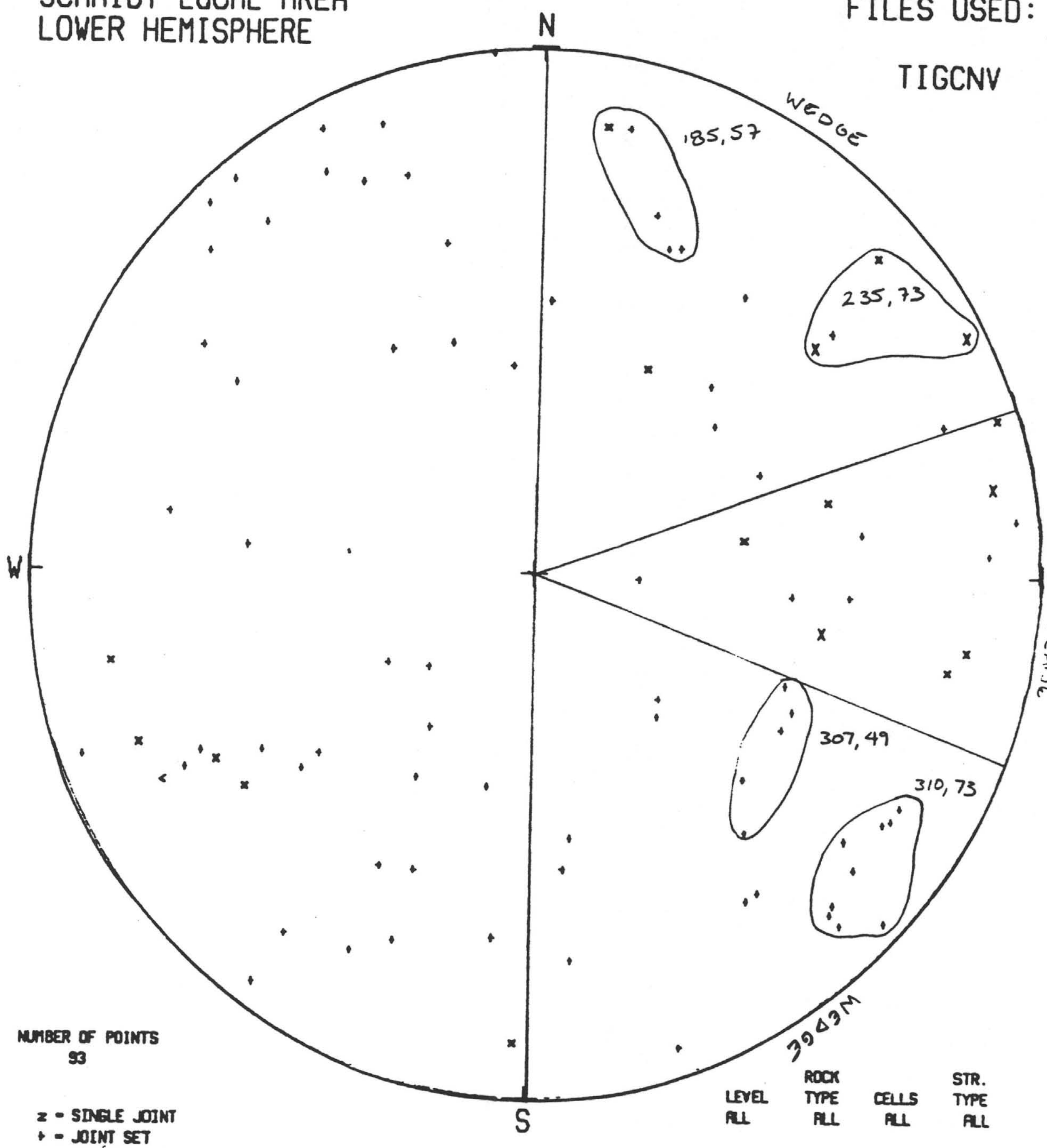
SECTOR 25

CALL & NICHOLAS, INC.

SCHMIDT EQUAL AREA
LOWER HEMISPHERE

FILES USED:

TIGCNV




NUMBER OF POINTS
93

| LEVEL | ROCK TYPE | CELLS | STR. TYPE |
|---|---|---|---|
| ALL | ALL | ALL | ALL |

z - SINGLE JOINT
+ - JOINT SET
o - BEDDING SET
O - BEDDING JOINT
x - FAULT
X - FAULT SET
# - FAULT CONTACT
< - CONTACT
S - SHEAR ZONE
B - BROKEN ZONE

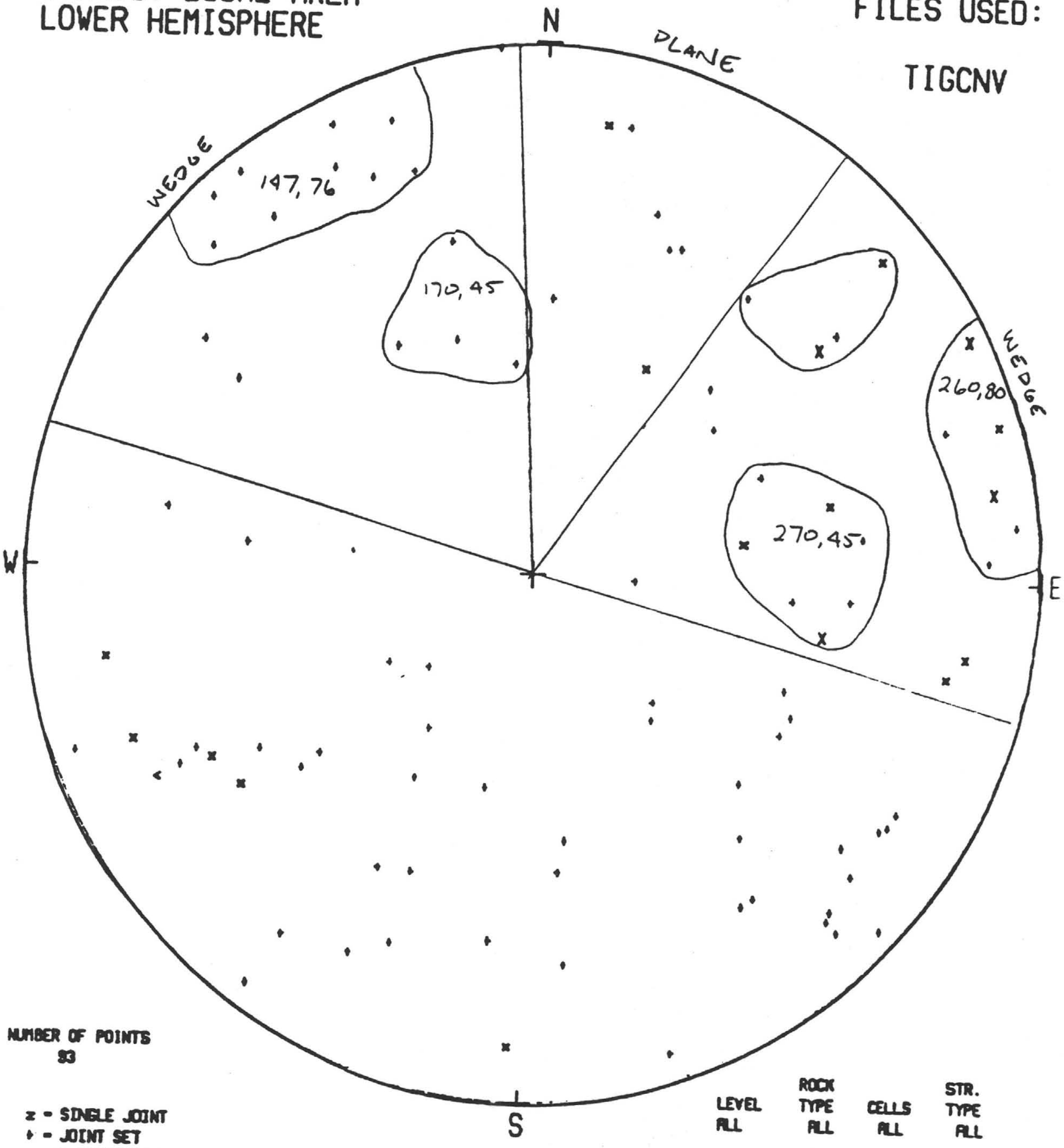
SECTOR 180

CALL & NICHOLAS, INC.

SCHMIDT EQUAL AREA
LOWER HEMISPHERE

FILES USED:

TIGCNV




NUMBER OF POINTS
93

| LEVEL | ROCK TYPE | CELLS | STR. TYPE |
|---|---|---|---|
| ALL | ALL | ALL | ALL |

z - SINGLE JOINT
+ - JOINT SET
o - BEDDING SET
O - BEDDING JOINT
x - FAULT
X - FAULT SET
# - FAULT CONTACT
< - CONTACT
S - SHEAR ZONE
B - BROKEN ZONE

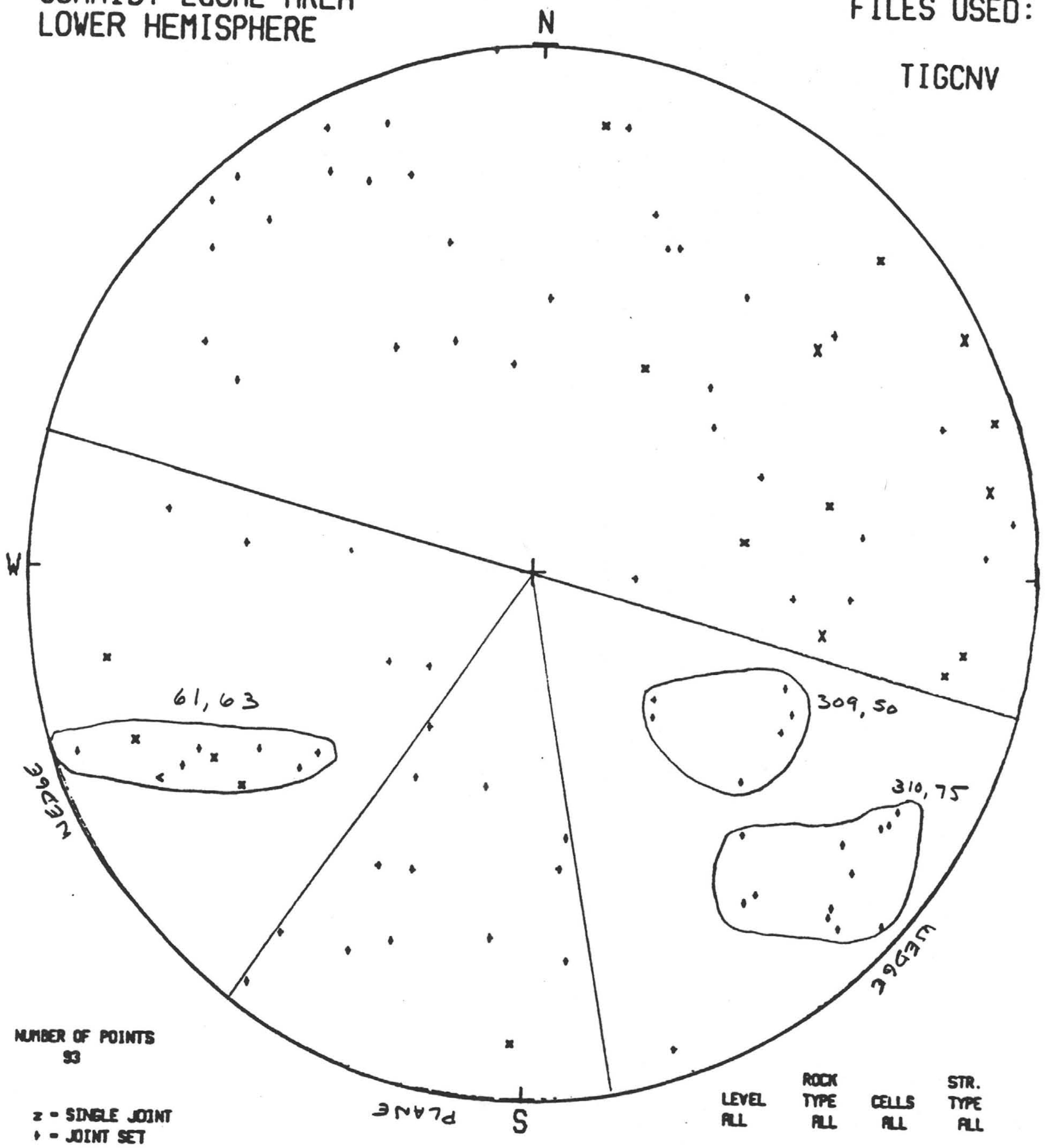
SECTOR 105

CALL & NICHOLAS, INC.

SCHMIDT EQUAL AREA
LOWER HEMISPHERE

FILES USED:

TIGCNV



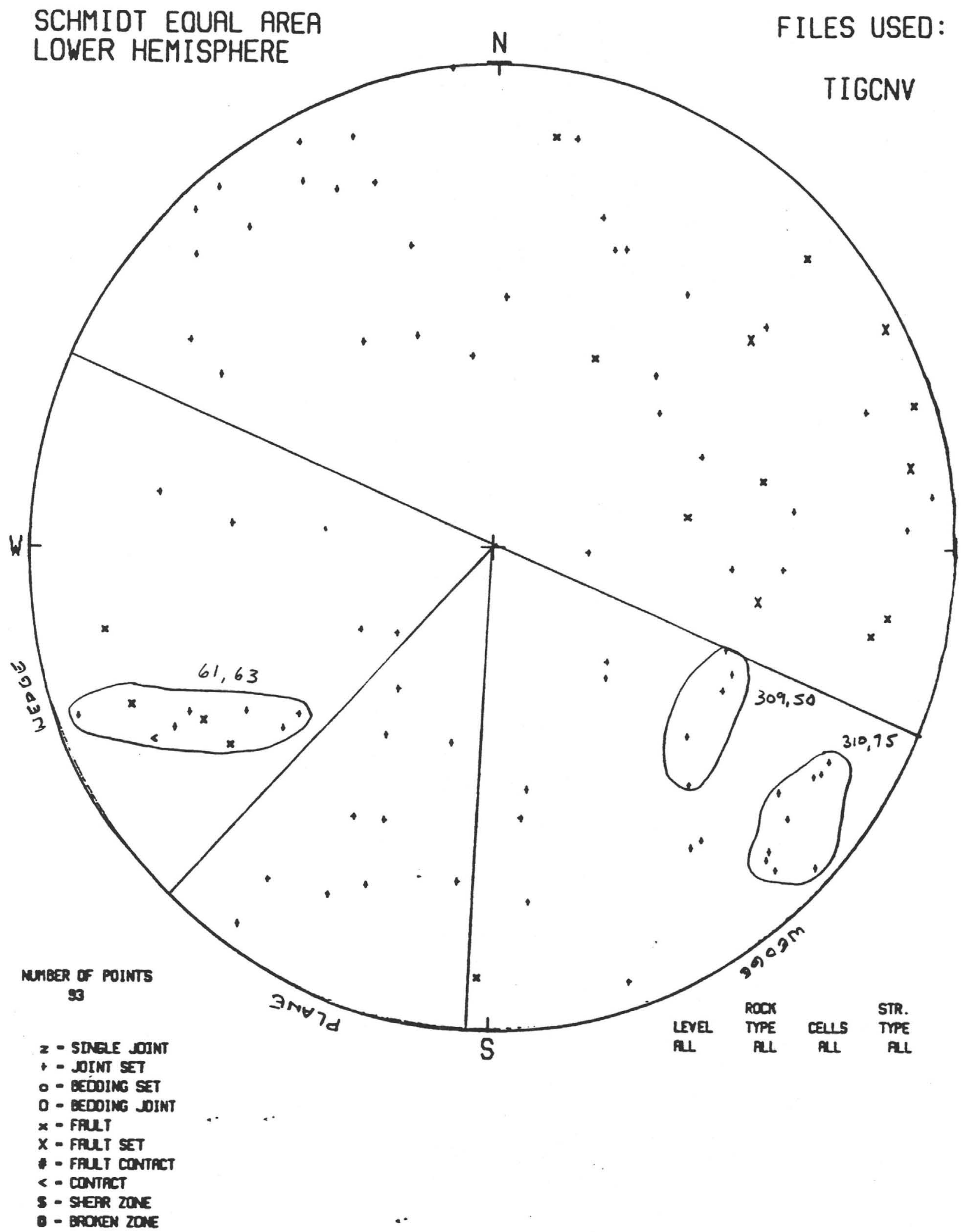

NUMBER OF POINTS
93

z - SINGLE JOINT
+ - JOINT SET
o - BEDDING SET
O - BEDDING JOINT
x - FAULT
X - FAULT SET
# - FAULT CONTACT
< - CONTACT
S - SHEAR ZONE
B - BROKEN ZONE

| LEVEL | ROCK TYPE | CELLS | STR. TYPE |
|---|---|---|---|
| ALL | ALL | ALL | ALL |

SECTOR 285

CALL & NICHOLAS, INC.



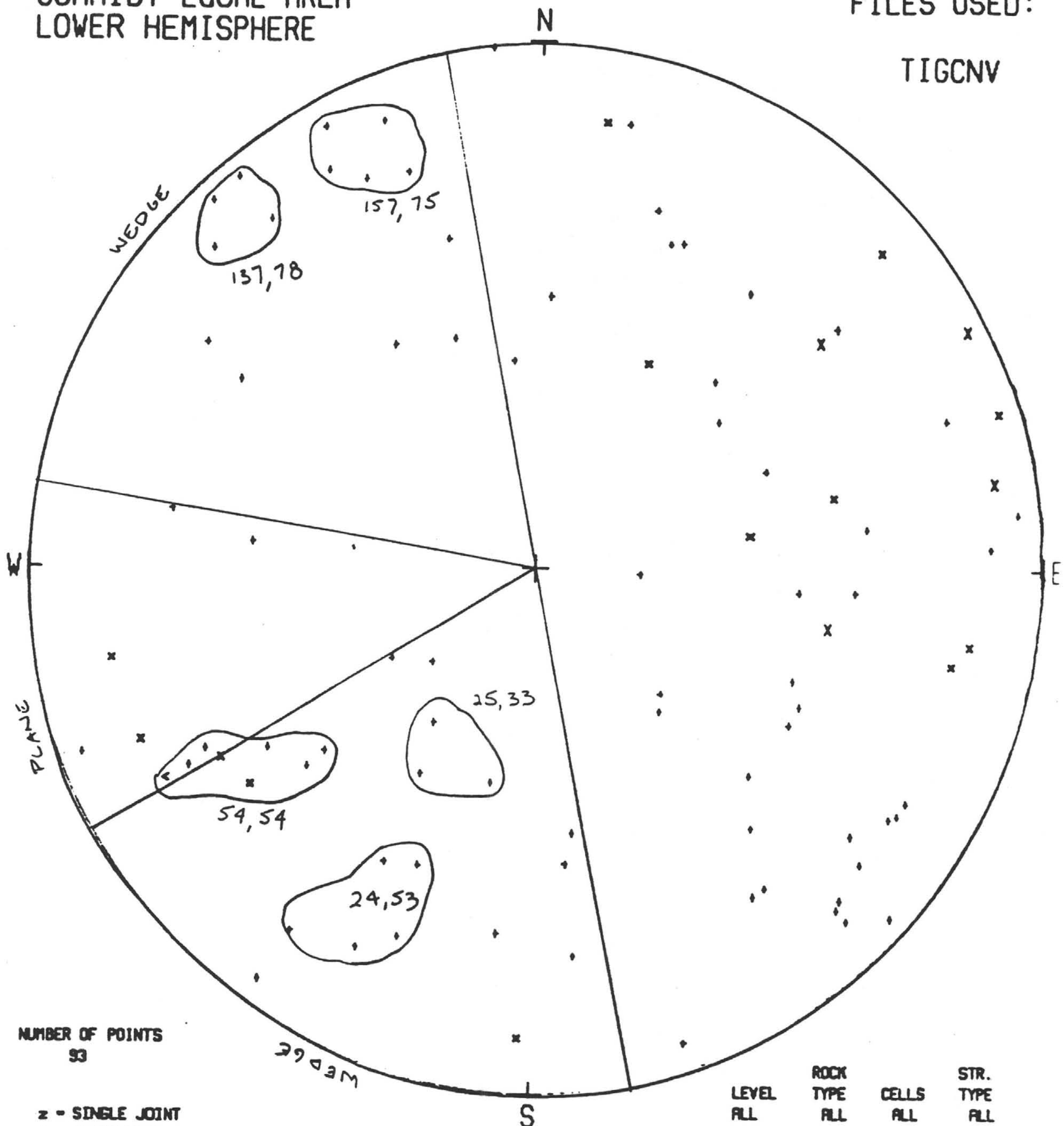

SECTOR 293

CALL & NICHOLAS, INC.

SCHMIDT EQUAL AREA
LOWER HEMISPHERE

FILES USED:

TIGCNV




NUMBER OF POINTS
93

| LEVEL | ROCK TYPE | CELLS | STR. TYPE |
|---|---|---|---|
| ALL | ALL | ALL | ALL |

z - SINGLE JOINT
+ - JOINT SET
o - BEDDING SET
O - BEDDING JOINT
x - FAULT
X - FAULT SET
# - FAULT CONTACT
< - CONTACT
S - SHEAR ZONE
B - BROKEN ZONE

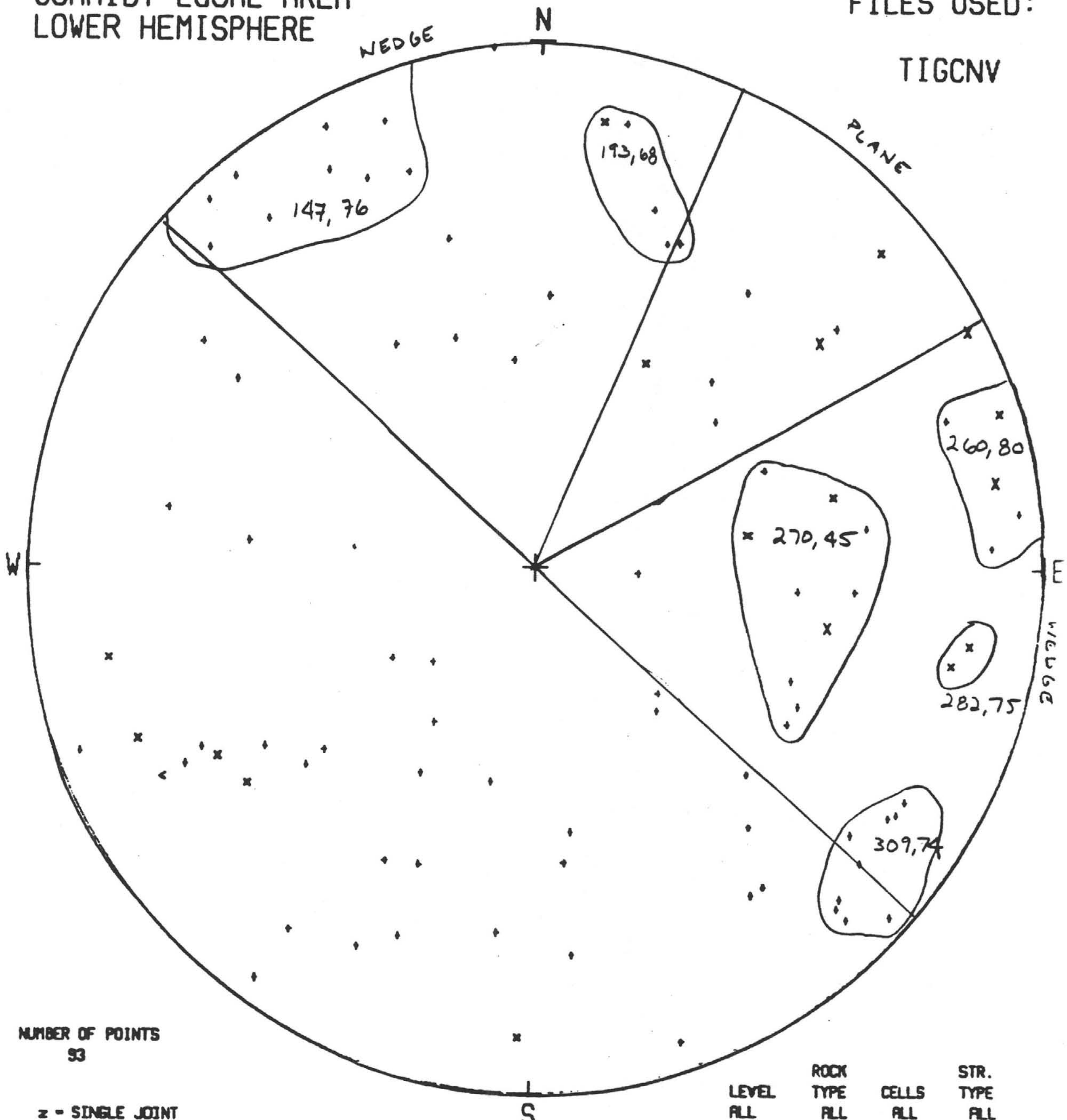
SECTOR 348

CALL & NICHOLAS, INC.

SCHMIDT EQUAL AREA
LOWER HEMISPHERE

FILES USED:

TIGCNV




NUMBER OF POINTS
93

| LEVEL | ROCK TYPE | CELLS | STR. TYPE |
|---|---|---|---|
| ALL | ALL | ALL | ALL |

z - SINGLE JOINT
+ - JOINT SET
o - BEDDING SET
O - BEDDING JOINT
x - FAULT
X - FAULT SET
# - FAULT CONTACT
< - CONTACT
S - SHEAR ZONE
B - BROKEN ZONE

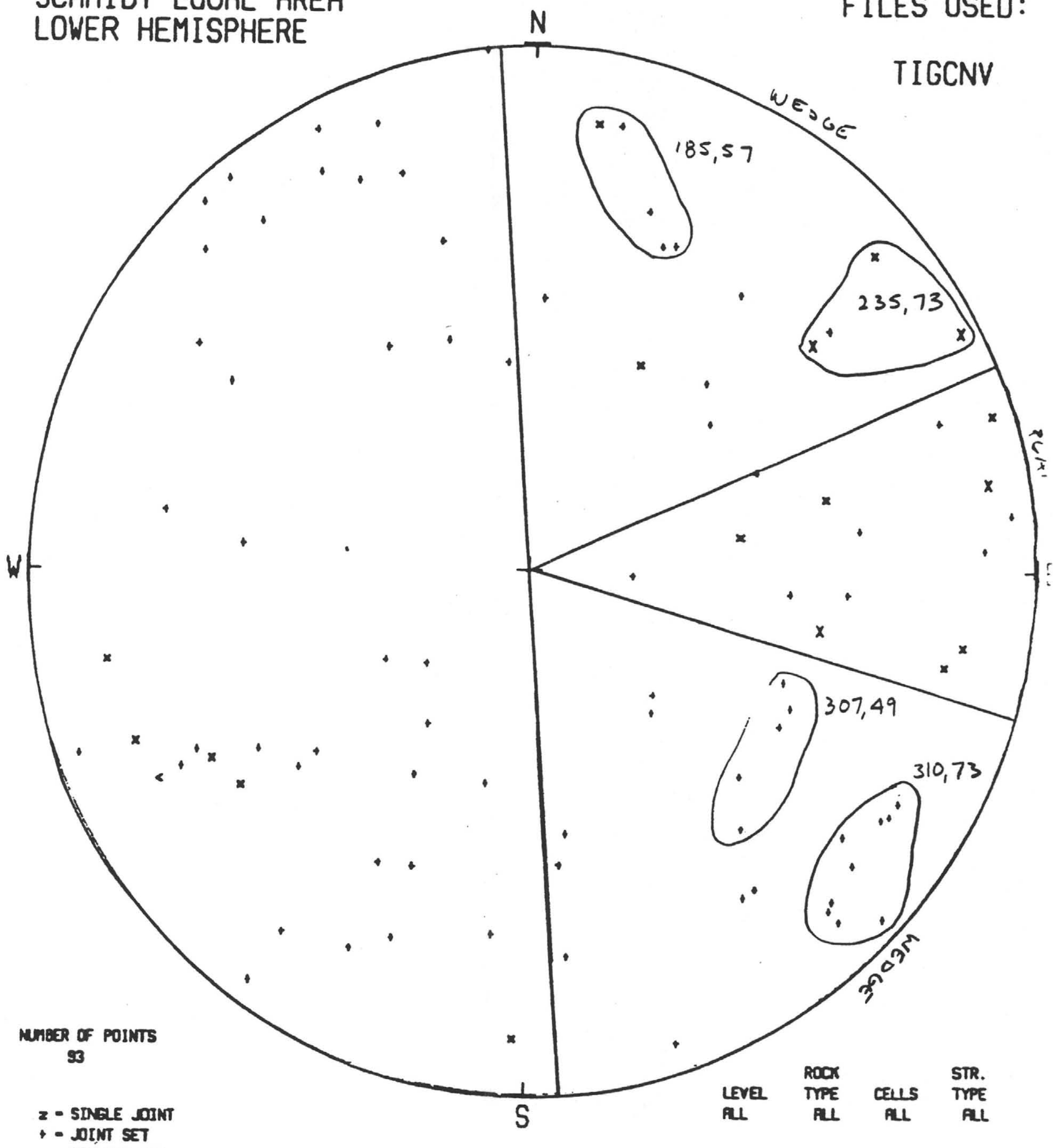
SECTOR B2

CALL & NICHOLAS, INC.

SCHMIDT EQUAL AREA
LOWER HEMISPHERE

FILES USED:

TIGCNV




NUMBER OF POINTS
93

z - SINGLE JOINT
+ - JOINT SET
o - BEDDING SET
O - BEDDING JOINT
x - FAULT
X - FAULT SET
# - FAULT CONTACT
< - CONTACT
S - SHEAR ZONE
B - BROKEN ZONE

| LEVEL | ROCK TYPE | CELLS | STR. TYPE |
|---|---|---|---|
| ALL | ALL | ALL | ALL |

SECTOR 176

CALL & NICHOLAS, INC.

## *APPENDIX C*

## *MEASURED BENCH FACE ANGLES IN FLUX PIT*

## TIGER FLUX PIT

### BENCH FACE ANGLE MEASUREMENTS

| LOCATION | ROCK TYPE | TAPE | BRUNTON |
|---|---|---|---|
| Cell #4 | Gr | | 74 |
| Cell #5 | Gr | | 69 |
| Cell #6 | Cb | | 68 |
| Cell #7 | Cb | | 68 |
| Cell #10 | Rh | | 78 |
| Cell #11 | Rh | 71° | 68 |
| | | | 78 |
| Cell #12 | Rh | 56° | 64 |
| | | | 70 |
| Cell #13 | Cb | 51° | 54 |
| | | | 58 |
| Cell #19 | Gr | 64° | 71 |
| Cell #20 | Gr | 65° | 74 |
| Cell #15 | Gr | | 67 |
| Cell #18 | Gr | | 70 |
| Cell #16 | Gr | | 65 |
| Cell #17 | Gr | | 62 |
| Top Bench | Gr | | 54,56 |
| | | | 60,62 |
| | | | 64,66 |

**Average Bench Face Angles**

Granite (Avg) = 65°
Rhyolite = 69°
Cloud Burst = 62°

*APPENDIX D*

*SCHMIDT HAMMER DATA*

# TIGER PIT

## SCHMIDT HAMMER RESULTS

| ROCK TYPE | LOCATION | REBOUND NUMBER |
|---|---|---|
| Rh | Cell #10 | 48, 31, 41, 48 |
| Rh | Cell #12 | 50, 45, 42, 48, 38, 30, 52 |
| Cb | Cell #13 | 45, 27, 40, 48, 45, 63 |
| Gr | Cell #15 | 30, 36, 35, 24, 24, 25 |
| Gr | Cell #16 | 30, 31, 33, 29, 39, 27, 25 |
| Gr | Cell #17 | 18, 29, 32, 28, 24, 30, 20 |
| Gr | Cell #18 | 20, 34, 28, 22, 24, 36, 24, 37 |
| Gr | Cell #19 | 36, 45, 36, 38, 40 |
| Gr | Cell #20 | 40, 42, 40, 36, 36 |

## UNCONFINED STRENGTH

| | | |
|---|---|---|
| Rh | L(Avg) = 42.7 | $\sigma u \approx$ 12,600 psi ± 4,000 |
| Cb | L(Avg) = 45.0 | $\sigma u \approx$ 13,300 psi ± 4,000 |
| Gr | L(Avg) = 32.0 | $\sigma u \approx$ 7,700 psi ± 3,000 |

CALL & NICHOLAS, INC.
3625 E. 42nd Strav.
Tucson, Arizona 85713 U.S.A.

(602) 745-8141
FAX: (602) 745-6674
Telex: 910 240-6828

FACSIMILE INFORMATION SHEET

TELECOPY CENTER
REC. ✓ SENT ____
MAY 05 1989
AM 7 8 9 10 11 12 1 2 3 4 5 6 PM

DATE: May 5, 1989

COMPANY NAME: Cyprus

ATTENTION: Howard Harlan

FACSIMILE NUMBER: 303-643-5049

FROM: Dave Nicholas

NO. OF PAGES (INCLUDING COVER SHEET) 3

Should any transmission or reception problems occur:
Call (602) 745-8141 or FAX (602) 745-6674

Howard —

Your cost of \$1.65/ton looks ok. Ben would add \$0.02 to 0.03 per ton for smooth wall bdasting. The total [redacted] material cost for meshing both pits is \$530,000. See attached sheets.

# COST ESTIMATE FOR TIGER BOLTS & MESH

## MAMMOTH PIT

| SECTOR | # BENCHES | LENGTH [A] | AREA |
|---|---|---|---|
| 132 A | 4 | 260' | 91,520 |
| 132 B | 8 | 380' | 267,520 |
| 176 A | 8 | 380' | 267,520 |
| 176 B | 4 | 380' | 133,760 |
| 293 A | 4 | 380' | 133,760 |
| 293 B | 8 | 380' | 267,520 |
| 348 A | 8 | 380' | 267,520 |
| 348 B | 5 | 360' | 158,400 |

1,587,500 $ft^2$ [B]

x \$0.23

**\$365,125.00**

## MOHAWK PIT

| | | | |
|---|---|---|---|
| 25, 105, 180, 285, | 6 | 1360' | 718,000 $ft^2$ [B] |

x \$0.23

**\$165,140.00**

NOTES:

A. LENGTHS BASED ON AVERAGE CIRCUMFERENCE LENGTH

B. AREAS BASED ON 70° BENCH FACE ANGLE, 24' WIDE CATCH BENCHES, 60' HIGH BENCHES.

$$\left( AREA = \left(24' + \frac{60'}{\sin(70°)}\right) * L * \#\,BENCHES \right.$$

## BOLT & MESH UNIT COST (MAT'LS ONLY)

Mechanical

A) BOLTS.

| | |
|---|---|
| \$ 1.99 | BOLT |
| \$ 0.66 | PLATE |
| \$ 2.65 | EACH |

ON A 4'x4' SPACING. $\$2.65/16\ ft^2 = \$0.17/ft^2$

B) MESH => 10 gage ≈ $\$.06/ft^2$

C) TOTAL MAT'L COST => \$ 0.23

APPENDIX III

**FEASIBILITY STUDY OF**
**<u>TIGER ORE GOLD HEAP LEACHING</u>**

**ML-1567**

SEPTEMBER 23, 1991

Dick Um

Dick Um
Sr. Metallurgist

cc: D. McGregor
A. Fernandez
S. Young
A. Liguori
M. Robinson
Met lab file

**FEASIBILITY STUDY OF**
**TIGER ORE GOLD HEAP LEACHING**

**ML-1567**

## PURPOSE

Preliminary feasibility study of Tiger ore gold heap leaching process by estimating capital and operating costs.

Leach solution flow rate of 1,000 gpm was studied for gold loaded carbon production from 2.5 million tons of ore reserve.

## CONCLUSION

Cost estimation results are shown in the following table:

| | |
|---|---|
| MINE LIFE ; YEAR | 3 |
| CAPITAL COST ; $MM | 8.1 |
| OPERATING COST ; $MM/A | 6.9 |
| OPERATING COST;$/TON ORE | 8.28 |
| REVENUE ; $MM/A | 12.2 |
| RATE OF RETURN BEFORE TAX; % | 46.3 |
| PRESENT WORTH(15) BEFORE TAX;$MM | 4.2 |

## RECOMMENDATIONS

Capital cost estimations were based on used equipment costs. Some equipment costs were approximated by using half the costs of new equipment. A search for used equipment should continue.

## DISCUSSION

A. DESIGN CRITERIA

| | | |
|---|---|---|
| Reserve | : | 2,500,000 tons |
| Grade | : | 0.065 oz Au/ton |
| Mine life | : | 3 years |
| Gold price | : | $375/oz on loaded carbon |
| Contract mining cost | : | $5.00<br>-6 inch ore loaded in crusher feed hopper, and ripping heap |

| | | |
|---|---|---|
| Heap Leaching | : | 120 days leaching<br>0.003 gpm/sq ft spray<br>15 ft each lift<br>Total 3 lifts<br>60 % gold extraction<br>1100 gpm leach solution flow rate |

## B. PROCESS DESCRIPTION

### CRUSHING AND AGGLOMERATION

Mine ore (-6 inch) is conveyed to an ore crushing plant, where two screens and two cone crushers are located. The ore is crushed to -3/8 inch, then conveyed to an agglomerator. Lime and cement are fed to the ore conveyor belt while ore is conveyed. Sodium cyanide solution is sprayed while ore is tumbled in the agglomerator. The agglomerator discharge is conveyed via stackers to heap the leaching pad.

### HEAP LEACHING

Gold concentration of PLS by single cycle leaching will be 0.5 ppm. Gold recovery plant is designed to process 1 ppm of gold feed by double cycle method. PLS solution is recycled back to heap leaching at 600 gpm, and 500 gpm of that is fed to the gold recovery plant. Leach solution spray rate is 0.003 gpm/sq ft.

### GOLD RECOVERY PROCESS

Gold PLS is pumped through carbon columns, where gold is loaded up to 200 oz/ton of carbon. The gold loaded carbon is sold directly without stripping.

## C. ECONOMIC ASSESSMENT

Salvage values are estimated at about half of new equipment purchase costs, and these values are included for cash flow analyses. Salvage values were estimated to be $1.5 MM.

Rate of return before tax is 46.3 %, and present worth of 15 % opportunity rate of return before tax is $4.2/MM.

## ANALYSIS

The following tables and figures for cost estimation studies are attached:

| | |
|---|---|
| Material flow balance flow diagram | : Figure 1 |
| Material balance flow data sheet | : Table 1 |
| Capital cost estimation sheet | : Table 2 |
| Operating cost estimation sheet | : Table 3 |
| Major equipment flow diagram | : Figure 2 |
| Major equipment list, size and cost | : Table 4 |
| Cash flow analysis | : Table 5 |

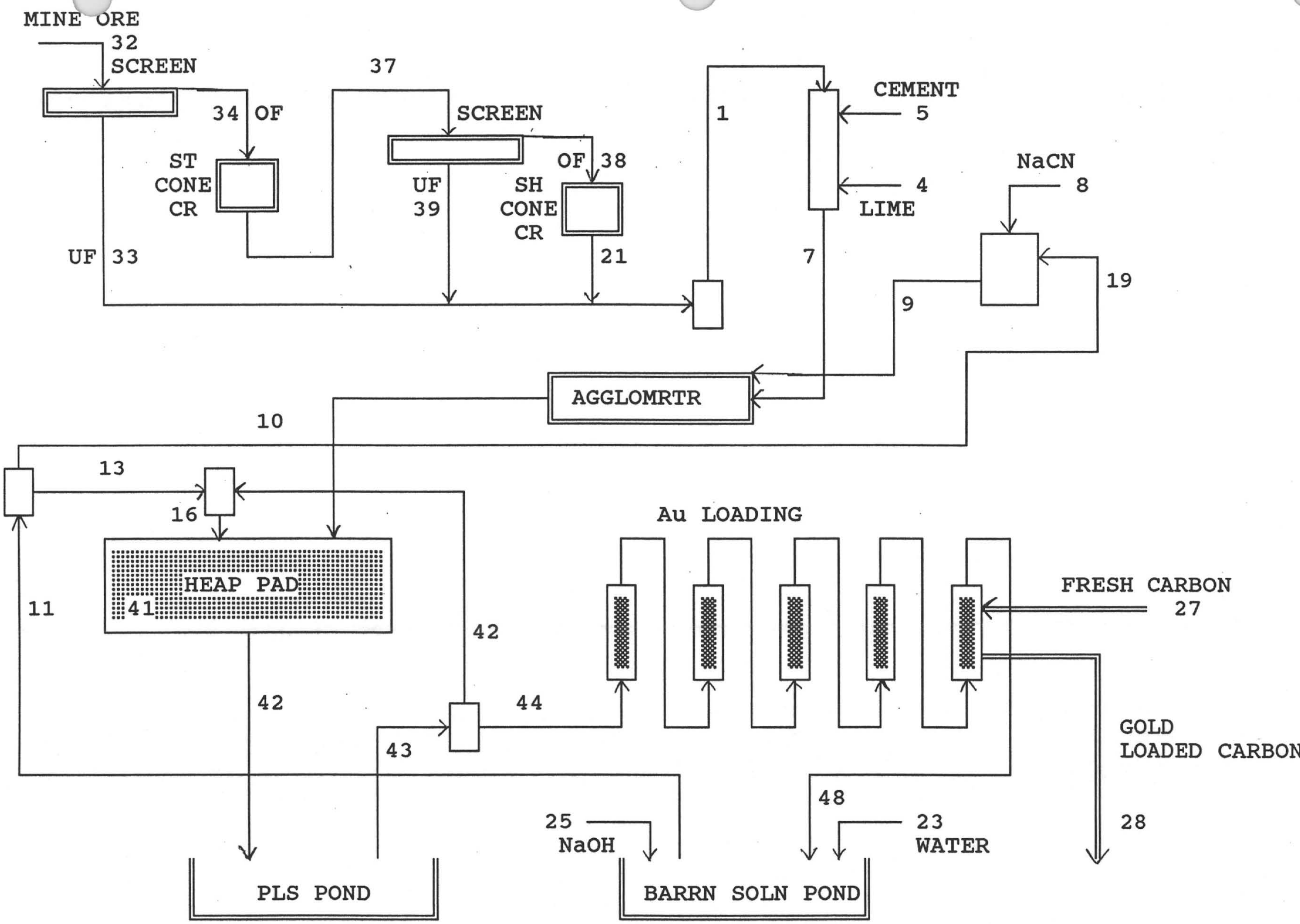



FIGURE 1. MASS BALANCE FLOW DIAGRAM

TABLE 1

MASS FLOW DATA

| NO | STREAM NAME | SOLID FLOW (STPD) | LIQUID FLOW (STPD) | TOTAL FLOW (GPM) | TOTAL FLOW (STPD) |
|---|---|---|---|---|---|
| 1 | ORE TO AGGLOMERATOR | 2,314.82000 | 20 | 149 | 2,335 |
| 4 | LIME | 1.28415 | 0 | 0 | 1 |
| 5 | CEMENT | 9.33928 | 0 | 1 | 9 |
| 7 | FEED TO AGGLOMTR | 2,325.44343 | 20 | 149 | 2,345 |
| 8 | NACN | 1.16741 | 0 | 0 | 1 |
| 9 | LIQ TO AGGLOMTR | 0.00000 | 235 | 39 | 235 |
| 10 | ORE TO HEAP | 2,325.45023 | 255 | 188 | 2,580 |
| 11 | BARREN SOLN TO HEAP | 0.00000 | 3,006 | 500 | 3,006 |
| 13 | BARREN TO HEAP | 0.00000 | 2,772 | 461 | 2,772 |
| 15 | MINE ORE | 2,314.82000 | 20 | 149 | 2,335 |
| 16 | SOLN TO HEAP | 1.24818 | 6,610 | 1,100 | 6,611 |
| 17 | EVAPORATION | 0.00000 | 0 | 71,040 | 343 |
| 19 | WATER TO NACN TK | 0.00000 | 233 | 39 | 233 |
| 21 | CRUSHER #2 DISCARGE | 771.55117 | 7 | 50 | 778 |
| 23 | WATER MAEKUP | 0.00000 | 323 | 54 | 323 |
| 25 | NaOH | 0.08756 | 0 | 0 | 0 |
| 27 | BARREN CARBON IN | 0.45140 | 0 | 0 | 0 |
| 28 | LOADED CARBON OUT | 0.45449 | 0 | 0 | 0 |
| 32 | MINE ORE | 2,314.82000 | 20 | 149 | 2,335 |
| 33 | SCREEN #1 UDS | 1,268.17056 | 11 | 81 | 1,279 |
| 34 | SCREEN #1 OVS | 1,046.64944 | 9 | 67 | 1,056 |
| 37 | CRUSHER #1 DISCHARGE | 1,046.64944 | 9 | 67 | 1,056 |
| 38 | SCREEN #1 OVS | 771.55117 | 7 | 50 | 778 |
| 39 | SCREEN #2 UDS | 275.09827 | 2 | 18 | 277 |
| 41 | ORE IN HEAP | 2,326.69532 | 0 | 146 | 2,327 |
| 42 | PLS TO HEAP | 0.00000 | 3,516 | 585 | 3,516 |
| 43 | PLS TO RECYCLE | 0.00000 | 6,522 | 1,085 | 6,522 |
| 44 | PLS TO CARBON COMN | 0.00000 | 3,006 | 500 | 3,006 |
| 48 | BARREN SOLN | 0.00000 | 3,006 | 500 | 3,006 |

TABLE 2

CAPITAL COST

| | |
|---|---|
| 000 SITE PREPARATION | 200,000 |
| 100 ORE PREPARATION PLANT | 1,527,450 |
| 200 LEACHING PLANT | 3,493,000 |
| 300 GOLD RECOVERY PLANT | 416,300 |
| | --------------- |
| TOTAL DIRECT COST | 5,636,750 |
| ENGINEERING MANAGEMT (25 %DC) | 1,409,188 |
| CONTINGENCY (15 %) | 1,056,891 |
| | --------------- |
| TOTAL CAPITAL COST | 8,102,828 |

TABLE 3

OPERATING COST ; TOTAL PLANT

| | UNIT COST | UNIT | | COST |
|---|---|---|---|---|
| RAW MATERIAL | | | | |
| SODIUM CYANIDE | $.79/# | 1 LB/TON | | 658,333 |
| PORTLANT CEMENT | $62/TON | 15 LB/TON | | 387,500 |
| SODIUM HYDROXIDE | $.351/LB | .075 #/TON ORE | | 21,938 |
| CARBON MAKEUP | $1.13/# | 163 TON | | 367,259 |
| DESCALE SOL | $1/# | 5 PPM | | 4,603 |
| CARBON BAG | $10/500# | 700 BAG | | 7,000 |
| | SUBTOTAL | | | 1,446,633 |
| UTILITIES | | | | |
| ELECTRIC | $55/MWH | 2120 MWH | | 116,600 |
| WATER | $0.40/K GAL | 29,000K | | 11,600 |
| LABOR | 16 LABOR | 30,000/LABOR | | 480,000 |
| MAINTENANCE | | | | 320,000 |
| OPERATING SUPPLIES | | | | 94,500 |
| GENERAL AND ADMINISTRATION | | | | 260,000 |
| OPERATING COST | | | | 2,729,333 |
| OPERATING COST PER TON | | | | 3.28 |
| CONTRACTOR COST | $5.00/TON | | 833,333 | 4,166,667 |
| TOTAL OPERATING COST | | | | 6,896,000 |
| TOTAL OPERATING COST PER TON | | | | 8.28 |

TABLE 3A

OPERATING COST : CRUSHING

| | UNIT COST | UNIT | COST |
|---|---|---|---|
| RAW MATERIAL | | | |
| SODIUM CYANIDE | $1.12/# | 0 | 0 |
| LIME | $100/T | 0 | 0 |
| PORTLANT CEMENT | $62/TON | 0 | 0 |
| SODIUM HYDROXIDE | $.351/LB | 0 | 0 |
| CARBON MAKEUP | $1.13/# | 0 | 0 |
| DESCALE SOL | $1/# | 0 | 0 |
| | SUBTOTAL | | 0 |
| UTILITIES | | | |
| ELECTRIC | $55/MWH | 1500 MWH | 82,500 |
| WATER | $0.40/K GAL | 0 | 0 |
| LABOR | 6 LABOR | 30,000/LABOR | 180,000 |
| MAINTENANCE | | | 250,000 |
| OPERATING SUPPLIES | | | 61,000 |
| GENERAL AND ADMINISTRATION | | | 126,000 |
| OPERATING COST | | | 699,500 |
| CONTRACTOR COST | $5.00/TON | | 0 |
| TOTAL OPERATING COST | | | 699,500 |
| TOTAL OPERATING COST PER TON | | | 0.84 |

TABLE 3B

OPERATING COST ; LEACHING

| | UNIT COST | UNIT | COST |
|---|---|---|---|
| RAW MATERIAL | | | |
| SODIUM CYANIDE | $.79/# | 1 LB/TON | 658,333 |
| PORTLANT CEMENT | $62/TON | 15 LB/TON | 387,500 |
| SODIUM HYDROXIDE | $.351/LB | .075 #/TON ORE | 21,938 |
| CARBON MAKEUP | $1.13/# | 0 | 0 |
| DESCALE SOL | $1/# | 5 PPM | 4,603 |
| | SUBTOTAL | | 1,072,374 |
| UTILITIES | | | |
| ELECTRIC | $55/MWH | 300 MWH | 16,500 |
| WATER | $0.40/K GAL | 28,000K | 11,200 |
| LABOR | 4 LABOR | 30,000/LABOR | 120,000 |
| MAINTENANCE | | | 30,000 |
| OPERATING SUPPLIES | | | 13,500 |
| GENERAL AND ADMINISTRATION | | | 54,000 |
| OPERATING COST | | | 1,317,574 |
| CONTRACTOR COST | $5.00/TON | | 0 |
| TOTAL OPERATING COST | | | 1,317,574 |
| TOTAL OPERATING COST PER TON | | | 1.58 |

TABLE 3C

OPERATING COST ; GOLD RECOVERY PLANT

| | UNIT COST | UNIT | | COST |
|---|---|---|---|---|
| RAW MATERIAL | | | | |
| SODIUM CYANIDE | $1.12/# | 0 | | 0 |
| LIME | $100/T | 0 | | 0 |
| PORTLANT CEMENT | $62/TON | 0 | | 0 |
| SODIUM HYDROXIDE | $.351/LB | 0 | | 0 |
| CARBON MAKEUP | $1.13/# | 163 TON | | 367,259 |
| DESCALE SOL | $1/# | 0 | | 0 |
| CARBON BAG | $10/500# | 700 BAG | | 7,000 |
| | SUBTOTAL | | | 374,259 |
| UTILITIES | | | | |
| ELECTRIC | $55/MWH | 320 MWH | | 17,600 |
| WATER | $0.40/K GAL | 1,000K | | 400 |
| LABOR | 6 LABOR | 30,000/LABOR | | 180,000 |
| MAINTENANCE | | | | 40,000 |
| OPERATING SUPPLIES | | | | 20,000 |
| GENERAL AND ADMINISTRATION | | | | 80,000 |
| OPERATING COST | | | | 712,259 |
| CONTRACTOR COST | $5.00/TON | | 833,333 | 0 |
| TOTAL OPERATING COST | | | | 712,259 |
| TOTAL OPERATING COST PER TON | | | | 0.85 |

APPENDIX IV

M A G M A   C O P P E R   C O M P A N Y

INTER-OFFICE CORRESPONDENCE

TO: AJ Fernandez

FROM: DA Deming

DATE: November 4, 1991

RE: Tiger Project Environmental Permits

Aquifer Protection Permit

Magma Copper Company-San Manuel Mining Division (Magma) currently has an Aquifer Protection Permit (APP) (#P-100421) applicable to the oxide copper operations (heap leach pad, solvent extraction-electrowinning circuit). As a Compliance Schedule Requirement at Part II.G.4 of the APP, Magma was to do the following:

> "The permittee shall submit necessary supplemental information such that a complete APP application for the entire San Manuel mining facility is filed at ADEQ, Water Permits Unit by December 1, 1992."

Magma's Environmental Affairs Department has gathered the remaining supplemental information for the remainder of the mine site and is currently finalizing the document for submittal to the Arizona Department of Environmental Quality (ADEQ). The intent was to also include the Tiger Gold facility discussion within the document, but this portion will now be deleted as the project has been put on hold. If and when the project is renewed, APP #P-100421 will require a "major modification" to the APP, requiring that the project design plans, process description, hydrological conditions description, monitoring plan, closure plan, etc. be submitted for ADEQ review. The information requested for an APP is shown on the form included as Attachment 1. Approximately one year is required for securing a permit, as shown on the previously prepared Gantt Chart included as Attachment 2.

The design of the Tiger Gold facility will be required to meet "Best Available Demonstrated Control Technology", or BADCAT, in order to obtain an APP major modification. Site specific conditions may be allowed to substitute for the specific BADCAT requirements. The BADCAT requirements for the Precious Metals Heap Leaching category are included as Attachment 3. The requirements typically call for synthetically lined leach pads, double lined with leak detection between the liners underneath pad areas where significant hydraulic head exists (i.e. solution collection ditches); double lined solution ponds with leak detection systems; surface water diversion ditches upstream of the facility; containment in ditches and ponds for process solutions plus the 100 year, 24-hour storm event; and monitoring the leak detection system plus groundwater wells. Upon commencement of the Tiger Project, Environmental

Affairs should be notified in order to prepare the APP application for submittal approximately one year in advance of leaching operations. The APP program currently allows construction of the facility in advance of obtaining a permit, however, leaching can not be conducted on a pad without having the permit in hand.

<u>NPDES Permit</u>
It is understood that current design plans would require no discharge from the facility to surface waters; therefore no NPDES Permit will be required.

<u>Air Quality Installation Permit and Operating Permit</u>
Before construction of a facility may proceed which might cause or control the emission of air pollutants, it may be necessary to obtain an installation permit from ADEQ. In the alternative, at least an operating permit would need to be obtained from the Pinal County Air Quality Control District. The time required to obtain either permit would be at least 90 days after receipt of a fully completed application.

<u>Well Construction Permits</u>
Applications must be filed with the Department of Water Resources for exploration boreholes, water supply wells and monitor wells associated with the project. The applications must be filed, and a drilling card issued by the Department, prior to drilling the well or borehole. Certain minimum well construction requirements apply to water supply wells and monitor wells.

<u>Dam Permit</u>
The Environmental Affairs group, after receiving a plan of operations for the facility, will determine whether or not a dam permit will be required. If this is the case, it is expected that obtaining such a permit would take less time than obtaining an APP for the facility.

<u>Section 404 Permit</u>
A US Army Corps of Engineers Section 404 (Dredge and Fill Permit) may be required if the affected area exceeds the threshold five acres of drainage. It is unlikely that the scope of the project is going to cause triggering of such a permit, but a facility plan will have to be analyzed prior to making this determination.

<u>Miscellaneous Permits (radios, hazardous waste, etc.)</u>
Miscellaneous permits such as those for radio operation and the handling of hazardous waste generated by the project can be taken care of under the existing permits for the mine facility. Environmental Affairs does not foresee any need to obtain special permits for these activities.

cc: AA Brodkey
ED Helmer
JC May

ATTACHMENT 1. ADEQ AQUIFER PROTECTION PERMIT APPLICATION FORM

Arizona Department of Environmental Quality

# AQUIFER PROTECTION PERMIT APPLICATION

(see guidance document for details)

## A. FACILITY DATA

1. NAME OF FACILITY

______________________________

______________________________

2. FACILITY CONTACT PERSON

______________________________

3. CONTACT PERSON'S MAILING ADDRESS

______________________________

______________________________

4. CITY, STATE, ZIP

______________________________

5. TELEPHONE NUMBER

______________________________

6. FACILITY OWNER

______________________________

7. OWNER'S ADDRESS

______________________________

8. CITY, STATE, ZIP

______________________________

9. FACILITY OWNER'S TELEPHONE NUMBER

______________________________

10. LAND OWNER

NAME ______________________________

ADDRESS ______________________________

PHONE ______________________________

11. FACILITY OPERATIONS START DATE

______________________________

B. <u>FACILITY</u> <u>LOCATION</u>

1. STREET, ROUTE NUMBER, OR OTHER SPECIFIC IDENTIFIER

______________________________

______________________________

2. COUNTY

______________________________

3. CITY or TOWN, STATE, ZIP

______________________________

______________________________

4. TOWNSHIP, RANGE, SECTION, 1/4, 1/4, 1/4

______________________________

5. LATITUDE ______________________________

6. LONGITUDE ______________________________

7. IF NATIVE AMERICAN LAND, NAME OF COMMUNITY

______________________________

C. <u>ENGINEER/CONSULTANT</u> (if applicable)

1. FIRM NAME

______________________________

2. CONTACT PERSON

______________________________

3. FIRM ADDRESS

______________________________

______________________________

4. CITY, STATE, ZIP

______________________________

______________________________

5. TELEPHONE NUMBER

______________________________

D. <u>OPERATIONS INFORMATION</u>

1. FACILITY MANAGER'S NAME

______________________________

2. FACILITY MANAGER'S TELEPHONE NUMBER

______________________________

3. OPERATOR'S NAME

______________________________

4. OPERATOR'S TELEPHONE NUMBER

______________________________

5. OPERATOR'S ADDRESS

______________________________

______________________________

6. OPERATOR'S CITY or TOWN, STATE, ZIP

______________________________

______________________________

E. <u>DESCRIPTION OF PROJECT</u> (include the expected life of facility)

______________________________

______________________________

______________________________

______________________________

______________________________

______________________________

F. EXISTING ENVIRONMENTAL PERMITS

1. NPDES NUMBER ________________________________
2. PSD NUMBER ________________________________
3. UST NUMBER ________________________________
4. RCRA NUMBER ________________________________
5. REUSE NUMBER ________________________________
6. ARMY CORPS OF ENGINEERS 404 NUMBER ______________
7. STANDARD INDUSTRIAL CLASSIFICATION (SIC) CODE
8. OTHER PERMITS (SPECIFY) ______________________

G. REQUIRED ATTACHMENTS

1. MAPS - TWO COPIES OF USGS 7.5 MINUTE QUADRANGLES. SHOW FACILITY LOCATION AND LAND USE WITHIN A 3 MILE RADIUS, ALL WELL LOCATIONS WITHIN 1/2 MILE RADIUS. PROVIDE WELL CONSTRUCTION DETAILS AND WELL USES.

2. SITE PLANS - TWO COPIES: PROPOSED AND/OR AS BUILT. SHOW CONFIGURATION OF BASINS, PONDS, WASTE STORAGE AREAS, DRAINAGE DIVERSION FEATURES, INJECTION WELLS, STRUCTURES, PROPERTY LINES, WATER WELLS, DRY WELLS, LOCATION POINT OF DISCHARGE. THE SITE PLAN MUST INCLUDE A DESCRIPTION OF THE LOCATION OF ALL KNOWN BORINGS

3. DESIGN PLANS - TWO COPIES SHOWING ALL ENGINEERED ELEMENTS OF THE FACILITY INCLUDING PROCESS FLOW DIAGRAMS AND CROSS SECTIONAL DIAGRAMS OF CONTROL STRUCTURES.

4. OPERATING PLAN - INCLUDE A CONSTRUCTION SCHEDULE AND A DESCRIPTION OF ALL RELEVANT OPERATING PRACTICES.

5. SUMMARY - OF EACH KNOWN PAST FACILITY DISCHARGE AND/OR THE PROPOSED FACILITY DISCHARGE INCLUDING:

   a. CHEMICAL, BIOLOGICAL, AND PHYSICAL CHARACTERISTICS OF THE DISCHARGE. INCLUDE LABORATORY REPORTS

   b. THE RATES, VOLUMES, AND FREQUENCY OF THE DISCHARGE FROM THE FACILITY.

6. BADCT DEMONSTRATION (THIS DOES NOT APPLY TO RECHARGE AND UNDERGROUND STORAGE & RECOVERY PROJECTS.) - A DESCRIPTION OF BADCT TO BE EMPLOYED AT THE FACILITY. DESCRIBE THE TECHNOLOGY EMPLOYED TO MEET THE REQUIREMENTS OF A.R.S. 49-243.B. DOCUMENT A DISCUSSION OF ALTERNATIVE DISCHARGE CONTROL MEASURES, THE TECHNICAL AND ECONOMIC ADVANTAGES AND DISADVANTAGES OF EACH ALTERNATIVE AND THE JUSTIFICATION FOR EACH SELECTION OR REJECTION OF EACH ALTERNATIVE. INCLUDE AN EVALUATION OF EACH ALTERNATIVE DISCHARGE CONTROL TECHNOLOGY IN RELATION TO DISCHARGE REDUCTION A CHIEVABLE, SITE SPECIFIC HYDROLOGICAL AND GEOLOGIC CHARACTERISTICS, ENVIRONMENTAL IMPACTS, AND WATER CONSERVATION OR WATER AUGMENTATION.

7. PROPOSED LOCATION OF EACH POINT OF COMPLIANCE

8. DEMONSTRATION - DESCRIBE WHY THE FACILITY WILL NOT CAUSE A VIOLATION OF AQUIFER WATER QUALITY STANDARDS AT THE POINT OF COMPLIANCE. ATTACH SUPPORT DATA.

9. TECHNICAL CAPABILITY - SUPPLY THE FOLLOWING INFORMATION.

   ~~a. ENGINEER/CONSULTANT ORGANIZATIONAL CHART~~

   b. PROFESSIONAL LICENSES OR CERTIFICATES HELD BY THE PERSON OR FIRM

   c. ANY PROFESSIONAL TRAINING RELATIVE TO THE DESIGN, CONSTRUCTION, OR OPERATION OF THE FACILITY.

   d. ANY WORK EXPERIENCE RELATIVE TO THE DESIGN CONSTRUCTION OR OPERATION OF THE FACILITY.

10. FINANCIAL CAPABILITY - SUPPLY THE FOLLOWING INFORMATION:

~~a. COMPANY ORGANIZATIONAL CHART~~

b. AN ESTIMATE OF THE TOTAL COST OF CONSTRUCTION, OPERATIONS, CLOSURE AND ASSURANCE OF PROPER POST CLOSURE CARE.

c. A STATEMENT BY THE CHIEF FINANCIAL OFFICER OF THE PROJECT THAT THE APPLICANT IS FINANCIALLY CAPABLE OF MEETING THE COSTS AS DESCRIBED IN PART E. THE STATEMENT MUST ADDRESS THE FINANCIAL ARRANGEMENTS FOR MEETING THE CLOSURE AND POST-CLOSURE PLANS.

d. FOR A PERSON OR FIRM OTHER THAN A STATE OR FEDERAL AGENCY OR A COUNTY, CITY, TOWN OR OTHER LOCAL GOVERNMENTAL ENTITY, THE DEMONSTRATION OF FINANCIAL CAPABILITY SHALL BE FURTHER SUPPORTED BY ANY ONE OF THE FOLLOWING:

1. THE MOST RECENT COPY OF THE PERSON'S 10K FORM FILED PURSUANT TO SECTION 13 OR 15 (D) OF THE FEDERAL SECURITIES AND EXCHANGE ACT OF 1934.

2. A REPORT THAT CONTAINS ALL OF THE FOLLOWING INFORMATION.

   a. A DESCRIPTION OF THE PERSON'S STATUS AS A CORPORATION, PARTNERSHIP OR OTHER LEGAL ENTITY.

   b. A DESCRIPTION OF THE PERSON'S BUSINESS

   c. AN INDICATION OF THE PERSON'S NET WORTH, INCLUDING A DESCRIPTION OF MAJOR ASSETS AND LIABILITIES.

   d. A BRIEF DESCRIPTION OF ANY JUDGEMENT EXCEEDING $100,000 RENDERED AGAINST THE PERSON DURING THE FIVE YEARS PRECEDING THE DATE OF THE APPLICATION.

e. A BRIEF DESCRIPTION OF ANY BANKRUPTCY OR INSOLVENCY PROCEEDINGS INSTITUTED BY THE PERSON DURING THE FIVE YEARS PRECEDING THE DATE OF THE APPLICATION.

f. IF THE PERSON IS A CORPORATION, THE NAMES OF ITS EXECUTIVE OFFICERS AND THEIR DATES OF BIRTH.

3. g. EVIDENCE OF A BOND, INSURANCE, OR A TRUST FUND ASSURING THAT THE APPLICANT WILL BE FINANCIALLY CAPABLE OF MEETING THE CLOSURE AND POST-CLOSURE REQUIREMENTS OF THE INDIVIDUAL AQUIFER PROTECTION PERMIT.

4 h. A BRIEF DESCRIPTION OF ANY ACTION FOR THE ENFORCEMENT OF FEDERAL, STATE LAW, RULE OR REGULATION; OR ANY COUNTY, CITY OR LOCAL GOVERNMENT ORDINANCE RELATING TO THE PROTECTION OF THE ENVIRONMENT, INSTITUTED AGAINST THE PERSON DURING THE FIVE YEARS PRECEDING THE DATE OF APPLICATION.

Separate section

11. A HYDROGEOLOGIC STUDY WHICH MAY INCLUDE: (Please see the application guidance document or contact the Water Permits Unit's staff. Requirements vary depending on facility design and site-specific characteristics of the location)

a. DESCRIPTION OF THE SURFACE AND SUBSURFACE GEOLOGY

b. THE LOCATION OF ANY PERENNIAL OR EPHEMERAL SURFACE WATER BODIES.

c. THE CHARACTERISTICS OF THE AQUIFER AND GEOLOGIC UNITS WITH LIMITED PERMEABILITY, INCLUDING DEPTH, HYDRAULIC CONDUCTIVITY AND TRANSMISSIVITY.

d. RATES, VOLUMES AND DIRECTIONS OF SURFACE WATER AND GROUND WATER FLOW, INCLUDING HYDROGRAPHS, IF AVAILABLE AND EQUI-POTENTIAL MAPS.

e. THE LOCATION OF THE 100-YEAR FLOOD PLAIN AND AN ASSESSMENT OF THE 100-YEAR FLOOD SURFACE FLOW AND POTENTIAL IMPACTS ON THE FACILITY.

f. A DOCUMENTATION OF THE EXISTING QUALITY OF THE WATER IN THE AQUIFERS UNDERLYING THE SITE, INCLUDING, WHERE AVAILABLE, THE METHOD OF ANALYSIS AND QUALITY ASSURANCE AND QUALITY CONTROL PROCEDURES ASSOCIATED WITH THE DOCUMENTATION.

g. DESCRIBE WATERSHED CHARACTERISTICS. INCLUDE SIZE, VEGETATION, SLOPE, SOIL TYPE,ETC. A DOCUMENTATION OF THE EXTENT AND DEGREE OF ANY KNOWN SOIL CONTAMINATION IN THE VICINITY OF THE FACILITY.

h. AN ASSESSMENT OF THE POTENTIAL OF THE DISCHARGE TO CAUSE THE LEACHING OF POLLUTANTS FROM SURFACE SOILS OR VADOSE MATERIALS.

i. ANY ANTICIPATED CHANGES IN THE WATER QUALITY EXPECTED AS A RESULT OF THE DISCHARGE.

j. A DESCRIPTION OF ANY EXPECTED CHANGES IN THE ELEVATION AND FLOW DIRECTIONS OF THE GROUNDWATER THAT MAY BE CAUSED BY THE FACILITY.

k. MAP OF THE FACILITY'S DISCHARGE IMPACT AREA.

l. THE CRITERIA AND METHODOLOGIES USED TO DETERMINE THE DISCHARGE IMPACT AREA.

12. A DETAILED PROPOSAL INDICATING THE ALERT LEVELS DISCHARGE LIMITATIONS, MONITORING REQUIREMENTS, CONTINGENCY PLANS, COMPLIANCE SCHEDULES AND TEMPORARY CLOSURE, CLOSURE AND POST-CLOSURE PLANS WHICH THE APPLICANT PROPOSES TO SATISFY THE REQUIREMENTS OF A.R.S. 49, CHAPTER 2, ARTICLE 3, AND A.C.C. R-18-9-101 THRU 130.

a. ALERT LEVELS MAY BE BASED UPON SITE-SPECIFIC CONDITIONS DESCRIBED BY THE APPLICANT OR MAY BE BASED UPON A POLLUTANT WHICH INDICATES THE POTENTIAL APPEARANCE OF ANOTHER POLLUTANT OR MAY BE PRESCRIBED TO BE MEASURED AT THE POINT OF RELEASE, THE POINT OF COMPLIANCE OR ANY INTERVENING POINT.

b. AN APPROPRIATE CONTINGENCY PLAN IF AN ALERT LEVEL IS EXCEEDED

c. DISCHARGE LIMITATION BASED UPON THE CONSIDERATIONS DESCRIBED IN ARS 49-243 A, B, C, D, AND APPROPRIATE CONTINGENCY PLAN IF DISCHARGE LIMITATION IS EXCEEDED.

13. <u>MONITORING PLAN</u>

a. TYPE AND METHOD OF MONITORING TO BE CONDUCTED.

b. FREQUENCY OF MONITORING.

c. ANY REQUIREMENTS FOR THE INSTALLATION, USE OR MAINTENANCE OF MONITORING EQUIPMENT, AND REPORTING INTERVALS. MONITORING RECORD AS PRESCRIBED BY THE DEPARTMENT.

d. MONITORING, RECORD-KEEPING, AND PROCEDURES WILL BE SUBMITTED ON THE FORM PROVIDED BY THE DEPARTMENT AND WILL INCLUDE THE FOLLOWING:

1. THE DATE, TIME, AND EXACT PLACE OF SAMPLING OR MEASUREMENT AND THE NAME OF EACH INDIVIDUAL WHO PERFORMED THE SAMPLING OR MEASURING.

2. METHODOLOGY AND PROCEDURES USED TO COLLECT THE SAMPLE OR MAKE THE MEASUREMENT.

a. THE DATE ON WHICH THE SAMPLE ANALYSIS WAS COMPLETED.

b. THE NAME OF EACH INDIVIDUAL OR LABORATORY WHO PERFORMED THE ANALYSIS.

c. THE ANALYTICAL TECHNIQUES OR METHODS USED TO PERFORM THE SAMPLING OR ANALYSIS.

d. THE CHAIN OF CUSTODY RECORDS.

e. ANY FIELD NOTES.

3. ANY DEVIATION FROM THE SAMPLING/MONITORING PLAN

14. <u>ZONING ORDINANCE</u> - EVIDENCE THAT THE FACILITY COMPLIES WITH APPLICABLE MUNICIPAL OR COUNTY ZONING ORDINANCES AND REGULATIONS.

G. CERTIFICATION

I certify under penalty of law that I have personally examined and am familiar with the information submitted in this application and all attachments and that based on my inquiry of those persons immediately responsible for obtaining the information contained in the application, I believe that the information is true, accurate and complete. I am aware that there are significant penalties for submitting false information, including the possibility of fine and imprisonment.

NAME AND OFFICIAL TITLE

______________________________________________

SIGNATURE

______________________________________________

DATE SIGNED

______________________________________________

TABLE 1

FEE SCHEDULE

Each individual Aquifer Protection Permit must be accompanied by a non-refundable fee. Make checks payable to the State of Arizona.

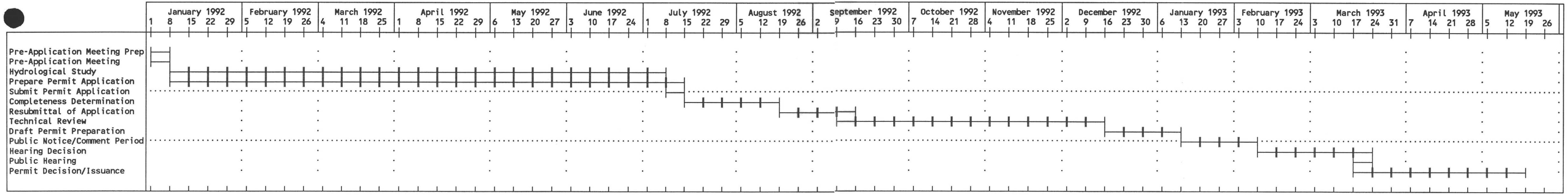
| Categories | Fee (In U.S. Dollars) |
|---|---|
| On-Site Sewage Disposal Systems (less than 20,000 gpd) | $1,200 |
| Wastewater Treatment Plants Where Influent is Predominantly Sewage | |
| Surface Impoundment | $1,400 |
| Discharge to Water of the U.S | $1,600 |
| Subsurface Discharge | $1,400 |
| Recharge and Underground Storage and Recovery Without Effluent | $2,200 |
| Recharge and Underground Storage and Recovery Using Effluent | $2,800 |
| Solid Waste Disposal Facility (Landfills) | $2,200 |
| Construction Debris Landfills | |
| Mines | |
| Surface Impoundments | $1,800 |
| Tailings Piles or Ponds | $2,200 |
| Base Metal Leaching Operations Including Collection and Process Ponds | $2,300 |
| Cyanide Leaching Including Collection and Process Ponds | $1,500 |
| In-Situ Leaching | $3,400 |
| Discharge to Water of U.S | $1,900 |
| Dry Wells | $ 900 |
| Industrial Wastewater Discharges | |
| Surface Impoundment | $2,200 |
| Discharge to Water of U.S | $1,700 |
| Subsurface Discharge | $1,900 |
| Other Discharging Facilities | $1,800 |
| Permit Transfer | $ 200 |

| | |
|---|---|
| Permit Modification that constitutes a major modification as described A.R.S. 49-201.18 | Same as for original permit in application according to type of facility |
| Permit Modification that is described as a minor modification under R18-9-121.D | 0 |
| Permit modification that is neither a major modification nor a minor modification | $ 200 |

ATTACHMENT 2. AQUIFER PROTECTION PERMIT APPLICATION GANTT CHART

November 12, 1991

Tiger Project Aquifer Protection Permit Schedule



## ATTACHMENT 3. BADCAT GUIDANCE DOCUMENT FOR THE MINING CATEGORY; PRECIOUS METALS HEAP LEACHING SECTION

[illegible] copy

# BADCT GUIDANCE DOCUMENT
# FOR THE
# MINING CATEGORY

However, vegetation established either directly in tailings or on a cover of natural materials can be considered as BADCT to minimize erosion from dam faces and the subsequent potential for impacts on groundwater. The need for irrigation to establish the vegetation must also be considered as it effects the opportunity for water conservation. Recontouring the slopes and/or the construction of benches and check dams can also aid in the control of erosion and sediment transport. Continued operation and maintenance of leachate collection systems are also important aspects of a closed facility.

## 3. Precious Metals Heap Leaching

### a. Facility Description

In the process of precious metals heap leaching, ore is crushed and often agglomerated to bind the fines and minimize channeling before it is placed on a leach pad. Water is usually mixed with caustic soda and sodium cyanide keeping the pH at 10 or higher to keep the formation of hydrogen cyanide gas to a minimum. The solution is then applied or sprayed onto the heap and pregnant solution is collected as it flows down the sloped lined pad base into a collection system of ditches or trenches feeding into a lined pond. The precious metals are stripped from the pregnant solution as it is circulated through beds of activated carbon. The barren solution which flows out of the carbon processing plant is normally held in a lined pond before being brought back to the approprate cyanide strength and recirculated back to the heap (Figure 4). Integrated facilities which have a tailings pond on site, or those with sufficient excess capacity in their pregnant pond, may not employ a barren solution pond.

Leaching may be conducted on single use ("dedicated") or multiple use ("restackable") pads. On a dedicated pad, spent ore is left in place upon closure. A restackable pad is loaded, leached, and the spent ore ("spoil") is rinsed and removed from the pad for disposal elsewhere. The pad can then be used for further leaching. The decision on which type of pad to use is based on a number of parameters including ore mineralogy and available terrain. Ores which can be leached rapidly may be leached on restackable pads. These pads are also utilized where available space for new leaching pads is at a minimum. In most other cases, the use of dedicated pads usually has a distinct economic advantage.

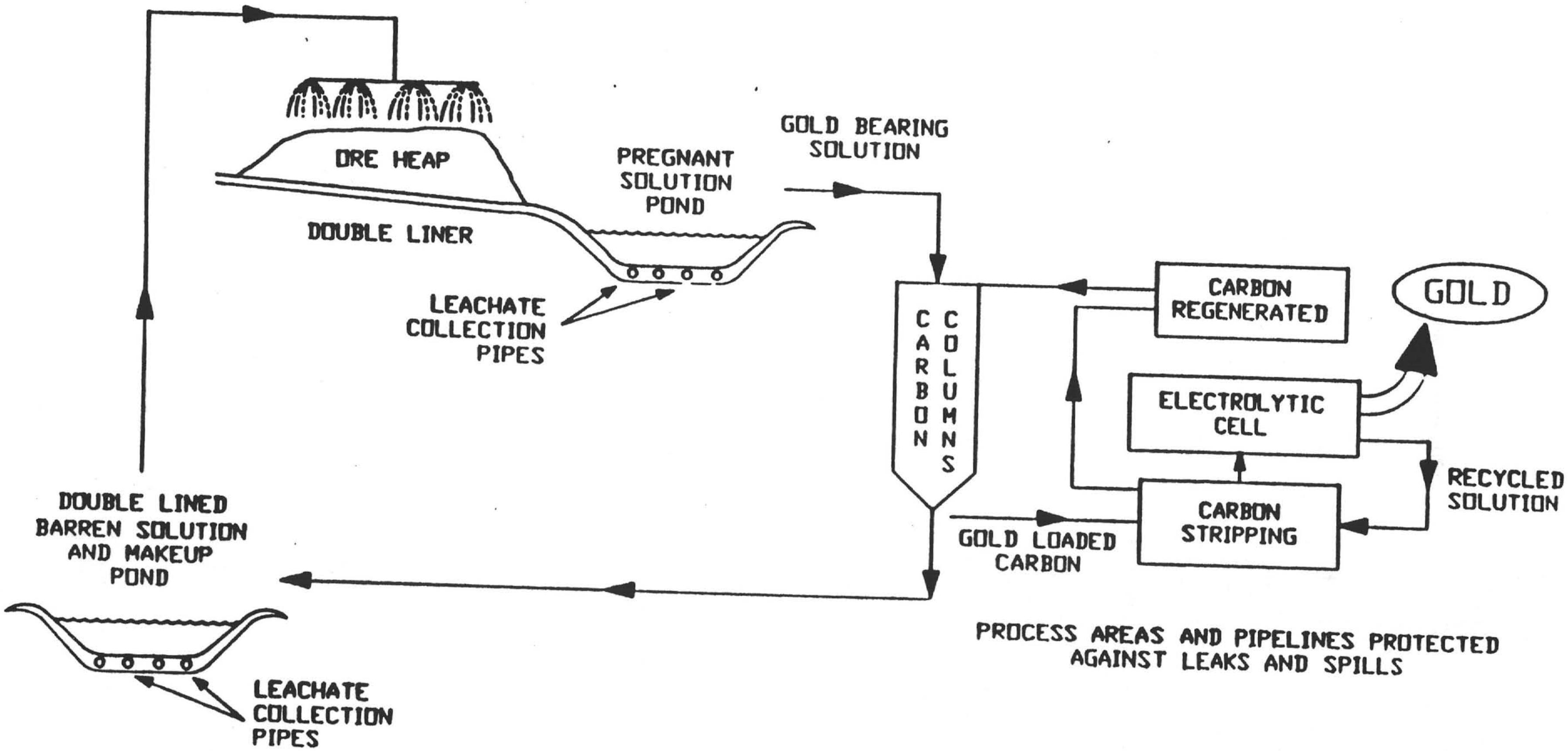



FIGURE 4: CYANIDE HEAP LEACH FLOW DIAGRAM

The key BADCT components in precious metals leaching operations are those design elements, operating practices and closure measures which eliminate the potential for significant discharge to an aquifer. These include liners and leachate collection systems for leach pads, ditches and solution ponds, surface water controls contouring and covering. Two items must be noted in regard to BADCT for precious metals leaching facilities. First, while it is commonplace to categorize these facilities as having "zero discharge" to groundwater, it must be recognized that all materials do in fact leak, and that the concept of a totally non-leaking facility is technically infeasible. These facilities can, however, be constructed such that they present no significant potential to discharge in a manner which could adversely impact aquifer quality. Secondly, in determining BADCT for this segment of the mining industry, cyanide compounds are not considered to be organic chemicals subject to the requirements of A.R.S. 49-243.D.

b. Design, Construction and Operation of Leach Pads

The technologies described in this section are generally only feasible for new leaching facilities. Those technologies also applicable to existing facilities are so indicated.

1) Site Preparation

Clearing and grubbing is generally necessary in preparation for installation of a synthetic liner. Compaction of the surface serves to inhibit discharge of leaching solution and to provide a firm, smooth subgrade on which to install the liner and construct the heap. The extent of surface preparation required is dependent on the characteristics of the liner (including whether the pad is dedicated or restackable), the nature of the overliner material, and the weight of ore which will be placed on the heap. The surface can then be treated with a biocide to eliminate plant growth which could adversely affect liner performance.

2) Liners

i. Design: In most cases, an overliner or drainage blanket must be place on top of the liner of a dedicated pad in order to protect it from punctures and to promote flow of pregnant solution to the collection ditch. Specific design elements are dependent on the conditions existing at the site. The topography of the available leach site generally determines the pad design. In relatively flat areas, pads can be designed to drain to a single collection ditch external to the heap. In such cases, internal berms can be constructed to segment the pad so that solution flows to the ditch as directly as possible. Perforated piping can also be installed within the drainage blanket to further promote flow to the collection ditches.

In mountainous or rolling terrain, valley-fill or modified valley-fill pads can be constructed. Valley-fill pads take advantage of existing topography; rather than constructing the pad as a "tabletop" sloped to one corner, the pad follows the contours of the natural ground surface and the pregnant solution is collected in internal ditches which are built in the natural drainages. The lay of the land and the existing natural gradients can eliminate the need for internal berms and piping. The valley-fill design also uses the pad as the pregnant pond. The downgradient end of the pad is constructed against a berm which functions as a dam. The pregnant solution is collected and stored within the heap, and is either extracted by a pipe through the liner and berm, or is pumped out along the upstream face of the berm. In this design, that portion of the pad which functions as an impoundment must be constructed with the same technology as a pond (See Section IV.E.). The valley-fill design is only feasible where the ore will not degrade the cyanide holding the gold or otherwise "rob" the pregnant solution. In the modified valley-fill design, the pad is similarly constructed, but the pregnant solution is stored in an external pond.

ii. Materials: Pad liners may be constructed from natural or synthetic materials. The type and thickness of a liner should be determined to maximize liner integrity based on consideration of the loading weight of the heap, the puncture properties of the subgrade and the resistance of the liner to chemical and ultraviolet degradation. For restackable pads, the liner must be constructed to withstand the stress of repeated vehicle traffic as the pad is loaded and unloaded. These types of pads are commonly constructed in layers using asphalt and rubberized membranes, and may be 6 inches or more in thickness.

The angularity of the material used as overliner, and the manner in which the pad will be loaded are also factors in determining the liner to be used. The overliner material must be sufficiently permeable to readily transport the pregnant solution with minimal head build-up, and must also be subangular to rounded so as not to risk puncturing the liner during loading of the pad. In some cases, run of mine or crushed and screened ore can provide a suitable drainage blanket.

iii. Quality Control and Quality Assurance: The effectiveness of any liner system can be increased by a program of quality control and quality assurance so that the liner functions as it was designed. Specifications and procedures for parameters such as the density testing of soil liners, and such activities as seaming of synthetic liners, should be determined, and a program should be established to monitor and document these activities and parameters during construction.

3) Operations of Restackable Pads: The operation of restackable pads involves a special BADCT consideration because spent leach ore is removed from these pads and disposed of prior to closure. Depending on the potential of the spoils to release residual cyanide, and their method of disposal, the spoils may require rinsing before they can be removed from the pad. In cases where the spoils will be placed within a lined facility such as a tailings pond, no rinsing or further pollutant removal is needed. However, in cases where the spent ore is to be disposed of on unlined ground, it will be necessary to rinse or otherwise detoxify the waste in a manner similar to that described below for closure of dedicated pads.

4) Leak Detection and Collection: Leak detection and collection systems for precious metals heap leach pads have only been demonstrated for dedicated pads. These systems generally focus on those areas of the pad upon which a significant hydraulic head is exerted. Since most pads are designed to promote the rapid flow of pregnant solution to the collection ditches, leak detection and collection systems are normally limited to these portions of the facility. The most sophisticated designs have been used on modified valley-fill pads where the internal collection ditches can neither be visually inspected nor easily repaired. These designs employ a second synthetic membrane beneath the primary liner under the solution collection ditches. Placed between these two liners is a drainage layer of sand or some other pervious material, and corrugated perforated piping is placed within the drainage layer. The system is arranged so that the operator is able to sample any solution found and to quantify the amount of flow prior to routing the solution back to the circuit.

   Less elaborate systems may be appropriate in cases where the collection ditches are external to the pad and allow for visual inspection and repair. Perforated piping can be installed beneath the ditch's primary liner and the compacted subgrade, and routed so that sampling and quantification of any flow is possible prior to routing the solution back to the circuit.

5) Surface Water Control: Control of surface water run-on is generally applicable to both new and existing precious metals heap leach operations. Design considerations are influenced by precipitation (intensity, duration, distribution), water shed characteristics (size, shape, topography, geology, vegetation), runoff (peak rate, volume, time distribution), and the degree of protection

warranted. Berms or ditches should be constructed capable of protecting a leach pad from the 100-year, 24-hour storm event.

c. Design, Construction and Operation of Solution Ponds and Ditches

The technologies presented in this section are generally appropriate for new facilities. However, it may be feasible at some existing sites to employ some of these controls depending on the amount of discharge reduction which could be achieved.

1) Liners

i. Design: A system consisting of two liners and a leak detection and collection system is normally considered to represent BADCT for precious metals pregnant and barren solution ponds. The pond must be of sufficient size to contain the operating volume of solution and the run-on and direct precipitation resulting from the 100-year, 24-hour storm event. For pregnant solution ponds, the area where the collection ditch enters the pond may be subject to extra stress, and energy dissipation measures or reinforcement may be necessary.

ii. Materials: Both liners used for these ponds are normally constructed of synthetic materials. Where site conditions allow, it may be possible to substitute a liner of natural materials for the secondary synthetic liner. The primary liner must be selected to be resistant to ultraviolet light.

iii. Quality Control and Quality Assurance: Quality control and quality assurance considerations for ponds and ditch liners are the same as for pad liners, but are more critical because hydraulic head is exerted on these components at all times.

2) Leak Detection and Collection: New ponds should be designed with leak detection and collection systems. These systems normally consist of a pervious layer installed between the liners, with any seepage collected in a manner which allows sampling and quantification of the flow. The drainage layer may consist of sand, fine gravel, geonet or other similar material. The system should be designed to maximize the volume of leakage which can be withdrawn so that hydraulic head is not transferred to the secondary liner.

3) Surface Water Controls: Ponds and ditches should be protected by berms, dikes or other diversion features capable of withstanding the 100-year, 24-hour storm event.

d. Closure of Precious Metals Leach Pads, Ponds and Ditches

In addition to protecting groundwater quality, the objective in closure of cyanide facilities should incorporate protection of the public from future exposure to this toxic substance. The following closure technologies are applicable to both new and existing precious metals leach pads, ponds and ditches.

1) Leach Pads: Upon closure of a dedicated leach pad, the spent ore must be left in a condition which will not result in a discharge with the potential to cause an exceedance of aquifer quality standards. In many cases, the potential for impacts to groundwater may be mitigated by the intact liner beneath the heap.

   The environment in which the heap is located, and the nature of the waste must also be assessed to determine approprate closure measures. The potential of a closed heap to discharge fluid depends in part on the amount and distribution of precipitation and evaporation, the proximity and pathways to surface waters, the depth to groundwater, and the nature of the subsurface lithology. The moisture retaining capacity of the spoils themselves may be factored in to assess the potential for any seepage from the closed heap.

   If a potential for seepage exists, the chemical nature of the potential seepage becomes an item of concern. Residual cyanide concentrations within the heap are normally reduced by rinsing with water or rinsing with water followed by a hypochlorite solution. In this manner, most facilities are able to achieve free cyanide concentrations below 0.2 mg/l in the rinsate.

   As noted above, rinsing prior to unloading the pad is important for restackable pads, particularly when their disposal will be in an unlined facility. It should also be noted that the process of unloading the pad will result in further degradation of cyanide. Physical agitation of the material will break down some of the more weakly held cyanide complexes, and the exposure of the waste to the air and its contained carbon dioxide will reduce pH and result in volatilization of hydrogen cyanide. Prior to land surface disposal, representative samples of the material must first be analyzed to confirm that soil cleanup levels specifed by ADEQ are met.

   Where sediment loading to surface water and subsequent impacts on groundwater are of concern, recontouring of the heap or construction and maintenance of berms may be necessary to limit erosion.

2) Ponds and Ditches: Solutions remaining in ponds after cessation of operations and rinsing of the heap can be allowed to evaporate. Pond liners may then be folded over

upon themselves, thereby encapsulating any solids which are left after evaporation of the solutions. The pond can then be backfilled to avoid future ponding and reduce the potential for any leaching from the liner or the solids. Ditches may be closed in a similar manner. Alternately, liners may be removed and disposed of in accordance with applicable solid waste regulations.

## 4. Precious Metals Vat Leaching

In the vat leaching process, after crushing and grinding, the finely pulverized ore enters a closed [illegible] of cyanide leaching and carbon in pulp (CIP) absorption (Figure 5). The spent ore is carried via pipeline to a tailings disposal area. The optimum technology invol[illegible] the system, including the use [illegible] disposal area wi[illegible] the tailings. If the cont[illegible] ions and water [illegible] is reduced before the tailings are sent to the tailings disposal area, the potential for affecting groundwater quality will be reduced even further.

Many of the [illegible] associated with vat leaching have been discussed ear[illegible] Chemical [illegible] age, Processing Areas, Pipelines, and General Closure Considerations sections of this document. (Sections IV.D.8. and 9.).