PART I

INTRODUCTION

DESCRIPTION OF OPERATIONS

THE 16-TO-1 MINE AND MILLING OPERATIONS NEAR SILVER PEAK, NEVADA HAS BECOME NOT ONLY SUNSHINE MINING COMPANY'S FIRST NEW GOLD AND SILVER PRODUCING FACILITY, BUT ALSO REPRESENTS A DRAMATIC DEPARTURE FROM THE MINING AND MILLING PRACTICES LONG EMPLOYED AT THE SUNSHINE MINE IN NORTH IDAHO. SEEKING TO FURTHER STRENGTHEN ITS POSITION AS ONE OF THE NATION'S LEADING SILVER PRODUCERS AND PROVIDE A SECOND SOURCE OF INCOME, SUNSHINE'S BOARD OF DIRECTORS MADE THE DECISION IN JUNE OF 1980 TO PROCEED WITH DEVELOPMENT OF THIS PROPERTY. UNDERGROUND MINE DEVOLOPMENT BEGAN IN SEPTEMBER OF THAT YEAR AND, FOLLOWING COMPLETION OF ALMOST 16,000 FEET OF DRIFTING AND CROSSCUTTING TOGETHER WITH A 550 TONS PER DAY MILL, THE FIRST BULLION WAS POURED IN MARCH OF 1982. THE SILVER PEAK OPERATIONS CONSISTS OF A CONVENTIONAL CYANIDE MILLING FACILITY AND AN UNDERGROUND MINE LOCATED 3½ MILES FROM THE MILL IN THE SILVER PEAK MOUNTAIN RANGE IN CENTRAL ESMERALDA COUNTY. PRESENT ANNUAL OUTPUT IS APPROXIMATELY ONE MILLION OUNCES OF SILVER AND 8500 OUNCES OF GOLD FROM 200,000 TONS OF ORE WITH A TOTAL EMPLOYMENT OF ABOUT 85 PERSONS.

PART II

BACKGROUND DATA

GEOGRAPHY

THE 16-TO-1 MILL AND MAIN OFFICE IS LOCATED 5½ MILES SOUTHWEST OF THE TOWN OF SILVER PEAK IN CENTRAL ESMERALDA COUNTY, NEVADA (FIGURE 1). AT AN ELEVATION OF 5200 FEET ABOVE SEA LEVEL, THE MILLSITE IS SITUATED IN AN AREA OF RELATIVELY EVEN, GENTLY SLOPING TERRAIN NEAR THE EDGE OF THE EASTERN FOOTHILLS OF THE SILVER PEAK MOUNTAIN RANGE, JUST ABOVE THE BROAD ALLUVIAL FANS WHICH LEAD TO THE FLOOR OF CLAYTON VALLEY. PRESENT ACCESS TO THE FACILITY IS BY WAY OF A GRAVEL ROAD ORIGINATING FROM SILVER PEAK. THIS ROAD CONTINUES PAST THE MILLSITE ANOTHER 3½ MILES WEST TO THE UPPER PORTAL AND SHOPS OF THE 16-TO-1 MINE WHICH IS AT AN ELEVATION OF 7000 FEET. THE MINE IS LOCATED IN A MOUNTAINOUS AREA WHICH IS CHARACTERIZED BY VERY RUGGED TOPOGRAPHY, WITH JAGGED CLIFFS AND DEEP, NARROW CANYONS AND WASHES. MAXIMUM ELEVATIONS IN THIS AREA APPROACH 9000 FEET.

THE REMOTENESS OF THE AREA AT TIMES MAKES OBTAINING SERVICES AND SUPPLIES DIFFICULT. THE NEAREST TRUCK FREIGHT TERMINALS ARE LOCATED 60 MILES AWAY IN TONOPAH. COMMERCIAL AIRLINE AND AIR-FREIGHT SERVICES ARE IN RENO AND LAS VEGAS, 230 AND 215 MILES AWAY, RESPECTIVELY. ALTHOUGH NOT USED BY THIS OPERATION, THE CLOSEST RAILHEAD IS AT MINA, NEVADA, 70 MILES TO THE NORTH.

THE HIGH, SEMI-ARID DESERT CLIMATE IN THIS REGION SUPPORTS A MODERATE TO SPARSE POPULATION OF VEGETATION WITH SAGEBRUSH,

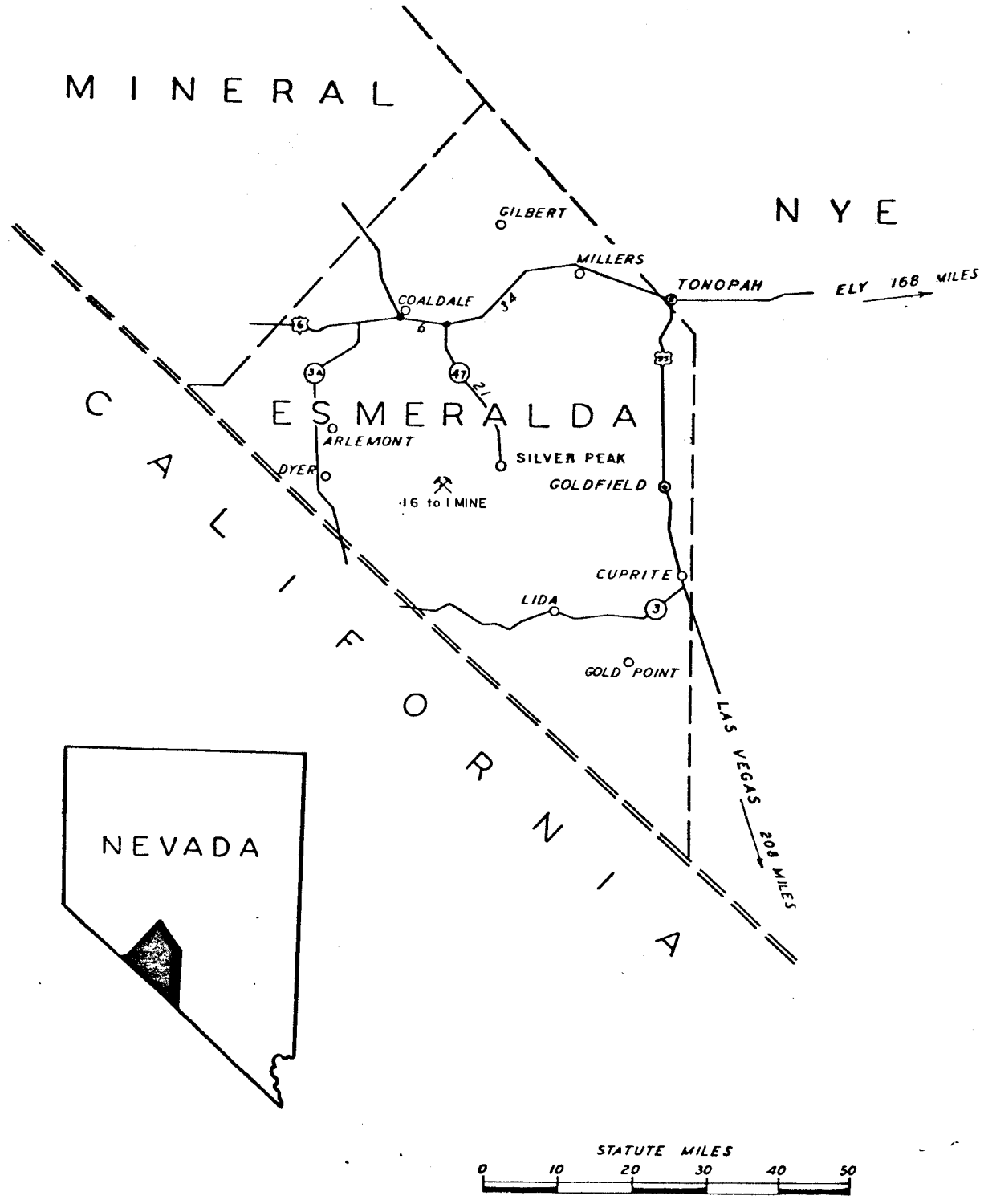


FIGURE 1

SUNSHINE MINING CO.	
LOCATION MAP - SILVER PEAK	
DRAWN 3-13-64 J.O.J.	FIGURE 1

GREASEWOOD, AND SOME GRASSES PREVALENT AT THE LOWER ELEVATIONS. PINION AND JUNIPER BECOME COMMON AT ELEVATIONS ABOVE 7000 FEET. WILDLIFE IN THE REGION IS RELATIVELY SPARCE DUE TO THE LACK OF VEGETATIVE COVER. THERE ARE NO THREATENED OR ENDANGERED SPECIES OF PLANTS NOR ANIMALS KNOWN TO INHABIT THE AREA. ANNUAL PRECIPITATION IN THE REGION IS BETWEEN FOUR AND FIVE INCHES WITH MUCH OF IT OCCURING AS FREQUENT CLOUDBURSTS DURING THE LATE SUMMER. FLASH FLOODS ARE NOT UNCOMMON. TOTAL SNOWFALL IN THE AREA OF THE MINESITE SELDOM EXCEEDS 24 INCHES ANNUALLY AND SNOW FROM INDIVIDUAL STORMS USUALLY MELTS OFF WITHIN A FEW DAYS. TEMPERATURES AT THE MILLSITE RANGE FROM A LOW OF ZERO TO HIGHS APPROACHING 100°. SUSTAINED PERIODS OF BELOW-FREEZING TEMPERATURES ARE RARE, HOWEVER, AT THIS LOCATION. ONLY ONE PERENNIAL STREAM OCCURS IN THE AREA WITH ITS ORIGIN BEING THE MINE DISCHARGE. A SMALL LAKE, PERIODICALLY STOCKED WITH TROUT, EXISTS ONE MILE BELOW THE MINE AND IS FED BY THIS WATERCOURSE WHICH SUBSEQUENTLY DISAPPEARS INTO THE GROUND APPROXIMATELY THREE MILES FURTHER DOWNSTREAM.

GEOLOGY

THE SILVER PEAK MOUNTAINS ARE THE LARGEST MOUNTAIN GROUP IN THE COUNTY, COVERING APPROXIMATELY 300 SQUARE MILES. TWO PARALLEL BELTS OF PRE-TERTIARY ROCKS INTRUDED BY QUARTZ MONZONITE PLUTONS FORM THE NORTHEAST AND SOUTHWEST PARTS. BETWEEN THESE BELTS OF OLDER ROCKS IS A THICK SECTION OF TERTIARY SEDIMENTARY AND VOLCANIC ROCKS OCCUPYING WHAT MUST HAVE BEEN A TOPOGRAPHIC LOW IN EARLY TERTIARY TIMES. IT IS IN THIS REGION THAT THE RED MOUNTAIN

MINING DISTRICT IS LOCATED, ENCOMPASSING NOT ONLY THE 16-TO-1 MINE BUT ALSO OTHER IMPORTANT, ONCE-PRODUCING PROPERTIES SUCH AS THE MOHAWK AND NIVLOC.

THE ROCKS OF THIS DISTRICT ARE PRIMARILY LATE MIOCENE EXTRUSIVE VOLCANIC TUFFS AND BRECCIAS, WITH INDIVIDUAL UNITS TYPICALLY HAVING A VERY LIMITED LATERAL EXTENT. THE OLDER VOLCANIC UNITS ARE GENERALLY ANDESITIC IN COMPOSITION, WHILE YOUNGER UNITS ARE PREDOMINENTLY RHYOLITES AND LATITES. RHYOLITIC DIKES AND PLUGS CAN BE FOUND LOCALLY. AGES FOR THE VOLCANIC UNITS RANGE FROM 22.8 M.Y. TO 11.1 M.Y. THE ANDESITES EXPOSED IN THE DISTRICT ARE ESTIMATED TO BE MORE THAN 800 FEET THICK, BASED UPON UNDERGROUND MAPPING AND DRILL HOLE DATA FROM THE 16-TO-1 MINE. THEY ARE THE PREDOMINENT HOST ROCK FOR THE OREBODY PRESENTLY BEING MINED. THE ANDESITES CAN BE DIVIDED INTO FIVE UNITS OF ALTERNATING CRYSTAL TUFF FLOWS AND TUFFACEOUS SEDIMENTS. THE RHYOLITES EXPOSED IN THE DISTRICT INCLUDE TUFFS, FLOW BRECCIAS, AND LOCAL INTRUSIVE DIKES AND PLUGS. THE SEQUENCE IS OVER 550 FEET THICK IN SOME AREAS, AND TYPICALLY FORMS THE CLIFFS AND JAGGED SPIRES PRESENT IN THE UPPER PORTION OF COTTONWOOD CANYON ABOVE THE 16-TO-1 MINE. RHYOLITES FORM THE HOST ROCKS FOR THE OREBODY ON PORTIONS OF THE UPPERMOST LEVELS OF THE MINE. LATITES CAP THE RIDGE ABOVE THE 16-TO-1 MINE IN COTTONWOOD CANYON, AND ARE MORE PREVALENT TOWARD THE WEST WHERE THEY FORM THE HOST ROCKS FOR THE MOHAWK MINE. THE UNITS CONSIST OF WATER-LAID AND AIR FALL TUFFS, INTRUSIVE DIKES, AND FLOWS, IN A SEQUENCE OVER 850 FEET THICK.

THE MAJOR STRUCTURAL SYSTEM IN THE DISTRICT CONSISTS OF NORMAL FAULTING WHICH STRIKES GENERALLY NORTHEAST. THESE FAULTS PROVIDED THE OPEN FISSURES FOR VEIN EMPLACEMENT BY HYDROTHERMAL FLUIDS. THE 16-TO-1 VEIN IS AN EPITHERMAL FISSURE VEIN WHICH STRIKES N. 40-70°E AND DIPS FROM 65° TO VERTICAL TO THE SOUTHEAST. THE WIDTH OF THE VEIN RANGES FROM FIVE FEET ON THE SURFACE TO 50 FEET ON THE 7000 AND 6890 LEVELS, BUT AVERAGES 22 FEET FOR THE ENTIRE OREBODY. THE VEIN IS WIDEST IN THE CENTRAL PORTION OF THE OREBODY, AND TAPERS TO AN AVERAGE OF EIGHT FEET ABOVE, BELOW AND TO THE SOUTHWEST ALONG STRIKE. TO THE NORTHEAST, THE VEIN NARROWS ABRUPTLY IN THE UPPER LEVELS AFTER THE DEPARTURE OF A PROMINENT FOOTWALL SPLIT CALLED THE COLORADO VEIN. THE DIMENSIONS OF THE ORE CHUTE PRESENTLY BEING MINED ARE ROUGHLY 1000 FEET ALONG STRIKE BY 525 FEET VERTICALLY, ALTHOUGH THE LOWER LIMIT OF THE CHUTE IS DEFINED BY THE LIMITATIONS OF THE MINING METHOD EMPLOYED, RATHER THAN AN ORE TENOR OR MINERALOGICAL CHANGE.

THE MINERALOGY OF THE 16-TO-1 VEIN IS RELATIVELY SIMPLE. THE DOMINANT GANGUE MINERALS ARE QUARTZ AND CALCITE, WITH QUARTZ COMPRISING 60% AND CALCITE 40%. MINOR AMOUNTS OF BARITE AND SIDERITE HAVE BEEN NOTED IN THE UPPER LEVELS. RARE ANHYDRITE HAS BEEN NOTED IN THIN SECTIONS. MUCH OF THE CALCITE IN THE UPPER LEVELS CONTAINS MANGANESE, GIVING IT A SIDERITE-LIKE APPEARANCE. BARITE HAS BEEN OBSERVED AS WHITE BLADED CRYSTALS IN LOCAL VUGS. SMECTITE IS COMMON IN BANDS ON THE LOWER LEVELS. ADULARIA HAS BEEN IDENTIFIED, BUT IS RARE. THE PRECIOUS METAL MINERALS ARE PREDOMINANTLY ARGENTITE (ACANTHITE), WITH LESSER AMOUNTS OF THE RUBY

SILVERS, PROUSTITE AND PYRARGYRITE. ADDITIONAL SILVER MINERALS WHICH HAVE BEEN TENTATIVELY IDENTIFIED MICROSCOPICALLY ARE POLYBASITE, STEPHANITE, TETRAHEDRITE, AND STROMEYERITE. NATIVE SILVER OCCURS IN MINOR AMOUNTS ON THE UPPER LEVELS AS WIRES AND FLAKES IN VUGS AND ALONG FRACTURES. GOLD IS ASSUMED TO OCCUR PRIMARILY IN NATIVE FORM. IT HAS BEEN OBSERVED IN THIN SECTION ONLY ONCE, AS AN EIGHT MICRON GRAIN WITHIN A GRAIN OF ACANTHITE. SILVER VALUES IN THE VEIN CAN RANGE FROM NIL TO +100 OUNCES PER TON, BUT OVERALL AVERAGES SIX TO SEVEN OUNCES PER TON FOR THE DEPOSIT. BASE METALS OCCUR AS SPALERITE, GALENA, AND CHALCOPYRITE. THE GALENA, WHICH IS NOT ARGENTIFEROUS, CAN LOCALLY BE FOUND IN AMOUNTS EXCEEDING 10% WITHIN INDIVIDUAL VEIN SECTIONS ON THE LOWER LEVELS. HOWEVER, OVERALL IT AVERAGES LESS THAN 1%. SPHALERITE OCCURS AS THE STRAW-COLORED, IRON POOR "HONEYJACK" VARIETY. LIKE GALENA, IT CAN APPROACH 10% IN INDIVIDUAL BANDS ON THE LOWER LEVELS, BUT AVERAGES LESS THAN 1%. CHALCOPYRITE IS FOUND LOCALLY ONLY IN FINE GRAINED DISSEMINATIONS IN AMOUNTS LESS THAN 0.2%. OTHER SULFIDES FOUND IN THE VEIN ARE PYRITE AND MARCASITE. THE PYRITE CONTENT OF THE VEIN IS VERY LOW (LESS THAN 0.3%). MARCASITE IS RARE, AND HAS BEEN OBSERVED ONLY IN POLISHED SECTION. MERCURY, ANTIMONY AND ARSENIC HAVE NOT BEEN FOUND IN SIGNIGICANT AMOUNTS IN THE ORE. VOLCANIC WALLROCK FRAGMENTS OF HORSES OF WASTE BETWEEN VEIN BANDS ARE COMMON VEIN CONSTITUENTS ON ALL LEVELS.

THREE MAJOR STAGES OF VEIN EMPLACEMENT CAN BE RECOGNIZED IN THE 16-TO-1 VEIN. STAGE I IS COMPOSED OF FRAGMENTS OF BARREN CRYPTOCRYSTALLINE QUARTZ FLOATING IN STAGE II AND III MATERIAL.

STAGE II CONSISTS OF PREDOMINENTLY QUARTZ WITH PRECIOUS AND BASE METALS AND MINOR CALCITE. STAGE III IS COMPRISED OF MOSTLY CALCITE WITH MINOR AMOUNTS OF QUARTZ AND LOW GRADE (LESS THAN 2 OZ./TON) PRECIOUS METALS. WELL DEVELOPED BANDING EXISTS IN STAGES II AND III IN THE UPPER LEVELS OF THE MINE, BUT DIMINISHES WITH DEPTH. WALL ROCK ALTERATION ADJACENT TO THE VEIN IS NOT WELL DEVELOPED. ON THE SURFACE, A ZONE OF FOOTWALL SILICIFICATION EXTENDS AT LEAST 20 FEET FROM THE VEIN. PETROGRAPHIC WORK SHOWS THE HANGINGWALL ALTERATION TO BE WEAK ARGILLIC, ALTHOUGH ALTERATION FROM A DIKE ADJACENT TO THE VEIN HANGINGWALL IMPLIES THAT THE VEIN ALTERATION IS MUCH MORE INTENSE AT DEPTH. ONLY A ONE TO TWO INCH ENVELOPE OF FOOTWALL SILICIFICATION IS PRESENT.

ALTHOUGH CONDITIONS VARY GREATLY THROUGHOUT THE OREBODY, THE ROCK WITHIN THE VEIN MAY BE DESCRIBED AS MODERATELY JOINTED AND FRACTURED WITH THESE STRUCTURES BEING PREDOMINATELY HIGH ANGLE. FRACTURES SYMPATHETIC WITH THE VEIN WALLS ARE COMMON. THE GROUND IS BLOCKY BUT USUALLY VERY TIGHT, REQUIRING LITTLE IF ANY GROUND SUPPORT. LARGE, MUD-FILLED WATERCOURSES ARE PRESENT IN THE UPPER LEVELS OF THE MINE. HANGINGWALL AND FOOTWALL ROCKS CONTAIN MINOR JOINTING AND VEINING GENERALLY PARALLELING THE MAIN VEIN STRUCTURE. OTHERWISE, THESE WALL ROCKS ARE QUITE HOMOGENEOUS. THE COMPRESSIVE STRENGTH OF THIS MATERIAL AVERAGES APPROXIMATELY 18,000 PSI, BUT MAY RANGE FROM 9000 PSI IN HIGHLY ALTERED RHYOLITES TO 26000 PSI IN COMPACT, UNALTERED, ANDESITE BRECCIA. THERE ARE NO SIGNIFICANT VERTICAL NOR LATERAL GROUND PRESSURES IN EVIDENCE UNDERGROUND.

History

The earliest mining in the Silver Peak area occurred in the 1860's and 1870's in the Mineral Ridge District located northwest of town. The Red Mountain District was born with the founding of the Nivloc Mine in 1907. The most productive years for the Nivloc, located about 1½ miles east of the 16-to-1, was from 1937 to 1943. During this period and under the directorship of Ira B. Joraleman, this mine produced approximately 300,000 tons of ore containing 12 ounces of silver and 0.05 ounces of gold per ton. The 16-to-1 property was originally staked in 1935 with a small amount of surface work being done by the locator around that time. The property saw very little activity until 1961, when Callahan Mining Corporation acquired the property under lease and option and completed five diamound drill holes into the 16-to-1 vein. Three of these holes intersected significant mineralization but the grade was thought not to be high enough to justify further work. In 1963, Mid-Continent Uranium Corporation (now Mid-Continent Mining Co.) gained control of the property and expended approximately $175,000 to drive a 1000-foot exploration adit to the 16-to-1 vein. They were rewarded by intersecting a significant width of moderate grade silver mineralization. In 1964, Sunshine Mining Company obtained a 2/3 interest in the 16-to-1 and during the following year completed additonal exploration drifting, crosscutting, and drilling with encouraging results. During 1970, New Park Mining Corporation took a short-term option on the property in an attempt to evaluate and perhaps buy out Sunshine's position. A successful drilling

PROGRAM WAS SUBSEQUENTLY COMPLETED BUT THEY DID NOT FEEL THAT THE PROPERTY WOULD SUSTAIN SUNSHINE'S EXPENSIVE BUY-OUT PRICE AND THEREFORE RELINQUISHED THEIR OPTION. EARLY IN 1973 AND AGAIN IN 1975, SUNSHINE CONDUCTED DIAMOND DRILLING PROGRAMS AT THE 16-TO-1 TO FURTHER DELINEATE THE BOUNDARIES OF THE OREBODY. A FINAL HOLE WAS DRILLED IN 1978 IN ORDER TO TEST THE FOOTWALL FOR ORE-BEARING STRUCTURES, THEREBY BRINGING THE TOTAL MONIES EXPENDED BY SUNSHINE ON EXPLORATION AT THE 16-TO-1 TO SLIGHTLY OVER $1 MILLION. DURING 1979, VARIOUS STUDIES WERE PERFORMED ON THE PROPERTY, INCLUDING FURTHER METALLURGICAL TEST WORK, ORE RESERVE ESTIMATIONS, AND PROJECT FEASIBILITY STUDIES. IN MARCH OF 1980, AN ENGINEER WAS ASSIGNED TO THE PROJECT FOR THE PURPOSE OF PREPARING A PROJECT FEASIBILITY REPORT AND BUDGET PRESENTATION TO SUNSHINE'S BOARD OF DIRECTORS IN JUNE OF THAT YEAR.

THE 2/3 INTEREST MENTIONED EARLIER AND WHICH SUNSHINE PRESENTLY ENJOYS WAS ACQUIRED BY WAY OF A 50-YEAR OPERATING AGREEMENT WITH THEIR PARTNER, MID-CONTINENT, WHEREBY SUNSHINE ASSUMES FULL AND EXCLUSIVE CONTROL OF THE THE JOINT PROPERTIES. NET PROFITS FROM THE OPERATION ARE DIVIDED ON A 2/3 TO 1/3 BASIS, BUT ONLY AFTER SUNSHINE HAS RECOUPED 1/3 OF ALL EXPLORATION AND DEVELOPMENT COSTS, INCLUDING INTEREST FIGURED AT 7% OR AT PRIME, WHICHEVER IS LARGER.

SUNSHINE PRESENTLY CONTROLS 257 LODE MINING CLAIMS ENCOMPASSING APPROXIMATELY 5,300 ACRES TOGETHER WITH 117 MILL SITE CLAIMS TOTALING 585 ACRES. ALL CLAIMS ARE PRESENTLY UNPATENTED AND ARE LOCATED ON LANDS ADMINISTERED BY THE BLM. THERE

ARE NO NATIONAL FOREST NOR WILDERNESS LANDS IN THE IMMEDIATE VACINITY. HOWEVER, THERE IS ONE AREA WITHIN FOUR MILES OF THE MINE WHICH IS CURRENTLY UNDER REVIEW AS TO ITS SUITABILITY AS A WILDERNESS STUDY AREA.

PART III

PROJECT FEASIBILITY AND PLANNING

ORE RESERVE ESTIMATION

THE ORE RESERVES OF THE 16-TO-1 MINE WERE CALCULATED BY EMPLOYING THE COMMONLY USED "POLYGON METHOD" OF ESTIMATING RESERVES. THE INPUT DATA FOR THIS CALCULATION CAME FROM BOTH UNDERGROUND MINE CHANNEL SAMPLING AND FROM DIAMOND DRILLING RESULTS. AS OF 1980, APPROXIMATELY 850 FEET OF DRIFT HAD BEEN DRIVEN ALONG THE HANGINGWALL PORTION OF THE 16-TO-1 STRUCTURE AT AN ELEVATION OF 7000 FEET. INCLUDED IN THIS DISTANCE WAS A 600-FOOT SECTION WHERE CROSSCUTS HAD BEEN DRIVEN THROUGH THE FOOTWALL OF THE VEIN AT 35 FOOT INTERVALS. THESE CROSSCUTS WERE EXTENSIVELY SAMPLED BY GEOLOGISTS FROM VARIOUS COMPANIES. IN ADDITION TO THESE CROSSCUTS, RAISES WERE ALSO DRIVEN IN A FEW LOCATIONS AND SAMPLED IN ORDER TO TEST THE VEIN ABOVE THE 7000 FOOT LEVEL. ALL ASSAY DATA FROM CROSSCUT AND RAISE SAMPLING WAS USED IN THE ORE RESERVE CALCULATION. POLYGONS WERE CONSTRUCTED ACCORDING TO THE GEOMETRY OF THESE WORKINGS AND THE ASSAY DATA WAS THEN GROUPED AND WEIGHTED IN ORDER TO ARRIVE AT AN AVERAGE GRADE FOR EACH POLYGON. ACTUAL VEIN WIDTHS MEASURED AT EACH CROSSCUT WERE USED IN ASSIGNING A WIDTH TO EACH POLYGON. THE MAXIMUM DISTANCE PROJECTED FROM SAMPLING LOCATIONS WAS APPROXIMATELY 100 FEET. THE ASSAY RESULTS FROM 20 DIAMOND DRILL HOLES WERE USED IN CONJUNCTION WITH THE CROSSCUT AND RAISE SAMPLING DATA IN ORDER TO ESTIMATE RESERVES BELOW THE 7000 FOOT LEVEL. THESE DIAMOND DRILL HOLES WERE BW AND AQ IN SIZE, WITH THEIR VEIN INTERCEPT LOCATIONS BEING IRREGULARLY

SPACED (FIGURE 2) AS A RESULT OF THE INCONSISTENT NATURE OF PAST DRILLING PROGRAMS AND DUE TO CONSTRAINTS IN SELECTING DRILL SITES BECAUSE OF THE RUGGED TOPOGRAPHY. POLYGONS WERE CONSTRUCTED AROUND THESE 20 INTERCEPT LOCATIONS AND THE AREAS THEN COMPUTED. GRADES WERE PROJECTED PAST PERIMETER HOLES FOR A DISTANCE EQUAL TO ONE-HALF THE DISTANCE TO THE NEAREST DRILL HOLE. A TONNAGE FACTOR OF 12 WAS USED ON ALL TONNAGE ESTIMATES. HIGH GRADE BANDS, SOMETIMES ASSAYING UP TO 100 OUNCES/TON AND OCCURRING IN CROSS-CUTS AND IN DRILL HOLES, WERE NOT DISCOUNTED TO OBTAIN THE GRADE OF THE POLYGON. A MINIMUM WIDTH OF 12 FEET WAS ASSUMED IN ALL RE-SERVE ESTIMATES.

IN ORDER TO FACILITATE SUBSEQUENT MINE PLANNING ACTIVITIES, THE ORE RESERVE ESTIMATE WAS DONE USING A VARIETY OF PARAMETERS. SEVERAL ESTIMATES WERE MADE USING DIFFERENT CUTOFF GRADES, DILU-TION FACTORS, AND ASSUMED ELEVATIONS FOR THE BOTTOM OF THE OREBODY, SINCE THE OREBODY WAS ESSENTIALLY OPEN-ENDED DOWN-DIP. EARLY ON IN THE PLANNING STAGES OF THE PROJECT, A DECISION WAS MADE TO USE A "STRUCTURAL CUTOFF" RATHER THAN A GRADE CUTOFF FOR ARRIVING AT THE ULTIMATE MINING WIDTH FOR EACH POLYGON. ASSUMING THAT A HIGHLY EFFICIENT MINING SYSTEM WOULD BE EMPLOYED, IT WAS FELT THAT THE VEIN, WHEN BLASTED, WOULD BREAK OUT TO THE HANGINGWALL AND FOOT-WALL CONTACTS, REGARDLESS OF HOW IT WAS DRILLED. THIS LATER PROVED TO BE A CORRECT PREDICTION.

ANOTHER EXERCISE WHICH WAS PERFORMED IN ORDER TO AID MINE PLANNING EFFORTS WAS A LIMITED GEOSTATISTICAL STUDY OF THE OREBODY.

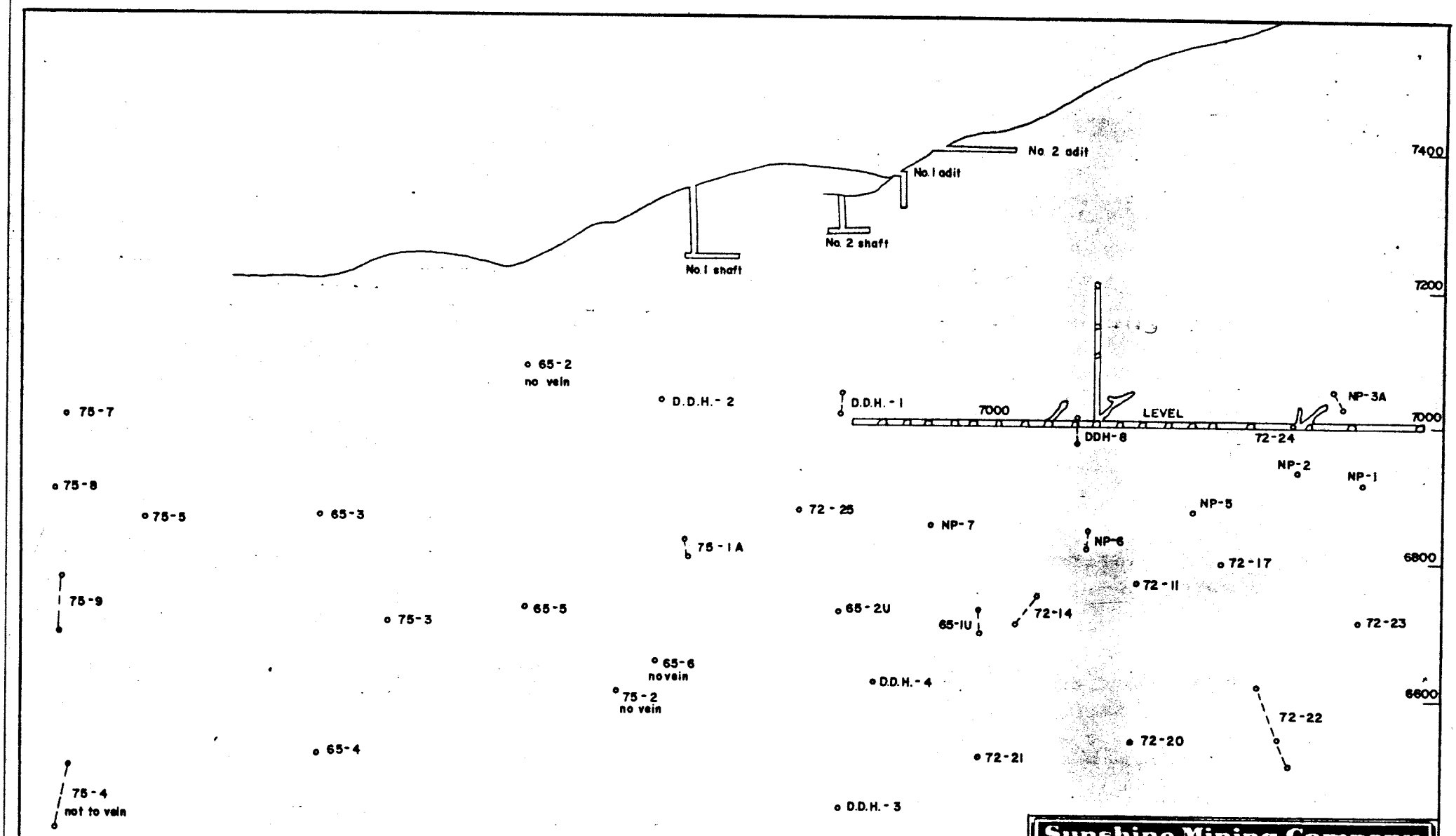


FIGURE 2

Sunshine Mining Company		
16 to 1 MINE		
Silver Peak, Nevada		
LONGITUDINAL SECTION		
R.A.F.	1" = 200'	11/19/76

A CONSULTANT, GOLDER ASSOCIATES, WAS HIRED TO DO AN EVALUATION OF THE TONS AND GRADE PROJECTIONS IN ORDER TO DETERMINE IF THE RELIABILITY OF THESE PROJECTIONS WAS SIGNIFICANTLY REDUCED WITH DEPTH AND, IF SO, BY HOW MUCH. RELIABILITY, EXPRESSED AS A PERCENT, WAS CALCULATED FOR CONFIDENCE INTERVALS OF ± 10% AND ± 15% FOR DIFFERENT ASSUMED ELEVATIONS FOR THE BOTTOM OF THE OREBODY. THIS INFORMATION WAS USEFUL IN CHOOSING THE OPTIMUM "BOTTOM" ELEVATION AND IT ALSO PROVIDED MANAGEMENT WITH DATA WHICH WOULD BE HELPFUL IN EVALUATING THE RISKS INVOLVED WITH THE PROJECT.

GENERAL DESIGN PARAMETERS

THE SELECTION OF THE DESIRED THROUGHPUT RATE IS CERTAINLY THE MOST BASIC OF ALL THE GENERAL DESIGN PARAMETERS FOR ANY PROJECT, SINCE IT AFFECTS ALL SUBSEQUENT PLANNING ACTIVITIES AND ECONOMIC ANALYSIS. ONCE IT BECAME APPARENT THAT THE MINEABLE ORE RESERVES AT THE 16-TO-1 WOULD BE IN THE 0.9 - 1.2 MILLION TON RANGE, THE QUESTION OF DESIRED THROUGHPUT WAS ADDRESSED. IN THEORY, IT IS POSSIBLE TO CALCULATE AN OPTIMUM THROUGHPUT RATE, BUT THIS APPROACH REQUIRES INFORMATION OF A PRECISION WHICH DID NOT EXIST FOR THIS PROJECT. THEREFORE, THE QUESTION OF DESIRED THROUGHPUT WAS ONE REQUIRING MORE INTUITIVE REASONING THAN MATHEMATICAL CALCULATION. TOO LOW A RATE SACRIFICES THE ADVANTAGES OF ECONOMY OF SCALE WHILE TOO HIGH A RATE MAY INCREASE THE INITIAL CAPITAL COSTS BEYOND THE ABILITY OF A PROJECT TO REPAY THEM WITHIN ITS SHORTENED LIFE. A THROUGHPUT RATE OF 150,000 TONS PER YEAR (430 TPD) WAS CHOSEN FOR THIS PROJECT. THIS FIGURE, IT WAS FELT, WAS HIGH ENOUGH TO RESULT

IN COMPETITIVE UNIT COSTS WHILE STILL ALLOWING SUFFICIENT LIFE IN WHICH TO EXPLORE AND DEVELOP OTHER ORE BODIES AND THEREBY SUSTAIN LONG-TERM, UNINTERRUPTED PRODUCTION.

PERHAPS A FOREGONE CONCLUSION BUT NEVERTHELESS A VERY BASIC DESIGN PARAMETER WAS THE SELECTION OF THE METHOD OF MINING FOR THE 16-TO-1 OREBODY; THAT IS, UNDERGROUND OR OPEN PIT. THE FOLLOWING FACTORS MADE THE CONSIDERATION OF ANY TYPE OF OPEN PIT METHOD EXTREMELY IMPRACTICAL:

- DEPTH OF BURIAL OF THE OREBODY WOULD REQUIRE A TREMENDOUS AMOUNT OF INITIAL STRIPPING.
- RUGGED TOPOGRAPHY IN THE AREA WOULD RESULT IN HIGH ACCESS ROAD COSTS AND WASTE DISPOSAL SITING PROBLEMS.
- ORIENTATION AND NARROW WIDTH (RELATIVE TO OTHER PITS) OF OREBODY WOULD RESULT IN AN UNBEARABLE STRIPPING RATIO ONCE PRODUCTION WAS UNDERWAY.

A THIRD GENERAL DESIGN PARAMETER WAS ONE INVOLVING PEOPLE. DUE TO THE REMOTENESS OF THE AREA, THERE WERE REAL CONCERNS THAT IT WOULD BE EXTREMELY DIFFICULT TO ATTRACT AND KEEP SKILLED LABORERS AT THIS OPERATION. IN ORDER TO NOT ONLY MINIMIZE OPERATING COSTS BUT ALSO REDUCE THE EFFECT OF THIS POTENTIAL PERSONNEL PROBLEM, A PHILOSOPHY OF DESIGN WHICH WOULD MINIMIZE MANPOWER REQUIREMENTS WAS ADOPTED. AN EQUIPMENT INTENSIVE RATHER THAN LABOR INTENSIVE MINING SYSTEM HAD TO BE DEVELOPED ALONG WITH A CONVENTIONAL MILLING SCHEME WITH ADEQUATE AUTOMATION AND INSTRUMENTATION. THIS PHILOSOPHY

WAS PREVALENT THOUGHOUT ALL PHASES OF DESIGN OF BOTH THE MINE AND THE MILL AND ULTIMATELY RESULTED IN A VERY MANPOWER EFFICIENT OPERATION.

Conceptual Mining Plan

SELECTING THE PROPER UNDERGROUND MINING SYSTEM WAS OBVIOUSLY THE FIRST STEP IN DEVELOPING A CONCEPTUAL MINING PLAN FOR THE 16-TO-1 OREBODY. AFTER CAREFUL CONSIDERATION OF MANY FACTORS, AND EXAMINATION OF THE EXISTING WORKINGS, LARGE DIAMETER BLASTHOLE STOPING WAS CHOSEN AS THE MINING SYSTEM. BRIEFLY, THIS SYSTEM INVOLVES DRILLING AND BLASTING VERTICAL SLICES OF ORE BETWEEN SUBLEVELS INTO THE VOID CREATED BY PREVIOUS BLASTS, WITH THE BROKEN MATERIAL BEING EXTRACTED FROM DRAWPOINTS AT THE BOTTOM OF THE OREBODY (FIGURE 3). THE CONDITIONS WHICH MADE THE SELECTION OF THIS HIGHLY EFFICIENT SYSTEM POSSIBLE WERE:

- ADEQUATE VEIN WIDTH; OVER 12 FEET IN MOST PLACES, AVERAGING 22 FEET.
- STEEPLY DIPPING OREBODY; GENERALLY 70 - 90°.
- VEIN CONTINUITY
- GOOD GROUND CONDITIONS; COMPETENT WALL ROCKS AND VEIN, DEFINITE HANGINGWALL AND FOOTWALL CONTACTS.

FOLLOWING A REVIEW OF THE LITERATURE AVAILABLE ON MINES USING SIMILAR SYSTEMS, A DILUTION FACTOR OF 10%, TO BE USED IN CALCULATING MINING RESERVES, WAS ESTIMATED FOR THE 16-TO-1. ASSAY VALUES OF

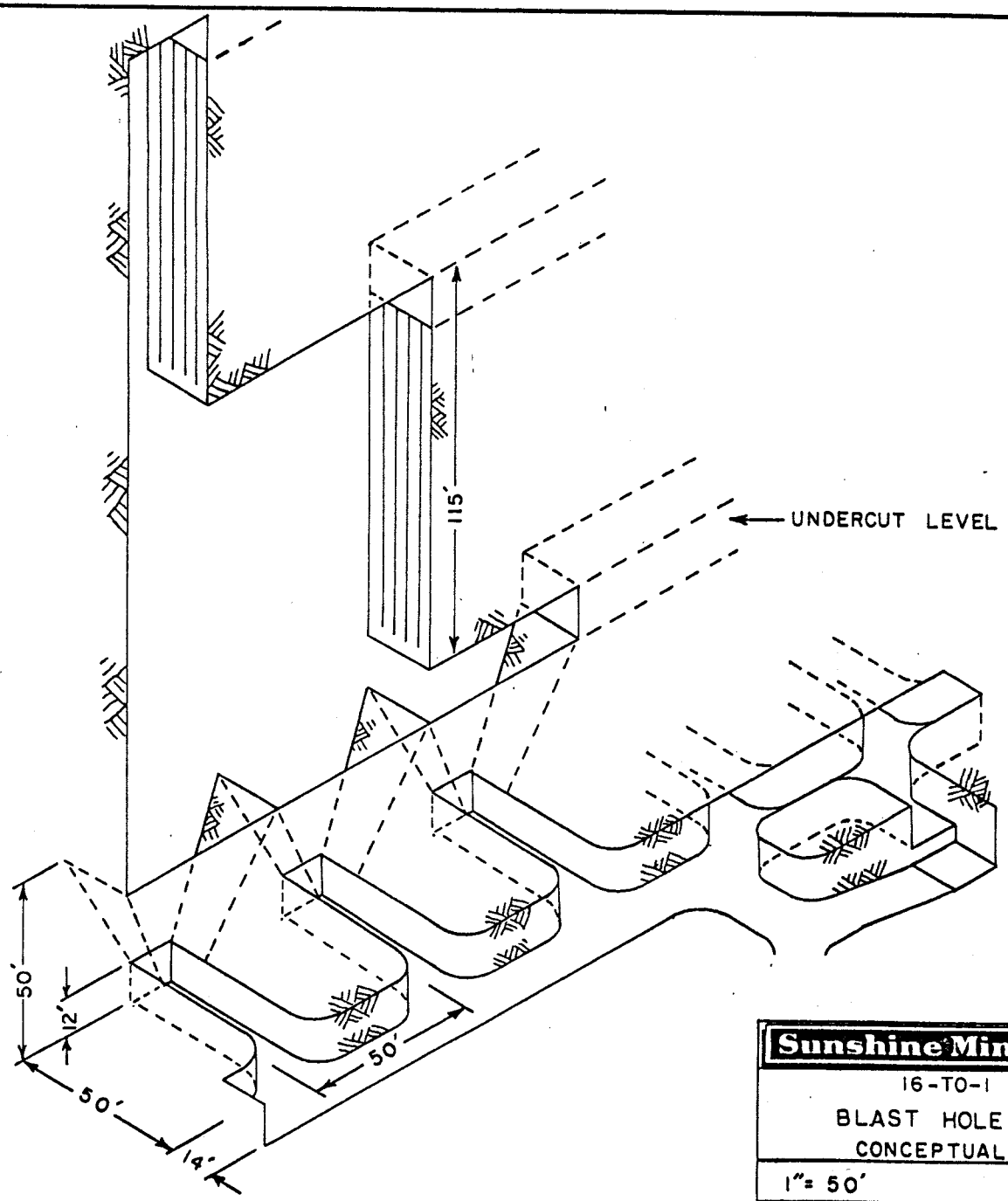


FIGURE 3

Sunshine Mining Company		
16-TO-1 MINE		
BLAST HOLE STOPING CONCEPTUAL DRAWING		
1"= 50'		
5-26-80		
THB		

1.00 OUNCE PER TON SILVER AND 0.01 OUNCE PER TON GOLD WERE APPLIED TO THIS DILUTION AS IT WAS FOUND THAT WALL ROCK IMMEDIATELY ADJACENT TO THE VEIN CONTAINED A SMALL AMOUNT OF BACKGROUND GOLD AND SILVER AS DISSEMINATIONS AND VEINLETS. THE 10% DILUTION FACTOR WAS APPLIED TO ALL RESERVE BLOCKS, REGARDLESS OF WIDTH.

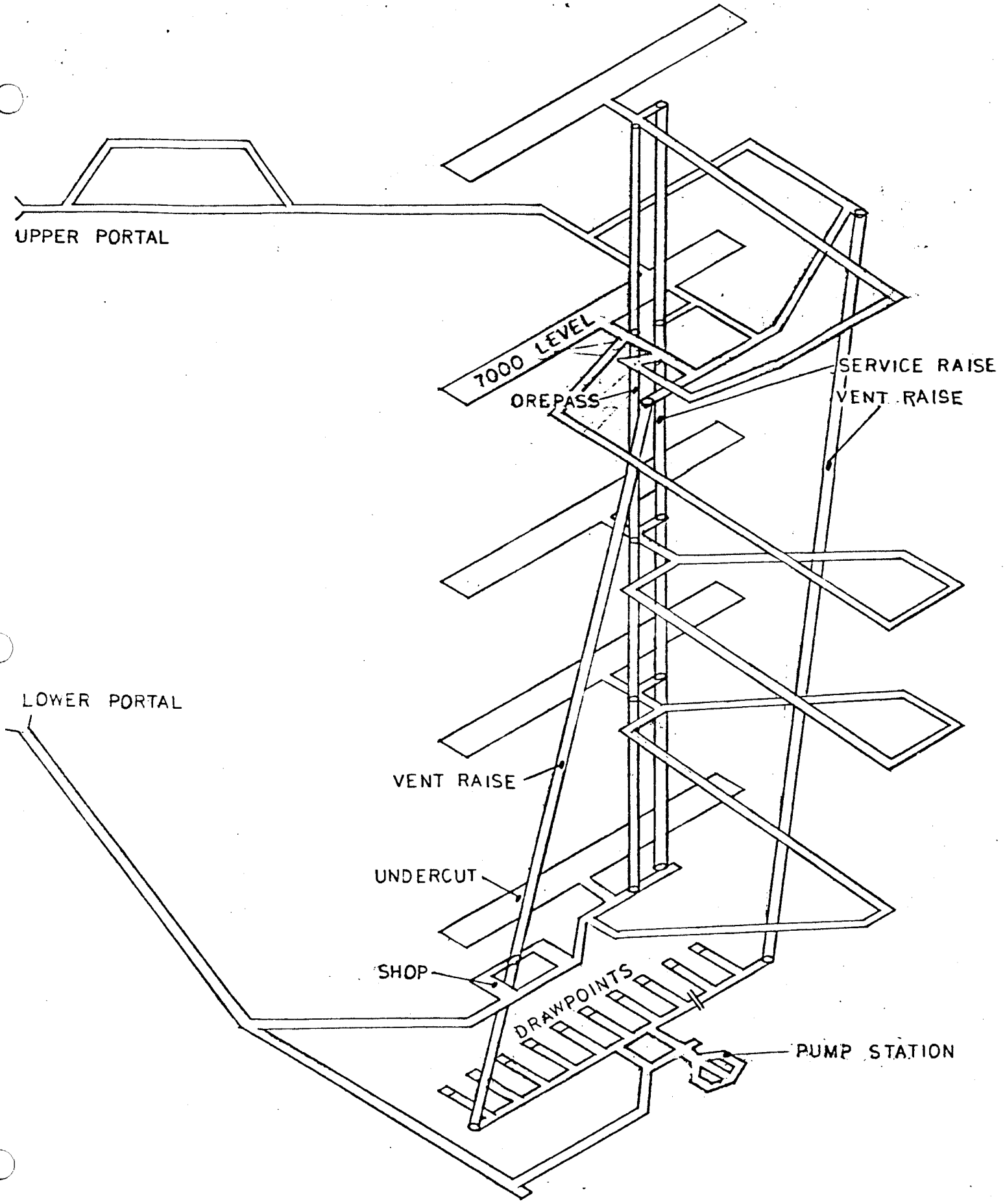
ANOTHER IMPORTANT DECISION WHICH HAD TO BE MADE DURING THE CONCEPTUAL MINE PLANNING PHASE PERTAINED TO THE METHOD OF ACCESS OF THE OREBODY. SINCE THE ORIGINAL 7000 LEVEL CROSSCUT WAS NOT ONLY TOO SMALL BUT ALSO INTERSECTED THE VEIN NEAR THE TOP OF THE OREBODY, A MEANS OF ACCESSING THE BOTTOM PORTION OF THE OREBODY HAD TO BE DETERMINED. THIS QUESTION INVOLVED THE CONSIDERATION OF SEVERAL ALTERNATIVES INCLUDING A CONVEYOR INCLINE, SHAFT, HORIZONTAL TRACK DRIFT FROM A LOWER ELEVATION, AND RAMP ACCESS FROM SEVERAL LOCATIONS. USING AN ASSUMED MINEABLE TONNAGE AND BOTTOM ELEVATION OF THE OREBODY, PRESENT-VALUE (I=20%) COST COMPARISONS WERE DONE FOR ALL ALTERNATIVES AND COMBINATIONS OF ALTERNATIVE. AMONG THE FACTORS WHICH WERE CONSIDERED IN THESE CALCULATIONS WERE ESTIMATED PUMPING COSTS, DEVELOPMENT COSTS OF DRIFTING AND SHAFT SINKING, OPERATING COSTS FOR ORE HANDLING THROUGHOUT THE LIFE OF THE MINE, AND INITIAL EQUIPMENT COSTS. THE RESULTS OF THIS COST STUDY INDICATED THAT RAMP ACCESS, WITH ITS PORTAL LOCATED APPROXIMATELY ¼ OF A MILE FROM THE UPPER LEVEL PORTAL AND 175 FEET LOWER IN ELEVATION, WAS THE PREFERRED METHOD. THIS ACCESS RAMP WOULD BE DRIVEN AT A MINUS 10½% GRADE THROUGH THE INTERSECTION AND INTO THE FOOTWALL OF THE VEIN, WITH SUBSEQUENT DEVELOPMENT OF THE DRAWPOINT AND UNDERCUT LEVELS.

SELECTING THE OPTIMUM BOTTOM ELEVATION OF THE OREBODY WAS THE NEXT IMPORTANT DECISION TO BE MADE. UP TO THIS POINT IN THE MINE PLANNING PHASE, THIS ELEVATION HAD ONLY BEEN ASSUMED. IT WAS, HOWEVER, IMPORTANT THAT IT BE DEFINITELY IDENTIFIED SO THAT WORK COULD PROCEED ON MORE DETAILED MINE LAYOUT AND PLANNING. PREVIOUS EXPLORATION WORK AT THE 16-TO-1 HAD DEFINED THE TOP AND LATERAL LIMITS OF THE OREBODY AS THESE WERE QUITE ABRUPT CHANGES IN GRADES OR WIDTHS. GRADE AND WIDTH CHANGES IN THE DOWN-DIP DIRECTION WERE MORE GRADUAL. SELECTION OF THE OPTIMUM BOTTOM ELEVATION DEPENDED ON TWO FACTORS; ECONOMICS AND RISK. AS MENTIONED PREVIOUSLY, ORE RESERVES HAD BEEN CALCULATED USING A VARIETY OF BOTTOM ELEVATIONS. AS LOWER ELEVATIONS WERE CHOSEN, THE MINEABLE TONNAGE, OF COURSE, INCREASED BUT SO DID DEVELOPMENT COSTS, BUT NOT NECESSARILY IN A PROPORTIONATE MANNER. THEREFORE, A TOTAL CASH FLOW CALCULATION WAS DONE FOR EACH ELEVATION AND THESE WERE COMPARED USING NET PRESENT VALUE METHODS (I = 20%). THE RESULTS OF THIS EXERCISE WERE THEN EVALUATED ALONG WITH THE STATISTICAL RELIABILITY DATA CORRESPONDING TO THE TONS AND GRADE PROJECTIONS AT THE VARIOUS ELEVATIONS. AN ELEVATION OF 6650 FEET, 350 FEET BELOW THE EXISTING WORKINGS, WAS CHOSEN AS BEING THE OPTIMUM LEVEL. THIS ELEVATION DEFINED A MINEABLE ORE RESERVE OF APPROXIMATELY 1.1 MILLION TONS HAVING A DILUTED GRADE OF 8.38 OZ/TON SILVER AND 0.03 OZ/TON GOLD.

IN ORDER TO MAKE AN INTELLIGENT CHOICE OF THE OPTIMUM SUBLEVEL SPACING FOR THE 16-TO-1 MINE, MANY FACTORS WERE CONSIDERED AND ANALYZED. A VARIETY OF MAXIMUM OR OPTIMUM INTERVAL FIGURES WERE

COMPUTED DEPENDING ON THE PARAMETER USED. SOME OF THE PARAMETERS WERE OF GREATER CONCERN OR IMPORTANCE THAN OTHERS. THEREFORE, SOME CALCULATED SUBLEVEL INTERVAL FIGURES WOULD BE WEIGHTED MORE HEAVILY THAN OTHERS. COMPUTED INTERVALS RANGED FROM 60 FEET TO 164 FEET WITH THE MEAN BEING 118 FEET. TWO OF WHAT ARE PERHAPS THE MOST IMPORTANT CONSIDERATIONS IS HOLE LENGTH VE. SUPERVISION REQUIRED TO MAINTAIN ACCURACY AND MINIMIZING DILUTION DUE TO CHANGES IN VEIN DIP. ANALYSIS OF THESE TWO CONSIDERATIONS INDICATED LEVEL SPACINGS OF 95 FEET AND 114 FEET RESPECTIVELY. DUE TO THE FACT THAT MOST OF THE OTHER COMPUTED VALUES OF LEVEL SPACING WERE GREATER THAN THE 114 FEET AND BECAUSE THE CALCULATED INTERVAL BASED ON MINIMIZING DILUTION HIGH SIDE OF THIS RANGE, 115 FEET SILL TO SILL WAS CHOSEN.

FOLLOWING THE SELECTION OF THE MINING SYSTEM, METHOD OF ACCESS, PHYSICAL BOUNDARIES OF THE OREBODY, AND LEVEL SPACING, THE JOB OF DETAILED MINE LAYOUT AND PLANNING WAS BEGUN. BECAUSE IT WAS THOUGHT THAT GROUND CONDITIONS WOULD BE BETTER ON THE FOOTWALL SIDE OF THE VEIN AND BECAUSE FOOTWALL DEVELOPMENT WOULD SHORTEN THE DRILLING DISTANCE TO OTHER EXPLORATION TARGETS, THIS METHOD OF ATTACK WAS CHOSEN. IN CONJUNCTION WITH THE DRIVING OF THE 2100-FOOT MAIN ACCESS RAMP, THE UPPER LEVEL CROSSCUT WOULD ALSO BE ENLARGED TO ACCOMMODATE DIESEL EQUIPMENT. ACCESS TO INTERMEDIATE SUBLEVELS WOULD BE BY 15% SPIRAL RAMPS DRIVEN FROM ONE SUBLEVEL TO THE NEXT. THESE RAMPS WOULD ULTIMATELY CONNECT THE LOWER AND UPPER LEVEL WORKINGS TOGETHER, THEREBY PROVIDING TWO MEANS OF ACCESS OR ESCAPE FROM ANY LOCATION WITHIN THE MINE (FIGURE 4). ALL OPENINGS WOULD BE A MINIMUM OF 12 FEET HIGH BY 14 FEET WIDE. DRAWPOINTS LOCATED



16-TO-1 MINE
CONCEPTUAL DEVELOPMENT PLAN

BY: A. YOUNG
5-17-80

FIGURE 4

AT THE BOTTOM OF THE MINE WOULD BE DRIVEN ON 50-FOOT CENTERS FROM A LATERAL DRIFT PARALLELING THE VEIN IN THE FOOTWALL. THIS LATERAL DRIFT WAS LOCATED A MINIMUM OF 50 FEET FROM THE FOOTWALL OF THE VEIN IN ORDER TO PROVIDE MAXIMUM GROUND STABILITY AND INSURE THAT THE LHD UNITS WOULD HAVE A STRAIGHT SHOT AT THE MUCKPILE WHEN MUCKING OUT OF DRAWPOINTS. EACH DRAWPOINT WOULD BE CONED UP TO AN UNDERCUT LEVEL, DRIVEN IN ORE APPROXIMATELY 35 FEET ABOVE THE DRAWPOINT BROWS, WITH THE FORMATION OF PILLAR APEXES AT THE UNDERCUT SILL ELEVATION. THE MINE IS LAID OUT WITH A CENTRALLY LOCATED RAMP SYSTEM SERVING EACH OF THE FIVE SUBLEVELS SPACED 115 FEET APART. THIS RAMP SYSTEM CONSISTS OF EXTREMELY ELONGATED SPIRALS LOCATED IN THE FOOTWALL OF THE VEIN WITH THEIR LONG AXIS' POSITIONED PERPENDICULAR TO THE STRIKE OF THE VEIN. THIS DESIGN MINIMIZES THE AMOUNT OF OPEN GROUND EXISTING ADJACENT TO AN ACTIVE STOPE AND ALSO PLACES THE FAR END OF EACH SPIRAL IN CLOSE PROXIMITY TO THE PREVIOUSLY MENTIONED DRILL TARGETS ON THE FOOTWALL SIDE OF THE 16-TO-1 STRUCTURE. THE RAMPS ARE CONNECTED TO EACH SUBLEVEL BY MEANS OF A SHORT CROSSCUT ALONG WHICH IS CONTAINED AN OREPASS DUMP HOLE. ALL ORE FROM SUBLEVEL DEVELOPMENT WOULD BE GRIZZLIED AT THESE POINTS WITH THE ROCK BEING LOADED INTO TRUCKS THROUGH A CHUTE AT THE BOTTOM. ALSO LOCATED ALONG THESE CROSSCUTS WERE CONNECTIONS TO A COMMON SERVICE RAISE FROM WHICH VENTILATION AIR, COMPRESSED AIR, DRILL WATER, AND ELECTRICAL POWER WOULD BE OBTAINED. ON THE DRAWPOINT LEVEL, A TRUCK LOADING STATION WAS LOCATED APPROXIMATELY MIDWAY ALONG THE LATERAL DRIFTS. HERE, TRUCKS WOULD BACK INTO A SHORT DRIFT WHICH WOULD BE RUN BELOW GRADE TO ESTABLISH A BENCH FROM WHICH LHD UNITS COULD QUICKLY AND COMPLETELY

LOAD THE TRUCK FOR ITS TRIP OUT OF THE MINE. VIRTUALLY ALL ORE WOULD BE HAULED OUT OF THE LOWER PORTAL OF THE MINE, WITH MOST OF THIS COMING FROM THE DRAWPOINT LEVEL AND THE BALANCE FROM DEVELOPMENT ORE PICKED UP AT THE OREPASS CHUTE ON THE UNDERCUT LEVEL. UPON REACHING THE SURFACE, THE ORE WOULD BE GRIZZLIED AND DUMPED INTO A 400-TON COARSE ORE BIN EXCAVATED OUT OF A ROCK OUTCROP PROTRUDING FROM ONE SIDE OF THE 16-TO-1 CANYON. AT THE BOTTOM OF THIS BIN A HYDRAULICALLY OPERATED CHUTE WOULD LOAD INTO 25-TON SURFACE HAULAGE TRUCKS FOR THE 3½-MILE TRIP TO THE MILL. AS ALL SUBLEVEL ACCESS RAMPS ARE LOCATED ON THE FOOTWALL SIDE OF THE VEIN AND BOTH ACCESS CROSSCUTS ORIGINATE ON THE HANGINGWALL SIDE, THE MINE LAYOUT HAD TO INCLUDE PROVISIONS FOR MAINTAINING BOTH ACCESSES AS THE VEIN WAS BEING MINED OUT. A RUN-AROUND WAS DRIVEN, IN THE CASE OF THE UPPER LEVEL CROSSCUT, TO ALLOW FOR THE EVENTUAL MINING OUT OF THE ORIGINAL ACCESS THROUGH THE VEIN. THE POINT AT WHICH THE LOWER LEVEL ACCESS RAMP INTERSECTS THE 16-TO-1 VEIN WAS CHOSEN TO BE IN A NARROW, WASTE SECTION OF VEIN LEAVING SUFFICIENT PILLAR BETWEEN IT AND THE STOPING AREA.

A MAIN MINE SUMP WAS PROVIDED FOR AT THE LOWEST ELEVATION IN THE MINE AND LOCATED NEAR THE TRUCK LOADING DOCK. DRAINAGE WATER FROM BOTH THE EAST AND WEST SIDES OF THE OREBODY WOULD BE COLLECTED HERE AND PUMPED OUT THE LOWER PORTAL. AN UNDERGROUND COMPRESSOR STATION WAS PLANNED AND WOULD BE LOCATED AT A SOURCE OF RELATIVELY CLEAN, FRESH AIR, BUT ALSO CLOSE TO THE MAIN EXHAUST AIRWAY.

THE DEVELOPMENT OF A MINE VENTILATION PLAN SHOULD BE AN

INTEGRAL PART OF ANY CONCEPTUAL MINING PLAN. THIS IS PARTICULARLY IMPORTANT IN A MINE WHERE DIESEL EQUIPMENT IS TO BE USED. VENTILATION REQUIREMENTS AT THE 16-TO-1 DICTATED THE LOCATION OF CERTAIN CROSSCUTS, LATERALS AND RAISES, SINCE IT WAS A PREREQUISITE THAT ALL WORKING FACES AND DRAWPOINT ENTRANCES BE SUPPLIED WITH FRESH UNCONTAMINATED AIR. THE GENERAL COURSE OF THE VENTILATION AIR THROUGH THE MINE WAS DOWNCAST, WITH ALL AIR BEING EXHAUSTED OUT THE LOWER PORTAL. ALTHOUGH THIS SCHEME WORKED AGAINST THE NATURAL VENTILATION PRESSURE IN THE MINE, IT DID HAVE THE FOLLOWING ADVANTAGES:

- MOST DIESEL SMOKE IS CREATED IN AREAS WHERE EQUIPMENT IS WORKING THE HARDEST; NAMELY, THE DRAWPOINTS AND MAIN HAULAGE RAMP. EXHAUSTING OUT THE LOWER PORTAL IS THE QUICKEST WAY TO GET RID OF THIS CONTAMINATED AIR.
- EXHAUSTING OUT THE LOWER PORTAL PREVENTS FREEZING OF THE HAULAGE SURFACE NEAR THE PORTAL DURING THE WINTER MONTHS.

IN ORDER TO PREVENT POSSIBLE INJURIES AND TEMPORARY LOSS OF VENTILATION DUE TO ACCIDENTAL RAMMING OF AIR DOORS AND BULKHEADS, THE MAIN MINE FAN AND AIRLOCK SYSTEM WAS PLANNED FOR THE UPPER LEVEL PORTAL WHERE THERE WOULD BE MUCH LESS TRAFFIC. MINE PRODUCTION HAULAGE CREWS, THEREFORE, WOULD NOT BE REQUIRED TO GO THROUGH ANY AIRDOORS DURING LOWER LEVEL HAULAGE OPERATIONS.

ALL UNDERGROUND MINING EQUIPMENT CHOSEN FOR THE 16-TO-1 PROJECT WAS DIESEL POWERED AND HIGHLY MOBILE. MUCKING AND HAULAGE

EQUIPMENT PURCHASED BY SUNSHINE FOR DEVELOPMENT WOULD ALSO BE USED EVENTUALLY IN PRODUCTION. THE MUCKING AND HAULAGE FLEET CONSISTED OF TWO 3½ CU. YD. LOAD-HAUL-DUMP UNITS AND TWO 14-TON UNDERGROUND HAULAGE TRUCKS FOR A PLANNED MINE OUTPUT OF 500 TONS PER DAY, SIX DAYS PER WEEK. DEVELOPMENT DRIFTING WOULD BE DONE UTILIZING A TWO-BOOM HYDRAULIC JUMBO. THIS HYDRAULIC UNIT WAS SELECTED OVER COMPARABLE AIR MACHINES ON THE BASIS OF POWER SAVINGS ALONE OVER ITS USEFUL LIFE. PRODUCTION DRILLING AT THE 16-TO-1 WOULD BE ACCOMPLISHED BY A SELF-PROPELLED DOWN-THE-HOLE DRILLING RIG. BECAUSE THE MINING PLAN CALLED FOR LEAVING BROKEN ORE IN THE STOPE FOR GROUND SUPPORT, THIS UNIT HAD TO BE CAPABLE OF DRILLING OUT MORE THAN TWICE AS MUCH ORE PER DAY AS WAS NEEDED FROM THE STOPE TO FEED THE MILL. OTHER PLANNED MOBILE EQUIPMENT INCLUDED AN EXPLOSIVES CARRIER, MAINTENANCE TRUCK, AND UTILITY UNIT.

IN ORDER TO COMPLETE OVER THREE MILES OF PREPRODUCTION DEVELOPMENT WORK WITHIN THE TIME SCHEDULE ALLOWED, A STRATEGY WAS ADOPTED WHICH WOULD UTILIZE AN EXPERIENCED CONTRACTOR TO DO A PORTION OF THE WORK FROM ONE PORTAL WHILE SUNSHINE CREWS FINISHED THE BALANCE FROM THE OTHER. THIS METHOD HAD AN ADVANTAGE OVER ONE USING A CONTRACTOR TO COMPLETE THE ENTIRE JOB IN THAT IT WOULD ALLOW SUNSHINE TO ESTABLISH AND TRAIN ITS OWN MINING CREW FOR EVENTUAL PRODUCTION AND ALSO WOULD EFFECT SOME SAVINGS IN DEVELOPMENT COSTS. ATTEMPTING THE ENTIRE JOB USING ONLY SUNSHINE CREWS WOULD NOT ONLY REQUIRE MORE EQUIPMENT AND MANPOWER THAN THAT NEEDED FOR EVENTUAL PRODUCTION BUT ALSO WOULD CERTAINLY LENGTHEN THE DEVELOPMENT PERIOD.

Conceptual Milling Plan

During the period of time beginning when Sunshine first acquired an interest in the 16-to-1 in 1964 and continuing through 1979, a number of metallurgical tests were conducted by a variety of labs in an effort to establish the best method for processing the 16-to-1 ores. As most of the tests were based on a very limited quantity of high-grade ore, a larger lot, some 20 tons, was collected from the surface ore stockpile in 1979 for the purpose of providing a better sample for additional testwork. Since the vein material from which this sample came was already heavily oxidized, it was assumed, and correctly so, that any further oxidation occuring on the dump would have been minimal. Thus, this large sample would be considered to be metallurgically representative of the ore from the 7000 level workings. The 20-ton sample was shipped to the Hazen Research testing facility where it was crushed, blended, and sampled to obtain an average assay for the lot. Fortunately, the grade of this material was very close to the projected grade of the orebody and test work was soon started. The results of this work indicated, as did those tests done previously, that both cyanidation and flotation were viable methods of recovery for this ore. In order to minimize the period of time from the go-ahead on construction until completion of the mill, a decision as to processing scheme had to be made during the conceptual planning phase. To delay engineering work until the vein was intersected at depth would have delayed plant start-up and environmental permitting by several months. An estimate of construction costs also

HAD TO BE COMPLETED DURING THIS PHASE AND, OBVIOUSLY, THE AMOUNT ESTIMATED DEPENDED ON WHICH SCHEME WAS CHOSEN.

SOME CONCERN WAS EXPRESSED AT THE TIME THAT PERMITTING COULD BE MORE DIFFICULT WITH A CYANIDE PLANT. A CHECK WITH THE APPROPRIATE STATE AND FEDERAL AGENCIES INDICATED THAT THIS WOULD NOT BE A PROBLEM. FLOTATION HAD AN ADVANTAGE OVER CYANIDATION IN THAT THERE WOULD BE SOME BASE METALS RECOVERED. SINCE COMBINED LEAD AND ZINC CONTENT AVERAGED LITTLE MORE THAN 1% OVERALL, HOWEVER, THIS ADVANTAGE WAS CONSIDERED INSIGNIFICANT. CHANGES IN PROCESSING CHARACTERISTICS OF THE ORE WITH DEPTH WAS ANOTHER CONCERN SHOULD A CYANIDE PROCESS BE THE CHOSEN SCHEME. METALLURGICAL TEST WORK DONE ON CORE SAMPLES FROM DEEP HOLES INDICATED COMPARABLE RECOVERIES, HOWEVER. IN ADDITION, NIVLOC RECORDS SHOWED EXCELLANT RECOVERIES BY CYANIDATION ON ORE WHICH WAS VERY SIMILAR TO OURS. BECAUSE RETURN-ON-INVESTMENT COMPARISONS OF THE TWO SCHEMES SHOWED VERY LITTLE DIFFERENCE AND BECAUSE MARKETING FLEXIBILITY WAS MUCH GREATER WITH A BULLION PRODUCT THAN WITH A LOW-GRADE CONCENTRATE, CYANIDATION WAS CHOSEN AS THE METHOD OF PROCESSING THE 16-TO-1 ORE. IT WAS PROPOSED THAT PRELIMINARY ENGINEERING WORK ON THE MILL BEGIN WITHIN TWO MONTHS FOLLOWING APPROVAL OF THE PROJECT. CERTAIN ASSUMPTIONS WERE MADE ON TYPES OF EQUIPMENT AND DESIGN FEATURES OF THE PLANT FOR THE PURPOSE OF DEVELOPING AN INITIAL CONSTRUCTION COST ESTIMATE. THIS ESTIMATE WOULD BE LATER REVISED AS ENGINEERING PROGRESSED AND FLOWSHEET REFINEMENTS MADE. MILL RECOVERY BY CYANIDATION WAS ESTIMATED AT 90.8% FOR SILVER AND 89.7% FOR GOLD FOR A MILLING RATE OF 150,000 TONS PER YEAR.

EVEN THOUGH MILL DESIGN ENGINEERING WOULD BEGIN SHORTLY FOLLOWING PROJECT APPROVAL, THE GO-AHEAD ON THE ACTUAL CONSTRUCTION OF THE MILL WOULD NOT BE GIVEN UNTIL THE VEIN WAS INTERSECTED AT DEPTH, PREDICTED TO TAKE PLACE IN EARLY 1981. THREE VERY IMPORTANT QUESTIONS WOULD BE ANSWERED AT THAT TIME:

- WHAT ARE THE GROUND CONDITIONS ADJACENT TO THE VEIN AND WILL GROUND CONDITIONS AT DEPTH SUPPORT THE CHOSEN MINING SYSTEM?
- ARE ACTUAL GRADES AND WIDTHS CONSISTANT WITH THOSE PROJECTED FROM DIAMOND DRILLING?
- ARE THE METALLURGICAL CHARACTERISTICS OF THE ORE CONSISTENT WITH THOSE DETERMINED FROM PREVIOUS TEST WORK?

A FAVORABLE RESPONSE TO THE ABOVE THREE QUESTIONS WOULD RESULT IN THE COMMENCEMENT OF MILL CONSTRUCTION. A NEGATIVE RESPONSE IN ANY ONE OF THE THREE WOULD RESULT IN ADDITIONAL ENGINEERING WORK AND POSSIBLY AN ENTIRE PROJECT RE-EVALUATION.

SELECTING A SITE FOR THE MILL AND TAILINGS DAM WERE VERY IMPORTANT DECISIONS AND WERE MADE DURING THE DEVELOPMENT OF THE CONCEPTUAL MILLING PLAN. CONSTRUCTION COST ESTIMATES WERE THEN DEVELOPED BASED ON THIS SELECTION. VARIOUS SITES WERE EXAMINED FOR BOTH THE MILL AND TAILINGS DAM FROM THE ASPECTS OF INITIAL COST, FUTURE EFFECTS ON OPERATING COSTS (HAULAGE AND TAILINGS DISPOSAL COSTS), AND ENVIRONMENTAL FACTORS. A SITE WAS FINALLY CHOSEN FOR BOTH WHICH WAS LOCATED APPROXIMATELY 3½ MILES FROM THE MINE AND WHICH ALLOWED FOR A MUCH LESS EXPENSIVE TAILINGS DAM AND MILLSITE

TO BE BUILT.

DEVELOPING A STRATEGY FOR LOCATING A DEPENDABLE AND SUFFICIENT QUANTITY OF WATER FOR USE IN MILL PROCESSING WAS THE NEXT LOGICAL STEP IN THE PLANNING PHASE. OBTAINING AN ADEQUATE SUPPLY OF WATER IN DESERT REGIONS CAN BE A VERY EXPENSIVE PROPOSITION AND THEREFORE DESERVES CAREFUL CONSIDERATION. EVEN THOUGH A HYDROGEOLOGIC STUDY TARGETING A NUMBER OF POSSIBLE WELL-DRILLING SITES WAS COMPLETED PRIOR TO THE PREPARATION OF CAPITAL COST ESTIMATES, THERE STILL REMAINED A GREAT DEAL OF RISK IN ARRIVING AT THESE ESTIMATES. FOR THIS REASON, TWO SEPARATE WATER SYSTEMS WERE PROPOSED IN ORDER TO INSURE THAT THE TOTAL ESTIMATED COST OF DEVELOPING MILL PROCESS WATER WOULD NOT BE EXCEEDED. THE PRIMARY SYSTEM INVOLVED A DEEP WELL LOCATED AT THE MILLSITE WHILE THE SECONDARY SYSTEM INVOLVED PIPING WATER FROM THE MINE, AS THERE APPEARED TO BE A DEPENDABLE SOURCE AT THAT LOCATION.

ALSO A PART OF THE CONCEPTUAL MILLING PLAN WAS THE LAYOUT OF THE ANCILLARY STRUCTURES AND SERVICES AT THE MILLSITE. BECAUSE OF ITS ACCESSIBILITY, THE MILLSITE WAS CHOSEN AS THE LOCATION FOR THE MAIN OFFICE. A SMALL PARTS WAREHOUSE, FENCED STORAGE YARD, CHANGEROOM FACILITIES, AND SHOP BUILDING WERE ALSO PLANNED FOR THIS SITE. COST ESTIMATES FOR THESE STRUCTURES WOULD BE QUITE APPROXIMATE AS NO PRELIMINARY ENGINEERING WORK WAS DONE ON THEM.

Other Planning Considerations

Providing dependable electrical power for the mine and mill was, of course, a very important consideration and one that would include a sizeable investment of capital. Since commercial power was available over five miles from the mill requiring a large up-front investment by Sunshine, and since power costs were relatively high (4.8¢/KWH) in the area, a second alternative was briefly examined. Producing our own power by diesel generation would have been significantly more expensive and therefore was dropped from further consideration. Due to maintenance reasons, it was proposed that power be supplied by a utility-owned line and two utility-owned sub-stations, one at the mine and one at the mill. This scheme necessitated the extra expense of a second metering point but it did eliminate the need for the people and equipment capable of working on high voltage lines, should a problem occur.

Early on in the planning stages of the project it became apparent that, in order to be able to attract quality personnel, something had to be done about the housing situation in Silver Peak. There was essentially no available housing in this town of 150 residents and it was felt that Tonopah was farther away than most people would be willing to drive to work. It was, therefore, proposed that Sunshine would provide a variety of housing alternatives to its employees including a boarding house, trailer park, house and mobile home rentals, and a lease/option program for double and single-wide units. To manage this program Sunshine

WOULD HIRE AN OUTSIDE CONTRACTOR WHO WAS EXPERIENCED IN THIS FIELD. PROPERTY ACQUISITION IN SILVER PEAK BEGAN PRIOR TO THE ANTICIPATED PROJECT AUTHORIZATION AND, AS A RESULT, SERVED TO MAINTAIN SOME DEGREE OF RATIONALITY IN THE LOCAL PROPERTY PRICES.

IN ORDER TO ACCURATELY PREPARE A CONSTRUCTION SCHEDULE AND THEREBY PREDICT A START-UP DATE, IT WAS IMPORTANT THAT ENVIRONMENTAL PERMITTING REQUIREMENTS BE INVESTIGATED EARLY ON IN THE CONCEPTUAL PLANNING PHASE. A FEASIBILITY REPORT COMPLETED IN JANUARY OF 1980 BY A CONSULTING FIRM, BEHRE DOLBEAR AND COMPANY, ADDRESSED THIS MATTER AND INDICATED THAT PERMITTING WOULD NOT BE CRITICAL PATH TO THIS PROJECT. PRIOR TO PROJECT APPROVAL, IT WAS, HOWEVER, NECESSARY TO BETTER DETAIL THESE REQUIREMENTS THROUGH CONSULTATIONS WITH APPROPRIATE COUNTY, STATE, AND FEDERAL AGENCIES. AS SUNSHINE DID NOT HAVE, AT THAT TIME, ANYONE EXPERIENCED IN ENVIRONMENTAL PERMITTING, A DECISION WAS MADE TO HIRE A CONSULTANT TO MANAGE THE PERMITTING PROCESS. THIS PROCESS BEGAN WITH AN ENVIRONMENTAL RECONNAISSANCE STUDY OF THE PROPERTY IN MARCH OF 1980. THE PURPOSE OF THIS STUDY WAS TO IDENTIFY NECESSARY PERMITS, APPROVALS, DATA REQUIREMENTS, AND AREAS OF ENVIRONMENTAL CONCERN, AND TO DEVELOP A PERMITTING STRATEGY. THE STRATEGY DOCUMENT WOULD IDENTIFY PERMITTING LEAD TIMES, KEY CONTRACTS, NECESSARY ENGINEERING AND ENVIRONMENTAL DATA, AND POTENTIAL PROBLEM AREAS. THIS DOCUMENT WAS COMPLETED IN MAY OF 1980 AND CONTAINED THE FOLLOWING INFORMATION REGARDING STATE AND FEDERAL REQUIREMENTS:

° SUNSHINE WOULD HAVE TO OBTAIN CERTAIN STATE AIR PERMITS. A

PERMIT TO CONSTRUCT AND A PERMIT TO OPERATE WOULD BE REQUIRED.

- SUNSHINE WOULD HAVE TO OBTAIN A STATE SURFACE WATER DISCHARGE PERMIT FOR THE MINE SITE AND A GROUNDWATER DISCHARGE PERMIT FOR THE TAILINGS POND.
- THE STATE DID NOT, AT THAT TIME, HAVE A SOLID WASTE PERMIT PROGRAM, BUT INTENDED TO SEEK AUTHORITY FOR THE FEDERAL RCRA PROGRAM.
- PLANS FOR THE CONSTRUCTION OF CERTAIN PROJECT SYSTEMS (TAILS DAM, WATER SUPPLY, SEWERAGE SYSTEM) REQUIRED STATE LEVEL PRECONSTRUCTION APPROVAL.
- THE PROPOSED 16-TO-1 FACILITY WOULD NOT REQUIRE FEDERAL PSD OR NONATTAINMENT NEW SOURCE REVIEW UNDER THE CLEAN AIR ACT.
- THE PROPOSED FACILITY WOULD NOT HAVE TO OBTAIN A FEDERAL NPDES WATER PERMIT UNDER THE CLEAN WATER ACT.
- THE PROPOSED FACILITY WOULD HAVE TO COMPLY WITH THE PROVISIONS OF THE RESOURCE CONSERVATION AND RECOVERY ACT, BUT IF IT WAS POSSIBLE TO SITE HAZARDOUS WASTE DISPOSAL FACILITIES (TAILING POND, BOILER ASH, ETC.) WITHIN CERTAIN CRITERIA, A "PERMIT-BY-RULE" WOULD BE ISSUED AND NO ENVIRONMENTAL REVIEW WOULD BE REQUIRED. A "PERMIT-BY-RULE" APPLICATION REQUIRED 180 DAYS FOR ISSUANCE.
- CERTAIN SURFACE RIGHT APPLICATIONS TO THE B.L.M.(RIGHT-OF-WAY, BORROW PITS, ETC.) WOULD INITIATE ENVIRONMENTAL REVIEW IN THE FORM OF AN ENVIRONMENTAL ANALYSIS. IT WAS NOT KNOWN IF ANY OF THESE APPLICATIONS WOULD BE REQUIRED.
- ALL WILDERNESS STUDY AREAS IN THE PROJECT VICINITY WERE

PROPOSED TO BE DROPPED FROM FURTHER WILDERNESS CONSIDERATION.

THERE WERE NO COUNTY PERMITS REQUIRED NOR WERE THERE ANY COUNTY ORDINANCES OR REGULATIONS APPLICABLE TO OUR PROJECT.

ESTIMATION OF CAPITAL COSTS

FOLLOWING COMPLETION OF THE CONCEPTUAL PLANNING FOR THE MINE, MILL AND OTHER IMPORTANT SEGMENTS OF THE PROJECT, THE PROCESS OF ESTIMATING CAPITAL COSTS COULD BEGIN. THIS ESTIMATE WAS DIVIDED INTO THREE MAIN PARTS; MINE DEVELOPMENT, MILLSITE CONSTRUCTION, AND GENERAL CONSTRUCTION AND DEVELOPMENT.

THE ESTIMATE FOR MINE DEVELOPMENT WAS MADE UP OF BOTH SURFACE AND UNDERGROUND COSTS. SURFACE COSTS WOULD BE INCURRED PRIMARILY DURING THE FIRST FEW MONTHS FOLLOWING AUTHORIZATION AND INCLUDED SUCH ITEMS AS PORTAL SITE PREPARATION, SHOP INSTALLATIONS, THE MINE WASTE WATER TREATMENT SYSTEM, THE MINE INDUSTRIAL WATER SYSTEM, ROADS, AND GENERATOR AND COMPRESSOR RENTALS. TOTAL MINE SURFACE DEVELOPMENT COSTS WERE ESTIMATED AT APPROXIMATELY ONE MILLION DOLLARS. UNDERGROUND DEVELOPMENT COSTS INCLUDED SUNSHINE'S AND AN ESTIMATE OF THE CONTRACTOR'S CHARGES FOR COMPLETING OVER 14,500 FEET OF CROSSCUT, DRIFT, RAISE, AND DRAWPOINT CONE DEVELOPMENT. ALSO INCLUDED WERE ESTIMATES OF UNDERGROUND EQUIPMENT COSTS AND INSTALLATION COSTS FOR MINE VENTILATION, PUMPING, COMPRESSED AIR, AND ELECTRICAL SYSTEMS. TOTAL UNDERGROUND DEVELOPMENT COSTS WERE ESTIMATED AT APPROXIMATELY SIX MILLION DOLLARS, PUTTING THE

TOTAL FOR MINE DEVELOPMENT AT SEVEN MILLION DOLLARS.

MILLSITE CONSTRUCTION COSTS CONSISTED PRIMARILY OF THOSE PERTAINING TO THE PROCESSING FACILITY AND TAILING DAM. AS VIRTUALLY NO ENGINEERING WORK HAD BEEN COMPLETED ON THE MILL PRIOR TO THE PREPARATION OF THE CONSTRUCTION AND DEVELOPMENT BUDGET, THIS PORTION OF THE COST ESTIMATE WOULD BE ONLY AN APPROXIMATION AT BEST. ONCE A TENTATIVE LIST OF MAJOR MILL EQUIPMENT HAD BEEN PREPARED ALONG WITH THEIR ESTIMATED COSTS, THE COST OF THE MILL WAS THAN CALCULATED ACCORDING TO A METHOD DEVELOPED BY A. L. MULAR AND DESCRIBED IN A PUBLICATION ENTITLED, MINERAL PROCESSING EQUIPMENT COSTS AND PRELIMINARY COST ESTIMATIONS, THE CANADIAN INSTITUTE OF MINING AND METALLURGY, 1978. CONSTRUCTION COSTS FOR THE TAILINGS DAM HAD BEEN PREVIOUSLY DEVELOPED BY DAMES AND MOORE DURING THE SITE SELECTION PROCESS AND THIS COST WAS USED IN THE MILL CONSTRUCTION COST ESTIMATE. THE TOTAL COST OF THE MILL AND TAILINGS DAM WAS ESTIMATED TO BE APPROXIMATELY EIGHT MILLION DOLLARS. IN ADDITION, COSTS OF CONSTRUCTION FOR THE OFFICE, WAREHOUSE, SHOP, AND LAB TOGETHER WITH CERTAIN MOBILE EQUIPMENT AND THE MILLSITE WATER SUPPLY SYSTEM WERE ESTIMATED AT AN ADDITIONAL ONE MILLION DOLLARS. THIS BROUGHT THE TOTAL FOR THE MILLSITE CONSTRUCTION SEGMENT OF THE PROJECT TO NINE MILLION DOLLARS.

GENERAL CONSTRUCTION AND DEVELOPMENT COSTS INCLUDED TWO MAJOR CAPITAL ITEMS; THE MAIN ELECTRICAL SERVICE INSTALLATION AND HOUSING. THESE AND OTHER MISCELLANEOUS COSTS ACCOUNTED FOR AN ESTIMATED TWO MILLION DOLLARS.

THE TOTAL OF THE ESTIMATED MINE, MILLSITE, AND GENERAL CONSTRUCTION AND DEVELOPMENT COSTS FOR THE 16-TO-1 PROJECT WAS APPROXIMATELY 18 MILLION DOLLARS AND WOULD BE EXPENDED OVER A 20-MONTH PERIOD BEGINNING IN JULY OF 1980. THE DEVELOPMENT OF THESE ESTIMATES TOOK LESS THAN 300 ENGINEERING MAN-HOURS TO COMPLETE AND, THEREFORE, WERE TO BE CONSIDERED PRELIMINARY. ONCE UNDERGROUND DEVELOPMENT WORK HAD STARTED AND ENGINEERING WORK ON THE MILL HAD BEGUN, THESE ESTIMATES WOULD BE REFINED AND A PROJECT RE-EVALUATION MADE. IN ORDER TO MITIGATE SOME OF THE RISK IN THIS ORIGINAL ESTIMATE, A 10% ERROR FACTOR WAS BUILT IN TO ALL COST CALCULATIONS. NO CONSIDERATION WAS GIVEN TO THE ESCALATION OF CAPITAL COST ITEMS OVER THE 20-MONTH CONSTRUCTION PERIOD, HOWEVER.

ESTIMATION OF OPERATING COSTS

THE ESTIMATED OPERATING COST FOR THE LIFE OF THE 16-TO-1 WAS COMPOSED OF THREE PARTS; MINING, MILLING, AND ADMINISTRATIVE. THESE COSTS WERE NOT ESCALATED OVER TIME FOR INFLATION AS IT WAS ASSUMED THAT PRECIOUS METAL PRICES WOULD INCREASE APPROXIMATELY AT THE SAME RATE.

BESIDES THE BLASTHOLE STOPING OPERATIONS, THE ESTIMATED MINING COST ALSO INCLUDED SOME ADDITIONAL STOPE DEVELOPMENT TOGETHER WITH DIAMOND DRILLING, MINE SUPPORT ACTIVITIES, AND HAULAGE OF ORE TO THE MILL. THESE COSTS WERE DEVELOPED BASED ON ESTIMATES OF LABOR REQUIREMENTS AND PROJECTIONS OF THE CONSUMPTION OF ITEMS SUCH AS DIESEL FUEL, ELECTRICAL POWER, EXPLOSIVES, TIRES, AND REPAIR

PARTS. THIS NUMBER COMPARED FAVORABLY WITH COSTS WHICH HAD BEEN MADE AVAILABLE FROM OTHER SIMILAR MINING OPERATIONS.

MILL OPERATING COSTS WERE MADE UP PRIMARILY OF LABOR, REAGENT, AND POWER COSTS. REAGENT USAGE HAD BEEN ESTIMATED IN PREVIOUS METALLURGICAL TEST WORK AND INCLUDED CYANIDE, LIME AND ZINC DUST.

ADMINISTRATIVE COSTS INCLUDED NOT ONLY STAFF LABOR CHARGES BUT ALSO INSURANCE, HOUSING, AND OUTSIDE MANAGEMENT EXPENSES. THIS COST WAS ADDED TO THE MINING AND MILLING COSTS IN ORDER TO ARRIVE AT A TOTAL OPERATING COST FOR USE IN THE FINAL CASH FLOW CALCULATIONS.

FINANCIAL ANALYSIS

THE CASH FLOW CALCULATION AND SUBSEQUENT FINANCIAL ANALYSIS WAS THE FINAL STEP INVOLVED IN THE PREPARATION OF THE PROJECT FEASIBILITY REPORT. THE RESULTS OF THESE COMPUTATIONS WOULD BE THE BASIS ON WHICH SUNSHINE MANAGEMENT WOULD MAKE THE DECISION WHETHER OR NOT TO PROCEED WITH THE PROJECT.

BEFORE A CASH FLOW CALCULATION FOR THE PROJECT COULD BE DONE, CERTAIN ADDITIONAL ASSUMPTIONS HAD TO BE MADE. ALTHOUGH OTHER METHODS MAY HAVE BEEN MORE ADVANTAGEOUS, THE UNIT-OF-PRODUCTION METHOD OF DEPRECIATION WAS EMPLOYED. A FEDERAL TAX RATE OF 48% WAS USED TOGETHER WITH A NET PROCEEDS AND PROPERTY TAX RATE OF 2.541 DOLLARS PER ONE HUNDRED DOLLARS. A 10% INVESTMENT TAX CREDIT

WOULD BE TAKEN DURING THE FIRST FULL YEAR OF PRODUCTION. A COST DEPLETION OF 15% WAS USED AS ALLOWED BY THE IRS FOR GOLD AND SILVER MINES. USING THE ABOVE ASSUMPTIONS TOGETHER WITH ESTIMATED CAPITAL AND OPERATING COSTS AND A PREDICTED SILVER AND GOLD PRICE, A TOTAL NET CASH FLOW WAS CALCULATED FOR THE LIFE OF THE PROJECT. AFTER DEDUCTING ESTIMATED ROYALTY PAYMENTS TO THE PARTNER, MID-CONTINENT, A NET CASH FLOW TO SUNSHINE WAS DETERMINED FOR EACH YEAR FROM 1980 THROUGH 1989. USING THESE FIGURES, A NET PRESENT VALUE (I=20%), PAYBACK PERIOD, AND DCF-ROI WAS CALCULATED.

THE RESULTS OF THESE CALCULATIONS INDICATED A DCF-ROI OF 38% WITH A PAYBACK PERIOD OF LESS THAN TWO YEARS FROM START OF PRODUCTION. THE RETURN ON INVESTMENT OF 38%, IT WAS FELT, WAS HIGH ENOUGH TO COMPENSATE FOR THE SUBSTANTIAL RISK INVOLVED WITH A NEW UNDERGROUND MINE. THEREFORE, IT WAS RECOMMENDED THAT SUNSHINE PROCEED WITH THE DEVELOPMENT OF THE 16-TO-1. FOR REASONS OF THE OBVIOUS ECONOMICS OF THE PROJECT TOGETHER WITH A CORPORATE STRATEGY AND RESOLVE TO REDUCE THE VULNERABILITY OF THE COMPANY TO PROBLEMS AT ANY ONE MINE, SUNSHINE'S BOARD OF DIRECTORS AUTHORIZED THE DEVELOPMENT OF THE 16-TO-1 ON JUNE 3, 1980.

PART IV

CONSTRUCTION AND DEVELOPMENT

GENERAL

THE ORIGINAL CONSTRUCTION AND DEVELOPMENT SCHEDULE FOR THE 16-TO-1 PROJECT CALLED FOR WORK TO BEGIN IN JULY OF 1980 WITH MILL START-UP ON A PRODUCTION BASIS BY MARCH 1, 1982. THE FIRST INCOME FROM SALES OF BULLION WOULD BE REALIZED IN APRIL OF THAT YEAR. THIS GENERAL SCHEDULE, DEVELOPED IN MAY OF 1980, WAS MET ALMOST TO THE DAY AS THE FIRST SHIPMENT OF BULLION WAS MADE ON APRIL 1, 1982.

IN MARCH OF 1981, FOLLOWING COMPLETION OF APPROXIMATELY 1/3 OF THE PLANNED UNDERGROUND DEVELOPMENT WORK TOGETHER WITH ALL PRELIMINARY AND SOME DETAILED ENGINEERING WORK ON THE MILL, A REVISED CAPITAL BUDGET WAS COMPLETED. THIS NEW BUDGET INDICATED THAT INCREASED EXPENDITURES, PRIMARILY IN THE AREA OF MINE DEVELOPMENT, WOULD BE NECESSARY IN ORDER TO COMPLETE THE PROJECT ACCORDING TO SCHEDULE. THE TOTAL ESTIMATED MINE DEVELOPMENT COST WAS REVISED TO NINE MILLION DOLLARS FROM SEVEN MILLION DOLLARS WHILE GENERAL CONSTRUCTION AND DEVELOPMENT WAS INCREASED TO THREE MILLION DOLLARS, DUE MAINLY TO INCREASED HOUSING AND MAIN ELECTRICAL SERVICE COSTS. SURPRISINGLY, THE ESTIMATED MILLSITE CONSTRUCTION COST OF NINE MILLION DOLLARS HAD NOT CHANGED, RESULTING IN A NEW ESTIMATED CAPITAL COST FOR THE PROJECT OF 21 MILLION DOLLARS. THESE FIGURES WERE THEN USED TO COMPLETE A SECOND CASH FLOW CALCULATION AND FINANCIAL ANALYSIS. THE RETURN ON INVESTMENT FOR THE PROJECT

HAD, OF COURSE, BEEN REDUCED BUT STILL REMAINED AT A RESPECTABLE LEVEL OF OVER 30%. RISKS INHERENT IN PROJECTING A RETURN AS PART OF THE MAY 1980 FINANCIAL ANALYSIS HAD, HOWEVER, ALSO BEEN REDUCED. THIS WAS DUE TO THE FOLLOWING CONDITIONS:

- A CONSIDERABLE AMOUNT OF ENGINEERING WORK HAD BEEN COMPLETED ON THE MILL RESULTING IN MORE RELIABLE CONSTRUCTION COST ESTIMATES.
- DEVELOPMENT COST DATA FOR THE MINE HAD BEEN ESTABLISHED THROUGH SEVERAL MONTHS OF EXPERIENCE. THESE COSTS COULD BE USED TO BETTER ESTIMATE THOSE ASSOCIATED WITH THE AMOUNTS OF DEVELOPMENT REMAINING.
- THE 16-TO-1 VEIN HAD BEEN INTERSECTED AT DEPTH AND FOUND TO BE CONSISTENT IN ALL RESPECTS TO WHAT HAD BEEN PREDICTED.

A DECISION WAS, THEREFORE, MADE IN APRIL OF 1981 TO PROCEED WITH THE COMPLETION OF THE CONSTRUCTION AND DEVELOPMENT OF THE 16-TO-1.

MINE DEVELOPMENT

FOLLOWING PROJECT AUTHORIZATION IN JUNE OF 1980, A MINING CONTRACTOR HAD TO BE CHOSEN TO COMPLETE THE UNDERGROUND DEVELOPMENT WORK FROM THE LOWER PORTAL SITE. FOUR COMPANIES, EACH EXPERIENCED IN RAMP DEVELOPMENT WORK, WERE ASKED TO BID ON THE JOB. AMERICAN MINE SERVICES WAS AWARDED THE CONTRACT IN JULY AND WOULD BE ON THE JOBSITE BY LATE AUGUST. THE CONTRACT CALLED FOR DRIVING RAMPS, DRIFTS, LATERALS, AND CROSSCUTS, 12 FEET BY 14 FEET IN SIZE, ON

A COST PER FOOT BASIS. GROUND CONTROL ACTIVITIES AND EXTRA WORK DUE TO UNFORESEEN PROBLEMS, SUCH AS WATER, WERE TO BE PAID FOR AT COST PLUS A FIXED PERCENTAGE. RAISE BORING ACTIVITIES WERE SUB-CONTRACTED OUT ON A COST PER FOOT BASIS.

DEVELOPMENT WORK BY THE CONTRACTOR BEGAN ON SCHEDULE WITH THE FIRST FULL FACE ROUND BEING TAKEN IN THE LOWER LEVEL DECLINE IN LATE SEPTEMBER. PRIOR TO THIS, SUNSHINE CREWS HAD FINISHED THE INSTALLATION OF THE INDUSTRIAL WATER SYSTEM FOR BOTH THE UPPER AND LOWER PORTALS, INITIAL SITE CLEARING FOR THE LOWER PORTAL, AND THE PIPING SYSTEM TO THE WASTEWATER SETTLING POND. THIS SETTLING POND, TOGETHER WITH A NEW ACCESS ROAD, HAD BEEN COMPLETED BY A LOCAL CONTRACTOR EARLIER IN THE SUMMER. SLABBING OF THE OLD 7000 LEVEL CROSSCUT BY SUNSHINE CREWS BEGAN IN EARLY OCTOBER.

THROUGHOUT THE ENTIRE MINE DEVELOPMENT PERIOD, ALL WORK DONE BY BOTH SUNSHINE'S AND THE CONTRACTOR'S CREWS WERE DIRECTLY OR INDIRECTLY SUPERVISED BY SUNSHINE'S MINE SUPERINTENDENT. ALL COST PLUS WORK TURNED IN BY THE CONTRACTOR WAS THERBY APPROVED BEFORE PAYMENT WAS MADE. THE QUALITY OF THE CONTRACTOR'S WORK WAS ALSO CAREFULLY SCRUTINIZED BY SUNSHINE'S SUPERINTENDENT. ONLY ONE MAJOR DESIGN CHANGE WAS EFFECTED DURING DEVELOPMENT OF THE MINE. THIS WAS A CHANGE IN SIZE OF ALL HEADINGS DRIVEN BY THE CONTRACTOR TO APPROXIMATELY 15 FEET BY 15 FEET AND WAS DONE IN ORDER TO BETTER ACCOMODATE THE HAULAGE EQUIPMENT WHICH WAS IN USE. THIS CHANGE, HOWEVER, DID NOT RESULT IN INCREASED COSTS PER FOOT OVER THAT WHICH HAD BEEN ORIGINALLY BID.

DEVELOPMENT OF THE 16-TO-1 MINE USING THE CONCEPT OF A CONTRACTOR WORKING FROM ONE PORTAL WHILE THE OWNER'S CREWS WORKED CONCURRENTLY FROM A SECOND PORTAL RESULTED IN A VERY SATISFACTORY ARRANGEMENT AND ENABLED THE UNDERGROUND DEVELOPMENT WORK TO BE COMPLETED SLIGHTLY AHEAD OF SCHEDULE. HOLE-THROUGH WAS MADE, TYING UPPER AND LOWER LEVELS TOGETHER UNDERGROUND ON APRIL 21, 1982, FOLLOWING COMPLETION OF A TOTAL OF 15,880 FEET OF DRIFTING AND CROSSCUTTING AND 1630 FEET OF RAISE BORING. FOLLOWING HOLE-THROUGH, THE CONTRACTOR WAS RELEASED FROM ANY FURTHER WORK WITH SUNSHINE'S CREWS CONTINUING TO DRIVE THE REMAINING ORE AND WASTE DEVELOPMENT HEADINGS.

THE ONLY MAJOR PROBLEMS EXPERIENCED DURING MINE DEVELOPMENT WERE WATER-RELATED. EVEN THOUGH TEST HOLES WERE MAINTAINED AHEAD OF THE FACES IN MOST HEADINGS, GROUND WATER ENCOUNTERS PROVED TO BE HIGHLY UNPREDICTABLE. THERE WERE CASES WHERE THREE TEST HOLES WOULD BE DRILLED IN A FACE WITH ALL BEING DRY; THE NEXT ROUND WOULD BE SHOT AND ANOTHER 100 GALLONS PER MINUTE WOULD BE PICKED UP. WATER INFLOWS DURING DEVELOPMENT PEAKED OUT AT OVER 1000 GALLONS PER MINUTE. THIS FLOW, IN CONJUNCTION WITH RECURRING PUMP PROBLEMS AND AN INADEQUATE TEMPORARY SUMP INSTALLATION, CAUSED A VERY SIGNIFICANT AMOUNT OF DOWNTIME EXPENSE DURING DEVELOPMENT. IT WAS STILL FAR CHEAPER, HOWEVER, THAN IT WOULD HAVE BEEN TO DRIVE A HORIZONTAL CROSSCUT TO THE VEIN TO ELIMINATE THE NEED FOR PUMPS. GROUND PROBLEMS WERE ALMOST NON-EXISTENT DURING THE DEVELOPMENT OF THE MINE. THE FIRST 1000 FEET OF RAMP DRIVEN FROM THE LOWER PORTAL REQUIRED NO GROUND SUPPORT WHATSOEVER. TIMBER WAS NEEDED IN A FEW

VERY RARE INSTANCES WITH ROCKBOLTING AND MATTING BEING REQUIRED FOR LESS THAN HALF THE FOOTAGE DRIVEN. WHERE BOLTING WAS NECESSARY, THE SPLIT SET TYPE ROCKBOLT WAS USED ALMOST EXCLUSIVELY. GROUND CONDITIONS IN THE VEIN WERE ALSO VERY GOOD, WITH UNSUPPORTED WIDTHS OF OVER 30 FEET BEING COMMON ON THE UPPER LEVELS.

ACTUAL MINE DEVELOPMENT COSTS FOR THE 16-TO-1 DURING THE PERIOD FROM JULY OF 1980 THROUGH MARCH OF 1982 WERE APPROXIMATELY 11 MILLION DOLLARS, OR TWO MILLION DOLLARS OVER THE REVISED ESTIMATE DONE IN EARLY 1981. THIS WAS DUE PRIMARILY TO THE 3800 FEET OF ADDITIONAL DRIFTING DONE BY SUNSHINE AND THE CONTRACTOR AND WHICH WAS NECESSARY BECAUSE OF THE DISCOVERY OF INCREASED RESERVES ON 7000 LEVEL, THE LONGER DRAWPOINT CROSSCUTS CAUSED BY A TURN IN THE VEIN, AND ADDITIONAL VEIN DEVELOPMENT DONE AHEAD OF SCHEDULE. ALSO CONTRIBUTING TO THE OVER-RUN WAS ALMOST 500 THOUSAND DOLLARS IN WATER-CAUSED DELAYS AND OVER 250 THOUSAND DOLLARS IN ADDITIONAL ROAD-BUILDING COSTS. INCLUDED IN THE 11 MILLION DOLLAR FIGURE PREVIOUSLY MENTIONED IS ALMOST TWO MILLION DOLLARS IN MINING EQUIPMENT WHICH WAS PURCHASED FOR EVENTUAL USE IN PRODUCTION. THE AMOUNT ACTUALLY SPENT STRICTLY ON UNDERGROUND DEVELOPMENT BY SUNSHINE AND THE CONTRACTOR, INCLUDING ALL EXTRA WORK AND DELAY COSTS, WAS APPROXIMATELY 6½ MILLION DOLLARS. DIVIDING THIS FIGURE BY THE TOTAL FOOTAGE INCLUDING RAISE BORING, OF 17,500 FEET, RESULTS IN AN AVERAGE COST PER FOOT FOR ALL UNDERGROUND DEVELOPMENT OF ABOUT 370 DOLLARS PER FOOT.

Mill Design

The design process for the 16-to-1 mill was divided into two parts; preliminary and detailed engineering. Preliminary engineering work included the development of a detailed flowsheet with mass balances, a list of equipment, a general layout of the plant and equipment, and an estimate of capital and operating costs for the plant. This work began in July of 1980 following the selection of Industrial Design Corporation as the engineering firm. This firm was selected on the basis of experience in this field and their estimated cost per drawing relative to other companies. In addition to the engineering firm, a consultant with many years of operating experience was also hired to assist in the preliminary and, eventually, the detailed engineering work. Before beginning preliminary engineering on this project, several design parameters had to be defined. These parameters were based on additional metallurgical work which had recently been completed by Hazen together with certain design criteria supplied by Sunshine and were as follows:

- Mine will produce 500 t/day, 6 days/week and truck to coarse ore hopper at mill.
- Mine ore will pass over a 24" x 24" square grizzly at the mine.
- Crusher will operate 6 days/week, 1 shift/day = approximately 100 tons/hour (6 hours/day and 28% design factor).
- Average mine rate = 3000 tons/week, hauled by truck, 1 shift/day, 6 days/week.

- AVERAGE CRUSHER RATE = 3000 ÷ 6 ÷ 6 = 83 TONS/HOUR.
- AVERAGE MILL RATE = 3000 ÷ 7 ÷ 24 = 18 TONS/HOUR.
- CRUSHER PRODUCT-CLOSED CIRCUIT ON 5/8" X 2" SCREEN = APPROXIMATELY 80% -5/8".
- BALL MILL PRODUCT = 80% MINUS 200M
- LEACHING TIME = 48 HRS AVERAGE + DESIGN FACTOR 15% = 55 HRS.
- GRINDING INDEX = 16.5.
- SPECIFIC GRAVITY OF ORE = 2.75.
- BULK DENSITY OF CRUSHED ORE = 85 #/CUBIC FOOT.
- BALL MILL CIRCULATING LOAD WILL BE 250%.
- THICKENER RATE = 2.0 SQUARE FOOT/TON/24 HOURS (INCLUDES FLOCCULANT).

IT IS OBVIOUS FROM THE ABOVE THAT THE DESIGN MILL THROUGHPUT WAS CONSIDERABLY MORE THAN THAT NEEDED FOR A PRODUCTION OF 150,000 TONS PER YEAR. A DESIGN THROUGHPUT OF 550 TONS PER DAY WOULD, IT WAS FELT, INSURE THAT THE DESIRED THROUGHPUT OF 430 TONS PER DAY, 350 DAYS PER YEAR, WOULD BE ACHIEVED.

OTHER DESIGN DECISIONS WHICH WERE MADE DURING THE PRELIMINARY ENGINEERING PHASE INCLUDED THE ARRANGEMENT OF THE CRUSHING PLANT AND WHETHER TO USE FILTRATION OR COUNTER-CURRENT DECANTATION. THE CRUSHING PLANT LAYOUT WAS SOMEWHAT UNIQUE IN THAT, BY LOCATING THE VIBRATING SCREEN DIRECTLY ABOVE THE FINE ORE BIN INSTEAD OF IN THE CRUSHING BUILDING, THE NEED FOR A THIRD CONVEYOR WAS ELIMINATED. THE DECISION TO GO WITH A CCD SYSTEM INSTEAD OF FILTRATION WAS MADE, EVEN THOUGH TEST WORK ON THE PULP MATERIAL INDICATED GOOD

FILTERING RATES, BECAUSE IT WAS FELT THAT A CCD SYSTEM WOULD BE LESS LABOR INTENSIVE TO OPERATE AND MAINTAIN. ADDITIONAL DESIGN AND CONSTRUCTION COSTS WERE SAVED BY USING PRE-ENGINEERED BUILDINGS FOR ALL MILLSITE STRUCTURES EXCEPT FOR THE CRUSHING PLANT. INDUSTRIAL DESIGN COMPLETED THE PRELIMINARY ENGINEERING REPORT IN SEPTEMBER AND IT INDICATED THAT THE TOTAL CAPITAL COSTS FOR THE MILL PLANT WOULD PROBABLY NOT EXCEED THAT WHICH WAS ESTIMATED IN THE ORIGINAL CONSTRUCTION AND DEVELOPMENT BUDGET. AS IT WAS ALSO IMPORTANT TO DEVELOP A BETTER ESTIMATE OF THE TAILINGS DAM COSTS IN ADDITION TO PLANT COSTS, A CONSULTANT, DAMES AND MOORE, WAS HIRED SOON AFTER PRELIMINARY ENGINEERING WORK BEGAN. SOILS STUDIES WERE PERFORMED IN THE AREA OF THE PROPOSED DAM ALONG WITH LOCATING A SUITABLE CLAY LINING MATERIAL. AS THE TAILINGS DAM WAS TO BE LOCATED IN A RELATIVELY SEVERE EARTHQUAKE ZONE, PROVISIONS FOR A SEISMIC ACCELERATION WERE ALSO MADE IN THE DESIGN. THE RESULTS OF THEIR INVESTIGATIONS AND PROPOSED DESIGN SPECIFICATIONS WERE COMPLETED IN NOVEMBER OF 1980 AND PRESENTED TO THE STATE ENGINEER'S OFFICE FOR APPROVAL. A PERMIT TO CONSTRUCT BY CENTERLINE CONSTRUCTION METHODS WAS ISSUED IN APRIL OF 1981.

DETAILED ENGINEERING WORK ON THE 16-TO-1 MILL BEGAN IN LATE NOVEMBER OF 1980 FOLLOWING THE DECISION TO CONTINUE USING INDUSTRIAL DESIGN FOR THIS NEXT PHASE OF THE PROJECT. A MILL SUPERINTENDENT WAS HIRED BY SUNSHINE IN ORDER THAT A PERSON WHO WOULD EVENTUALLY OPERATE THE PLANT COULD HAVE INPUT INTO THE DESIGN AND BECOME INTIMATELY FAMILIAR WITH THE PLANT'S LAYOUT AND CONSTRUCTION. THIS INDIVIDUAL WORKED CLOSELY WITH THE ENGINEERING FIRM THROUGHOUT

THE DETAILED DESIGN PHASE AND UNTIL THE DECISION WAS MADE TO PROCEED WITH CONSTRUCTION, AT WHICH TIME THIS PERSON RELOCATED IN SILVER PEAK. AS PROCUREMENT OF MAJOR EQUIPMENT FOR THE MILL WAS IDENTIFIED EARLY AS POSSIBLY BEING CRITICAL PATH SHOULD THE GO-AHEAD ON CONSTRUCTION BE GIVEN, EQUIPMENT BIDS WERE SOLICITED IN ANTICIPATION OF THIS BEGINNING IN JANUARY OF 1981. THE POSSIBILITY OF OBTAINING SOME MAJOR PIECES OF USED QUIPMENT WAS INVESTIGATED BUT IT WAS DISCOVERED THAT GOOD QUALITY USED EQUIPMENT AVAILABLE AT THAT TIME WAS NOT SIGNIFICANTLY LOWER IN PRICE THAN NEW EQUIPMENT. A DECISION WAS, THEREFORE, MADE TO PURCHASE ALL NEW EQUIPMENT FOR THE MILL AND THEREBY, HOPEFULLY, MAXIMIZE MILL AVAILABILITIES DURING THE FIRST FEW YEARS OF OPERATION.

THE PURPOSE OF THE DETAILED ENGINEERING PHASE OF THE PROJECT WAS TO COMPLETE ALL NECESSARY DESIGN COMPUTATIONS AND PROVIDE THE HUNDREDS OF WORKING DRAWINGS NEEDED TO BUILD THE MILL. THESE ACTIVITIES CONTINUED ON A LIMITED BASIS WELL INTO THE ACTUAL CONSTRUCTION PHASE. IN ADDITION TO THE DETAILED DESIGN WORK, THE ENGINEERING FIRM ALSO PROVIDED ALL EQUIPMENT PROCUREMENT SERVICES TOGETHER WITH CONTRACT SPECIFICATIONS AND BID SOLICITATION FOR THE VARIOUS PORTIONS OF THE CONSTRUCTION. INDUSTRIAL DESIGN WAS ULTIMATELY HIRED TO PROVIDE CONSTRUCTION MANAGEMENT SERVICES FOR THE PROJECT. THE TOTAL COSTS FOR ALL PRELIMINARY AND DETAILED ENGINEERING, EQUIPMENT PROCUREMENT, AND CONTRACT AND CONSTRUCTION MANAGEMENT WAS APPROXIMATELY ONE MILLION DOLLARS.

Mill Construction

Site preparation work began at the millsite during the last week of April 1981, only a month following the decision to proceed with construction. This was possible due to the fact that bids for portions of the early work had already been received prior to the decision to proceed in March. As this was a relatively small job, it was felt that a general contractor, as such, would not be required. Instead, Sunshine acted as their own general contractor during the construction phase and, utilizing a construction manager supplied by the engineering firm, sub-contracted out the various portions of the project. A total of six different contractors were used to complete the entire job, including the office, lab, warehouse, shop, and tailings dam, in a little over ten months time. The crushing plant was commissioned in December of 1981 while actual mill start-up was on February 8, 1982.

Prior to the decision to proceed with construction of the mill, a limited amount of pre-construction development work was required at the millsite. A source of potable and process water of sufficient quantity and quality had to be developed in the vacinity of the mill. A consultant, Hydrosearch Inc., was hired to identify possible wellsites and a drilling program was begun in September of 1980. In January of 1981, a successful well, 1300 feet deep, was finally completed and a pump was installed. In order to provide sufficient storage of water for eventual construction purposes, including a concrete batch plant and compacted

FILL WORK, THE MAIN 170,000 GALLON STORAGE TANK WAS ERECTED IN FEBRUARY OF 1981. ALSO COMPLETED IN FEBRUARY WAS A 40-UNIT TEMPORARY TRAILER PARK LOCATED ADJACENT TO THE MILLSITE. THE PURPOSE OF THIS FACILITY WAS TO HOUSE THE EMPLOYEES OF THE VARIOUS CONTRACTORS DURING THE CONSTRUCTION PERIOD.

TOTAL MILLSITE CONSTRUCTION COSTS FOR THIS PROJECT WERE APPROXIMATELY NINE MILLION DOLLARS OR ESSENTIALLY THE SAME AS THAT WHICH WAS ESTIMATED IN MAY OF 1980 AND AGAIN IN MARCH OF 1981. NO MAJOR PROBLEMS WERE ENCOUNTERED DURING CONSTRUCTION OF THE MILL AND SUNSHINE WAS QUITE PLEASED WITH THE PERFORMANCE OF ALL CONTRACTORS INVOLVED IN THE PROJECT.

OTHER DEVELOPMENT AND CONSTRUCTION ACTIVITIES

CONSTRUCTION OF THE NINE MILE LONG, 24.6 K.V. POWER LINE FROM SILVER PEAK TO THE MILLSITE AND MINE BEGAN IN MAY OF 1981, FOLLOWING OBTAINMENT OF A RIGHT-OF-WAY FROM THE B.L.M. THE LINE AND SUBSTATIONS WERE COMPLETED BY SIERRA PACIFIC POWER COMPANY CREWS IN MID-JULY AT A TOTAL COST OF SLIGHTLY OVER 800 THOUSAND DOLLARS. SUNSHINE WILL RECOUP APPROXIMATELY HALF OF THIS AMOUNT OVER THE LIFE OF THE MINE, WITHOUT INTEREST, IN THE FORM OF ANNUAL REFUNDS.

THE DEVELOPMENT OF HOUSING FACILITIES IN SILVER PEAK BEGAN SHORTLY AFTER PROJECT AUTHORIZATION WAS GIVEN IN JUNE OF 1980. THE INITIAL CONSTRUCTION CONSISTED OF A 21-SPACE TRAILER PARK, A SMALL BOARDING HOUSE, AND FOUR MODULAR HOUSES. AN APARTMENT

COMPLEX TOGETHER WITH SEVERAL DOUBLE-WIDE MOBILE HOMES WERE ADDED LATER. ONE IMPORTANT FEATURE OF THE HOUSING PROGRAM WAS A VERY AFFORDABLE LEASE/OPTION ARRANGEMENT WHEREBY AN EMPLOYEE COULD COME TO OWN A MOBILE HOME FOLLOWING EXPIRATION OF THE SEVEN-YEAR TERM OF THE LEASE. SUNSHINE'S GOAL IN THE ADMINISTRATION OF THE ENTIRE HOUSING PROGRAM WAS TO BREAK EVEN ON AN OPERATING COST BASIS, AND ALL RENTAL RATES WERE ESTABLISHED WITH THIS IN MIND. THE CONTRACTOR WHICH WAS HIRED TO MANAGE THE HOUSING PROGRAM WAS EVENTUALLY RELEASED AS IT WAS LEARNED THAT, BECAUSE OF THE SMALL SIZE OF THE OPERATION, IT WAS MUCH MORE EFFICIENT FOR THE COMPANY TO MANAGE IT USING THEIR OWN PERSONNEL. BESIDES INVESTING A CONSIDERABLE AMOUNT OF MONEY IN HOUSING IN SILVER PEAK, APPROXIMATELY 1.8 MILLION DOLLARS, SUNSHINE ALSO HELPED BRING ABOUT MUCH NEEDED IMPROVEMENTS IN THE CITY WATER SUPPLY SYSTEM. BY DONATING SEVERAL THOUSANDS OF DOLLARS IN ENGINEERING WORK AND COORDINATING WITH COUNTY OFFICIALS, RECURRING WATER SHORTAGES BECAME A THING OF THE PAST.

SUNSHINE UTILIZED THE SERVICES OF TWO FIRMS, TRC ENVIRONMENTAL CONSULTANTS AND, LATER, ENVIRONMENTAL MANAGEMENT SERVICES COMPANY TO MANAGE THE ENVIRONMENTAL PERMITTING PROCESS WHICH BEGAN IN 1980 AND CONTINUED INTO 1982 (FIGURE 5). FEDERAL APPROVALS INCLUDED THE BUREAU OF LAND MANAGEMENT'S 3809 PLAN OF OPERATIONS, CONSTRUCTION BORROW MATERIAL PURCHASE AGREEMENT OF THE TAILINGS POND LINING, AND RIGHT-OF-WAY FOR THE TRANSMISSION LINE. BECAUSE OF THE SMALL SIZE OF THE OPERATION, THE ENVIORNMENTAL PROTECTION AGENCY ISSUED AN AIR EMISSIONS NON-APPLICABILITY DETERMINATION

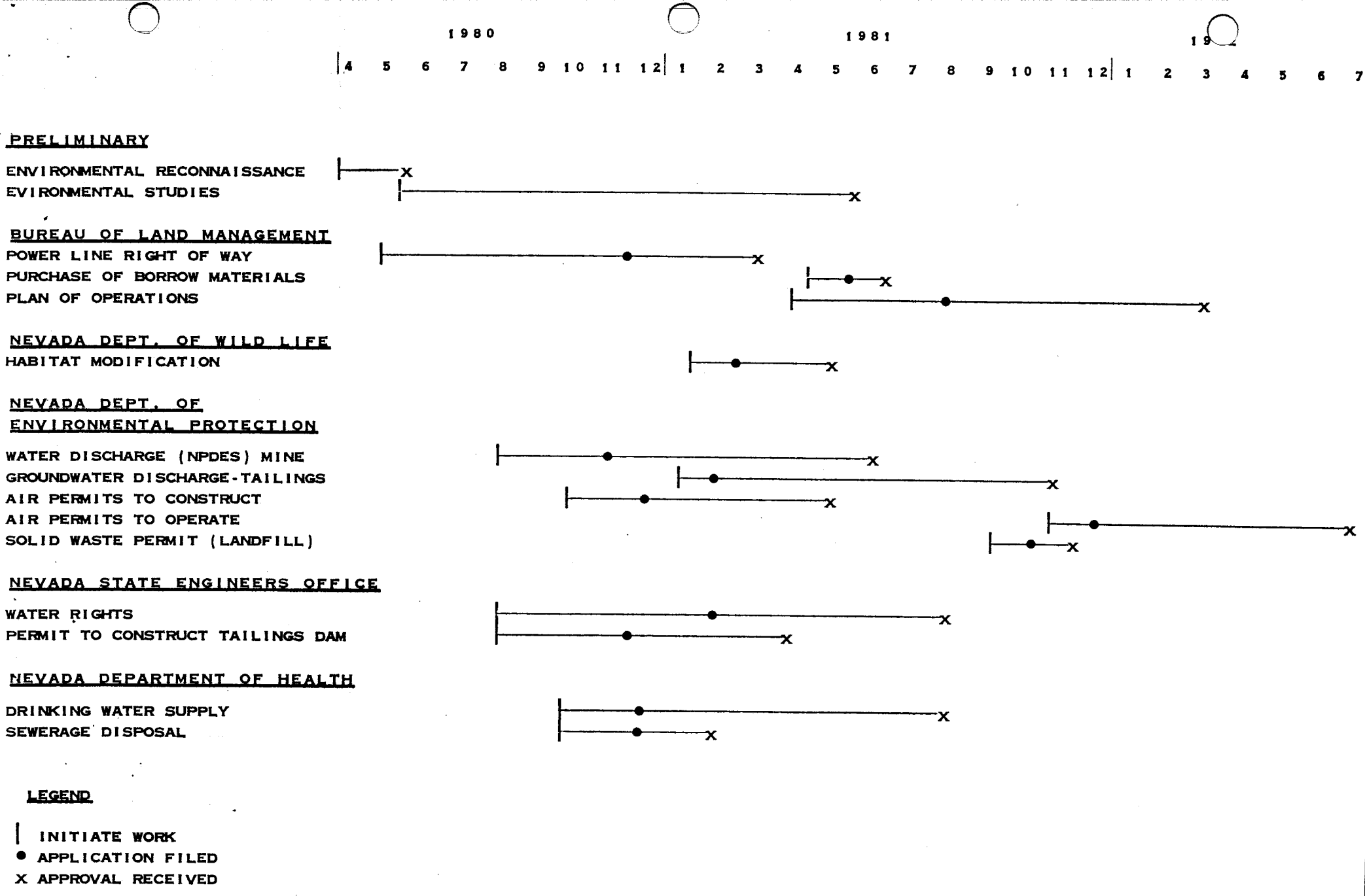

1980
1981
19
4 5 6 7 8 9 10 11 12 1 2 3 4 5 6 7 8 9 10 11 12 1 2 3 4 5 6 7
PRELIMINARY
ENVIRONMENTAL RECONNAISSANCE
EVIRONMENTAL STUDIES
BUREAU OF LAND MANAGEMENT
POWER LINE RIGHT OF WAY
PURCHASE OF BORROW MATERIALS
PLAN OF OPERATIONS
NEVADA DEPT. OF WILD LIFE
HABITAT MODIFICATION
NEVADA DEPT. OF ENVIRONMENTAL PROTECTION
WATER DISCHARGE (NPDES) MINE
GROUNDWATER DISCHARGE-TAILINGS
AIR PERMITS TO CONSTRUCT
AIR PERMITS TO OPERATE
SOLID WASTE PERMIT (LANDFILL)
NEVADA STATE ENGINEERS OFFICE
WATER RIGHTS
PERMIT TO CONSTRUCT TAILINGS DAM
NEVADA DEPARTMENT OF HEALTH
DRINKING WATER SUPPLY
SEWERAGE DISPOSAL
LEGEND
| INITIATE WORK
● APPLICATION FILED
X APPROVAL RECEIVED

FIGURE 5

UNDER THE CLEAN AIR ACT AND REVIEWED THE MINE SURFACE WATER DISCHARGE APPLICATION UNDER THE FEDERAL CLEAN WATER ACT. THE STATE OF NEVADA ISSUED AIR EMISSION PERMITS TO CONSTRUCT AND OPERATE THE PROJECT, SURFACE AND GROUNDWATER DISCHARGE PERMITS, A SANITARY LANDFILL PERMIT, DAM CONSTRUCTION AND OPERATING PERMIT, AND APPROVED THE DESIGN OF THE PUBLIC DRINKING WATER AND SANITARY SEPTIC SYSTEM. ALTHOUGH ESMERALDA COUNTY DID NOT HAVE LAND USE OR ENVIRONMENTAL REGULATIONS APPLICABLE TO THE PROJECT, SUNSHINE KEPT APPROPRIATE COUNTY OFFICIALS APPRAISED OF ANY ACTIONS AFFECTING THE COUNTY. IN ORDER TO GAIN THE NECESSARY PERMITS AND APPROVALS, SUNSHINE WAS COMMITTED TO PERFORM MONITORING OF VARIOUS PARAMETERS PRIOR TO AND DURING CONSTRUCTION, DURING OPERATION, AND AS PART OF THE ABANDONMENT PLAN. THESE MONITORING PROGRAMS RESULTED IN INSURING THAT THE 16-TO-1 WILL BE OPERATED IN AN ENVIRONMENTALLY SAFE FASHION. THE KEY TO SUCCESSFULLY PERMITTING THIS PROJECT WAS THE EARLY IDENTIFICATION OF ENVIRONMENTALLY SENSITIVE ISSUES, THE DEVELOPMENT OF AN ENVIRONMENTAL STRATEGY, AND CLOSE COMMUNICATION BETWEEN SUNSHINE, THE ENVIRONMENTAL CONSULTANTS AND THE VARIOUS REGULATORS. TOTAL COSTS FOR ALL PERMITTING WAS LESS THAN 70 THOUSAND DOLLARS. THE 16-TO-1 IS PROOF THAT A NEW MINING AND MILL COMPLEX CAN BE DEVELOPED IN A TIMELY AND COST-EFFECTIVE FASHION EVEN IN TODAY'S REGULATORY CLIMATE.

PART V
OPERATIONS

START-UP

THE TRANSFORMATION OF THE MINE FROM A DEVELOPMENT TO AN OPERATING MODE WAS ONE WHICH TOOK PLACE GRADUALLY AS THE NUMBER OF ORE HEADINGS INCREASED AND AS THE PRODUCTION DRILLING PROGRAM GOT UNDERWAY. UNTIL OCTOBER OF 1982, ALL ORE PROCESSED THROUGH THE MILL HAD COME FROM SUBLEVEL DEVELOPMENT HEADINGS. PRODUCTION STOPING ACTIVITIES THEREAFTER BEGAN TO SUPPLY MILL FEED AS DRILLING AND BLASTING CREWS WERE DEVELOPING NEW TECHNIQUES AND PROCEDURES. AS PREDICTED, DILUTION IN THE STOPE SOON BECAME THE MOST IMPORTANT CONCERN AND IT WAS QUICKLY LEARNED THAT LOADING AND BLASTING PROCEDURES GREATLY AFFECTED EFFORTS TO CONTROL THIS PROBLEM. DAMAGED WALLS WERE USUALLY THE RESULT OF TOO MUCH EXPLOSIVE BEING USED WHILE TOO LITTLE SOMETIMES CAUSED BENCHES AND PILLARS TO FORM IN THE ORE. ALMOST ONE MONTH OF PRODUCTION WAS LOST DUE TO A LARGE BENCH FORMING AS A RESULT OF A MISSED HOLE. FOLLOWING THIS, THE PROCEDURE OF DOUBLE-PRIMING WAS ADOPTED AS IT WAS BELIEVED THAT A PRIMACORD MAY HAVE BEEN CUT OFF IN A HOLE. ANOTHER CAUSE OF DILUTION WAS UNEXPECTED GEOLOGIC STRUCTURES LOCATED IN THE WALLS OF THE STOPE WHICH WEAKENED THE ROCK SURROUNDING THE VEIN. THIS WAS ESPECIALLY A PROBLEM IN THE HANGING WALL AND RANDOM PILLARS WERE LEFT IN THE VEIN IN AN ATTEMPT TO ARREST THE SITUATION.

REGARDLESS OF HOW MUCH PLANNING AND ENGINEERING WORK IS DONE ON ANY NEW MILL FACILITY, START-UP PROBLEMS ARE ALMOST ALWAYS ENCOUNTERED. THE 16-TO-1 FACILITY WAS NO EXCEPTION. THE PROBLEMS ENCOUNTERED HERE, HOWEVER, WERE RELATIVELY MINOR AND EASY TO REMEDY. ACTUAL START-UP OF THE MILL WAS ON FEBRUARY 8, 1982, WITH MOST OF THAT MONTH BEING DEVOTED TO TESTING AND ADJUSTMENT OF MILL EQUIPMENT AND LOADING OF THE THICKENERS AND LEACH TANKS WITH PULP. WASTE MATERIAL WAS PROCESSED FOR THIS PURPOSE WITH THE FIRST ORE BEING INTRODUCED INTO THE MILL IN MARCH. IT BECAME IMMEDIATELY APPARENT THAT A PROPER FLOCCULENT SYSTEM WAS NEEDED TO ACHIEVE ACCEPTABLE SETTLING RATES AND CLARITY IN OUR THICKENER TANKS. THIS SYSTEM WAS QUICKLY DESIGNED AND INSTALLED AND RESULTED IN ALMOST AN IMMEDIATE IMPROVEMENT IN OUR THICKENER CLARITIES. A SECOND PROBLEM WHICH WAS ENCOUNTERED ALSO INVOLVED CLARITY. A KEY REQUIREMENT IN ANY CYANIDE MILL USING MERRILL-CROWE PRECIPITATION IS THAT THE PREGNANT SOLUTION BE OF "SPARKLING" CLARITY PRIOR TO PRECIPITATION OF THE PRECIOUS METALS WITH ZINC DUST. THE CARTRIDGE FILTERS WHICH WERE ORIGINALLY SUPPLIED FOR THIS PURPOSE DID NOT PERFORM AS WAS HOPED. THEY WERE SUBSEQUENTLY REPLACED BY THE MORE CONVENTIONAL PRESSURE LEAF FILTERS AND THE RESULTS WERE DRAMATIC. MILL RECOVERY IMPROVED AND THE PRECIPITATE QUALITY INCREASED FROM 30% TO OVER 90% PRECIOUS METALS. START-UP PROBLEMS WITH MAJOR MILL EQUIPMENT WERE, FORTUNATELY, NON-EXISTENT AS ALL UNITS CHECKED OUT PERFECTLY.

Present Operation

The 16-to-1 Mine output presently stands at approximately 755 tons per day on a five day per week basis. With a workforce of 28 miners, this translates into a productivity of 27 tons per underground man-shift. Because daily output requirements from the mine have increased along with higher mill capacity by about 50%, additional mine haulage equipment was recently purchased. One additional five cubic yard load-haul-dump unit together with one 25-ton underground haulage truck enable the mine to now produce over 1200 tons in a day if it is required. All production loading and haulage equipment currently in use at the 16-to-1 are manufactured by Wagner Mining Equipment. Availabilities on the LHD units have been good, considering the amount of usage, at about 85%. Availabilities on the trucks have been excellent at over 95%. Presently, all ore and waste development headings are being driven with a Tamrock two-boom Minematic jumbo equipped with Tamrock HLR 438 drills. Availability, approximately 90%, along with productivity have both been excellent during the two years since it was purchased. This unit will typically drill out a 45-hole round, 1-3/4 inch diameter holes 10½ feet deep, in andesite, in one hour and 15 minutes. Our blasthole stoping program is being carried out using two TRW Mission 6200-U down-the-hole drilling rigs, each being equipped with an 800 CFM, 250 PSI booster compressor. Each drilling unit is equipped with a diesel tramming engine and is normally operated off an electric jumper from the compressor which is, in turn, fed from a nearby transformer through a trailing cable. This

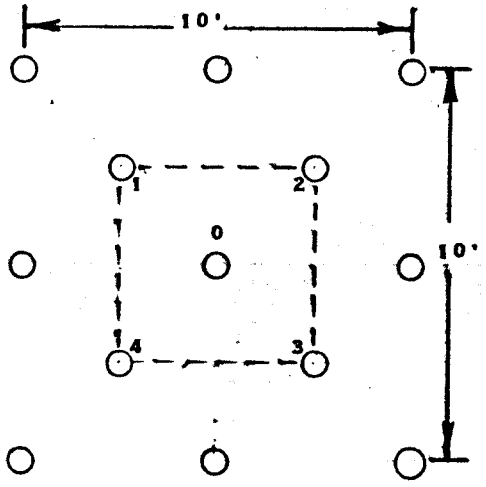
TRAILING CABLE USES THE SAME STYLE PLUG-ENDS AS THE JUMBO. THE UNIT CAN ALSO DRILL WITH MINE AIR USING NO EXTERNAL POWER SOURCE AS IT HAS THE CAPABILITY OF SWITCHING FROM ELECTRIC OVER HYDRAULIC TO DIESEL OVER HYDRAULIC TO OPERATE THE ROTATION MOTOR AND FEED. HOLES DRILLED IN THIS MANNER ARE KEPT SHORT AND OF A LIMITED NUMBER SINCE THIS PROCEDURE IS EXTREMELY SLOW. UNDER NORMAL OPERATING CONDITIONS, PENETRATION RATES OF BETWEEN 50 AND 70 FEET PER HOUR IN A QUARTZ-CALCITE VEIN ARE COMMON. ACCURACY, SO IMPORTANT IN THIS TYPE OF MINING, HAS ALSO BEEN GOOD WITH DEVIATIONS USUALLY NO MORE THAN ONE FOOT PER 100 FEET OF HOLE.

ELECTRICAL POWER IS DISTRIBUTED THROUGHOUT THE MINE BY MEANS OF PORTABLE, SELF-CONTAINED OHIO BRASS TRANSFORMERS WHICH NOT ONLY STEP-DOWN VOLTAGES BUT ALSO FEED THROUGH POWER TO OTHER TRANSFORMERS. THESE TRANSFORMERS ARE EQUIPPED WITH THE LATEST IN GROUND-FAULT MONITORING EQUIPMENT AND CAN BE TAKEN OFF LINE WITHIN SECONDS SINCE ALL CONNECTIONS ARE THE PLUG TYPE. AS DRILLING ON ONE SUBLEVEL IS COMPLETED, THESE UNITS MAY BE EASILY MOVED TO ANOTHER LOCATION FOR COMMENCEMENT OF DRILLING ACTIVITIES THERE.

THE MINE VENTILATION SYSTEM HAS BEEN DESIGNED SO THAT ALL WORKING AREAS WITHIN THE MINE ARE SUPPLIED WITH FRESH, UNCONTAMINATED AIR. ONE MAIN MINE FAN LOCATED IN THE UPPER LEVEL CROSSCUT PRESSURIZES THE ENTIRE MINE AND PROVIDES APPROXIMATELY 186,000 CFM OF FRESH AIR. AIR IS DISTRIBUTED THROUGHOUT THE MINE USING VENTILATION RAISES AND REGULATIONG DOORS.

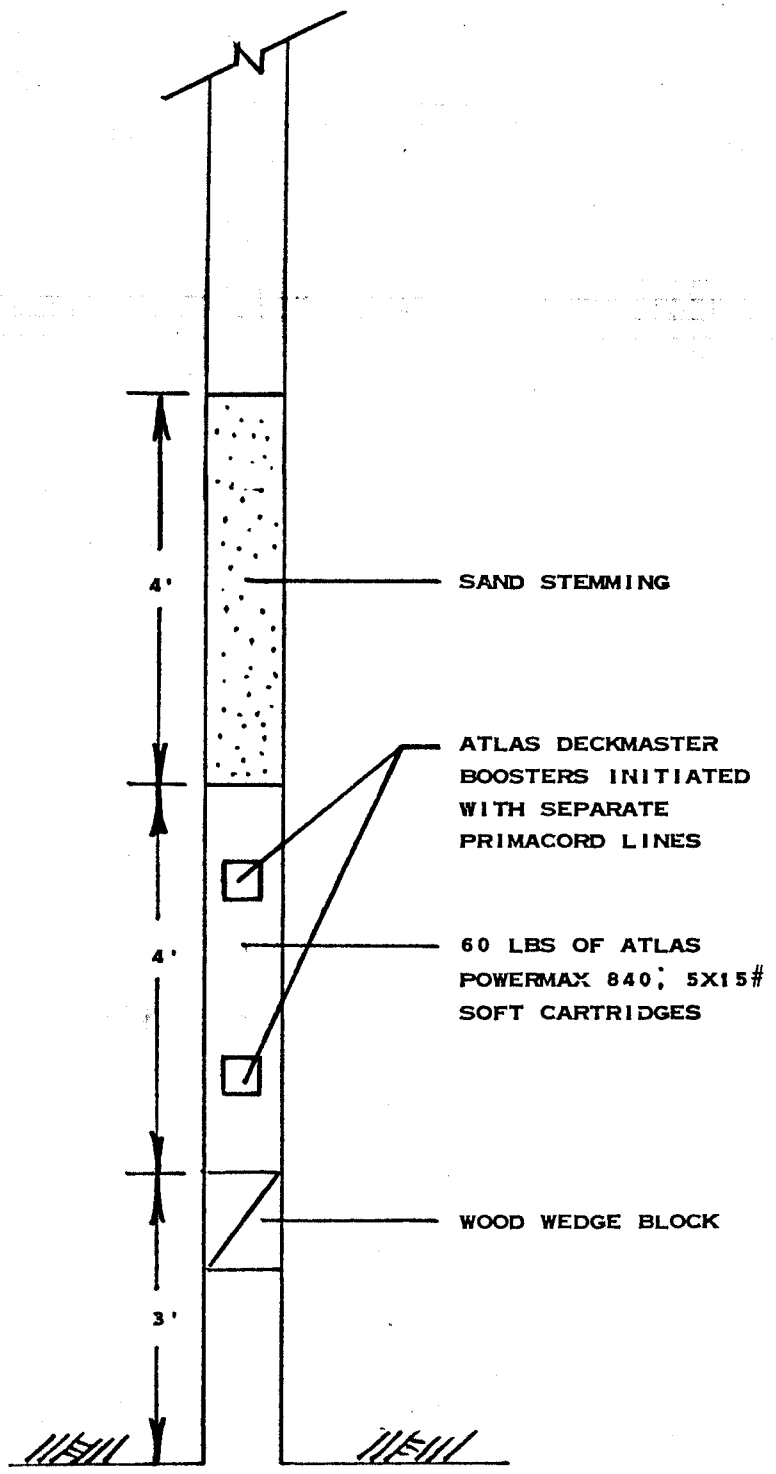
THE MINE GETS ITS DRILL WATER FROM STRUCTURES WHICH HAVE BEEN TAPPED WITHIN THE MINE AND PUMPED INTO STORAGE TANKS LOCATED ON THE SURFACE. THESE STRUCTURES ARE EXPECTED TO BE PRODUCTIVE FOR SEVERAL YEARS. THE MINE WATER DRAINAGE SYSTEM CONSISTS OF A MAIN SUMP INSTALLATION AT THE LOWEST LEVEL OF THE MINE WITH DUAL SETTLING AREAS WHICH CAN BE MUCKED OUT WITH AN LHD UNIT. DURING NORMAL OPERATION, ONE SETTLING DAM DECANTS INTO A SECOND WHICH DECANTS INTO A PUMP PIT. FROM HERE, TWO 100 HORSEPOWER VERTICAL TURBINE PUMPS LIFT THE WATER TO THE SURFACE.

MINING AT THE 16-TO-1 BY BLASTHOLE STOPING METHOD BEGINS BY EXCAVATING A SLOT RAISE BETWEEN SUBLEVELS AT THE EXTREMITY OF THE ORE BLOCK. THE RAISE PROVIDES THE FIRST FREE FACE TO WHICH THE PRODUCTION BLASTHOLES MAY BREAK TO AND IT IS COMPLETED USING DROP-RAISING TECHNIQUES. DROP-RAISING IS AN APPLICATION OF THE VERTICAL CRATER RETREAT (VCR) MINING SYSTEM AND INVOLVES THE PLACEMENT OF CHARGES NEAR THE COLLARS OF LARGE DIAMETER HOLES WHICH, WHEN SHOT, RESULT IN CRATERING OUT TO THE NEAREST FACE. PERIMETER HOLES ARE THEN SHOT INTO THIS CRATER IN ORDER TO SQUARE UP THE NEW FACE. DROP-RAISING AT THE 16-TO-1 BEGINS BY DRILLING 13 - 6½ INCH HOLES ON A 10 FOOT BY 10 FOOT PATTERN IN THE VEIN (FIGURE 6). ONLY THE FIVE CENTER HOLES ARE LOADED ACCORDING TO VCR TECHNIQUES. THE REMAINING HOLES ARE USED FOR SQUARING UP THE RAISE AND FOR LOADING AS A VCR HOLE IN THE EVENT OTHER HOLES ARE PLUGGED. THE FIVE CENTER HOLES ARE LOADED WITH A HIGH DENSITY EMULSION EXPLOSIVE TO WITHIN THREE FEET OF THE COLLAR, DOUBLE-PRIMED WITH MILLISECOND



PLAN VIEW

DROP RAISE DRILLING PATTERN



SECTION VIEW

DROP RAISE LOADING SCHEME

FIGURE 6

DELAY BOOSTERS, AND SHOT ONE LIFT AT A TIME. BLASTING CREWS ARE FREQUENTLY ABLE TO SHOOT TWO - SIX TO SEVEN FOOT LIFTS IN A SHIFT. FOLLOWING MEASUREMENT OF EACH BLASTED HOLE, THIS PROCESS IS REPEATED UNTIL THE RAISE IS HOLED THROUGH.

ONCE THE DROP RAISE IS COMPLETED, THE DRILLING AND BLASTING OF PRODUCTION BLASTHOLES BEGINS. THESE HOLES ARE DRILLED AT A MINIMUM OF THREE HOLES ACROSS FROM FOOTWALL TO HANGINGWALL AND ARE PARALLEL TO THE TRUE DIP OF THE VEIN. ROW BURDEN VARIES FROM SEVEN FEET TO TEN FEET, DEPENDING ON VEIN WIDTH. POWDER FACTORS ARE APPROXIMATELY 1.7 POUNDS PER TON IN NARROW AREAS AND LESS THAN THAT IN WIDER ZONES. PRODUCTION BLASTHOLES ARE DECK LOADED WITH AN EMULSION EXPLOSIVE IN CARTRIDGE FORM AND DOUBLE-PRIMED USING TIME DELAY SLIDING BOOSTERS (FIGURE 7). BLASTING PROCEEDS FROM THE DROP RAISE TOWARDS THE ACCESS DRIFTS WHICH CROSSCUT THE VEIN APPROXIMATELY MIDWAY ALONG THE STRIKE OF THE OREBODY. ENOUGH MUCK IS TAKEN OUT OF THE DRAWPOINTS BELOW TO EXPOSE EACH VERTICAL FREE FACE PRIOR TO BLASTING. STOPING PROCEEDS IN THIS MANNER FROM ONE SUBLEVEL TO THE NEXT WITH THE LOWEST ORE BLOCK BEING MINED OUT FIRST. IN AREAS OF BAD GROUND, PILLARS MAY BE LEFT TO SUPPORT THE WALLS AND A NEW DROP RAISE EXCAVATED. ONCE THE OREBODY HAS BEEN STOPED OUT, THE ORE WILL BE DRAWN EVENLY SO AS TO KEEP THE TOP OF THE BROKEN MUCKPILE WITHIN THE STOPE AS UNIFORM AS POSSIBLE. ANY SUBSEQUENT CAVED MATERIAL, THEN, WILL HOPEFULLY NOT START SHOWING UP IN THE DRAWPOINTS UNTIL MOST OF THE BROKEN ORE HAS BEEN REMOVED. THE METHOD OF STOPING PRESENTLY BEING EMPLOYED AT THE 16-TO-1 REQUIRES A GREAT AMOUNT OF PARTICIPATION FROM BOTH ENGINEERING AND GEOLOGY

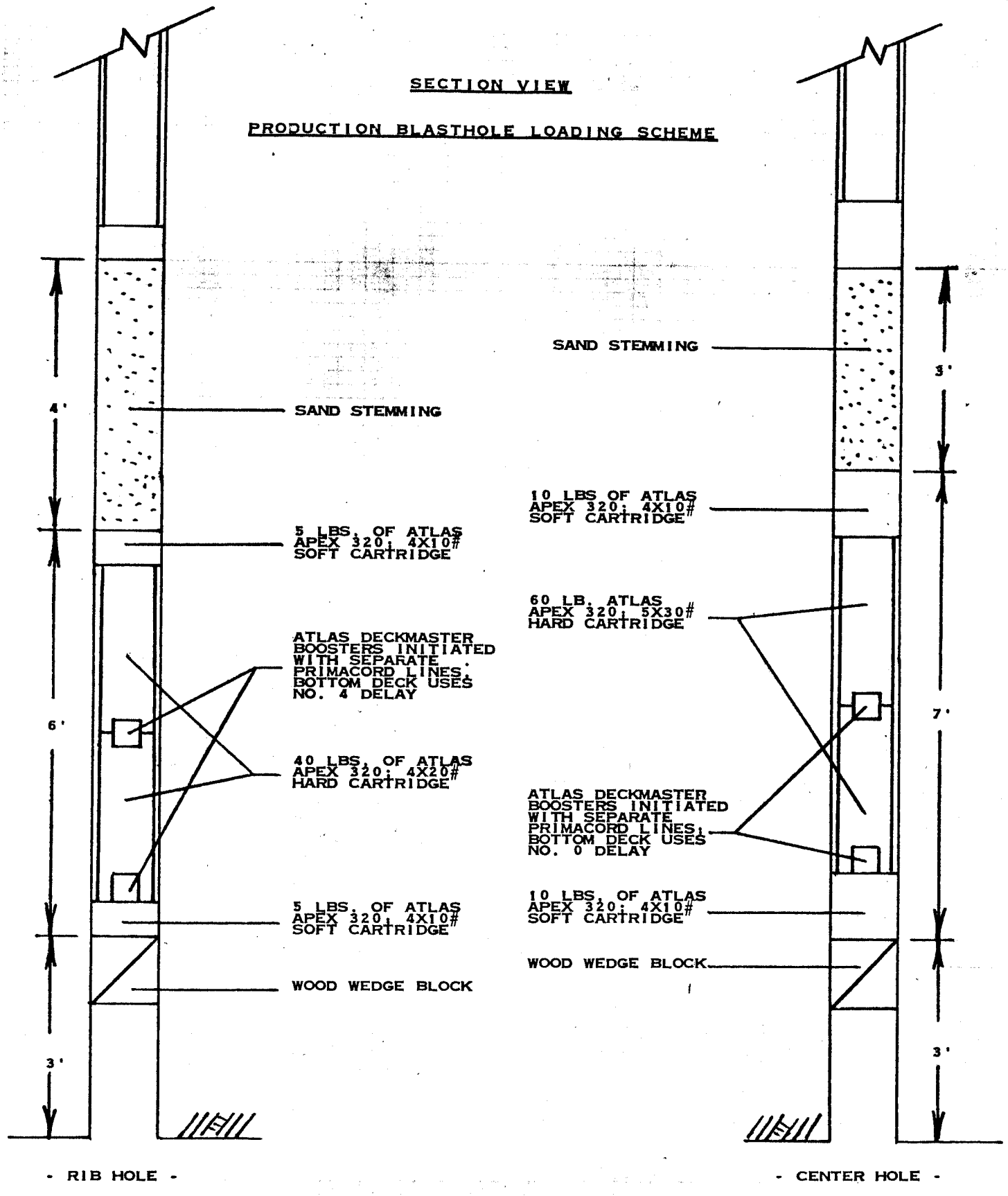

SECTION VIEW
PRODUCTION BLASTHOLE LOADING SCHEME
4'
SAND STEMMING
5 LBS. OF ATLAS APEX 320; 4X10# SOFT CARTRIDGE
6'
ATLAS DECKMASTER BOOSTERS INITIATED WITH SEPARATE PRIMACORD LINES; BOTTOM DECK USES NO. 4 DELAY
40 LBS. OF ATLAS APEX 320; 4X20# HARD CARTRIDGE
5 LBS. OF ATLAS APEX 320; 4X10# SOFT CARTRIDGE
WOOD WEDGE BLOCK
3'
- RIB HOLE -
SAND STEMMING
3'
10 LBS OF ATLAS APEX 320; 4X10# SOFT CARTRIDGE
60 LB. ATLAS APEX 320; 5X30# HARD CARTRIDGE
7'
ATLAS DECKMASTER BOOSTERS INITIATED WITH SEPARATE PRIMACORD LINES; BOTTOM DECK USES NO. 0 DELAY
10 LBS. OF ATLAS APEX 320; 4X10# SOFT CARTRIDGE
WOOD WEDGE BLOCK
3'
- CENTER HOLE -

FIGURE 7

DEPARTMENTS TO BE SUCCESSFUL. EACH ROW OF HOLES IS DESIGNED BY FIRST SELECTING THE PROPER BURDEN, BASED ON WIDTH, FOLLOWED BY PLOTTING THE LOCATION OF THE NEW ROW ON A PLAN MAP. A GEOLOGIC SECTION IS DRAWN THROUGH THE VEIN WITH THE PROPOSED DRILL HOLES THEN POSITIONED ACCORDINGLY. A MAP SHOWING THE HOLE COLLAR LOCATIONS TOGETHER WITH THE DIPS IS GIVEN TO THE DRILL CREW FOR USE IN SETTING UP THE MACHINE. LINE PLUGS ARE ALSO SET BY THE MINE SURVEYOR FOR EACH ROW OF HOLES. DRILL CUTTINGS FROM EACH HOLE ARE SAVED WITH THIS MATERIAL BEING SUBSEQUENTLY LOGGED BY THE GEOLOGIST. THIS INFORMATION IS PLOTTED ON THE ORIGINAL DRILL SECTION AND IS SOMETIMES USED TO DICTATE WHICH PORTION OF A HOLE IS TO BE LOADED WITH EXPLOSIVES.

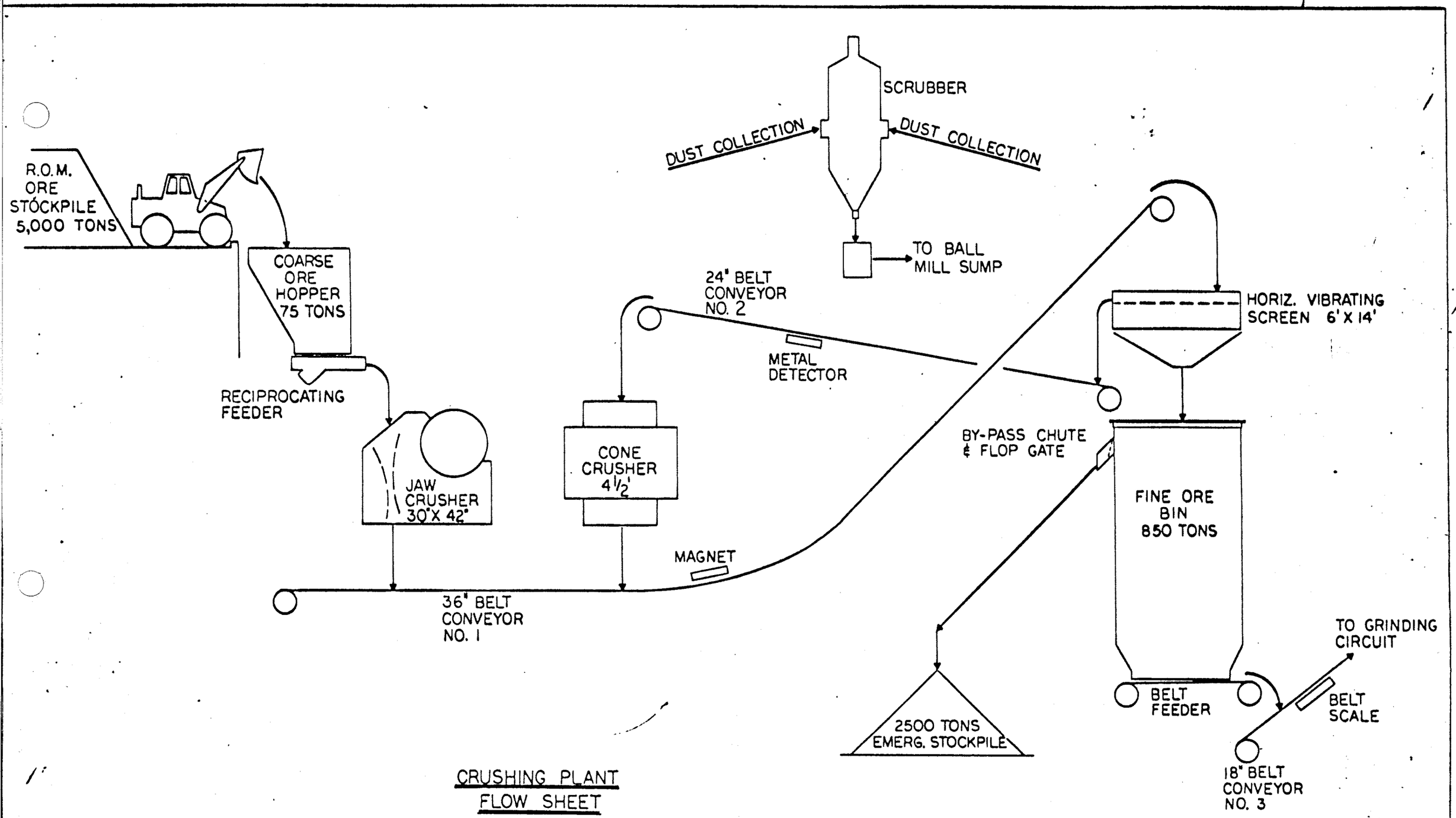
SINCE UNDERGROUND MINING OPERATIONS BEGAN, VARIOUS DESIGN AND OPERATING PROBLEMS HAVE BECOME EVIDENT. ONE OF THE MOST IMPORTANT HAS BEEN THAT OF EXCESS MINE DILUTION. THE PRIMARY CAUSE OF THIS, RECENTLY, HAS BEEN THE EXISTENCE OF STRUCTURES IN THE HANGINGWALL WHICH ALLOW LARGE BLOCKS OF UNSUPPORTED WALLROCK TO OCCASIONALLY FALL INTO THE STOPE. BECAUSE OF THE UNPREDICTABLE NATURE OF THIS PROBLEM, A POLICY HAS RECENTLY BEEN ADOPTED WHICH INVOLVES LEAVING ORE PILLARS IN THE STOPE AT REGULAR INTERVALS. THIS PROGRAM SHOULD EFFECT A SIGNIFICANT REDUCTION IN STOPE DILUTION OVER THE LIFE OF THE OREBODY. ANOTHER PROBLEM, ONE OF DESIGN, INVOLVES THE LOCATION OF THE MAIN MINE SUMP AND PUMPING STATIONS. THIS FACILITY IS PRESENTLY HANDLING APPROXIMATELY 700 GALLONS PER MINUTE ON A 24-HOUR BASIS. ALTHOUGH THERE ARE SPARE PUMPS AND BACKUP GENERATOR POWER IS AVAILABLE, THIS PUMPING STATION IS

NEVER-THE-LESS IN A VERY VULNERABLE LOCATION SHOULD A MAJOR PROBLEM ARISE. BECAUSE IT IS NEAR THE TRUCK LOADING DOCK, THE SUSPENDED SOLIDS LOAD IS ALSO VERY HIGH REQUIRING FREQUENT AND TIME-CONSUMING CLEANUP OF THE SUMPS. A BETTER LOCATION FOR THE MINE SUMP AND PUMPING STATION WOULD HAVE BEEN ON THE UNDERCUT LEVEL, APPROXIMATELY 50 FEET ABOVE ITS PRESENT LOCATION. HIGH VOLUME-LOW HEAD SUMP PUMPS COULD THEN HAVE BEEN USED TO PUMP WATER FROM THE BOTTOM OF THE MINE TO THE PUMPING STATION. THIS WOULD HAVE RESULTED IN CLEANER WATER TO THE VERTICAL TURBINE PUMPS AND LESS SUSEPTABILITY TO FLOODING. ROAD MAINTENANCE, ESPECIALLY IN THE MAIN HAULAGE RAMP, WAS ALSO A MAJOR PROBLEM FOR MANY MONTHS. THIS WAS MAINLY DUE TO THE SMALL AMOUNT OF WATER PRESENT IN THE RAMP TOGETHER WITH THE NATURE OF THE WASTE MATERIAL BEING USED AS ROAD BED. THIS PROBLEM WAS ALL BUT ELIMINATED BY HAVING COARSE GRAVEL HAULED IN AND APPLYING IT TO THE BAD SECTION OF RAMP. THIS MATERIAL WAS ESSENTIALLY GRAVEL PLANT REJECT, WITH THE SIZE RANGING FROM TWO TO FOUR INCHES, AND WAS FREE OF FINES. ONCE APPLIED, THIS MATERIAL HAD A DRAMATIC EFFECT ON THE QUALITY OF THE ROAD SURFACE.

THE 16-TO-1 MILL THROUGHPUT IS CURRENTLY AT APPROXIMATELY 650 TONS PER OPERATING DAY WITH A MILL AVAILABILITY OVER 95%. THE MILL OPERATES THREE SHIFTS PER DAY, SEVEN DAYS PER WEEK WHILE THE CRUSHING PLANT RUNS TWO SHIFTS PER DAY, FIVE DAYS PER WEEK. EMPLOYMENT IN THE MILL AND CRUSHING PLANT IS 15, NOT INCLUDING MAINTENANCE PERSONNEL. PRODUCTION CURRENTLY RANGES BETWEEN 75,000 AND 85,000 OUNCES OF SILVER PER MONTH AND 700 TO 1000 OUNCES OF

GOLD IN A DORE' WHICH IS TYPICALLY OVER 99.5% PRECIOUS METALS. RECOVERIES, BASED ON MILL HEAD GRADES, HAVE BEEN EXCELLENT AT 90.3% FOR SILVER AND 98.7% FOR GOLD. THESE NUMBERS CORRESPOND TO METALLURGICAL ACCOUNTABILITIES OF 104.8% AND 103.1%, RESPECTIVELY.

THE 16-TO-1 MILL IS A CONVENTIONAL CYANIDE FACILITY UTILIZING A COUNTER-CURRENT DECANTATION CIRCUIT AND MERRILL-CROWE PRECIPITATION. THIS PLANT DIFFERS LITTLE FROM THOSE OF 50 YEARS AGO EXCEPT FOR THE DEGREE OF AUTOMATION AND INSTRUMENTATION PRESENT. THE CRUSHING SYSTEM AT THE 16-TO-1 IS A TWO-STAGE ONE USING A 30X 42 KUE-KEN JAW AS THE PRIMARY AND A SYMONS 4¼ SHORT HEAD CONE AS THE SECONDARY. MINE-RUN ORE, UP TO 24 INCHES IN SIZE, IS FED INTO THE JAW CRUSHER BY WAY OF A NICO RECIPROCATING FEEDER LOCATED UNDERNEATH A 70-TON COARSE ORE HOPPER. ALL MATERIAL PASSING THROUGH THE JAW CRUSHER DROPS ONTO THE NUMBER ONE BELT AND IS THEN CONVEYED TO AN ALLIS-CHALMERS VIBRATING SCREEN LOCATED OVER THE FINE ORE BIN. MINUS 5/8 INCH MATERIAL PASSES THROUGH THE SCREEN INTO THE BIN WHILE ALL OVERSIZE DROPS ONTO THE NUMBER TWO BELT. THIS BELT TAKES THE OVERSIZE BACK INTO THE CRUSHER BUILDING WHERE IT IS FED INTO THE CONE AND THEN DROPS BACK ONTO THE NUMBER ONE BELT TO COMPLETE THE CYCLE (FIGURE 8). PRESENT CRUSHING CAPACITY IS 125-150 TONS PER HOUR. CRUSHED MATERIAL LEAVES THE FINE ORE BIN VIA A BELT FEEDER WHICH IS DRIVEN BY A VARIABLE SPEED D-C DRIVE MOTOR. THIS DRIVE UNIT IS INTERFACED WITH THE BELT SCALE SO THAT A PRE-SET FEED TONNAGE RATE MAY BE MAINTAINED. MATERIAL DROPS FROM THE END OF THIS FEEDER ONTO THE CONVEYOR WHICH FEEDS THE BALL MILL. LIME IS ADDED ALONG THIS BELT WITH MILL SOLUTION BEING ADDED AT THE BALL

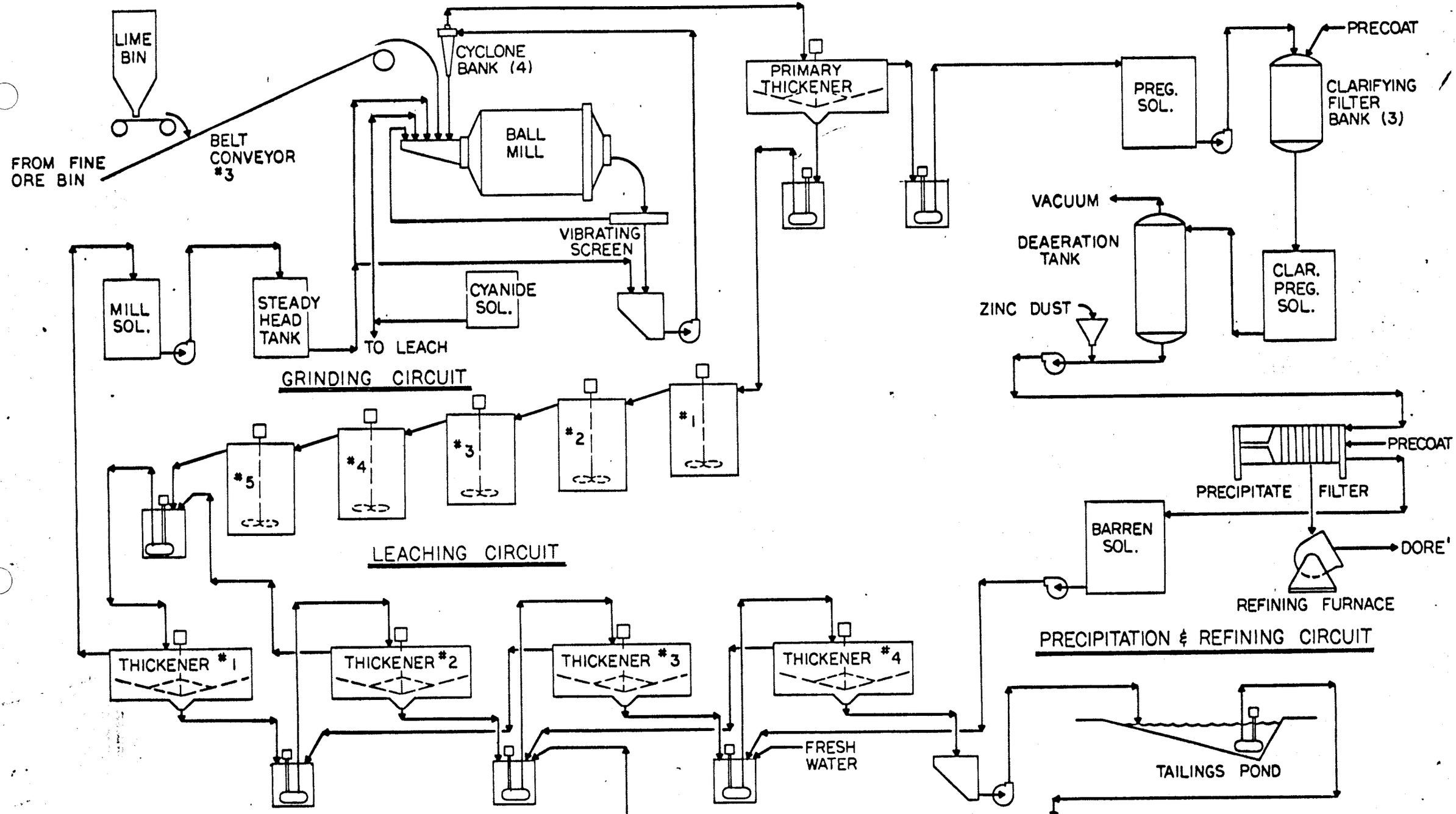


CRUSHING PLANT
FLOW SHEET

FIGURE 8

SUNSHINE MINING COMPANY
KELLOG, IDAHO
10 TO 1 MILL
SILVER PEAK, NEVADA

MILL FEED CHUTE. AN 800 HORSEPOWER, 10 X 14 MARCY BALL MILL IS USED TO ACHIEVE THE 80% MINUS 200 MESH GRIND WHICH IS NECESSARY FOR GOOD RECOVERY. THE BALL MILL DISCHARGE IS PUMPED THROUGH CYCLONES WITH THE OVERFLOW BEING SENT TO THE PRIMARY THICKENER. UNDERFLOW FROM THE PRIMARY THICKENER IS PUMPED INTO A SERIES OF FIVE LEACH TANKS GIVING A RETENTION TIME OF APPROXIMATELY 48 HOURS. A SUFFICIENT QUANTITY OF 20% SODIUM CYANIDE SOLUTION IS ADDED TO THE NUMBER ONE LEACH TANK TO MAINTAIN A FREE CYANIDE LEVEL OF BETWEEN 14 AND 18 POUNDS PER TON. AS WAS INDICATED FROM PREVIOUS METALLURGICAL TEST, THIS HIGH LEVEL OF CYANIDE PROVIDES THE NECESSARY ENVIRONMENT FOR GOOD LEACH EXTRACTION OF THE SILVER. DISCHARGE FROM THE LAST LEACH TANK ENTERS THE COUNTER-CURRENT DECANTATION CIRCUIT COMPRISED OF FOUR 40-FOOT DIAMETER THICKENERS. UNDERFLOW FROM THE LAST IN THIS SERIES OF THICKENERS GOES TO TAILS. OVERFLOW FROM THE FIRST IS RETURNED TO THE GRINDING CIRCUIT (FIGURE 9). BARREN SOLUTION FROM THE PRECIPITATION SECTION ALONG WITH FRESH-WATER MAKEUP IS ADDED TO THE FEED OF THICKENER NUMBER FOUR. TAILS RETURN SOLUTION IS ADDED TO THICKENER NUMBER THREE. ALL VALUES IN THE PLANT ARE PULLED OFF AS PRIMARY THICKENER OVERFLOW AND THIS IS PUMPED TO A PREGNANT SOLUTION TANK WITHIN THE MILL BUILDING. PREGNANT SOLUTION IS CLARIFIED USING PRESSURE LEAF FILTERN MANUFACTURED BY U.S. FILTER. THE SOLUTION THEN GOES INTO A CLARIFIED TANK WHICH IN TURN FEEDS THE DEAERATION TANK. PRECIPITANT FILTRATION IS ACCOMPLISHED USING THREE 36-INCH PLATE AND FRAME FILTERS. FINALLY, DRIED PRECIPITATION CAKE IS FLUXED AND MELTED INTO 800-1000 OUNCE BUTTONS AND SHIPPED TO KELLOGG, IDAHO, FOR REFINING.



C C D CIRCUIT FIGURE 9

SUNSHINE MINING COMPANY
KELLOG, IDAHO
1970 MILL
SILVER PEAK, NEVADA

WITH THE EXCEPTION OF THE FURNACE ROOM AND CRUSHING PLANT, OPERATIONS OF THE ENTIRE MILLING PROCESS MAY BE MONITORED FROM ONE CONTROL PANEL LOCATED INSIDE THE MILL. DESIRED PULP DENSITIES OF THICKENER UNDERFLOWS ARE MAINTAINED BY THE USE OF NUCLEAR DENSITY GUAGES IN COMBINATION WITH HYDRAULIC THROTTLING VALVES WHICH ARE REGULATED FROM THE MAIN CONTROL PANEL. THE UNDERFLOW FROM THE PRIMARY THICKENER IS MAINTAINED AT A 45% DENSITY WHILE ALL CCD UNITS ARE OPERATED AT 50%. THE MAIN CONTROL PANEL ALSO DISPLAYS BALL MILL BEARING TEMPERATURES, MILL FEED RATES, AND pH LEVELS. IN ADDITION, THERE ARE WARNING LIGHTS WHICH ALERT THE OPERATOR TO PROBLEMS SUCH AS HIGH TORQUE AND THE RAKES BEING UP ON A THICKENER. LIGHTS PLACED AT DIFFERENT LOCATIONS ON A PRINTED FLOWSHEET ON THE PANEL INDICATE WHETHER OR NOT THE VARIOUS PUMPS AND OTHER EQUIPMENT ARE RUNNING. DAILY QUANTITIES OF GOLD AND SILVER PRODUCED AS PRECIPITATE ARE CALCULATED BASED ON PREGNANT AND BARREN SOLUTION COMPOSITE SAMPLES COLLECTED BY AUTOMATIC SAMPLERS. THE CORRESPONDING TONS OF SOLUTION PROCESSED ARE RECORDED BY A FLOWMETER INSTALLED JUST AHEAD OF THE ZINC CONE. THESE CALCULATED FIGURES ARE TYPICALLY WITHIN FIVE PERCENT OF THE QUANTITY OF ACTUAL METAL PRODUCED. ZINC ADDITION IS REGULATED SO THAT SILVER VALUES IN THE BARREN SOLUTION RANGE FROM .01 TO .04 OUNCES PER TON. THIS IS REGULARLY MONITORED BY MEANS OF AN ATOMIC ABSORPTION UNIT LOCATED IN THE MILL AND WHICH THE MILL OPERATORS HAVE BEEN TRAINED TO USE. ALL ASSAYS USED FOR METALLURGICAL PURPOSES TOGETHER WITH ASSAYS ON MINE SAMPLES ARE DONE BY FIRE ASSAYING METHODS.

CYANIDE CONSUMPTION AT THIS PLANT IS VERY HIGH MAINLY AS A

RESULT OF OXIDATION LOSSES BROUGHT ABOUT BY THE HIGH LEVELS OF CONCENTRATION WHICH MUST BE MAINTAINED. THE PRESENT CONSUMPTION RATE IS APPROXIMATELY 6.9 POUNDS PER TON, WHICH EXPLAINS WHY CYANIDE REPRESENTS THE SECOND HIGHEST OPERATING COST AT THIS OPERATION. GRINDING BALL USAGE IS 3.3 POUNDS PER TON PROCESSED, WITH THREE INCH BALLS COMPRISING ALMOST 3/4'S OF THE TOTAL AND 1½ INCH BALLS MAKING UP THE REMAINDER. LIME CONSUMPTION IS FOUR POUNDS PER TON AND IS USED TO MAINTAIN A pH OF 11.5 TO 12.0. FLOCCULENT CONSUMPTION IS PRESENTLY AT 0.40 POUNDS OF NALCO 7873 PER TON OF ORE. FILTERING AID USAGE IS 0.49 POUNDS PER TON. ZINC CONSUMPTION AVERAGES 1.26 OUNCES OF ZINC PER ONE OUNCE OF PRECIOUS METAL PRECIPITATED.

PROCESSING CHANGES WHICH HAVE BEEN RECENTLY MADE INCLUDE A CHANGE IN THE POINT OF CYANIDE ADDITION TO THE NUMBER ONE LEACH TANK INSTEAD OF THE BALL MILL. THIS CHANGE REDUCED THE CYANIDE CONSUMPTION AND, THEREFORE, OPERATING COSTS WITHOUT AFFECTING RECOVERIES. BY INCREASING THE SIZE OF CERTAIN PIPES AND REPLACING A NUMBER OF PUMPS, MILL THROUGHPUT CAPACITY HAS BEEN INCREASED TO APPROXIMATELY 700 TONS PER DAY. IT REMAINS TO BE SEEN, HOWEVER, IF GOOD RECOVERIES CAN BE MAINTAINED AT THIS RATE. THE DEPENDABILITY OF THE SUPPLY OF PROCESS WATER WAS RECENTLY IMPROVED BY DIVERTING ALL MINE DISCHARGE WATER INTO THE MAIN STORAGE TANK AT THE MILL. THE 60 HORSEPOWER DEEP-WELL PUMP IS NOW USED ONLY FOR POTABLE WATER AND RUNNING TIME HAS BEEN REDUCED TO ½ HOUR PER DAY INSTEAD OF CONTINUOUS DUTY. ONE IMPORTANT FUTURE PROJECT AT THE MILL INVOLVES DEVELOPING A METHOD BY WHICH A STACKABLE TAILINGS

CAN BE ACHIEVED AND THEREBY REDUCE OUR TAILINGS DISPOSAL COSTS. CYANIDE AND PRECIOUS METALS LOSSES TO TAILINGS WOULD ALSO BE REDUCED IF THIS PROJECT IS SUCCESSFUL. CURRENTLY MAKING UP THE DIFFERENCE BETWEEN MILL THROUGHPUT AND MINE OUTPUT IS ORE WHICH IS BEING PURCHASED FROM OTHER SMALL MINING OPERATIONS NEARBY. THIS ORE PURCHASING PROGRAM WAS STARTED ONLY RECENTLY AND PROMISES TO BE MUTUALLY BENEFICIAL TO SUNSHINE AND THE SMALL OPERATION.

SUMMARY

THERE IS STILL MUCH ROOM FOR IMPROVEMENT AT THE 16-TO-1 IN THE AREAS OF REDUCING OPERATING COSTS AND IMPROVING PRODUCTIVITY AT THE MINE AND THE MILL. IN ADDITION, THE SEARCH FOR ADDITIONAL ORE RESERVES WILL CONTINUE IN EARNEST SINCE, GIVEN THE PRESENT RESERVE PICTURE THE USEFUL LIFE OF THE MILL WILL LONG OUTLAST THE MINE AS IT NOW EXISTS. IT IS THROUGH THESE EFFORTS, TOGETHER WITH AN OPTIMISTIC OUTLOOK FOR PRECIOUS METALS, THAT SUNSHINE HOPES TO BE AN IMPORTANT PRESENCE IN THE SILVER PEAK AREA FOR MANY YEARS TO COME.