PRINTED: 02/01/2002

ARIZONA DEPARTMENT OF MINES AND MINERAL RESOURCES AZMILS DATA

PRIMARY NAME: MIAMI EAST

ALTERNATE NAMES:

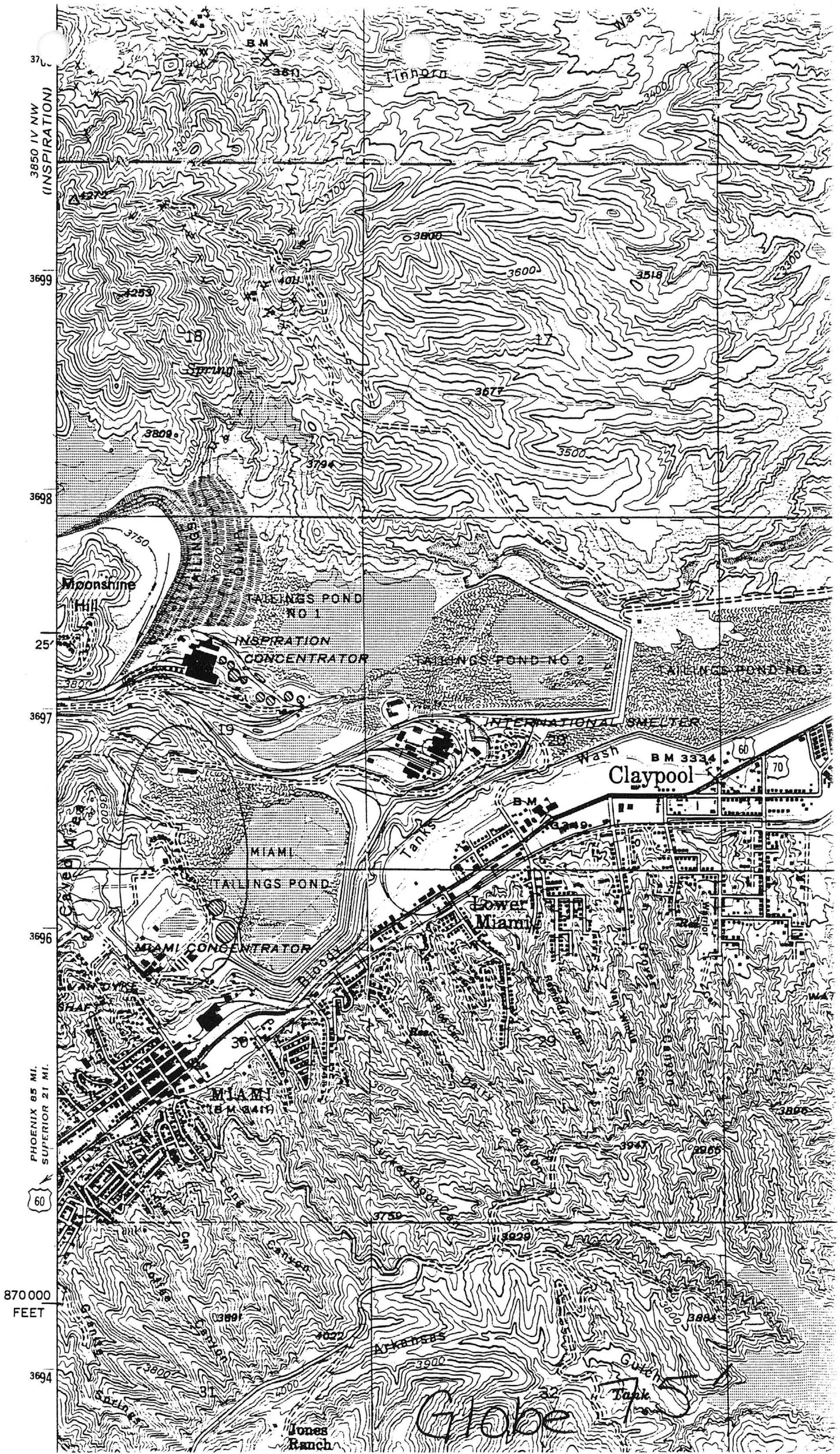
GILA COUNTY MILS NUMBER: 161C

LOCATION: TOWNSHIP 1 N RANGE 15 E SECTION 19 QUARTER S2
LATITUDE: N 33DEG 24MIN 30SEC LONGITUDE: W 110DEG 52MIN 00SEC
TOPO MAP NAME: GLOBE - 7.5 MIN

CURRENT STATUS: PRODUCER

COMMODITY:
COPPER SULFIDE
COPPER OXIDE

BIBLIOGRAPHY:
ADMMR MIAMI EAST FILE
MSHA MINE INFORMATION SUPP
OPERATIONS EXTEND INTO SEC 30
ADMMR 22 U/G PLAN MAPS (FLAT FILE DRAWER 18)


3850 IV NW
(INSPIRATION)
3699
3698
3697
3696
3694
25'
870 000
FEET
PHOENIX 85 MI.
SUPERIOR 21 MI.
60
70
Spring
Moonshine
Hill
TAILINGS POND
NO 1
INSPIRATION
CONCENTRATOR
TAILINGS POND NO 2
TAILINGS POND NO 3
INTERNATIONAL SMELTER
Wash
B M 3334
Claypool
MIAMI
TAILINGS POND
MIAMI CONCENTRATOR
Bloody
Tanks
Lower
Miami
MIAMI
(B M 3411)
Arkansas
Gulch
Canyon
Tank
Jones
Ranch
Globe 7

ARIZONA DEPARTMENT OF MINES AND MINERAL RESOURCES

INFORMATION FROM MINE CARDS IN MUSEUM

ARIZONA

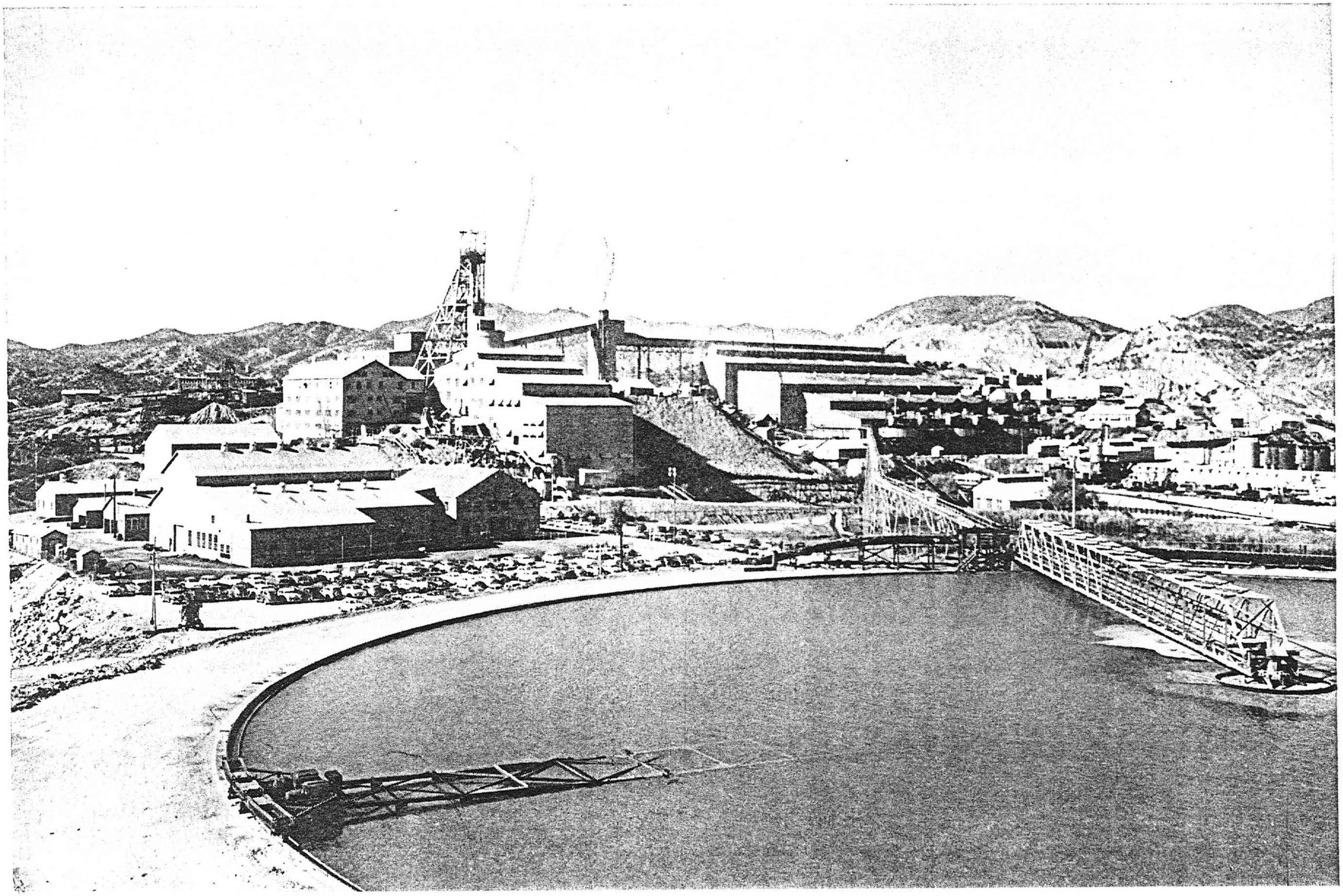
Gila Co

Miami

MM-9984	Chalcocite
9985	Chalcocite
9986	Native Copper

~~Miami East Mine~~ MILS # 161C
~~formerly Pinto Valley~~
~~formerly Sleeping Beuaty~~

MIAMI EAST (file)
O-AKA'C



Miami Copper Co. Concentrator and Mineshaft.

MIAMI EAST MINE   PINTO VALLEY COPPER CORPORATION   GILA COUNTY

NJN WR 5/13/83: Attended the AIME Technical Session held at Pinto Valley Copper Company's Miami East Underground mine. The highlight of this session was a demonstration of the Dosco Mechanical Miner. Copies of the talks presented were added to the Miami East mine file.

MIAMI EAST MINE

A.I.M.E. SPRING SESSION

May 13, 1983

TECH SPEC OF EQUIP

BATTERY LOCOMOTIVE

Model: 8-Ton Greensburg
Track Gauge: 36"
Wheel Base: 49"
Wheel Dia.: 20"
Height Overall: 54½"
Width Overall: 48"
Length Overall: 148"
Minimum Curve Radius: 17 Ft.

Equipment
Two-Motor Series Type - 22 H.P. Each @ 80 Volts
Control: Full Magnetic Type with 8 Speeds
Forward and Reverse
Locomotive Equipped with Battery Box Large
Enough to Accomodate the Following Batteries:
75X Type, 48 Cells, 23 Plates, 792 AMP-Hrs.
Which Weigh Approx. 7,632 Lbs.

Tractive Effort In Pounds

Speed in MPH	Lbs.
4.5	4,000
5	4,100
5.5	3,100
6	2,400
8	1,280

Braking
Mechanical Brakes, Fully Adjustable Brake Shoes
Are Cast Iron Steel Inserts.

#11 SHAFT-SEPARATION PLANT

(1) Tailings Line (Incoming Feed)
Flow Rate: 1400 GPM,220 TPH, 45% Solids
Length: 7,000 Ft.
Head: 80 Ft.-Static, 380 Ft.-TDH
Pipe: 8" Ø, Class 150 Transite
8" Ø, Sch.80 Steel Victaulic
Valves: 8" Ø Pneumatic Knife Gate,
Fabrivalve 45C
8" Ø Pneumatic Knife Gate,
Rovang 17A

(2) Surge Tank (Cyclone Feed)
Capacity: 5,000 Gal.
Dimensions: 6 Ft. Ø
Valves: 8" Ø Pneumatic Knife Gate,
Fabrivalve 45C
Level Controls: Warrick Sensor (High Level)
B & W Probe (Low Level)

(3) Cyclone Feed Pumps
Type: Warman 8/6 FAH Centrifugal
Impeller: Rubbert
Liners: Rubber
Seal: Gland Water
Drive Assembly: V-Belt, 545 Driven RPM
Pumping Rate: 1400 GPM, 220 TPH, 45% Solids
Head: 50 Ft.-Static, 60 Ft.-TDH
Motor: Louis-Allis, 75 H.P., 1200 RPM

(4) Cyclone
Type: Krebs D26B-1085
Size: 26"
Inlet: 8" Ø, 45" Rubber Head Liner
Vortex Finder: 10" Rubber Lined
Overflow: 12" Ø
Cone: 3 Piece, Rubber Lined
Apex Orifice: 3½" Rubber Adjustable-Hydraulic

LHD 4-WHEEL DRIVE SCOOPTRAM
(Front End Loader)

Model: ST-3½
Operating Weight: 31,940 Lbs.
Break Out Force: 16,993 Lbs.
Tramming Capacity: 12,000 Lbs.

Bucket Standard
Capacity, Heaped: 3.5 Cu.Yds.
Capacity, Struck: 2.9 Cu.Yds.
Raising Time: 5.7 Seconds
Lowering Time: 4.7 Seconds

Vehicle Speeds
1st Gear: 3.28 MPH.
2nd Gear: 6.64 MPH.
3rd Gear: 12.52 MPH.

Engine: Deutz - Model F6L413FW
Maximum Power @ 2300 RPM: 139 H.P.
Maximum Torque @ 1500 RPM: 325 Ft/Lb
No. of Cylinders: V6
Displacement: 584 Cu. In.
Cooling: Air
Electrical System: 12/24 Volts

Steering
Articulated With Self-Aligned Thrust
Turning Angle: 85° (42.5° Each Side)
System Pressure: 1500 PSI

(8) Slime Return Line
Flow Rate: 1100 GPM, 130 TPH, 45% Solids
Length: 7,000 Ft.
Head: 140 Ft. TDH, -15 Ft. Static (Average)
Pipe: 8" Ø, Class 100 Transite
8" Ø, Sch.40 Steel Victaulic
Valves: 8" Ø Pneumatic Knife Gate-Fabrivalve 45C
8" Ø Pneumatic Knife Gate-Rovang 17A

(9) Cement Plant
Silo: Ross 800 BBL with Aerator System
Feeder: Carter Day Rotary Valve BCI6
8" Ø, 6 Vane, Flexible Tips
Feeder Motor: Baldor, 1/2 H.P., Variable Speed, 1775 RPM Max.
Slurry Cone: 24" Ø, 36" HG, 70 Gal., 3/8" Urethane Lining, Unijet Spray Nozzles - 8 each
Slurry Piping: 1½" Ø Steel Vict. Pipe, 30 Ft. Length

(10) Density Control System
Nuclear Density Gage: Texas Nuclear SGH, 4"
Charter Recorder: Taylor 122RTI, 12"
Control Valve: Masoneilan Camflex II, 1" Ø Plug

(11) Air Compressor
Sullair 25-150L, 150 H.P.

DOSCO MECHANICAL MINER

Model: SL-120
Weight: 37 Tons
Length: 38'-7" (Including Conveyor)
Width: 7'-10" Over Apron 5"-0" Over Main Frame
Height: 5'-7" Overall
Track Speed: 23 Ft/Min. --- 54.0 Ft/Min.
Track Ground Pressure: 18 P.S.I.
Sumping Force: 26 Tons
Maximum Gradient: 1 in 4

Power Pack Assembly

No. of Pumps: 5
Pumping Capacity: 132 Gal/Min
Maximum Working Pressure: 2000 P.S.I.
Power Pack Motor: (Water Cooled) 110 H.P.

Gathering Arm Assembly

Speed: 40 Cycles/Min
Motor: 2 x HTL M350

Maximum Cutting Profile

Height Above Ground: 13'-5"
Depth Below Ground: 3 3/4"
Width (Boom Horizontal): 14'-1"
Width (Boom Raised): 10'-10"
Width (Boom Lowered): 13'-5"

Boom Assembly

Pick Speed (Max.): 35/70 RPM
Pick Force: 4.4/11 Tons
Cutting Head Dia.: 2'-0"
Cutting Head Motor: (Water Cooled) 130 H.P.

Scraper Conveyor Assembly

Conveyor Width: 1'-8"
Conveyor Speed: 190 Ft/Min.
Chain Size: 2 5/8 in. Pitch

HOISTING FACILITIES

Man Hoist

Allis Chalmers, Single Divided Drum
Drum Size: 8' Dia. by 2 x 36½"
Motor: 1, DC, 425 H.P., 550 Volt
Ropes: 1 1/8" x 4500', 6x21 Rt. Lang Lag
1 1/8" x 4500', 19x7 Non-Rotating
Cntrwt: 12,000 Lbs.
Conveyance Speed: 900 FPM

Muck Hoist

Nordberg, Double Drum
Drum Size: (2) 10' Dia. x 58½ Face with Lebus Grooving
Motors: (2), DC, 1400 H.P., 550 Volt
Skip Speed: 2250 FPM
Gnrtrs: (2) AC/DC, 1100 H.P., 4160V/650V

Shaft

Collar Elevation: 3635' Total Depth: 3550'
Upper 1100': 13' x 16' Rctnglr, Concrete Lined.
Lower 2450': 18'-2" Dia., Concrete Lined
Skip Sheaves: 12' Dia.
Cage Sheaves: 10' Dia.
Compressed Air Line: 12" Dia.
Pump Column: 8" Dia.
Guides: 5½" x 7½" x 20', Kerri Wood
Skips: 200 Cu.Ft., 10 Ton Capacity
Manacage: 5'-3" x 9'-4" x 32' O/A
23' Under the Hook
20 Person Capacity
10,000 Lb. Normal Load
U.G. Station: 2900 Lvl. 773' Elev.
3300 Lvl. 373' Elev.
Skiploader Station:

Ore Pockets: 800 Ton Capacity
Headframe Ht.: 148 Feet

Main Pump Room: Elev.
2-Wilson/Snyder Plunger Pumps, 500 H.P., 480 GPM, 1500 PSI

Total Depth: 3550' Elevation 85'

JOY 72" FAN

Model: M-72-50D
Diameter: 6'-0"
RPM: 1200
Length: 5'-8"
Overall Length of Fan and Evase':32'-4"
Dia. at Exhaust End of Evase': 9'-0"

Performance Capability

At 1200 RPM and Pulling 140,000 CFM, Fan Static Pressure Equals 13.2 Inches wg.
Fan Capability at 1200 RPM: 15" wg @ 65,000 CFM
Fan Capability at 1200 RPM: 9" wg @ 140,000 CFM

Motor

Model: Westinghouse HSW 1
H.P.: 700
RPM: 1200
Volts: 700/4160 3 Phase
Drive Shaft: Floating (12¼" Dia. x 12 Ft. Long)

NORTH TRANSFER RAISE

Driven Conventionaly With 8 Ft. Dia. Liner Plate.

Length: 39 Ft.
Dip: 64 Degrees

Raise Was Later Lined With 6 Ft. Dia. Plate and Concreted.

INSPIRATION - PUMPHOUSE

(1) Tailings Tap-In
    Saddle: 8" Ø Smith-Blair
    Valves: 8" Ø Pneumatic Gate Valve, Rovang 17A
            8" Ø Manual Gate Valve, Fabrivalve 45C
    Piping: 8" Ø Sch.80 Steel Vict. Pipe
    Air Blast: 1" Ø ASCO Solenoid Valve

(2) Tailings Pumps
    Type: Warman 6/4 FHH Centrifugal
    Impeller: NI-Hard
    Liners: NI-Hard
    Seal: Centrifugal
    Drive Assembly: V-Belt, 925 RPM Driven
    Pumping Rate: 1400 GPM, 220 TPH, 45% Solids
    Head: 80 Ft. Static, 380 Ft. TDH
    Motor: Allis-Chalmers, 350 H.P., 1180 RPM

(3) Pump Discharge Valves
    In-Line: Dezurik 198, 8" Ø Pneumatic Plug Valve
    Dump: Dezurik 198, 4" Ø Pneumatic Plug Valve

(4) Mass Flow System
    Nuclear Density Gage: Ohmart Ed-8, 8" Ø
    Magnetic Flow Meter: Great Lakes Ft.-76,8" Ø
    Electronic Calculator: Ohmart LmT-3

(5) Air Compressor
    Quincy D-350, Size - 3 1/4 x 3 1/2, 10 H.P.
    40 SCFM at 100 PSIG

GETMAN TRUCK

Model: 1248-13
Length: 24 Ft.
Width: 7 Ft.
Height Operator Head: 6'-11"
Wheel Base: 12 Ft.
Ground Clearance: 17 In.
Weight: 22,000 Lbs.

Dump Box
    Demensions: 6'-8" Wide x 11'-6" Long
    Loading Height: 4'-1" Over End
                    6'-1" Over Sides

Performance
    Turning Clearance: 10'-4" Inside
                       19' Outside
    Swept Path: 8'-8"
    Gradeability: Approx. 30% Maximum
    Maximum Speed: 9.3 MPH

Load Capacity
    13 Tons
    Dump Box Heaped: 9.0 Cu.Yds.

Engine
    Deutz - Model F6L413 FW
    Horsepower: 139 @ 2300 RPM
    Catalytic Exhaust Conditioners
    Ventilation Requirements: 12,000 CFM

Transmission
    Forward Speeds: Three
    Reverse Speeds: Three

(5) Cyclone Underflow Tank
Capacity: 450 Gal.
Size: 4 Ft. Ø
Flow Rate: 300 GPM, 90 TPH, 72% Solids
Valves: 4" Ø Pneumatic Gate Valve,
Febrivalve 45C
4" Ø Manual Pinch Valve-Clarkson B4
Discharge Piping: 4" Ø, Sch.40, with 1/2"
Urethane Lining

(6) Slime Tank
Capacity: 30,000 Gal.
Size: 16 Ft. Ø
Flow Rate: 1100 GPM, 130 TPH, 35% Solids
Valves: 8" Ø Pneumatic Gate Valve
Fabrivalve 45C
Level Controls: Warrick Sensor (High Level)
B & W Probe (Low Level)
Wesmar SLM-15 Level Sensor

(7) Slime Return Pump
Type: Warman 8/6 FAH Centrifugal
Impeller: Rubber
Liners: Rubber
Seal: Gland Water
Drive Assembly: V-Belt, Variable Speed,
835 RPM-Max.
Pumping Rate: 1100 GPM, 130 TPH, 35% Solids
Head: 140 Ft.-TDH, -15 Ft.Static (Average)
Motor: Borg Warner, 75 H.P., Variable Speed,
1775 RPM Max.
Controls: Borg Warner Electronic Speed
Controller

DIESEL LOCOMOTIVE

Plymouth - 15 Ton   Model: DMD 24
Gauge: 36"
Wheel Dia.: 24"
Wheel Base: 59.25"
Length: 13'-9"
Width: 58"
Height: 65"

Performance
Drawbar Pull on Level: 3,000 Lbs.
Maximum Capable Drawbar Pull: 7,500 Lbs.
Maximum Theoretical Gradient: 1.5%
Maximum Speed:

Drive Assembly
Chain Driven
Allison Torque Converter
2-Speed Power Shift Transmission

Braking
Air Brakes
Mechanical Hand Brake

Engine
Deutz - Model F6L413FW
Horsepower: 139 @ 2300 RPM
Catalytic Exhaust Conditioners
Ventilation Requirements: 12,000 CFM

<u>SOUTH TRANSFER RSE</u>

Size: Manway - 4' x 4½'
Muck Compartment - 4½ x 4½
Dip: 60°
Length: 397 Ft.
* Support: 10x10 Timber on 5' Centers
Started Raise January 1980 - Completed June 1980

* Section of raise from 3300 Level to Sub-Level 1 has since been lined with 4' x 9' eliptical liner plate and concreted.

Stop 14

North Transfer Raise

- . Driven Conventionally with 8' Diameter Liner Plate
- . Lined with 6' Diameter Plate and 1' Concrete
- . Carry Raise from Lift to Lift in Sand

Stop 15

3300 Shop

- . Refreshment Stop
- . See Handout for Equipment Specifications

Stop 16

South Transfer Raise

- . 10 Ton Granby Cars
- . Chute
- . See Handout for Raise Specifications

South Transfer Raise to 3300 Station

- . 33-7XC Repeat of 29-6XC (Miami Fault Zone)
- . Track Re-worked (10x10 Ties, Riveted Fish Plates)
- . Battery Barn

Stop 17

3300 Station Area

- . Pocket
- . Camelback Dump
- . Transformer Room

  13.8 KV Down #5 Shaft - Step Down to 4160 V and 480 V - 3000 KVA

MIAMI EAST UNDERGROUND TOUR

Spring Meeting A.I.M.E.

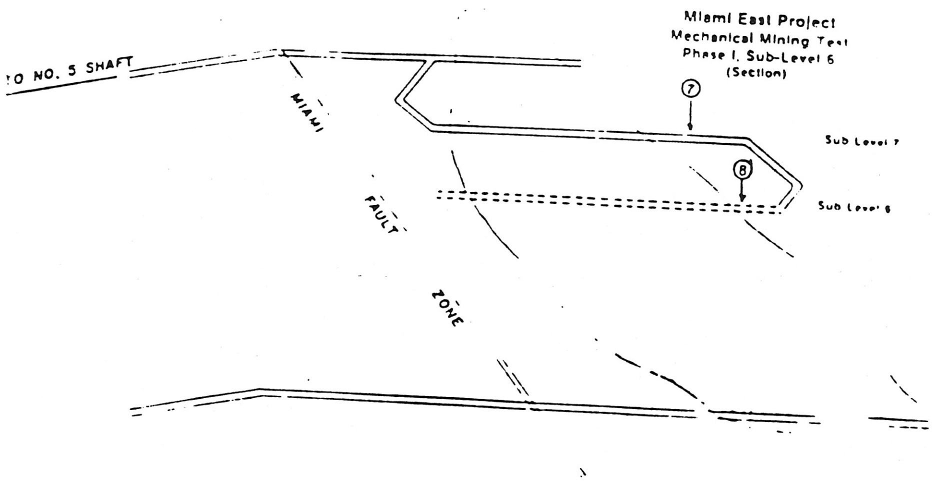
May 13, 1983

Miami, Arizona

Stop 7

Sub 7

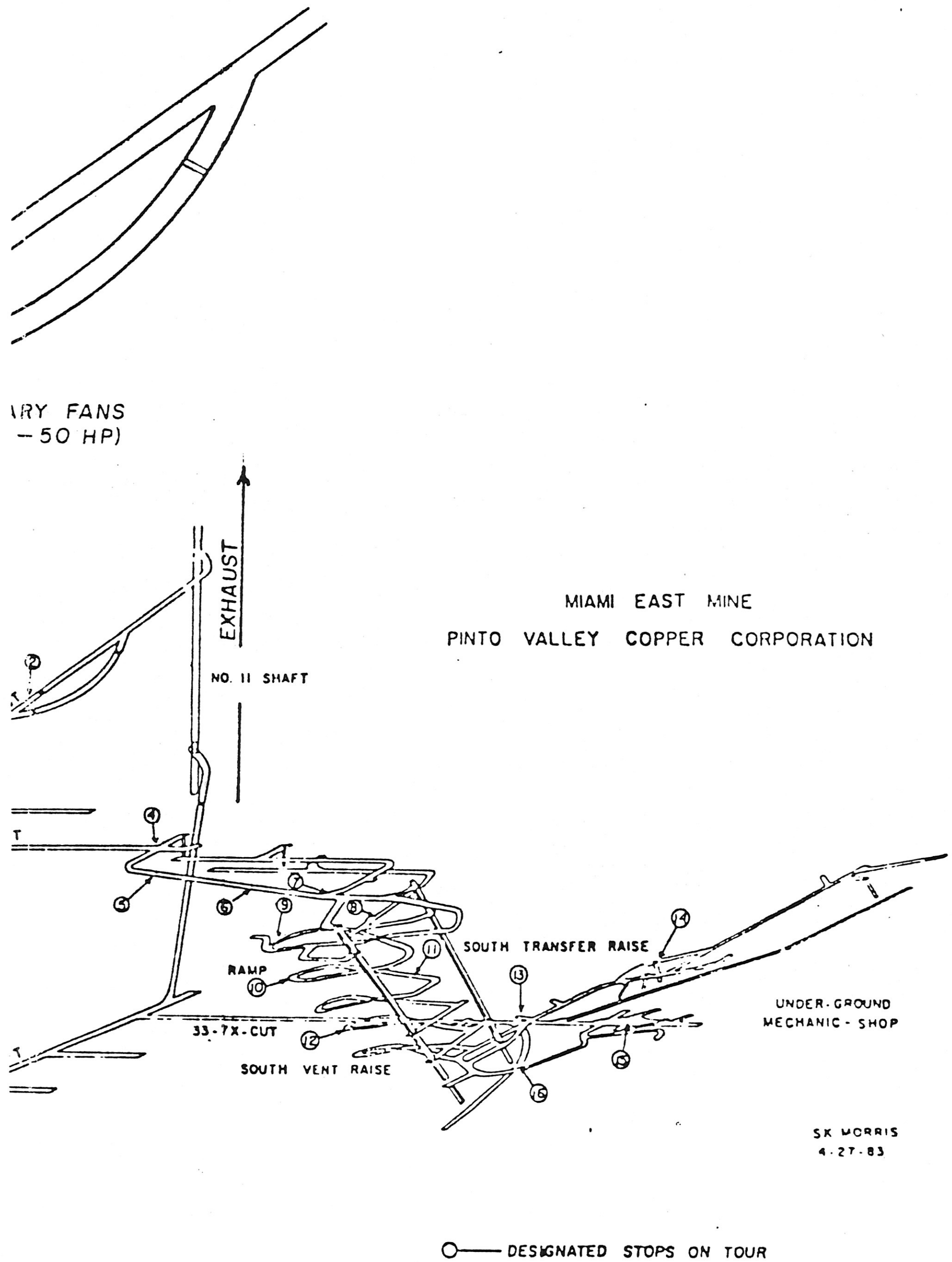
- Section thru Sub-Levels 7 and 6
- Sub 7 Driven Conventionally for Conventional Stoping
- Note Change to Lighter Weight Dosco Sets
- Point out Vent and Transfer Raise



Stop 8

Sub 6

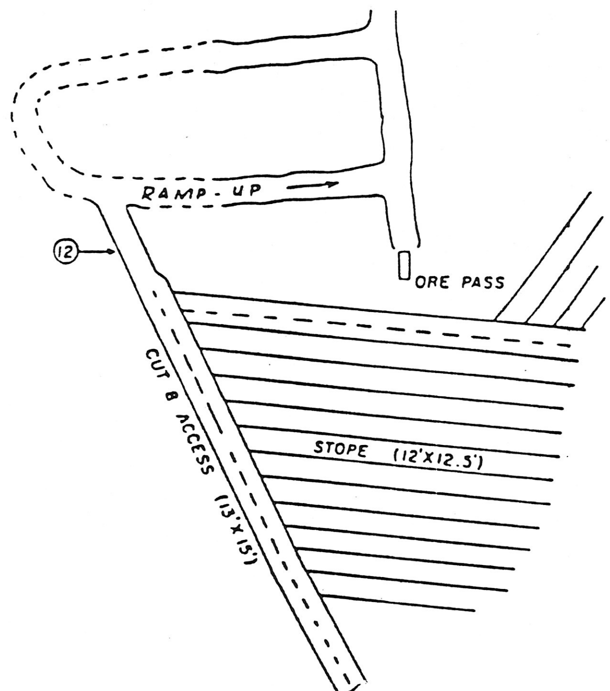
- Section thru Sub-Levels 7 and 6
- Dosco Assembled and Tested
- Advance Rate Doubled that Experienced on 7
- Cut from HW to FW, Total Footage 696' in all Rock Types, Ground Conditions, Ore and Waste



Stop 12

Cut 8 - Dosco #3

- Layout of Cut 8
- Observe and Discuss Machine and Ground
- See Handout for Equipment Specifications



PLAN VIEW-CUT 8 ACCESS
SOUTH STOPING AREA
SCALE 1"=50'

Stop 1

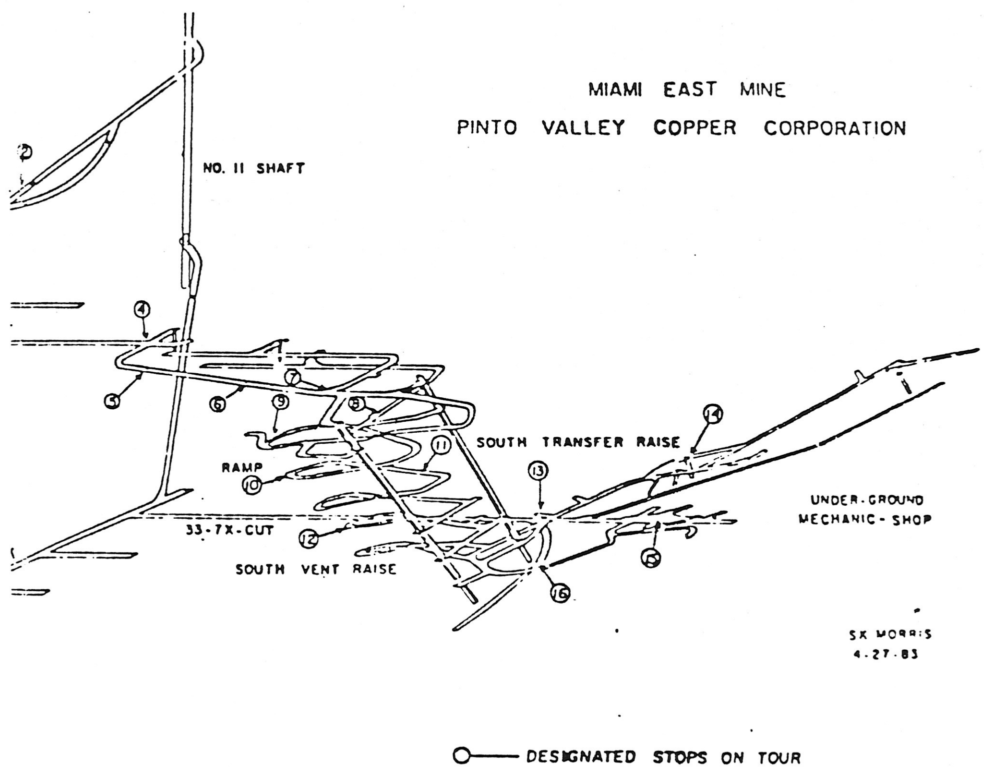
2900 Station

- Review Map of Mine and Scheduled Stops

2900 Station To 29-11XC Fan

General Information

- 800 Ton Pocket, 20' Ø x 60' Deep
- 60# Rail, 36" Gauge
- 8" Airline and 4" Waterline
- Power Distribution; 4160 V and 480 V
- Shotcrete Ground Support
- Air Doors - Fresh Air to 3300 Level



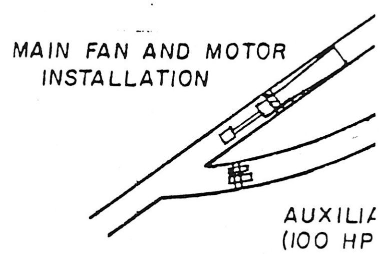
Stop 2

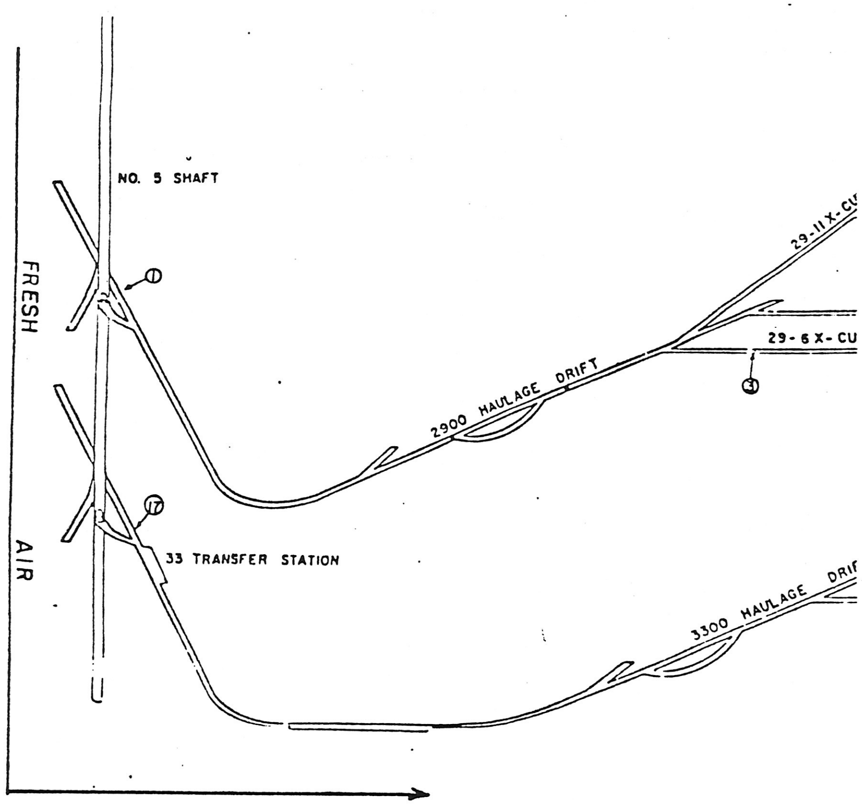
29-11XC Main Fan Installation

- 700 H.P. - 4160 Volt Motor
- Joy 72" Ø Fan
- Running at 225 H.P., Pulling 110,000 CFM at 3.2" wg
- Safety Controls Include Vibration, Temperature, and Over Current Sensors
- See Handout for Detailed Specifications

Auxiliary Fans, 100 H.P. and 50 H.P.

Note Sandfill Lines





Stop 9

Sub ½ - Ramp Down

- Changed from Conventional to Dosco
- Note Less Overbreak, Less Cribbing, and Wider Spaced Sets

Stop 10

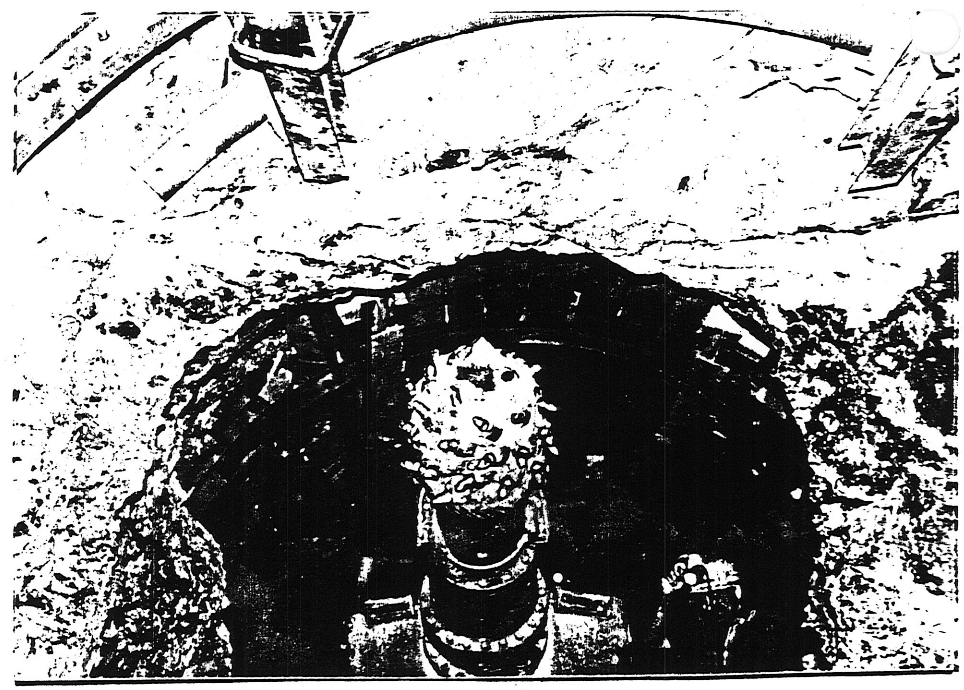
Sub 4½ - Ramp Down

- Note Hard Ground, ± 16,000 PSI Pick Marks, Drift Profile, Support
- Advanced 40' Before Bolting
- Hard Ground Required Different Bit and Slower Cutting Head RPM

Stop 11

Sub 4

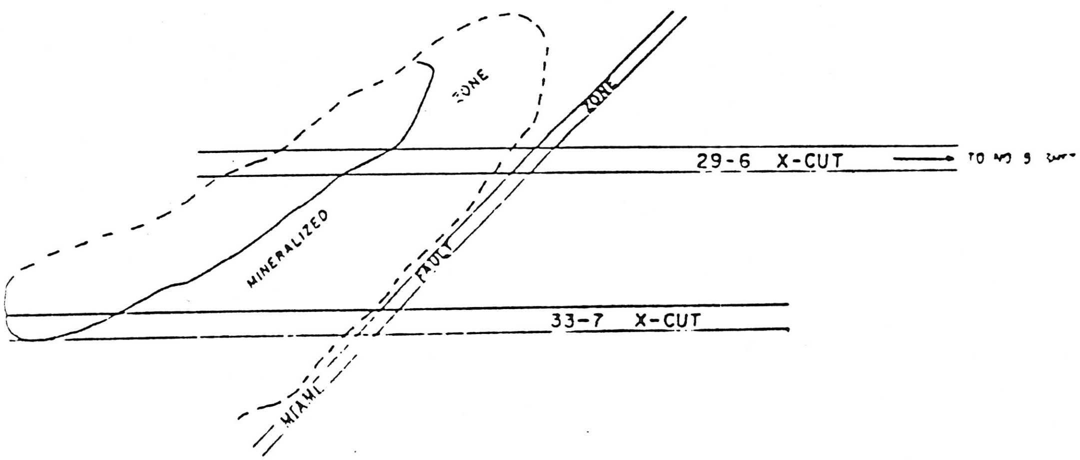
- Conventional Ramp Up Met Dosco Ramp Down
- Note Change in Support, Overbreak, etc.
- Total Ramp Lenth; Up 1960', Down 1924' Time of Drivage, 1 Year



Stop 3

29-6XC - Miami Fault Zone

. Section thru 29-6XC Showing Fault, Showing Mineralization @ + .6% and Orebody
. Steel Set Support in Fault Zone 4 Piece Sets, (8") Commercial Shearing



SECTION
THRU 29-6 X-CUT
LOOKING SOUTH
(NO SCALE)

Stop

Top Of Ramp

. End of Rail Development - Start of Rubber Tire
. Ramp Down Varies from -12% to -15%
. Sub-Levels Developed on 50' Vertical Intervals
. Ramp Legs Vary with Grade from 308' to 491'

Stop 5

Ore Intercept - 2900 Ramp Down

. Note Shotcrete and Wide Spaced Steel Sets (Now 2 Piece Sets on Minimum 2½' Centers)

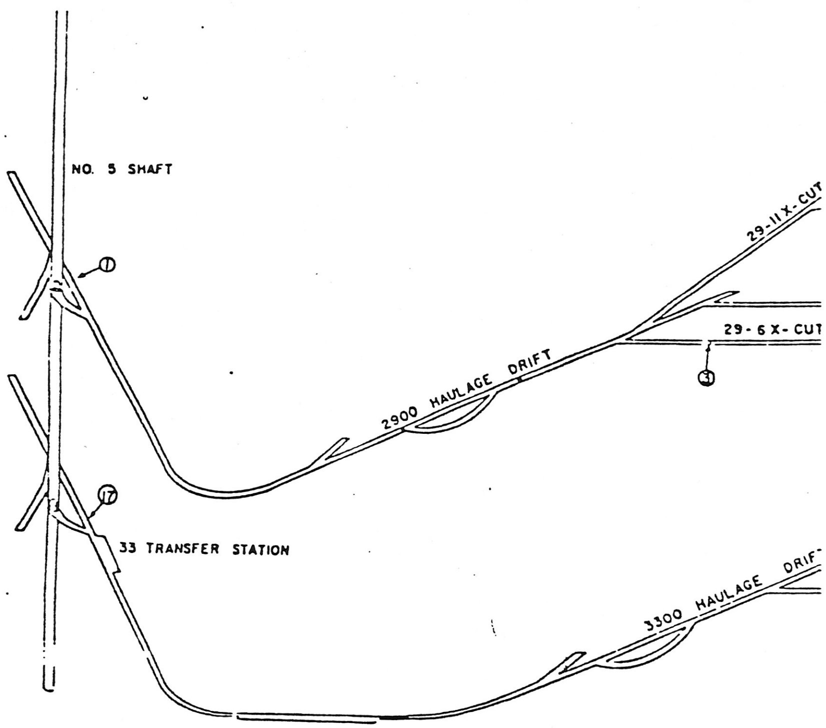
Stop 6

Ramp Down at Strike - July, 1980

. Note Grouting
. Ground Problems in Quartzite Capping, Highly Fractured with H_2O

#5 SHAFT INFORMATION

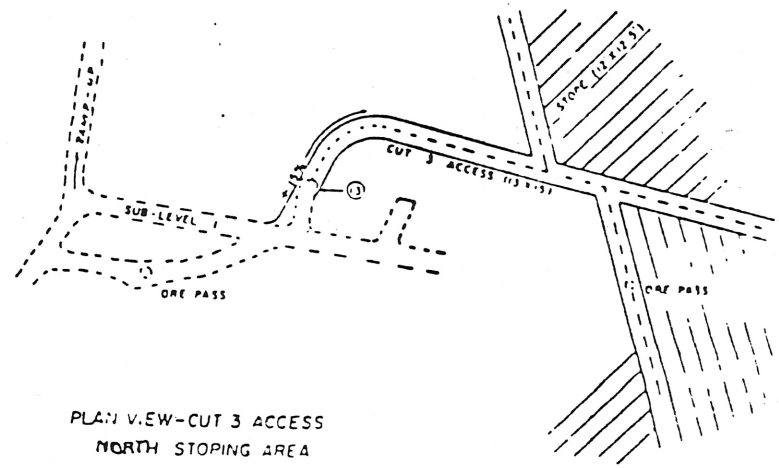
- Main shaft used for production hoisting, men and materials, and fresh air intake
- 10 ton skips, cage - 20 men/deck
- Collar - 1120'; 3 compartment, rectangular 13'x16' concrete lined
- 1120' - 3550; 18' Ø concrete lined
- Pump room at 3350
  Two 500 H.P. Wilson-Snyder Piston Pumps in one lift to surface - 8" discharge column



Stop 13

Sub-Level 1 - Dosco #2 (optional)

- Layout of Cut 3 Access
- Observe and Discuss Machine and Ground
- See Handout for Equipment Specifications



PLAN VIEW-CUT 3 ACCESS
NORTH STOPING AREA
SCALE 1"= 50'

A

SANDFILL SYSTEM AT

THE MIAMI EAST MINE

BY

Randy L. Seppala

PINTO VALLEY COPPER CORPORATION

Miami East Project

Prepared for the
1983 Spring Meeting of the
Underground Mining Division
ARIZONA SECTION OF THE A.I.M.E.

May 13, 1983

Miami, Arizona

INTRODUCTION

CEMENTED SANDFILL, ALSO REFERRED TO AS CEMENTED HYDRAULIC BACKFILL, HAS BECOME AN INTEGRAL PART OF MANY UNDERGROUND MINING OPERATIONS. THE NEED TO PREVENT SURFACE SUBSIDENCE, TO INCREASE EXTRACTION RATES, AND TO IMPROVE WORKING CONDITIONS HAS, IN MANY CASES, MADE SANDFILLING AN IMPORTANT CONSIDERATION IN THE SELECTION OF A MINING METHOD. FOR THESE REASONS, A MINING METHOD UTILIZING A SANDFILL SYSTEM WAS CHOSEN FOR THE MIAMI EAST MINE.

THE PURPOSE OF THIS PRESENTATION IS TO GIVE A GENERAL OVERVIEW OF THE SURFACE PLANT FACILITIES, AS WELL AS A DESCRIPTION OF THE TECHNIQUES INVOLVED IN THE UNDERGROUND SECTION OF THE SANDFILL SYSTEM.

PLANT HISTORY

DURING THE INITIAL PROJECT PLANNING IT WAS DECIDED THAT THE ORE FROM MIAMI EAST WOULD NOT BE PROCESSED AT THE MINE SITE. THEREFORE, AN ALTERNATE SOURCE OF TAILINGS FOR THE SANDFILL PLANT HAD TO BE ESTABLISHED, AND AN AGREEMENT WITH INSPIRATION CONSOLIDATED COPPER COMPANY WAS REACHED WHICH ALLOWED FOR THE EXTRACTION OF TAILINGS DIRECTLY FROM THEIR TAILINGS LINE. PILOT PLANT TESTING AND DESIGN WORK BASED UPON THIS TAILINGS SOURCE WAS INITIATED IN 1972, AND BY 1974 THE FINAL DESIGN WAS COMPLETED FOR A PLANT CAPABLE OF SUPPORTING A 2000 TPD OPERATION.

THE PLANT WAS DESIGNED AS A CONTINUOUS THROUGHPUT SYSTEM, WHICH INVOLVES PUMPING THE TAILINGS A DISTANCE OF ABOUT 1.3 MILES TO

PUMPHOUSE

THE SANDFILL PLANT SYSTEM BEGINS AT A PUMPING FACILITY LOCATED ON INSPIRATION'S PROPERTY ADJACENT TO THEIR TAILINGS LINE. OUTSIDE THE PUMPHOUSE THERE ARE TWO TAPPING ARRANGEMENTS ON INSPIRATION'S LINE (ONE SERVING AS A STANDBY) THROUGH WHICH THE TAILINGS ARE DRAWN OUT BY GRAVITY FLOW. EACH INSTALLATION CONSISTS OF AN 8" TAPPING SADDLE WITH AN 8 " PNEUMATIC GATE VALVE AND AN 8" MANUAL GATE VALVE. THE TAILINGS ARE THEN FED TO ONE OF TWO WARMAN 6/4 FHH CENTRIFUGAL SLURRY PUMPS INSIDE THE PUMPHOUSE. THESE PUMPS HAVE NI-HARD LINERS AND IMPELLERS AND ARE CENTRIFUGALLY SEALED. THEY ARE DRIVEN BY 350 HP MOTORS AT A SPEED OF 925 RPM THROUGH A V-BELT DRIVE ASSEMBLY.

THE TAILINGS ARE PUMPED TO NO. 11 SHAFT AT A RATE OF 1400 GPM AGAINST A TOTAL DYNAMIC HEAD OF 380 FEET (165 PSI). THE SLURRY HAS AN AVERAGE DENSITY OF 45% SOLIDS BY WEIGHT, WHICH AMOUNTS TO 220 TPH OF SOLIDS BEING DELIVERED.

THE TAILINGS LINE ENROUTE TO NO. 11 SHAFT IS 8" DIAMETER, CLASS 150 TRANSITE PIPE. IT IS SUPPORTED ON CONCRETE PIERS ON ABOUT 5 FOOT CENTERS AND IS SECURED WITH STAINLESS STEEL BANDING. THE PIPELINE ROUTE COVERS APPROXIMATELY 7000 FEET WITH AN ELEVATION INCREASE OF 80 FEET AND TRAVERSES SOME CONSIDERABLY RUGGED TERRAIN. THE LINE TRAVELS THROUGH TWO TUNNELS (EACH ABOUT 300 FEET LONG), CROSSES OVER A RAIL LINE ALONG A 70 FOOT TRESTLE, AND IS BURIED FOR ABOUT 200 FEET IN CROSSING UNDER ANOTHER RAIL LINE.

AND LINERS ARE RUBBER, AND THE PUMPS UTILIZE A GLAND WATER SEAL SYSTEM.

THE SLIME RETURN LINE IS 8" DIAMETER CLASS 100 TRANSITE PIPE. IT RUNS BACK TO INSPIRATION PARALLEL TO THE TAILINGS LINE AND IS SECURED TO THE SAME CONCRETE PIERS. THE SLIMES ARE RETURNED DIRECTLY INTO INSPIRATION'S TAILINGS LINE THROUGH A HEAD BREAKER ADJACENT TO THE PUMPHOUSE.

THE LOCAL PIPING AT THE SEPARATION PLANT IS 8" DIAMETER SCH. 80 AND SCH. 40 PIPE. THE SCHEDULE 40 PIPE IS USED ONLY IN THE SLIME RETURN LINE. THIS IS BASED ON WEAR CONDITIONS RATHER THAN PRESSURE REQUIREMENTS.

CEMENT PLANT

THE FINAL STEP IN THE PREPARATION OF THE TAILINGS TAKES PLACE AT THE CEMENT PLANT. THE PLANT CONSISTS OF THREE BASIC COMPONENTS: AN 800 BBL SILO, A CEMENT FEEDER, AND A SLURRY CONE. THE CEMENT IS FED FROM THE SILO BY A ROTARY FEEDER INTO A SLURRY CONE WHERE IT IS MIXED WITH WATER. FROM THE CONE THE RESULTING SLURRY FLOWS BY GRAVITY THROUGH A 1-½" DIAMETER STEEL LINE FOR A DISTANCE OF 30 FEET TO A MIXING LATERAL IN THE SANDFILL LINE AT THE SHAFT COLLAR.

THE PLANT IS PRESENTLY SET UP TO MIX AND DELIVER CEMENT IN THE RANGE OF 3TPH TO 9 TPH, WHICH CORRESPONDS TO SAND/CEMENT RATIOS OF 30/1 AND 10/1. AT A DENSITY OF 50% SOLIDS BY WEIGHT THIS RESULTS IN SLURRY FLOW RATES OF 48 GPM AND 16 GPM, RESPECTIVELY. THE PLANT IS CAPABLE OF PRODUCING A MAXIMUM OF 14 TPH OF DRY CEMENT IF NECESSARY.

PRESSURE, HIGH OIL TEMPERATURE, OR LOW SEAL WATER FLOW.

MEASUREMENTS OF FLOW RATES AND DENSITIES ARE TAKEN AT TWO POINTS IN THE SYSTEM. AT THE PUMPHOUSE, A MASS FLOW SYSTEM CONSISTING OF NUCLEAR DENSITY GAGES AND MAGNETIC FLOW METERS MEASURE THE DENSITY AND FLOW RATE IN THE TAILINGS LINE AND THE SLIME RETURN LINE. THIS UNIT IS PRESENTLY SET UP TO DETERMINE AND RECORD THE WATER CONSUMPTION. AT NO. 11 SHAFT, A DENSITY CONTROL SYSTEM WILL MEASURE AND RECORD THE DENSITY OF THE SANDFILL PRODUCT GOING UNDERGROUND. THE UNIT CONSISTS OF A NUCLEAR DENSITY GAGE AND A CIRCULAR CHART RECORDER, AND IS TIED INTO A CONTROL VALVE WHICH CAN ADD DILUTION WATER TO THE SANDDILL FOR DENSITY CONTROL. THE COMPONENTS OF THIS SYSTEM HAVE NOT YET BEEN INSTALLED.

PLANT START UP PROBLEMS

CYCLONE

DURING THE TESTING PERIOD WE DETERMINED THAT THE PLANT WAS ONLY PRODUCING ABOUT 50% OF THE DESIGNED CLASSIFIED SAND CAPACITY. THIS IS DUE TO AN ACTUAL CYCLONE RECOVERY RATE OF 20%, WHICH IS MUCH LOWER THAN THAT WHICH WAS ESTIMATED FROM THE PILOT PLANT STUDIES IN 1973. THIS PROBLEM WAS PRESENTED TO THE CYCLONE MANUFACTURER, KREBS ENGINEERS, AND IT WAS DETERMINED THAT THE DECREASE IN THE RECOVERY RATE COULD BE ATTRIBUTED TO A CHANGE IN THE PARTICLE SIZING OF THE CYCLONE FEED. SCREEN ANALYSIS OF THE CYCLONE FEED IN 1973 SHOWED THAT IT CONTAINED ABOUT 15% -325 MESH, WHILE NOW THE FEED CONSISTS OF AS MUCH AS 40%-325 MESH. THIS HIGH PERCENTAGE OF SLIMES RESULTS IN A HIGH VISCOSITY OF THE FEED, IN TURN CAUSING A COURSE

PIPELINES

AS WAS PREVIOUSLY MENTIONED, WE EXPERIENCED A GREAT DEAL OF TROUBLE WITH THE TRANSITE SECTIONS OF BOTH LINES DURING THE EARLY PLANT TESTING. THE LINES WERE CONTINUALLY BURSTING UNDER PRESSURE, ESPECIALLY WHEN THE PUMPS WERE FIRST STARTED OR SHUT DOWN. THESE LINE FAILURES WERE INITIALLY ATTRIBUTED TO INADEQUATE TIE DOWNS ON THE PIPE (IN THE CASE OF COUPLING FAILURE) AND DAMAGED OR DEFECTIVE PIPE (IN THE CASE OF PIPE RUPTURE). ADDITIONAL SUPPORTS WERE INSTALLED AND MOST OF THE OLD BANDING WAS THEN REPLACED. THIS REDUCED THE FREQUENCY OF LINE FAILURE, BUT MANY OF THE TEST RUNS STILL RESULTED IN A BROKEN LINE.

IT WAS NOT UNTIL WE STARTED ADJUSTING THE VALVE SEQUENCING THAT WE DISCOVERED THE MAJOR CAUSE OF THE PIPE FAILURES. WE FOUND THAT SEVERAL OF THE VALVES WERE CLOSING SIMULTANEOUSLY WITH THE PUMP SHUT DOWNS, RESULTING IN A VERY QUICK CLOSURE AGAINST A VELOCITY OF ABOUT 9 FT/SEC. THIS CAUSED SHOCK PRESSURES CLOSE TO 600 PSI, WHICH ALONE IS IN EXCESS OF THE PIPE MANUFACTURERS TEST PRESSURE. THROUGH THE USE OF ADDITIONAL TIMERS WE WERE ABLE TO DELAY THE VALVE CLOSURES, THEREBY SIGNIFICANTLY REDUCING THE SHOCK PRESSURE IN THE LINES. SINCE THIS WAS COMPLETED WE HAVE NOT EXPERIENCED A SINGLE LINE FAILURE IN THE LAST 16 TEST RUNS. THIS IS VERY PROMISING IN THAT IT MEANS A COMPLETE LINE RENNOVATION OR REPLACEMENT WILL NOT BE NECESSARY IN THE NEAR FUTURE.

AS EACH STOPE IS MINED OUT, THE MECHANICAL MINER WILL BE MOVED TO A NEW LOCATION TO BEGIN CUTTING ANOTHER STOPE. MEANWHILE, THE FIRST STOPE WILL BE SANDFILLED. THIS PATTERN WILL CONTINUE UNTIL THE ENTIRE LENGTH IS MINED AND FILLED, INCLUDING THE MAIN ACCESS, AT WHICH POINT THE PROCEDURE IS REPEATED ON THE CUT ABOVE.

THE MINING SEQUENCE WILL GENERALLY ALLOW FOR TWO OR MORE STOPES TO BE SANDFILLED AT ONE TIME, THUS MAKING THE FILLING PROCESS MORE EFFICIENT.

DISTRIBUTION LINES

SANDFILL SLURRY IS FED 2800 FEET DOWN NO. 11 SHAFT THROUGH A 4" DIAMETER PIPE WITH ½" URETHANE LINING. THE PIPE IS IN 20 FOOT JOINTS CONNECTED WITH STANDARD VICTAULIC COUPLINGS. DUE TO PRESSURE REQUIREMENTS THE UPPER 1200 FEET IS SCHEDULE 40 PIPE AND THE LOWER 1600 FEET IS SCHEDULE 80 PIPE. URETHANE LINED ORIFICES WITH A 2" INSIDE DIAMETER ARE INSTALLED AT 200 FOOT INTERVALS TO REDUCE FREE FALL. AT THE PRESENT TIME ONLY ONE LINE HAS BEEN INSTALLED IN THE SHAFT, BUT THE INSTALLATION OF A SECOND LINE IS PLANNED FOR COMPLETION SOMETIME AFTER START UP. THIS WILL REDUCE DOWNTIME CREATED BY A PLUGGED OR BROKEN LINE IN THE SHAFT.

AT THE 2900 LEVEL THE SANDFILL LINE LEAVES THE SHAFT WITH A 5 FOOT RADIUS SWEEP. THE FIRST 500 FEET ACROSS THE LEVEL IS URETHANE LINED, AT WHICH POINT IT CHANGES TO STANDARD SCHEDULE 40 VICTAULIC PIPE. THE LINE CONTINUES THROUGH A SERIES OF CURVES ACROSS THE 2900 LEVEL FOR A DISTANCE OF 1600 FEET TO THE TOP OF THE RAMP. IT THEN RUNS DOWN

FILTERWALLS

AFTER THE STOPES ARE MINED OUT, FILTERWALLS ARE CONSTRUCTED AT THE STOPE ENTRANCE AS CLOSE AS POSSIBLE TO THE MAIN ACCESS, DEPENDING ON THE GROUND CONDITIONS. A MAIN FRAMEWORK CONSISTING OF 6" X 8" TIMBER IS BUILT AND SECURED BY HITCHES AND ROCKBOLTS. 3" X 12" LAGGING IS THEN INSTALLED ON ABOUT 2 FOOT CENTERS, WITH 4" X 4" WIRE MESH NAILED ON THE INSIDE, FOLLOWED BY A LAYER OF 7-½ OZ.

BURLAP. THE RIBS ARE SEALED BY CUTTING AND INSTALLING LAGGING TO THE CONTOUR OF THE FROUND. FINAL SEALING IS THEN ACCOMPLISHED BY SIMPLY CHINKING THE HOLES WITH BURLAP OR ADHERING THE BURLAP TO THE RIBS WITH AN UNDERCOATING EMULSION SPRAY. THE BROW MAY ALSO BE SEALED IN THE SAME MANNER TO GET A TIGHT FILL IF OVER - BREAK IN THE STOPE NECESSITATES THIS.

THE CONSTRUCTION OF THESE WALLS HAS PROVEN TO BE VERY TIME CONSUMING. TO OVERCOME THIS PROBLEM, INITIAL DESIGN WORK HAS BEEN COMPLETED IN THE DEVELOPMENT OF A PORTABLE AND LIGHT WEIGHT FILTERWALL SYSTEM.

POURING

A POUR IS INITIATED FROM UNDERGROUND BY DIRECT COMMUNICATION WITH THE OPERATOR BY A PAGING PHONE. THE SANDFILL LINE IS FIRST FLUSHED WITH WATER TO WET THE LINE, AND IT THEN TAKES 10 TO 15 MINUTES FOR THE SAND TO ARRIVE. USUALLY, TWO OR MORE STOPES WILL BE POURED AT ONE TIME TO ALLOW FOR SETTLING AND TO PROVIDE FOR A DIVERT IN THE EVENT OF A LEAKING STOPE.

THESE LINES ARE CONSTRUCTED OF PERFORATED POLYETHYLENE PIPE (3"-4" DIAMETER) WRAPPED WITH 7-½ OZ. BURLAP AND ARE USUALLY INSTALLED NEAR THE SILL AND AT THE MIDPOINT OF THE STOPE HEIGHT.

THE WATER WILL BE COLLECTED AT THE FRONT OF EACH STOPE IN A SMALL DAM AND PICKED UP BY A 3" AIR POWERED DIAPHRAGM PUMP. IT IS THEN DELIVERED ACROSS THE SUBLEVEL THROUGH A 4" POLYETHYLENE PIPELINE TO EITHER THE NORTH TRANSFER OR THE SOUTH VENT RAISE. FROM THERE IT WILL FLOW DOWN TO THE 3300 LEVEL AND ACROSS THE LEVEL FOR A DISTANCE OF ABOUT 1700 FEET TO ONE OF TWO SETTLING SUMPS. THE DECANT WATER FROM THESE SUMPS WILL AGAIN BE PICKED UP BY AN AIR POWERED DIAPHRAGM PUMP AND DELIVERED THROUGH A 4" POLYETHYLENE PIPELINE TO THE MAIN SUMP AT NO. 5 SHAFT. AT THIS POINT THE WATER WILL BE PUMPED TO THE SURFACE IN ONE LIFT BY A WILSON-SNYDER 4-½ X 8 QUINTUPLEX PLUNGER PUMP.

SANDFILL PRODUCT QUALITY

WE HAVE NOW SEEN THE COMPLETION OF ALL THE PHASES INVOLVED IN THE SANDFILLING OF A STOPE. IT IS IMPORTANT NOW TO LOOK AT THE CHARACTERISTICS OF THE SANDFILL SLURRY WHICH HAVE TO BE CONTROLLED TO MAINTAIN A SATISFACTORY FILL MATERIAL. THE QUALITY OF OUR FINAL SANDFILL PRODUCT IS BASED UPON TWO MAIN PARAMETERS: PERCOLATION RATES AND STRENGTH CHARACTERISTICS. THESE PARAMETERS ARE DIRECTLY RELATED TO THE SLIME CONTENT, DENSITY, AND CEMENT CONTENT OF THE SLURRY.

TO BEGIN WITH, THE PERCOLATION RATE IS DIRECTLY RELATED TO THE SLIME CONTENT. PERCOLATION RATE TESTS HAVE SHOWN THAT AS THE SLIMES CONTENT

THESE NUMBERS MAY SEEM SOMEWHAT LOW, ESPECIALLY FOR THE 30/1 RATIO. HOWEVER, THE CEMENT CONTENT OF THE MAIN POUR IS BASED UPON THE AMOUNT REQUIRED TO MAINTAIN A COHESIVE RIB WHEN MINING NEXT TO THE FILL, NOT NECESSARILY TO ACHIEVE HIGH COMPRESSIVE STRENGTHS. BUT, THE STRENGTH CHARACTERISTICS OF THE CAP WILL BE CRITICAL IN TERMS OF SUPPORTING THE MINING EQUIPMENT ON THE NEXT CUT ABOVE.

FURTHER TEST STOPING AND FILLING WILL BE REQUIRED TO DETERMINE THE ACTUAL INSITU STRENGTH CHARACTERISTICS OF THE SANDFILL AT VARIOUS SAND/CEMENT RATIOS. THESE SUBSEQUENT STUDIES WILL ESTABLISH THE FINAL CEMENT CONTENT OF THE SANDFILL, AND SINCE CEMENT USAGE IS PREDICTED TO BE THE HIGHEST OPERATING COST IN OUR SANDFILL OPERATION, CONTINUING CONTROL OVER THESE CEMENT RATIOS WILL BE A NECESSITY.

SUMMARY

IN CONCLUSION, IT IS OBVIOUS THAT THE SANDFILL SYSTEM AT MIAMI EAST IS STILL BASICALLY IN THE EARLY STAGES. THERE ARE UNDOUBTEDLY MANY PROBLEMS THAT WILL BE ENCOUNTERED THROUGHOUT THE LIFE OF THE SYSTEM. BUT THROUGH A PROCESS OF LEARNING AND REFINING, THIS SYSTEM CAN DEVELOP INTO A VERY EFFICIENT OPERATING UNIT.

most often they know the rules and procedures but choose to do the unsafe act for reasons of their own.

Conditions that propose a threat of injury need to be recognized, then these conditions can be corrected.

The safety program at Miami East in 1980 and early 1981, incorporated all of the ingredients to make a good safety program. Eight hours of annual safety training, required by law, was being conducted at Gila Pueblo College, weekly safety meetings were being held, an excellent incentive program was in effect, and the mine was clean and orderly. Still we were suffering a 47.2 injury incidence rate.

In may of 1981, the training program was modified for two reasons; 1) we could personalize required training to fit our needs and, 2) by using the thirty minutes safety meetings for training sessions, we could save three (3) full shifts of lost production per year.

Our incidence rate dropped 30% and held through July. In August of 1981, the supervisors were relieved of their daily duties to spend a two-week training period in the safety department. The training stressed observation of employees for poor work habits and craftmanship. Company policy concerning enforcement of safety rules, rules of conduct and absenteeism were also reviewed at this time. Improvement in plant safety was almost immediate. The employees began to enjoy the safety incentives and a spirit of competition developed. A record 67 days without a lost time injury was accomplished. An injury occured on

EASILY IDENTIFIABLE INJURY COSTS

First Aid

Emergency Room	$ 37.66
Evaluation	48.82
Office Visit and Release	35.34
TOTAL	$121.82

Reportable Injury Costs

Emergency Room	$ 37.66
Evaluation	72.07
Debridement	18.64
Office Visit and Release	35.34
Laceration	46.15
TOTAL	$209 86

Lost Time

Emergency Room (Extended)	$ 60.45	
Comprehensive Evaluation	86.02	
Surgery		$93.20
Room Care	30.22	
Consultation		72.07
Daily Visit (M.D.)	16.27	
Room, per Day	140.00	
X-Rays	39.60+	
Dressing and Medication		
	$372.56	$165.27

1980 - 101 EMPLOYEES

25 RECORDABLE OR LOST-TIME INJURIES

25% OF THE WORK FORCE

112 LOST-TIME DAYS

1981 - 131 EMPLOYEES

28 INJURIES

21% OF THE WORK FORCE

123 LOST-TIME DAYS

1982 - 85 EMPLOYEES (AVERAGE)

3 INJURIES

2.5% OF THE WORK FORCE

0 LOST-TIME DAYS

Comparing the first six months of 1982 with the first six months of 1980; times when the mine operated with a full work force, we realized a 63% reduction in accident costs.

We are also enjoying a 14% cost reduction in our premium for Workmen's Compensation Insurance.

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GEOLOGY OF THE

MIAMI EAST MINE

BY

Stuart K. Morris

PINTO VALLEY COPPER CORPORATION

Miami East Project

Prepared for the
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Underground Mining Division
ARIZONA SECTION OF THE A.I.M.E.

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Miami, Arizona

INTRODUCTION

THE MIAMI-INSPIRATION DISTRICT IN SOUTH-CENTRAL ARIZONA IS ONE OF THE MOST PRODUCTIVE COPPER DISTRICTS IN THE SOUTHWESTERN UNITED STATES. COPPER HAS BEEN MINED ALMOST CONTINUOUSLY SINCE 1911 FROM A BROAD MINERALIZED ZONE ALONG A CONTACT BETWEEN PRECAMBRIAN SCHIST AND A TERTIARY GRANITE PORPHYRY. SEVERAL FAULTS DISPLACING THE MINERALIZED ZONE HAVE CAUSED THE ORE TO BE DISCONTINUOUS. THE EASTERN-MOST EXTENSION OF THE MINERALIZED ZONE HAS BEEN DISPLACED VERTICALLY OVER 2000 FEET BY THE MIAMI FAULT. THE MIAMI EAST ORE BODY IS LOCATED WITHIN THIS DISPLACED PORTION OF THE DISTRICT-WIDE MINERALIZED ZONE.

THIS ZONE TRENDS NORTHEAST IN THE WEST AND SWINGS TO A MORE EASTERLY DIRECTION TO THE EAST, FOLLOWING ROUGHLY THE NORTH SIDE OF THE GRANITE PORPHYRY CONTACT. THE EASTERN-MOST ORE FOUND NEAR THE SURFACE WAS THE MIAMI COPPER ORE BODY. THIS ORE BODY WAS TERMINATED BY THE NORTHEAST STRIKING MIAMI FAULT. ON THE HANGING WALL SIDE OF THE FAULT IS THE GLOBE VALLEY BASIN, WHICH IS STRUCTURALLY DEPRESSED BY FAULTING AND COVERED BY A THICK DEPOSIT OF CONGLOMERATE. EXPLORATION IN THIS BASIN FOR THE DISLOCATED ORE, THAT WAS DISPLACED BY THE MIAMI FAULT, LED TO THE DISCOVERY AND DEVELOPMENT OF THE MIAMI EAST ORE BODY.

BUT ONE HAS WELL DEVELOPED FOLIATION AND THE OTHER IS GRANULAR AND CATACLASTIC. THE GRANULAR SCHIST IS LOCATED IN AREAS OF STRUCTURAL DEFORMATION.

QUARTZITE

NORTHWEST OF THE MINE, ON MOONSHINE HILL, ARE OUTCROPS OF THE PIONEER FORMATION RESTING ON THE ERODED SURFACE OF THE PINAL SCHIST. THE PIONEER IS OVERLAIN BY DIABASE. IN SOME LOCATIONS THE DIABASE IS FOUND BETWEEN THE PINAL SCHIST AND THE PIONEER FORMATION.

IN THE DIABASE SILL COMPLEX AT THE MINE IS A QUARTZITE THAT LOOKS MEGASCOPICLY SIMILAR TO THE QUARTZITES FOUND ON MOONSHINE HILL. ABSOLUTE CORRELATION WITH THE PIONEER IS SPECULATIVE BECAUSE THE QUARTZITE FOUND UNDERGROUND AT MIAMI EAST IS IN A STRUCTURALLY COMPLEX LOCATION AND ENVELOPED BY DIABASE. SINCE PINAL SCHIST IN THE MIAMI EAST MINE OVERLIES THE SILL COMPLEX AND QUARTZITE, THE QUARTZITE IS STRATAGRAPHICALLY OUT OF PLACE TO BE IN-PLACE PIONEER FORMATION. IF THE QUARTZITE IS INDEED PIONEER, IT HAD TO BE RAFTED IN BY THE DIABASE FROM A STRATAGRAPHICALLY HIGHER LOCATION INTO THE PINAL SCHIST: THIS IS POSSIBLE BUT UNLIKELY. THE QUARTZITE COULD BE A DISCREET PORTION OF THE PINAL SCHIST WHICH HAS POORLY DEVELOPED SCHISTOSITY, OR THERMAL METAMORPHISM CAUSED BY THE DIABASE COULD HAVE ALTERED THE CHARACTER OF THE SCHIST TO A QUARTZITE. IN ORDER TO DETERMINE THE GEOLOGIC HISTORY OF THE MIAMI EAST ORE BODY WITH RESPECT TO THE MOVEMENT ON THE MIAMI FAULT, A RELIABLE STRATAGRAPHIC HORIZON IS NECESSARY. THIS HORIZON MUST BE RECOGNIZED ON BOTH SIDES OF THE FAULT. THE QUARTZITE COULD BE A HELPFUL STRATAGRAPHIC HORIZON IF CORRELATABLE ACROSS THE MIAMI FAULT, BUT MORE WORK IS NEEDED TO DETERMINE THE TRUE ORIGIN OF THIS FORMATION.

BELIEVED TO BE THE PARENT MAGMA FROM WHICH THE GRANITE PORPHYRY AND IT'S ASSOCIATED HYDROTHERMAL MINERALIZATION ORIGINATED.

GRANITE PORPHYRY PHASE

ALONG THE NORTH AND NORTHWEST SIDE OF THE SCHULTZE GRANITE STOCK IS A LOBE OF QUARTZ MONZONITE PORPHYRY. LOCALLY THIS INTRUSIVE ROCK IS DESIGNATED AS GRANITE PORPHYRY. THE GRANITE PORPHYRY IS REGARDED AS A SEPARATE MARGINAL INTRUSIVE PHASE OF THE SCHULTZE GRANITE. AGE DATING OF THE GRANITE PORPHYRY AND THE SCHULZE GRANITE ALSO INDICATE THE SAME RELATIONSHIP. THE SCHULTZE GRANITE HAS BEEN ASSIGNED A DATE OF 62 M.Y. AND THE GRANITE PORPHYRY 58 M.Y.

THE HYDROTHERMAL SOLUTIONS WHICH FORMED THE COPPER DEPOSITS AT INSPIRATION AND MIAMI COPPER ARE BELIEVED TO HAVE ORIGINATED FROM THE GRANITE PORPHYRY. THE MIAMI EAST DEPOSIT IS ALSO THOUGHT TO HAVE BEEN FORMED BY THE SAME HYDROTHERMAL EPISODE.

AT THE MIAMI EAST PROJECT A GRANITOID ROCK IS ENCOUNTERED ON THE 3300 AND 2900 LEVELS IN THE FOOTWALL BLOCK OF THE MIAMI FAULT. IT IS BELIEVED TO BE A FACIES OF THE GRANITE PORPHYRY.

A GRANITE PORPHYRY OF DIFFERENT PETROLOGIC CHARACTER IS FOUND IN THE SILL COMPLEX. THIS PORPHYRY OCCURS AS SILLS AND DIKES GENERALLY BETWEEN THE TWO LOWEST DIABASE SILLS.

IN THE AREA OF THE ORE BODY THIS FAULT ROUGHLY PARALLELS THE MIAMI FAULT AND PROBABLY JOINS THE MIAMI FAULT AT DEPTH. OFFSETS BY THE FOOTWALL FAULT OF THE LEACHED CAP AND THE CHALCOCITE BLANKET SUGGEST A DISPLACEMENT OF ABOUT 300 TO 500 FEET, BUT POST FAULT LEACHING AND THE EXISTENCE OF POSSIBLE CROSS-CUTTING FAULTS MAKES THESE DISPLACEMENT ESTIMATES VERY SPECULATIVE.

ABOUT 500 FEET EAST OF THE FOOTWALL FAULT IS THE MIAMI EAST FAULT. THIS FAULT HAS AT LEAST 500 FEET OF DISPLACEMENT. IN THE MINING AREA ON THE FOOTWALL SIDE OF THE FAULT IS THE ORE BEARING DIABASE SILL COMPLEX. THE SILL COMPLEX AND THE ORE BODY TERMINATES AT THE FAULT. ON THE HANGING WALL SIDE OF THE FAULT IS PINAL SCHIST. THE SCHIST HAS TRACES OF SULFIDE MINERALIZATION, BUT HAS VERY LOW COPPER VALUES.

SEVERAL NORTHWEST STRIKING FAULTS HAVE BEEN ENCOUNTERED DURING THE DEVELOPMENT OF THE MINE. THESE FAULTS HAVE DISPLACEMENTS RANGING FROM A FEW INCHES UP TO 150 FEET. THE "A" FAULT IS THE BEST DOCUMENTED NORTHWEST STRIKING FAULT. THIS FAULT DIPS NORTHEAST 50 DEGREES AND OFFSETS THE ORE 150 FEET VERTICALLY. OTHER NORTHWEST FAULTS THAT DISPLACE ORE ARE FOUND THROUGHOUT THE ORE BODY.

COPPER MINERALIZATION

THE KNOWN COPPER MINERALIZATION COVERS AN AREA 3000 FEET BY 1500 FEET AND HAS AN AVERAGE THICKNESS OF 320 FEET. THE GENERAL STRIKE OF THE DEPOSIT IS NORTH 35 DEGREES EAST, ROUGHLY PARALLEL TO THAT OF THE DIABASE SILL COMPLEX.

ABUNDANT NEAR THE DIABASE IN THE ENRICHED ZONE AND DECREASES WITH DEPTH AS THE HYPOGENE ZONE IS APPROACHED. SOME SUPERGENE BORNITE WAS DEPOSITED WITH THE CHALCOCITE. SUPERGENE BORNITE IS COMMONLY FOUND IN THE TRANSITION ZONE BETWEEN THE HYPOGENE SULFIDES AND THE WELL DEVELOPED SUPERGENE CHALCOCITE ORES.

ALTHOUGH SUPERGENE ENRICHMENT OF COPPER OCCURS IN ALL ROCK TYPES, IT WAS PARTICULARLY EFFECTIVE IN THE DIABASE. THE SUSCEPTIBILITY OF THE DIABASE TO SUPERGENE ENRICHMENT WAS CAUSED BY SEVERAL FACTORS. THE MOST SIGNIFICANT AND OBVIOUS FACTOR IS THE LOCATION OF THE DIABASE WITH REGARDS TO THE DEPTH OF OXIDATION. THE LEACHED CAP, WHICH COVERS THE MIAMI EAST DEPOSIT, RANGES FROM 50 TO 800 FEET THICK AND AVERAGES ABOUT 400 FEET THICK. IT EXTENDS FROM THE BASE OF THE GILA CONGLOMERATE (AN OLD EROSION SURFACE) DOWN THROUGH A THICK SECTION OF PINAL SCHIST AND THE UPPER PART OF THE DIABASE SILL COMPLEX. ALTHOUGH THE DEPTH OF OXIDATION IS QUITE VARIABLE, AS SHOWN BY THE VARIABLE THICKNESS OF THE LEACHED CAP, THE LOWER LIMITS OF LEACHING IS USUALLY CLOSE TO THE HANGING WALL OF THE LOWEST DIABASE SILL IN THE SILL COMPLEX. REMOVAL OF COPPER FROM THE LEACHED CAP WAS QUITE THOROUGH AND THE ABSENCE OF A SIGNIFICANT ENRICHED OXIDE COPPER ZONE, ALONG WITH A VERY SHARP CONTACT BETWEEN THE LEACHED CAP AND THE SUPERGENE ENRICHED SULFIDE ZONE, SUGGEST A VERY STABLE WATER TABLE DURING THE TIME OF SUPERGENE ENRICHMENT. THE DIABASE SILL STRUCTURES COULD HAVE BEEN A FACTOR CONTRIBUTING TO THE STABILITY OF THE WATER TABLE THROUGHOUT THE ENRICHMENT CYCLE.

TOWARD THE CENTER OF THE DIABASE SILL BIOTITE IS MORE COMMON. CHLORITE IS MORE PREVALENT ALONG THE MARGINS OF THE SILL.

IN PARTS OF THE DIABASE SILL, WHERE PROPYLITIC ALTERATION HAS NOT BEEN DESTROYED OR MASKED BY LATER SERICITIC ALTERATION, VEINLETS AND FRACTURES ARE FILLED WITH DOLOMITE, CALCITE, EPIDOTE, CHLORITE, QUARTZ, AND PYRITE. THESE VEINLETS ARE ALMOST BARREN OF COPPER SULFIDES AND PROBABLY FORMED PRIOR TO THE MAJOR HYPOGENE COPPER MINERALIZATION.

SERICITE-QUARTZ-CLAY ALTERATION

IN THE HANGING WALL BLOCK OF THE MIAMI FAULT THE MOST OBVIOUS HYDROTHERMAL ALTERATION MINERAL ASSEMBLAGE IS THAT OF SERICITE, QUARTZ, AND CLAYS. THIS ALTERATION AFFECTS ALL ROCK TYPES, BUT IS BEST EXPRESSED IN THE GRANULAR PORTIONS OF THE PINAL SCHIST AND IN THE DIABASE. THE INTENSITY OF ALTERATION MAY RANGE FROM WEAK TO STRONG AND IS PERVASIVE THROUGHOUT THE MINE.

IN THE SCHIST, HYDROTHERMAL ALTERATION HAS CONVERTED THE MUSCOVITE AND BIOTITE TO SERICITE AND CLAY. SOME REMOBILIZATION OR ADDITION OF QUARTZ IS ALSO PRESENT. THE SERICITIC ALTERATION IS MORE ADVANCED OR COMPLETE IN THE SCHIST BELOW THE DIABASE SILL COMPLEX THAN IN SCHIST FOUND WITHIN OR ABOVE THE SILL COMPLEX. SILICIFICATION IS MORE PREVALENT THAN SERICITIZATION IN SCHIST WITHIN THE SILL COMPLEX.

IN THE DIABASE, PLAGIOCLASE IS ALTERED TO FINE-GRAINED AGGREGATES OF SERICITE AND QUARTZ. THIS ALTERATION OF PLAGIOCLASE IS PERVASIVE THROUGHOUT THE DIABASE FOUND IN THE MINE. IN LOCATIONS OF MORE

THE INTRODUCTION OF MECHANICAL MINING AT MIAMI EAST

BY

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MIAMI EAST PROJECT

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MIAMI COPPER CO. BEGAN PRODUCTION FROM THE MIAMI INSPIRATION OREBODY IN 1911. AS MINING PROGRESSED IT BECAME EVIDENT THAT THE OREBODY TERMINATED AT THE MIAMI FAULT, AND AN EXPLORATION PROGRAM EAST OF THE FAULT WAS INITIATED. IN 1920 THE #5 SHAFT WAS DEEPENED 800 FEET FROM THE 1000 LEVEL TO THE 1800 LEVEL, AND EXPLORATION DRIFTS DRIVEN ALONG THE FAULT. IN 1922 A WINZE WAS SUNK 800 FEET TO THE 2600 LEVEL AND AGAIN EXPLORATION DRIFTS DRIVEN ALONG THE FAULT. FROM 1923 TO 1925 AN UNDERGROUND DRILLING PROGRAM WAS CARRIED OUT FROM THESE LEVELS, BUT FAILED TO PROVE UP AN ECONOMIC TONNAGE. IN 1927 THE LEVELS WERE ABANDONED, AND THE SHAFT BACKFILLED TO THE 1000 LEVEL POCKET. ALTHOUGH THE 1920'S PROGRAM WAS AN APPARENT FAILURE, IT ACTUALLY TOUCHED THE FRINGE OF WHAT WE NOW KNOW AS THE MIAMI EAST DEPOSIT.

A SECOND ATTEMPT TO LOCATE THE DEPOSIT WAS BEGUN IN 1949 WHEN TWO CHURN DRILL HOLES WERE COLLARED, BUT THE PROBLEM OF CHURN DRILLING 3000' - 4000' DEEP AND SUBSEQUENT HIGH COSTS WERE TOO MUCH.

IN 1968 A SURFACE DIAMOND DRILLING PROGRAM WAS BEGUN TO "ONCE AND FOR ALL" DETERMINE THE EXISTENCE OF AN OREBODY AT DEPTH. COMPLETED IN 1970, THE DRILLING PROGRAM HAD SUCCESSFULLY DEFINED A MINERALIZED ZONE 3000 FEET BELOW SURFACE OF APPROXIMATELY 100 MILLION TONS.

THE DRILLING PROGRAM SUCCESSFULLY DEFINED A ZONE OF MORE COMPETENT ROCK (DIABASE) ON THE HANGINGWALL OF THE MINERALIZED AREA THAT WAS ALSO CONSISTENT WITH "HIGHER GRADE" ZONES OF MINERALIZATION.

WITH THIS ADDITIONAL INFORMATION A NEW MINE PLAN WAS DEVELOPED TO RECOVER 7 MILLION TONS OF HANGINGWALL ORE USING A CONVENTIONAL STOPE AND FILL METHOD. MINING WOULD BE CARRIED OUT ON 40' - 50' SUB LEVELS IN ONLY THE MOST COMPETENT ROCK. DEVELOPMENT RESUMED IN LATE 1979.

DRIFTING IN THE DIABASE ORE ZONE ESTABLISHED THE ROCK TO BE RELATIVELY MORE COMPETENT, BUT VARYING DEGREES OF GROUND SUPPORT WERE STILL REQUIRED. INDICATIONS WERE THAT WHEN MINING COMMENCED, THE PROPOSED STOPE OPENINGS WOULD PROBABLY NEED SIGNIFICANT SUPPORT AND COULD LEAD TO SEVERE GROUND PROBLEMS. REDUCING THE SIZE OF THE STOPES WOULD DECREASE THE SUPPORT AND POTENTIAL GROUND PROBLEMS, BUT THE INCREASED DEVELOPMENT AND PRODUCTION COSTS COULD NOT BE OFFSET.

AS MORE INFORMATION BECAME AVAILABLE, OTHER MODIFICATIONS TO THE MINE PLAN WERE BEING MADE; HOWEVER, IT BECAME OBVIOUS THAT ALTERNATIVE MINING PLANS WOULD ALSO HAVE TO BE EVALUATED.

RATES MIGHT BE IMPROVED BY REDUCING GROUND SUPPORT AND OVERBREAK (NO BLAST EFFECT), DECREASING CYCLE TIMES, AND REDUCING EXPOSURE OF PERSONNEL.

INITIAL INVESTIGATION

DURING EARLIER PROJECT EVALUATIONS SOME INVESTIGATION INTO MECHANICAL MINING HAD BEEN MADE, INCLUDING ROCK STRENGTH TESTS, GROUND OBSERVATIONS, AND FIELD VISITS, BUT THE SYSTEM DID NOT LOOK APPLICABLE. HOWEVER, A REVIEW OF CURRENT LITERATURE INDICATED SIGNIFICANT ADVANCES IN THE "STATE OF THE ART" TOWARD HARD ROCK MINING, AND A THOROUGH INVESTIGATION WAS JUSTIFIED.

INITIALLY MINE OPERATORS, SUPPLIERS, AND CONSULTANTS WERE CONTACTED TO DISCUSS MACHINE CAPABILITIES, GROUND CONDITIONS, AND POSSIBLE APPLICATIONS. WITH FAVORABLE OPINIONS, ON-SITE INVESTIGATIONS WERE MADE AND ADDITIONAL ROCK-STRENGTH TESTING INITIATED. RESULTS WERE ENCOURAGING.

FIELD VISITS WERE THEN MADE TO CALIFORNIA, UTAH, AND WYOMING TO FURTHER DISCUSS MACHINE CAPABILITIES WITH MINE OPERATORS, AND TO ABSERVE A VARIETY OF CONTINUOUS MINING SYSTEMS IN OPERATION.

EVALUATE THE MACHINES CAPABILITIES, THE CUTTABILITY OF OUR GROUND, AND VARIOUS OPERATING AND MAINTENANCE RESULTS. THE MACHINE ARRIVED IN JUNE OF 1981.

THE TEST PROGRAM WOULD BE DIVIDED INTO A SERIES OF THREE(3) PHASES AS FOLLOWS:

PHASE I

THE FIRST PHASE CONSISTED OF MOVING THE MACHINE UNDERGROUND AND ADVANCING THE SUB LEVEL 6 DRIFT FROM THE HANGINGWALL OF THE OREBODY TO THE FOOTWALL. TO PROVIDE REPRESENTATIVE INFORMATION, THE MACHINE WOULD BE CUTTING THROUGH ALL THE ROCK TYPES, IN BOTH ORE AND WASTE, EXPECTED TO BE ENCOUNTERED DURING THE LIFE OF THE MINE. THIS DRIFT WOULD ALSO PARALLEL THE SUB LEVEL 7 DRIFT, WHICH WAS DRIVEN CONVENTIONALLY, AND WOULD ALLOW DIRECT COMPARISONS BETWEEN THE TWO METHODS.

DURING THIS PHASE, WORK WOULD PROCEED ON DEVELOPING MINE PLANS DESIGNED FOR USING THE MECHANICAL MINER ON BOTH DEVELOPMENT AND PRODUCTION.

PHASE II

WITH THE COMPLETION OF THE SUB LEVEL 6 HEADING, THE MACHINE WOULD BE MOVED TO THE RAMP AND TESTED ON 12% - 15% GRADES. THIS WOULD PROVIDE NOT ONLY ADDITIONAL ROCK CUTTING INFORMATION, BUT ALSO THE MACHINES LOADING AND TRAMMING

RIPPABLE (RIP TO FRACTURES).

2. ADVANCE RATES WERE SUBSTANTIALLY FASTER THAN ESTABLISHED CONVENTIONAL RATES.

   a.) IN ALL ROCK TYPES ENCOUNTERED REGARDLESS OF SUPPORT REQUIREMENTS, THE RATES EXPERIENCED MECHANICALLY WERE DOUBLE THE ESTABLISHED CONVENTIONAL RATES.

   .SUB 7 (CONVENTIONAL) AVERAGES 29'/WK
   .SUB 6 (MECHANICAL) AVERAGED 60'/WK
   IN STEEL SETS (2½' - 5' CENTERS):
      .SUB 7(CONVENTIONAL) AVERAGED 25'/WK
      .SUB 6 (MECHANICAL) AVERAGED 48'/WK
   MAXIMUM ADVANCE (MECHANICAL) IN WEEK - 120'
   MAXIMUM ADVANCE (MECHANICAL) IN SHIFT - 18'
   (12' OF ROCK BOLT AND MESH SUPPORT)

IT WAS NOT UNCOMMON TO ADVANCE 10' IN THE SAME TIME IT TOOK TO DRILL OUT A 10' ROUND.

3. REDUCED GROUND SUPPORT REQUIREMENTS

   .DUE TO NO BLAST EFFECT, LESS OVERBREAK, FASTER CYCLE TIMES, AND ABILITY TO CUT DESIRED DRIFT PROFILE.

   .ABLE TO CUT FOR THE ARCH ONLY, BOOM OUT, AND SUPPORT BACK BEFORE ADVANCING FULL FACE.

	CONVENTIONAL	MECHANICAL
a.)	STEEL SETS 2½' CENTERS	3' - 5' CENTERS
b.)	SHOTCRETE	ROCK BOLTS
c.)	ROCK BOLT EVERY 5'-10'	18' - 20'

4. WE EXPERIENCED SOME PROBLEMS LOADING ON CORNERS (CONVEYOR INTO RIB).

PHASE III - TEST STOPING

1) ALL ROCK IN STOPING AREA CUTTABLE AND RIPPABLE.

2) CONFIRMED ADVANCE RATES AND SUPPORT REQUIREMENTS ESTABLISHED IN PHASE I AND II. (ACTUALLY 2 OF 3 TEST STOPES CUT WITHOUT SUPPORT.)

3) BASICS OF MINE PLAN WERE FEASIBLE, HOWEVER:

   a) MAXIMUM ANGLE OF STOPE TO ACCESS DRIFT 60°. (LIMITED BY LENGTH OF MACHINE AND DRIFT WIDTH.)

   b) STOPE PROFILE WOULD VARY WITH GROUND CONDITIONS:

      ... FLAT BACK IN GOOD GROUND - PROVIDE SUFFICIENT SUPPORT AND MAXIMIZE RECOVERY.

      ... ARCH BACK IN BAD GROUND - MORE SUPPORT, SACRIFICE RECOVERY.

   c) VENTILATION SHOULD BE AN EXHAUST SYSTEM TO REMOVE DUST FROM WORK AREAS AND SCRUBBED. (ALSO GOOD SPRAY SYSTEM TO KNOCK DUST DOWN AT FACE.)

4) MOST CHAIN PROBLEMS SOLVED WITH HEAVIER CHAIN AND SPROCKET, AND MORE TENSION ADJUSTMENTS.

5) OTHER MAINTENANCE PROBLEMS IDENTIFIED.

   a) EXPERIENCED

      ... GATHERING ARMS (GEARBOX AND MAINTENANCE) (LUBRICATION AND ABRASIVENESS)

      REDESIGNED THE GATHERING ARMS TO A "SPINNER" ARRANGEMENT WHICH RESULTED IN NOT ONLY REDUCED GEARBOX MAINTENANCE BUT ALSO MORE EVEN LOADING TO THE CHAIN.

      ... ELECTRICAL CONNECTIONS (VIBRATION)

SUMMARY

THE TEST PROGRAM SUCCESSFULLY DETERMINED THE MACHINES CAPABILITIES AND THE CUTTABILITY OF OUR GROUND, BOTH OF WHICH WERE POSITIVE. ADDITIONALLY, THE PROGRAM REVEALED A FEW POTENTIAL PROBLEM AREAS, AND SOME POSSIBILITIES FOR IMPROVEMENT.

THE MAINTENANCE PROBLEMS WERE LIMITED AND WILL BE REDUCED THROUGH MACHINE MODIFICATIONS AND GOOD MAINTENANCE PROGRAMS.

BIT (PICK) CONSUMPTION VARIED WITH GROUND CONDITIONS, AVERAGING $.13/TON. THIS COST WILL BE REDUCED THROUGH CONTINUED EXPERIMENTING WITH BIT (PICK) DESIGN AND THE MACHINE MODIFICATION TO A 2-SPEED CUTTING HEAD. (RANGING FROM $.034/TON TO $1.12/TON AVERAGING $.13/TON, ORIGINAL ESTIMATES OF $.15/TON.)

THE USE OF THE MECHANICAL MINER WILL ALSO HAVE AN EFFECT ON SAFETY AND MINEABLE RESERVES. SINCE (1) THE OPERATOR IS 18' FROM THE FACE, (2) NO EXPLOSIVES ARE REQUIRED, AND (3) GROUND CAN BE QUICKLY SUPPORTED, THE OVERALL EXPOSURE OF PERSONNEL IS SIGNIFICANTLY REDUCED. MACHINE CAPABILITIES WOULD ALSO ALLOW FOR NEARLY 100% RECOVERY OF THE RESERVES, AND CAN SELECTIVELY MINE ADDITIONAL ORE IN AREAS PREVIOUSLY CONSIDERED TOO EXPENSIVE TO SUPPORT IF MINED CONVENTIONALLY.

ONE FURTHER PROBLEM AREA WORTH MENTIONING WAS THAT OF THE CHANGE IN MINING METHODS AND HOW IT WOULD BE ACCEPTED

MIAMI COPPER COMPANY

DIVISION TENNESSEE CORPORATION

IN PLACE LEACHING MIAMI MINE

MIAMI, ARIZONA

Presented before

MILLING DIVISION, ARIZONA SECTION

A. I. M. E.

April 6, 1962

IN PLACE LEACHING MIAMI MINE

by

J. B. FLETCHER

April 6, 1962

In order to give a background to leaching it is necessary to go back to the mining at Miami. The Miami mine started mining in 1910 and finished in July, 1959. (Slide #1) The orebody was divided as follows:

High Grade	24,400,000	tons - Top slicing sub-level caving, etc.
Low Grade	102,000,000	tons - Block Caving
Mixed Ore	9,800,000	" " "
Low Grade #2	23,000,000	" " "
TOTAL	159,000,000	tons

GEOLOGY:

Ore minerals are largely in pre-Cambrian Pinal schist intruded by tertiary Schultz granite porphyry and covered to some extent by Quarternary Gila conglomerate. The structures are highly faulted and shattered. The Miami fault on the east cuts off the ore and the Pinto fault on the southwest caused reoxidation of enriched sulphides producing mixed ore.

MINERALOGY:

The chief mineral is chalcocite, with chalcopyrite, bornite, covelight, malachite, azurite, chrysocola, cuprite, native copper, and molybdenite as minor minerals. The ore minerals occur in seams, veinlets and disseminated particles.

If the ore body was completely mined out in 1959 what was left to leach? (Slide #2) In block caving there is always some dilution and drawing of ore stops when the grade drops past an economical point. This results in a small amount of copper left in the stopes below the capping. There is some copper in the capping, probably below 0.1% or less than 2 lbs. per ton. Over the low grade #2 orebody and portions of the high grade there was an oxide capping which was not mined. The crushed pillars are another source of copper. The leaching at Miami is an attempt to recover this last remaining copper of a worked-out mine.

The enrichment at the Miami mine had gone more nearly to completion than most ore bodies in the southwest. There is practically no pyrite in the capping and very little in the ore.

HISTORY:

Leaching started on a small scale in Dec. 1941. After 20 years of leaching all the answers should be known. This is far from true. Some of the early problems that were met are as follows:

(1) The first leaching was tried through the conglomerate over the high grade mining. No solution came through the fault. There has been mining below the high grade since the first leaching, which has broken the conglomerate for the second time, but this problem has not been solved to date.

(2) The next leaching was tried over the mixed ore body with water. The solution came through with very little copper. Water was tried over the sulphide stopes with no recovery of copper. Then acid was added to the solution over the mixed ore. The first leaching was started with a 3% H_2SO_4 solution which was immediately dropped to a .6% solution. At the present time we try to control the H_2SO_4 feed by the amount of free acid in the pregnant solution. If the acid in the pregnant solution drops much below 0.5 lbs. H_2SO_4 per ton of solution the pumps, pipe lines, sumps, etc., are clogged with iron salts.

(3) In order to leach sulphides it was realized that Fe''' would be necessary, and several attempts were made to manufacture a high Fe''' solution. It was discovered that by adding acid to the off-solution from the precipitating plant, Fe'' was converted to Fe''' as the solution went through the caved ground.

The underground mine was closed down June 24, 1959, at which time full scale leaching was started.

OPERATIONS:

(1) Miami reports assay values in terms of lbs. per ton of solution. Practically, it is a simple term, because all pumps are rated in G.P.M. and we talk in G.P.M. and shifts. 1 gallon = 8 1/3 lbs., therefore 1 G.P.M. for 24 hrs. $= \frac{8\ 1/3 \times 1440}{2000} = 6$ tons.

(2) Requirements:

The Miami plant was designed to treat 2,000 G.P.M. and to produce 1,500,000 lbs. Cu per month or 50,000 lbs. Cu per day. Therefore

2,000 G.P.M. x 6 = 12,000 tons of solution per day

$\frac{50,000}{12,000}$ = 4.17 lbs. Cu per ton of solution

+ 0.04	Tail
4.21	
+ 0.07	Smelter loss
4.28	Required Assay

(3) Leaching can best be thought of in 4 dimensions. The first two dimensions are the surface to be covered. (Slides 3, 4, and 5) We have approximately 5,000,000 sq. ft. to cover. In 20 years we have covered 1/2 the area. 6 years should cover the balance, although that is not the end point of leaching.

Tailing water from the precipitating plant constitutes the most important source of leach solution. Water losses occur principally from evaporation and at Miami they approximate 10%. Fresh water is used as wash water in the precipitating plant and usually supplies the make-up water to maintain the 2,000 G.P.M. from underground. Acid is immediately added to the off-solution at the sump to bring the solution to .5 lbs. H_2SO_4 per ton of solution. This keeps the iron from precipitating out, which gives us a clean sump, pump, and pipe lines to the caved ground. This strength acid does not deteriorate the transite pipe line. When the solution reaches the caved ground it is measured in weir boxes and additional acid is added to bring the strength up to approximately 10 lbs. per ton. From this point on the solution is carried in polyethylene plastic pipe to the point of application. We have used ponds, sprays, and drill holes to introduce the solution to the caved ground. Due to the uneven surface slope encountered in the caved area, sprays are the most satisfactory. Sprays also allow the solution to be introduced slower than by ponding. Drill holes were used in an attempt to introduce the solution below the conglomerate cover east of the Miami fault.

The 3rd dimension is the amount of ore and waste to be penetrated. This averages about 600 ft. Of this 600 ft. the bottom 150 ft. contains the mixture of ore and waste we are leaching. The capping will average about 0.03% Cu. We try to maintain the underground collecting system so that we have a free flow of water in the main haulage ways and we have a few key sampling points. (Slides 6, 7 & 8)

The 4th dimension is the time factor. It takes from 3 to 4 weeks after a spray is turned on to the surface for it to come through on the 1000 level. The same spray will drop off in about 2 weeks after it is turned off.

(1) The next point is how long can the spray remain at one place before the grade drops.

This depends on the amount of ore in the area being leached. It is practically impossible to correlate the mining extraction records with the material to be leached. So when a new area is to be leached the spray is turned on and there is a waiting period of 3 to 4 weeks before a sample can be taken underground.

The sprays are left in one place until the grade from that area drops below 3 lbs. Cu per ton. The spray is then turned off and the area allowed to lie idle for a period of time. The rest and leach periods are alternated over and over. The second time over an area we can remain about one-half the time used originally and recover about one-half the amount of copper. The third time over it is one-half the second, the fourth is one-half the third, etc.

There are several explanations for this. During the rest period oxidization of the sulphides may take place, but since we are leaching with an acid ferric sulphate solution and from observed action on the surface it appears that a reverse capilliary action is the best explanation. A copper bearing boulder can be put in a flow of leaching solution, then the solution removed, and as the boulder begins to dry, a copper oxide coating forms on its surface. When a copper bearing rock is broken, it usually cracks on the mineralized seams and shows that the solution is actually penetrating the rock. (Slide #9)

The sulphide areas do not give as high grade a pregnant solution as the oxide areas. The leaching of chalcocite, the predominate ore at Miami, seems to take two steps, which are expressed in the following reactions:

(1) $Cu_2S + Fe_2(So_4)_3 = CuSo_4 + 2\ FeSo_4 + CuS$

(2) $CuS + Fe_2(So_4)_3 = CuSo_4 + 2\ FeSo_4 + S$

The first reaction takes place fairly rapidly. The second reaction is much slower. Chalcopyrite, a minor ore mineral at Miami, does leach but it is very difficult to set up equations for the reactions.

The time to allow for an area to rest is difficult to determine due to the varying depth of the column being leached, the type and amount of mineralization. With a depth of 600 ft. the minimum rest period seems to be three months.

To date the Miami Mine has produced by leaching 140,000,000 lbs. of copper. The iron to copper ratio = 1.3 and the acid to copper ratio = 2.4, both figures based on net smelter returns. It is apparent that the largest factors affecting the cost per lb. of Cu are the iron and acid consumption. If the acid in the pregnant solution becomes too high the iron consumption per lb. of Cu will go up. (The ferric iron in the pregnant solution also affects the iron consumption.) If the acid strength is too high the gangue consumption of acid will go up. The acid strength is kept below 15 lbs. even in the high grade oxide areas unless the ore is very close to the surface. The time lag between the solution entering the caved ground and the same solution appearing underground where it can be sampled and assayed makes this problem of control difficult. The best way we have found is to keep graphs of the assays of the underground sample points. From these graphs we can check the trend and anticipate a drop in grade or a decrease in acid consumption.

Iron salts do precipitate as the solution goes through the caved material. From the attached typical assay report it can be seen that iron loss from the cave area feed to the precipitation plant feed is 4.2 lbs. Fe per ton of solution. It has been our experience that there is no plugging action of the cave ground from these basic iron salts. It is not known where the iron precipitates, but it is probably high in the column.

The precipitating plant (Slides 10 to 19) at Miami is similar to other wash down plants in the southwest. The only difference is the method of cell loading. De-tinned shredded cans are received by rail and unloaded by a magnet. The magnet drops the cans into a hopper which feeds a belt conveyor. The belt travels over the center of the cells, and by means of a traveling tripper the cells are evenly charged with iron. The precipitated copper is washed from the cans through wooden screens by high pressure water. The precipitates discharge onto a decant slab. Washing is done on the day shift only. Solutions are drained from the precipitated copper and pumped back to the cells on afternoon shift. The copper is moved from the decant slab to a drying slab early the next morning before washing starts. The copper is moved with a front-end loader. The same loader loads the dry copper into the railroad car. The copper is shipped to the smelter at Douglas.

The plant operates every day in the year. The copper recovery at the plant = 99% or an 0.04 lbs. copper per ton solution tail. The precipitates shipped average 23.5% moisture and assay 79% copper.

The operating crew consists of the following:

Precipitating Plant 7-day operation	(1 operator or pumpman	3 shifts
	(1 equipment operator	day shift
	(1 cave area pumpman	day shift
	(3 cell washers	day shift
	(1 foreman	day shift
Underground 5-day operation	(3 timber crews 6 men	day shift
	(2 hoist engineers	day shift
	(1 foreman	day shift
Maintenance crews from Copper Cities 5-day operation	(1 pipeman	day shift
	(1 machinist	day shift
	(1 mechanic	day shift

The over-all recovery by leaching at Miami will be impossible to determine as we do not know exactly how much copper we had to start. A percent recovery figure for leaching is a misleading figure. A 50% recovery of a 0.2% copper orebody would be considered good, but a 50% recovery of a 1% orebody would be poor.

The first 15% or 20% of the copper in place is very easy to recover. It is only possible through careful attention to details to maintain a steady production and to get a good over-all recovery by this method.

In order to give a complete picture of the leaching at Miami a flow sheet, a typical assay report and a sheet of leaching reactions are attached.

MIAMI COPPER COMPANY
DIVISION TENNESSEE CORPORATION

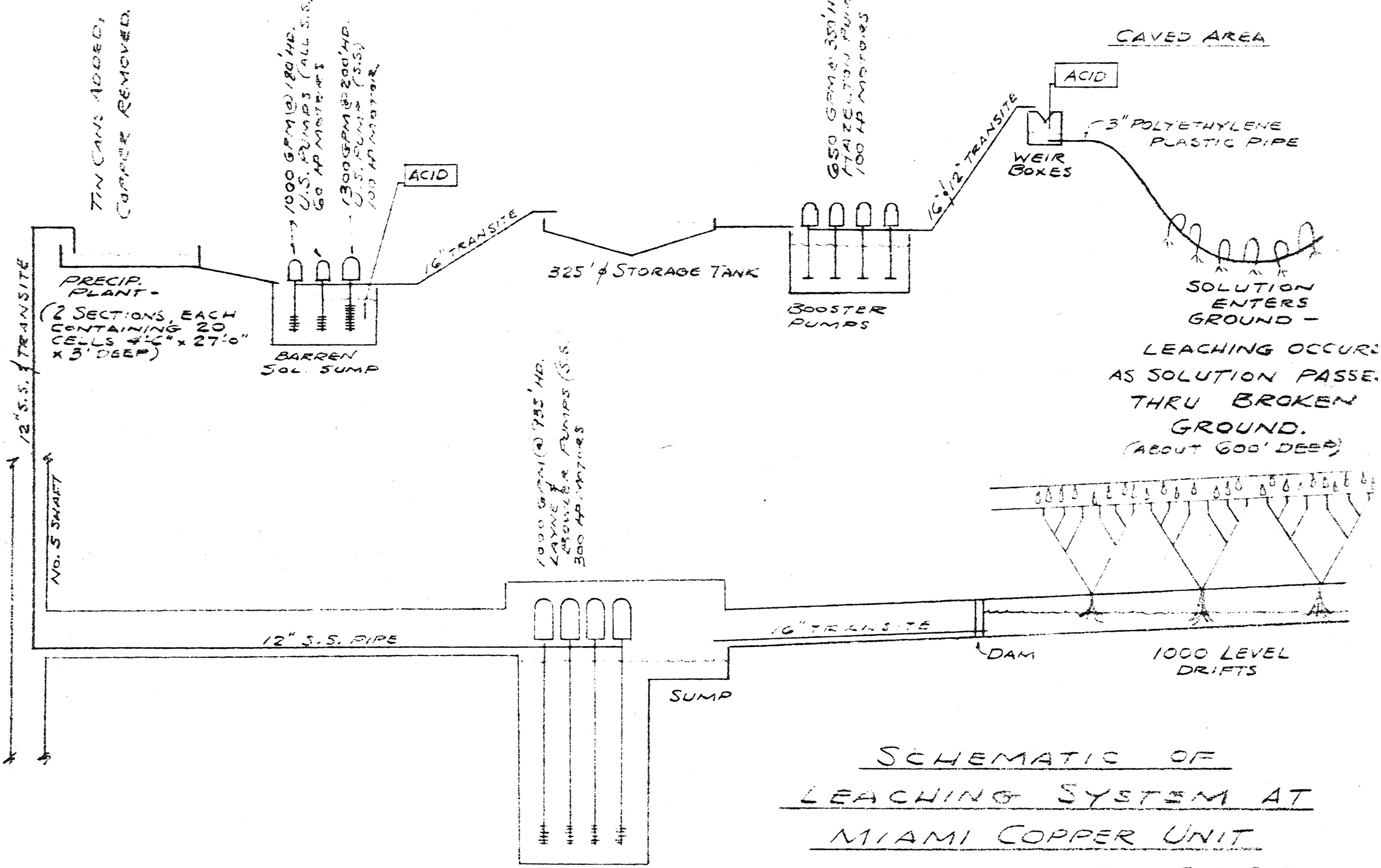
CERTIFICATE OF ASSAY

TYPICAL 1962

LEACHING PLANT

Beaker Nos.	MARKS, ETC.	SAMPLE	CU	FE	FE''	FE'''	H_2SO_4	P.H.
		GMS	- POUNDS PER TON -					
	Feed ABC		4.28	4.4	2.3	2.1	0.5	2.4
	Tail A		0.04	9.4	9.0	0.4	Tr.	4.0
	B		0.05	9.2	9.1	0.1	Tr.	4.2
	C		0.03	9.7	9.5	0.2	Tr.	4.3
	C A Feed ABC		0.03		8.6		10.0	1.4
	Sump A		0.04	9.3	8.5	0.8	0.6	2.7
	B		0.03	9.2	8.5	0.7	0.5	2.8
	C		0.06	9.6	8.8	0.8	0.5	2.8

_______________ Assayer



SCHEMATIC OF LEACHING SYSTEM AT MIAMI COPPER UNIT

ALL PUMPS ARE TYPE 316 S.S.

LEACHING REACTIONS

$CuO + H_2SO_4 = CuSO_4 + H_2O$
$Cu_2O + Fe_2(SO_4)_3 + H_2SO_4 = 2\ FeSO_4 + 2\ CuSO_4 + H_2O$
$3\ CuO + Fe_2(SO_4)_3 + 3\ H_2O = 3\ CuSO_4 + 2\ Fe\ (OH)_3$
$4\ CuO + FeSO_4 + 6\ H_2O + O_2 = 4\ CuSO_4 + 4\ Fe\ (OH)_3$
$CuO + FeSO_4 + H_2O = CuSO_4 + Fe(OH)_2$

PRECIPITATION

$CuSO_4 + Fe + FeSO_4 = Cu + 2\ FeSO_4$

SULPHIDES

$Cu_2S + Fe_2(SO_4)_3 = 2\ CuSO_4 + 4\ FeSO_4 + S$
$Cu_2S + Fe_2\ (SO_4)_3 = CuSO_4 + 2\ FeSO_4 + CuS$
$CuS + Fe_2\ (SO_4)_3 = CuSO_4 + 2FeSO_4 + S$

CONVERSION - FERROUS TO FERRIC

$4\ FeSO_4 + 2\ H_2SO_4 + O_2 = 2\ Fe_2\ (SO_4)_3 + 2\ H_2O$

OXIDATION OF PYRITE

$4\ FeS_2 + 11\ O_2 = 2\ Fe_2\ O_3 + 8\ SO_2$
$(2\ FeSO_4 + SO_2 + O_2 = Fe_2\ (SO_4)_3$
$(Fe_2(SO_4)_3 \quad + SO_2 + H_2O = 2\ F_2SO_4 + 2\ H_2SO_4$
$2\ FeS_2 + 2\ H_2O + 7\ O_2 = 2\ FeSO_4 + 2\ H_2SO_4$

MINERALS

AZURITE - $Cu_3\ (OH)_2.(CO_3)_2 + 3H_2SO_4 = 3\ CuSO_4 + 2\ CO_2 + 4\ H_2O$

MALACHITE - $Cu_2\ (OH)_2 \cdot CO_3 + 2\ H_2SO_4 = 2\ CuSO_4 + CO_2 + 3\ H_2O$

CHRYSOCOLLA - $CuSiO_3 \cdot 2\ H_2O + H_2SO_4 = CuSO_4 + SiO_2 + 3\ H_2O$

CUPRITE - $Cu_2O + H_2SO_4 = CuSO_4 + Cu + H_2O$
$Cu + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4$
$Cu_2O + H_2SO_4 + Fe_2(SO_4)_3 = 2\ CuSO_4 + H_2O + 2\ FeSO_4$

CHALCOCITE - $Cu_2S + Fe_2(SO_4)_3 = CuS + CuSO_4 + 2\ FeSO_4$
$CuS + Fe_2(SO_4)_3 = CuSO_4 + 2\ FeSO_4 + S$
$Cu_2S + 2\ Fe_2(SO_4)_3 = 2\ CuSO_4 + 4\ FeSO_4 + S$

COVELLITE - $CuS + Fe_2(SO_4)_3 = CuSO_4 + 2\ FeSO_4 + S$

BORNITE - $Cu_5FeS_4 = FeS + 2\ Cu_2S + CuS$. Reactions are probably as for chalcocite. Iron is also attacked.

CHALCOPYRITE AND PYRITE - Not attacked by acid ferric sulphate solutions.

HYDROLYSIS

$4\ FeSO_4 + O_2 + 2\ H_2O = 4\ FeSO_4(OH)$
$3\ Fe_2(SO_4)_3 + 14\ H_2O = Fe_2\ O_3 \cdot 4\ SO_3 \cdot 9\ H_2O - 5\ H_2SO_4$
$Fe_2(SO_4)_3 + 6\ H_2O = Fe_2O_3 \cdot 5\ H_2O + H_2SO_4$
$2\ Fe_2(SO_4)_3 + 18\ H_2O = Fe_2O_3 \cdot 5\ SO_3 \cdot 17\ H_2O + H_2SO_4$