MAGMA COPPER COMPANY

THE SAN MANUEL MINE

GENERAL INFORMATION BOOKLET

San Manuel Division

San Manuel, Arizona

MAGMA COPPER COMPANY
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SAN MANUEL, ARIZONA

Orebody

Disseminated mineralization in monzonite porphyry and quartz monzonite.

Principal Sulphide Minerals: Chalcopyrite, chalcocite, pyrite and molybdenite. The grade of the ore is less than 1%.

Overburden: 0 to 1,900 ft. thick. Average over the Southeast (Main) Orebody is 670 ft. thick, and consists mainly of a very competent rock called Gila Conglomerate.

[The last two pages of this booklet show sections of the orebody.]

Mine Power

Power is supplied to the Mine No. 3 Hoist Area Substation at 115 Kv from the Plant Substation over a feeder approximately 6.5 miles long. The Mine feeder conductors are 795 MCM ACSR with a current carrying capacity of 900 amps.

The No. 3 Hoist Area Substation Main Transformer is rated at 50/66.5 MVA, 115/46 Kv, 3-phase. This transformer supplies the 46 Kv bus at the Substation and the feeder to the No. 1 and No. 4 Shaft Area located approximately one mile away. The 46 Kv bus at the No. 3 Hoist Area Substation supplies several banks of single phase 46/2.4 Kv, 2,500 Kva and 7,500 Kva transformers for all surface and underground power requirements in this area.

The 46 Kv feeder to the No. 1 and 4 Shaft Area feeds three secondary substations supplying all surface and underground power requirements. These include No. 1 and No. 4 hoists, compressor building and all service requirements. In addition, at the powerhouse substation in this area, provision is made to tie in the mine standby generating system to the 46 Kv system.

No. 1 Shaft

Depth: 2,833 ft. (For the Third Lift.)

Dimensions: 25'-5 7/8" x 6' -- 4 Compartments:
2 Hoisting, 6'-5½" x 6'-0".
1 Manway, 5'-0 1/8" (Below 1475).
1 Service Cage, 5'-0" (Below 1475).

Ventilation downcast.

Structural steel sets in reinforced concrete.

Headframe: 100'-6" to ℄ sheaves.
Two 12' diameter steel sheaves.

Skips: 4-ton capacity in counterbalance, Kimberly type.

Hoists: Main Hoist, two 200-hp motors, double drum.
Rope speed = 800 fpm, with Lebus wind.
Rope size = 1-1/8".

Service Hoist, single drum, 200-hp.
Rope speed = 545 fpm.
Rope size = 1".

This shaft is used to hoist rock from the development headings. Manway compartment contains main pump discharge columns carrying most of the water pumped from the mine and power transmission lines. The service compartment has three 8-in. heavy-duty pipes for passing concrete from the automated concrete batching plant at the collar of the shaft to the underground mining levels.

No. 3A-3B Shafts

Twin ore hoisting shafts.

Depth: 2,305 ft. (For the Second Lift.)

Dimensions: 29' x 7' inside concrete.
Four compartments, each 6'-6" x 7'-0".

Ventilation exhaust shafts, 195' apart.

Structural steel sets poured in reinforced concrete for smooth lining. Reinforced concrete curtain walls between each compartment, with 3'-7½" x 4' windows in each set.

No. 3A-3B Shafts -- continued

In each of the two hoisting compartments steel hat-section guides are of Corten steel. These guides are supported every three ft.

Timber guides in service compartment are supported every 6'-0".

Headframe: 181' to ℄ sheaves.
14' diameter cast steel sheaves.

Coarse Ore Bins:

Diameter:	60'
Height:	66'-6"; top of bin 92' above collar.
Capacity:	750 tons total.
Loading Gates:	2 on each track per bin, air operated. 2 loading tracks.

Fine Ore Bins:

Diameter:	65'
Height:	81'-6"; top of bin 107' above collar.
Capacity:	10,000 tons total.
Loading Gates:	6 on each track per bin, air operated. On same two tracks as coarse ore bins.

Loading Pockets:

One in 3A; one below 2075 Level, capacity = 1,500 tons.
One in 3B below 2075 Level, capacity = 1,500 tons.

Skips: Bottom - dump Corten steel skips, with alloy steel liners, running on solid rubber tires.

Capacity:	23 tons with $\pm$ 4.0% moisture.
Dimensions:	35'-5" long, 6'-1" wide, 6'-3" deep.
Weight:	30,000 lbs. (Approximate)

Automation: Skip loading and hoisting are fully automated. Skip loading and dumping are viewed on closed circuit television, and the car dumping and hoisting systems are monitored on a control panel.

Hoists: Double drum with Lebus wind; automatic or manual operation; 15' diameter drums having a 109" face, spooling 4,700 ft. of 2-1/4" rope in two layers.
Drum Shaft: 27" diameter through drums.
Hoisting Speed: 2,800 fpm.
Two 3,000-hp DC motors equipped with MG set consisting of one 4,000-hp motor and two 2,500 KW DC generators. Steel plate flywheel, 44 tons (approximate) for 80% power peak equalization.

Service hoists are equipped with an 18-passenger cage and 45 cubic ft. skip combination with solid rubber tires running on timber guides. Hoisting rope is 1-1/8". The cage and skip are counterbalanced by an 8,400-lb. counterweight in the shaft manway.

Primary Crusher: The 460' x 50' crusher building houses three 42 x 65 gyratory crushers. There are three 96" x 62'-0" pan feeders, with a capacity of 1,500 tons per hour; one fed from the 3A, one from the 3B and the other from the 3C coarse ore bins. At 3A and 3B, 48" conveyor belts transfer the crushed ore from the crusher building to transfer towers (222'), and back to the fine ore bins (421'). At 3C, 48" conveyor belts transfer the crushed ore from the crusher building to transfer towers (169'), and back to the fine ore bins (365'). Skip dumping, fine ore bin capacity, and transfer belts are viewed on closed circuit television. The entire automated crusher is monitored from control panels at each crusher location.

No. 3C-3D Shafts

Identical ore hoisting shafts. 3C is 195 ft. north of 3A; 3D is 195 ft. south of 3B.

Depth: 3C = 2,859 ft. The sinking and furnishing of this shaft was completed April 12, 1971. Hoisting from 3C started in August, 1971.
3D = 3,700 ft. Shaft sinking is in progress.

Dimensions: 22'-0" inside diameter, circular, concreted.
Two ore hoisting compartments, service cage, manway and pipe compartments.

Ventilation exhaust shafts.

Steel box-section Corten guides supported every 12 ft. in hoisting compartment.

Douglas fir timber guides in service compartment supported every 6 ft.

Headframe: Structural box members of Corten seeel.
15' diameter fabricated steel sheaves.

Skips: Bottom - dump Corten steel skips, alloy steel liners with solid rubber tires running on Corten steel box-section guides.
Capacity: 29 tons.
Dimensions: Length, 37'-4½"; Width, 6'-8"; Depth 6'-3½"; Weight: 30,860 lbs.

Automation: Same as 3A-3B.

Hoists: Double drum with Lebus wind; automatic or manual operation, 15' diameter drums having a 109" face, spooling 2-1/4" wire rope.
Hoisting Speed: 2,800 fpm.
Each hoist has two 3,500-hp DC motors powered from a MG set consisting of one 6,000 - hp 514 rpm synchronous motor driving two 2,800 KW generators. MG set has no flywheel.

3C Service Hoist: Single drum, manually operated, powered by a 250-hp DC motor capable of handling a 8,000-lb. load at a depth of 3,600 ft. Adjustable DC voltage for driving the hoist motor will be supplied by silicomatic I-power conversion equipment converting AC to DC power through silicon control rectifier cells.

No. 4 Shaft

Depth: 2,730 ft.

Dimensions: 26'-6" x 14'. Structural steel sets are poured in concrete for smooth lining.

Two cage compartments, each 14' x 8'; two rounded end compartments for manway, pipes, electric cables and ventilation. Manway compartment also contains the main compressed air line supplying the mine.

Man and supply shaft, and downcast ventilation.

Cages: 2 decks, 50 men per deck.
Inside dimensions: 6'-9½" x 13'-6".
Rated Capacities: 20,000 lbs. supplies per deck.
12,000 lbs. men per deck.
Cage Weight: 20,000 lbs. (approximate)

Headframe: 109 ft. to ℄ sheaves.
Two 14' cast steel sheaves.

Hoist: Double drum.
15' diameter, 90" face drums.
2-1/4" hoisting rope.
Maximum hoisting speed: 1,500 fpm.
Single reduction drive, two 700-hp DC motors equipped with MG set consisting of one 1,750-hp AC motor and two 600-KW DC generators. Hoisthouse equipped with 30-ton crane, with 5-ton auxiliary.

No. 5 Shaft

Depth: 4,384 ft. at final sinking.

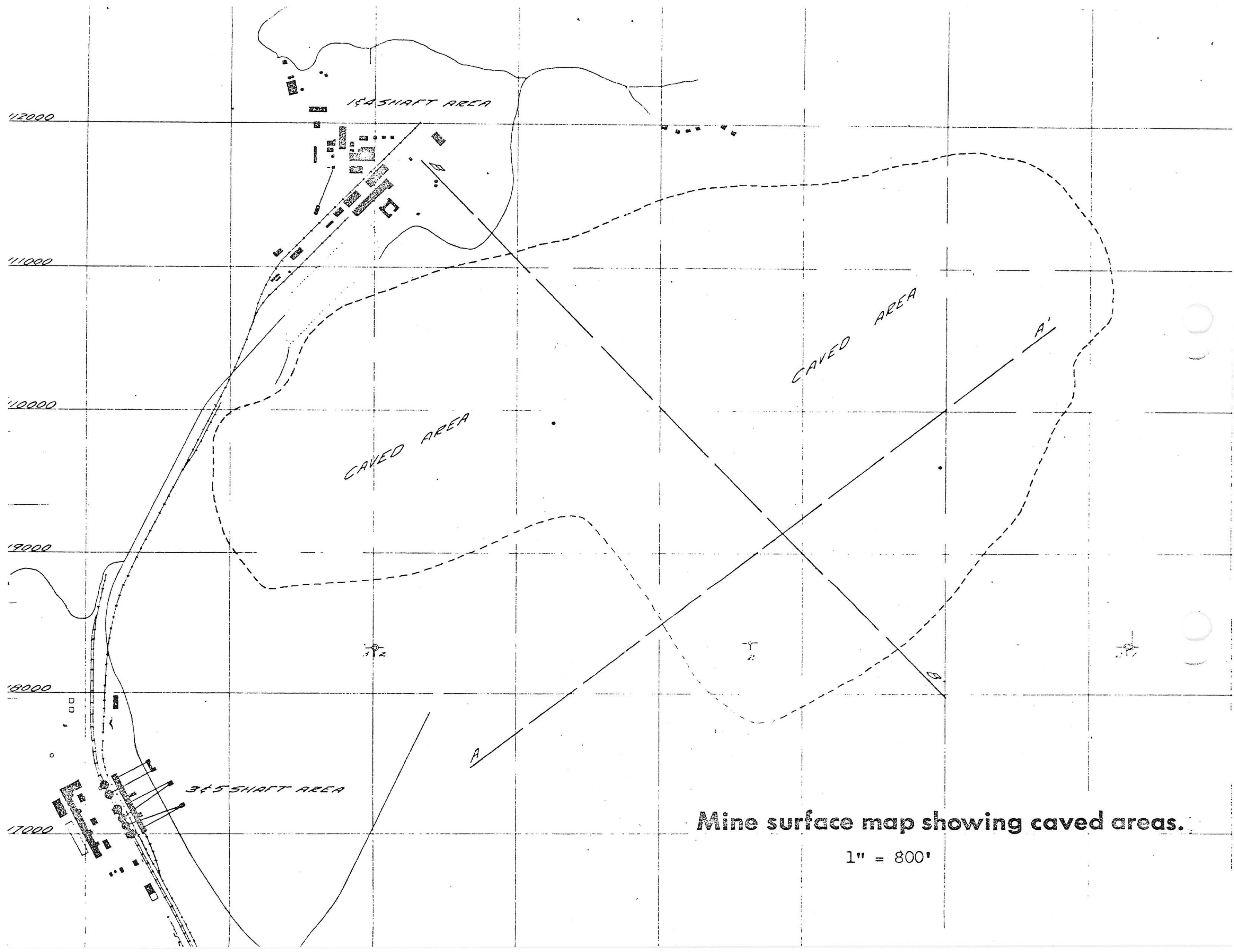
Dimensions: 25' diameter inside the concrete lining.

Of the four hoisting compartments, two will be used for handling men and supplies, and two will be used for hoisting waste rock. The shaft will also have compartments for manway, pipe lines, electric cables, and concrete transportation lines.

Man and supply shaft, downcast ventilation.

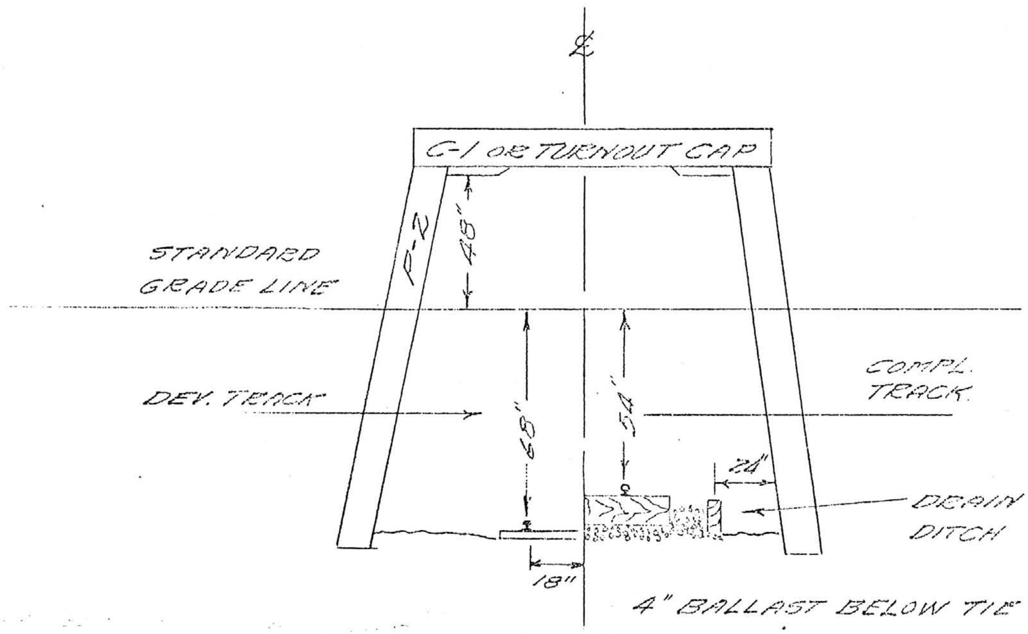
Hoists:
Double drum production hoist powered by one 1,000-hp DC motor.
9' diameter, 103.5" face.
1-3/8" hoisting rope.
Hoist speed: 1,925 fpm.
Skip capacity: 5.5 tons.
A static thrystor rated at 1,000 KW will replace the normally-used MG set to supply power to the production hoist motor.

Service Hoist:
Double drum service hoist (for men and materials), powered by two DC motors, each having 1,000-hp.
15' diameter, 105" face.
2-1/8" rope.
Hoist speed: 1,856 fpm.
Man cage: Similar to that at No. 4 Shaft.



Mine surface map showing caved areas.

1" = 800'

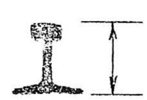


POSTS: 12" x 12" x 10'-4"
CAPS: 12" x 12" x 10'
Sets on 5' centers.

CREWS: Single Heading - 3 men.
Two Headings - 4 men.
44-48 holes drilled per set.
50 lbs. ± of Ammonium Nitrate primed with a stick of 60% Amogel per hole. Usually fuse blasted.

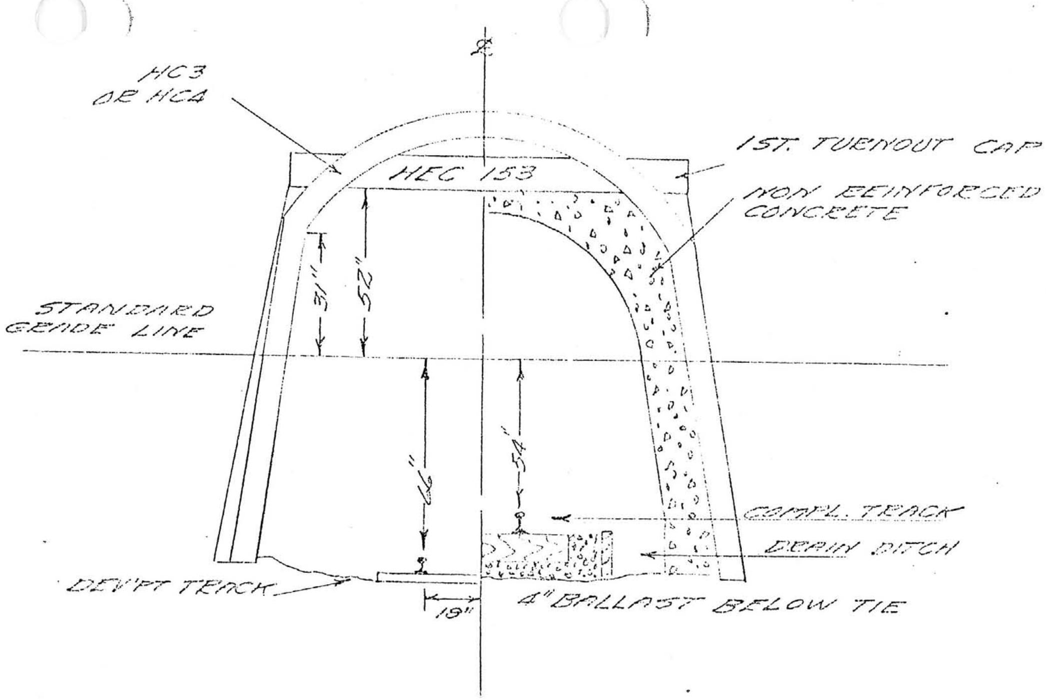
Timber Haulage Drift - 45# Rail for Development.
- 90# Rail finished track.

Timber Ladder Drift - If 75# rail is used, 64" is the grade.
45# rail for Development.
75# rail finished track.



Rail Measurements

3-3/4" for 45# rail.
4-13/16" for 75# rail.
5-5/8" for 90# rail.

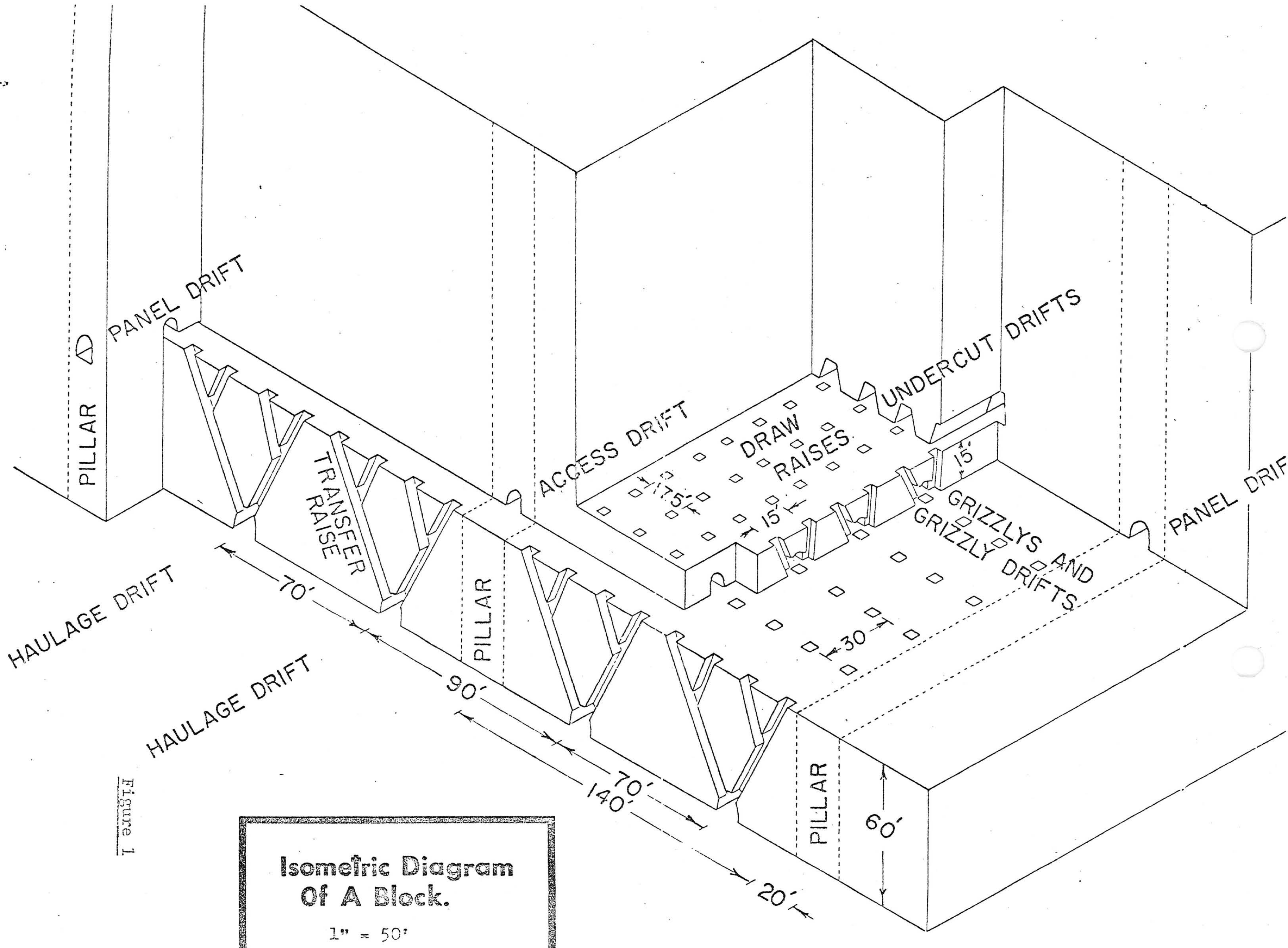


4" or 6" W.F. arch caps and
9'-6" posts for initial
ground support.
Sets on 5' centers.
Crews same as Timber Haulage Drift.

Initial Ground Support
4" Steel Set (HC-3) or
6" Steel Set (HC-4)

Concrete Ladder Drift - If 75# rail is used, 64" is the dev'pt grade.
45# rail for Development.
75# rail finished track.

Concrete Haulage Drift - 45# rail for Development
90# rail finished track.



Isometric Diagram Of A Block.

1" = 50'

Figure 1

Gravity Flow Transfer Raise

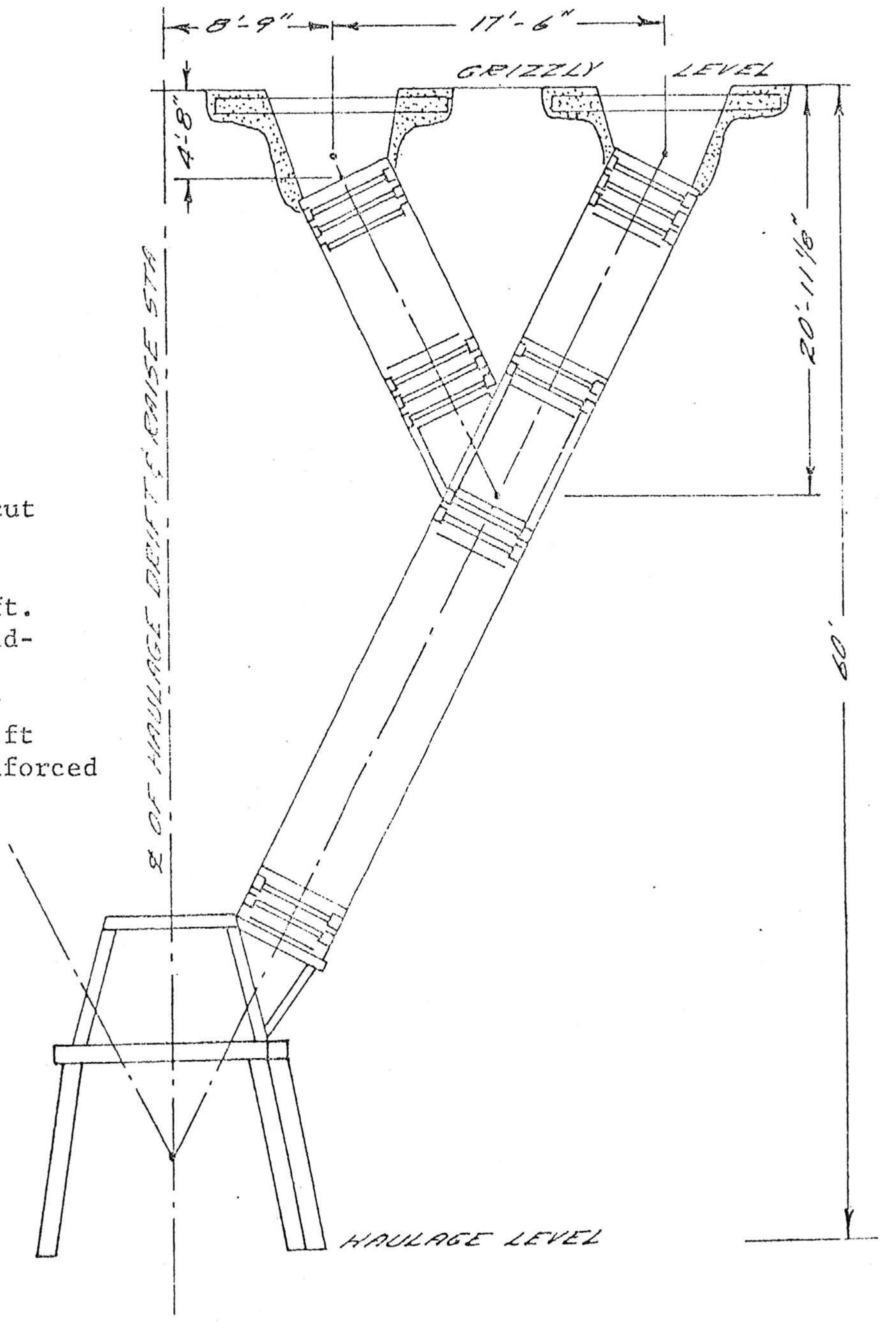
4' x 4' inside.

LINING:
   6" x 8" cribbing, armored with 3" x 4" x 1/4" steel angles and "T"-irons.

INCLINATION:
   63°

LENGTH:
   48"

CREWS:
   2-man crews advance two raises at once, using 3" stopers.
   14 holes drilled 5' deep; 35 lbs. 60% Amogel used per round, detonated with electric caps.
   Ventilation with compressed air.

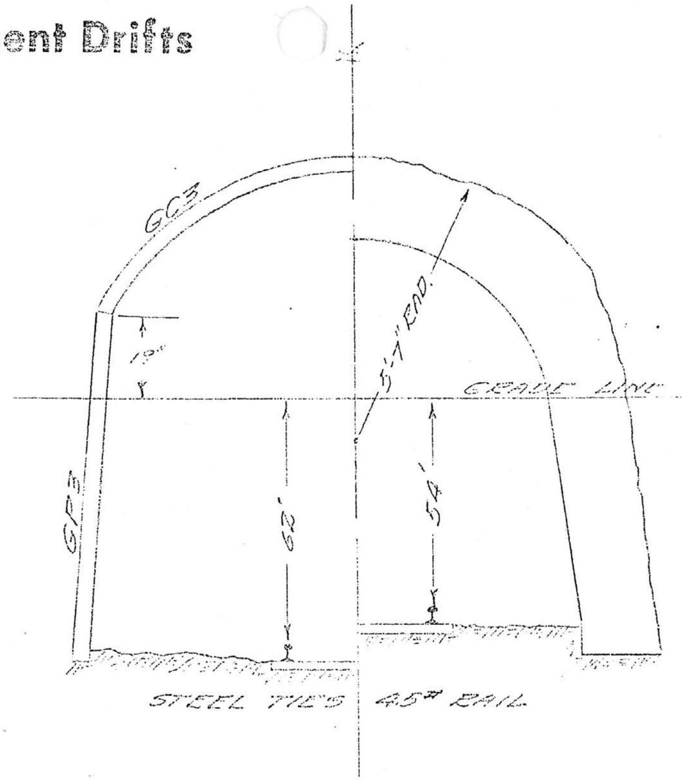


Transfer Raise Stations are cut by the regular drift crew using the same equipment they use to drive the drift.
They are cut as the drift heading is advanced.
Raise Stations are steel pony sets on top of special drift sets. Lined with non-reinforced concrete.

Grizzly Level Prime Development Drifts

Excavation only:
Rock bolts and steel straps
GC-3, GP-3 are used
as pre-concrete support.
SIZE: 10'-3" x 9'-1".
Each round marked on face by Boss.

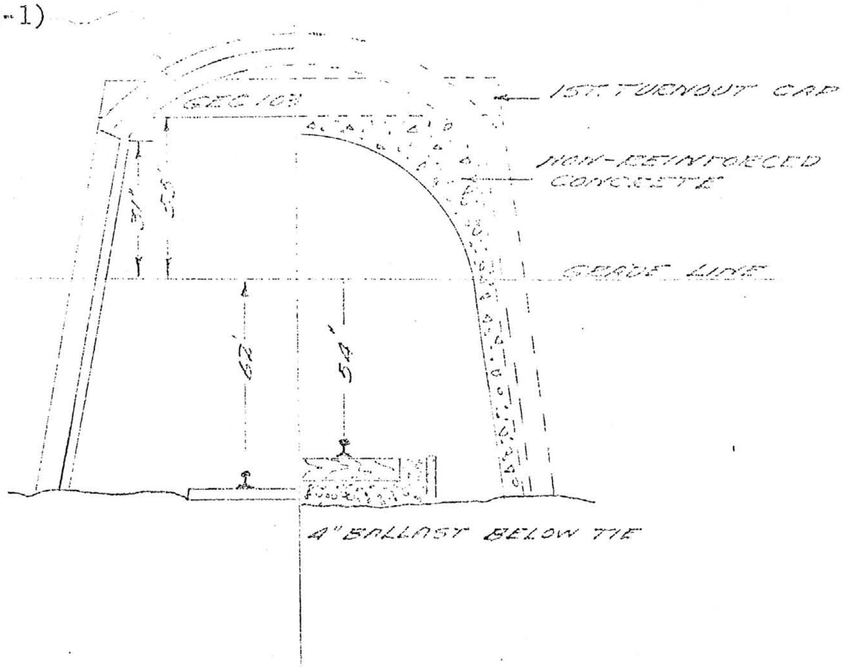
Two-man crews, or 3 men in two headings.
44-48 holes drilled 6' deep.
40 lbs. ammonium nitrate primed
with a stick of 60% Amogel per
hole. Usually fuse blasted.
Ventilation with fans and tubing.



CONCRETE PANEL DRIFT

Initial Ground Support:
4" Steel Set (GC-3) or
6" Steel Set (GC-1)

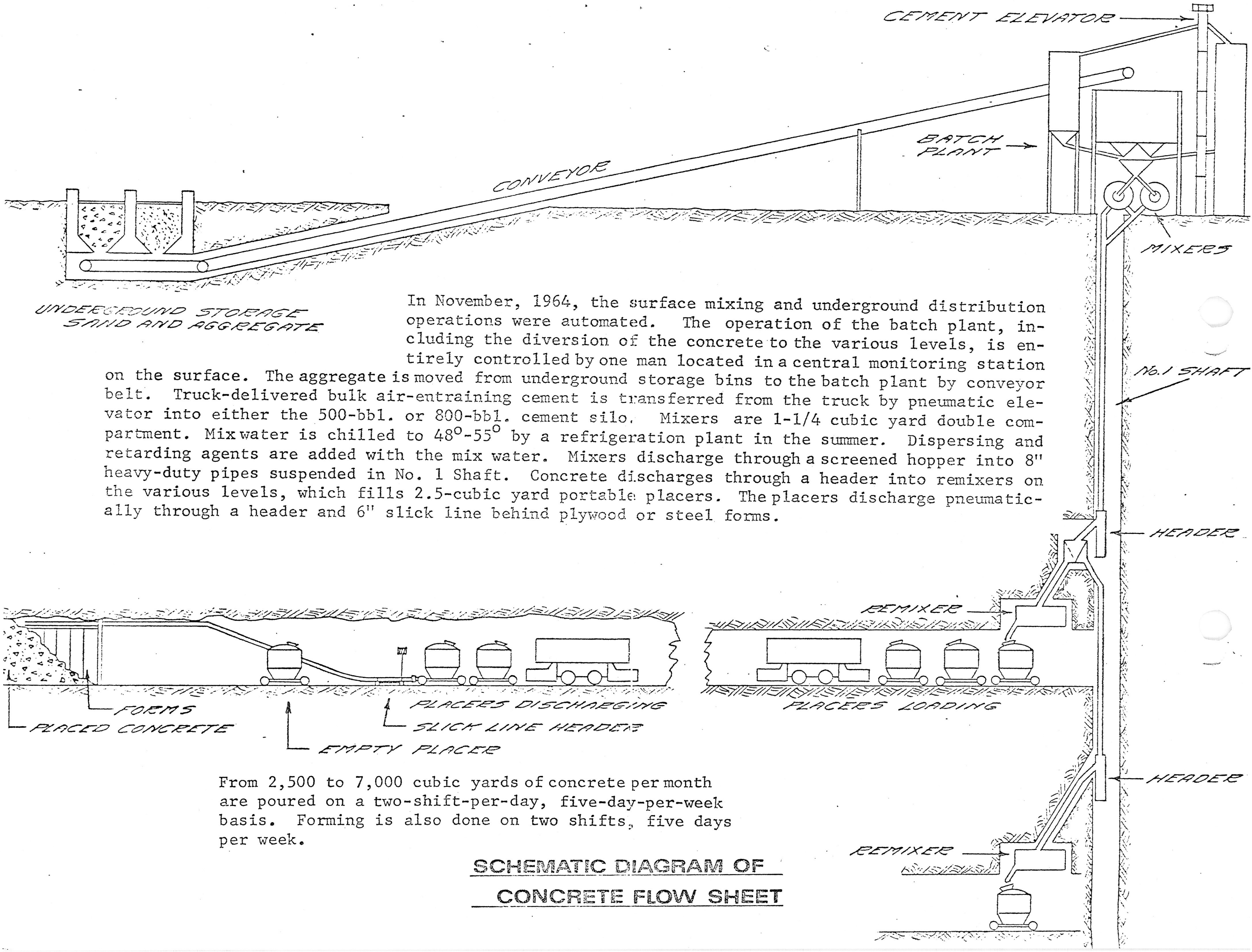
CONCRETE FRINGE DRIFT



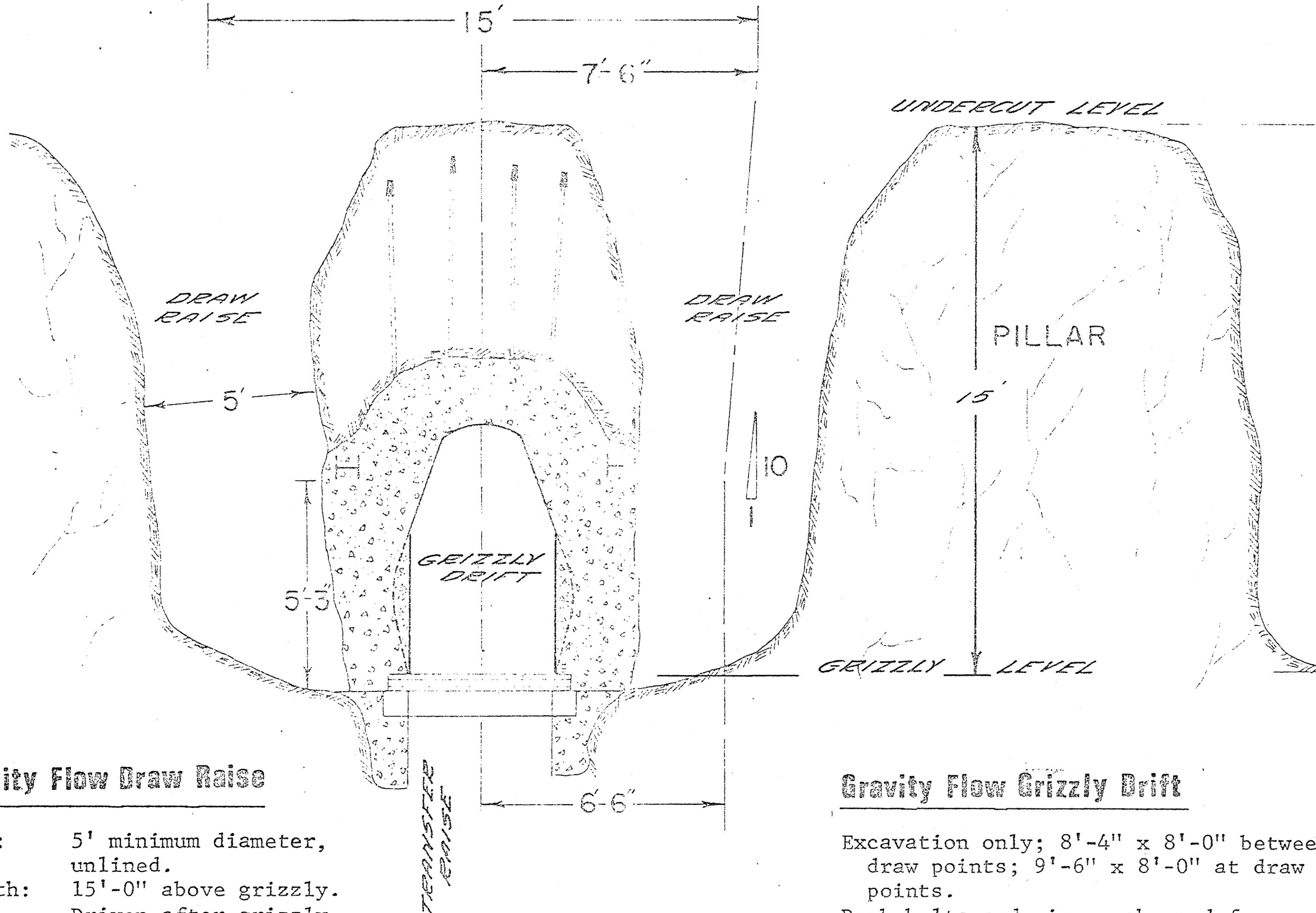
45# rail and steel ties for
Development and Finished Track.

In November, 1964, the surface mixing and underground distribution operations were automated. The operation of the batch plant, including the diversion of the concrete to the various levels, is entirely controlled by one man located in a central monitoring station on the surface. The aggregate is moved from underground storage bins to the batch plant by conveyor belt. Truck-delivered bulk air-entraining cement is transferred from the truck by pneumatic elevator into either the 500-bbl. or 800-bbl. cement silo. Mixers are 1-1/4 cubic yard double compartment. Mix water is chilled to 48°-55° by a refrigeration plant in the summer. Dispersing and retarding agents are added with the mix water. Mixers discharge through a screened hopper into 8" heavy-duty pipes suspended in No. 1 Shaft. Concrete discharges through a header into remixers on the various levels, which fills 2.5-cubic yard portable placers. The placers discharge pneumatically through a header and 6" slick line behind plywood or steel forms.

From 2,500 to 7,000 cubic yards of concrete per month are poured on a two-shift-per-day, five-day-per-week basis. Forming is also done on two shifts, five days per week.



SCHEMATIC DIAGRAM OF CONCRETE FLOW SHEET



Gravity Flow Draw Raise

Size: 5' minimum diameter, unlined.

Length: 15'-0" above grizzly. Driven after grizzly drift is concreted.

Crews: 2-man crew drives several raises at once. 11 to 24 holes drilled 4' to 6' deep. 27 lbs. ammonium nitrate with each hole primed with a stick of 60% Amogel. Electrically blasted.

Gravity Flow Grizzly Drift

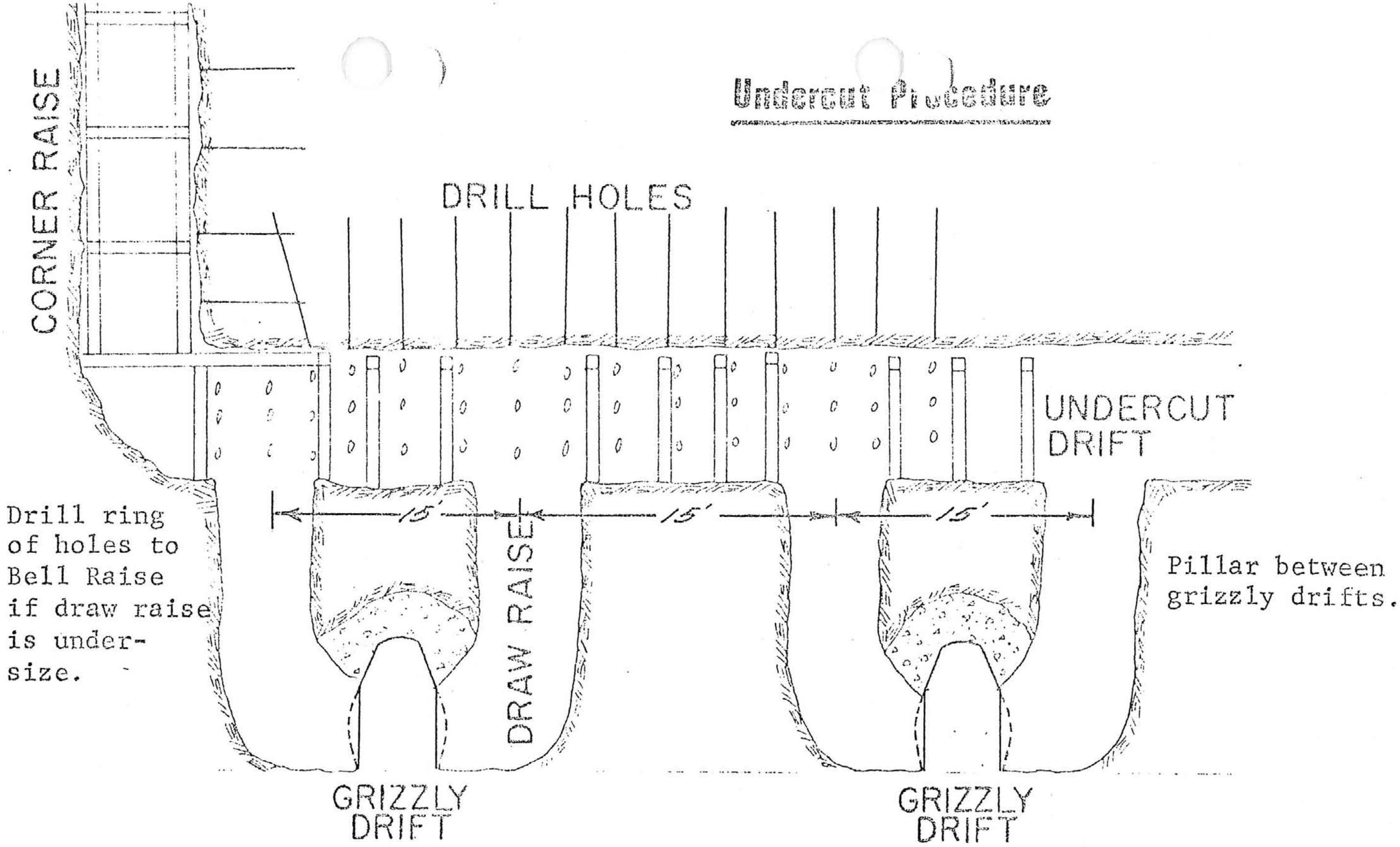
Excavation only; 8'-4" x 8'-0" between draw points; 9'-6" x 8'-0" at draw points.

Rock bolts and wire mesh used for pre-concrete support.

Size: 5' wide, 6-3/4' high finished drift with concrete reinforced at the draw points only.

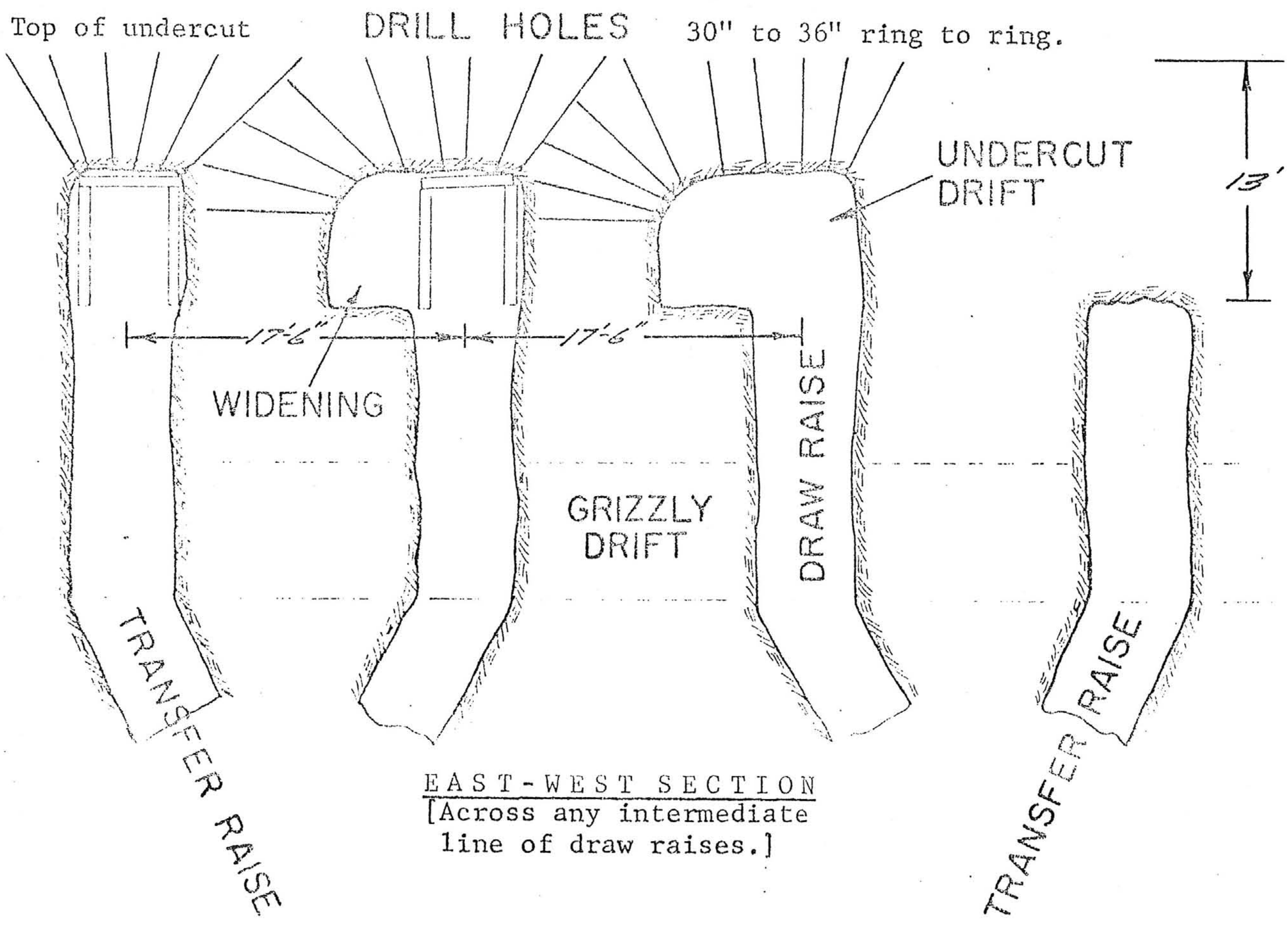
2-man crews used. 13 to 21 holes drilled 6' deep. 30 lbs. ammonium nitrate with each hole primed with a stick of 60% Amogel powder used per round; electric primer blasted. Drilling is done with Jackleg. Ventilation with air mover.

Undercut Procedure



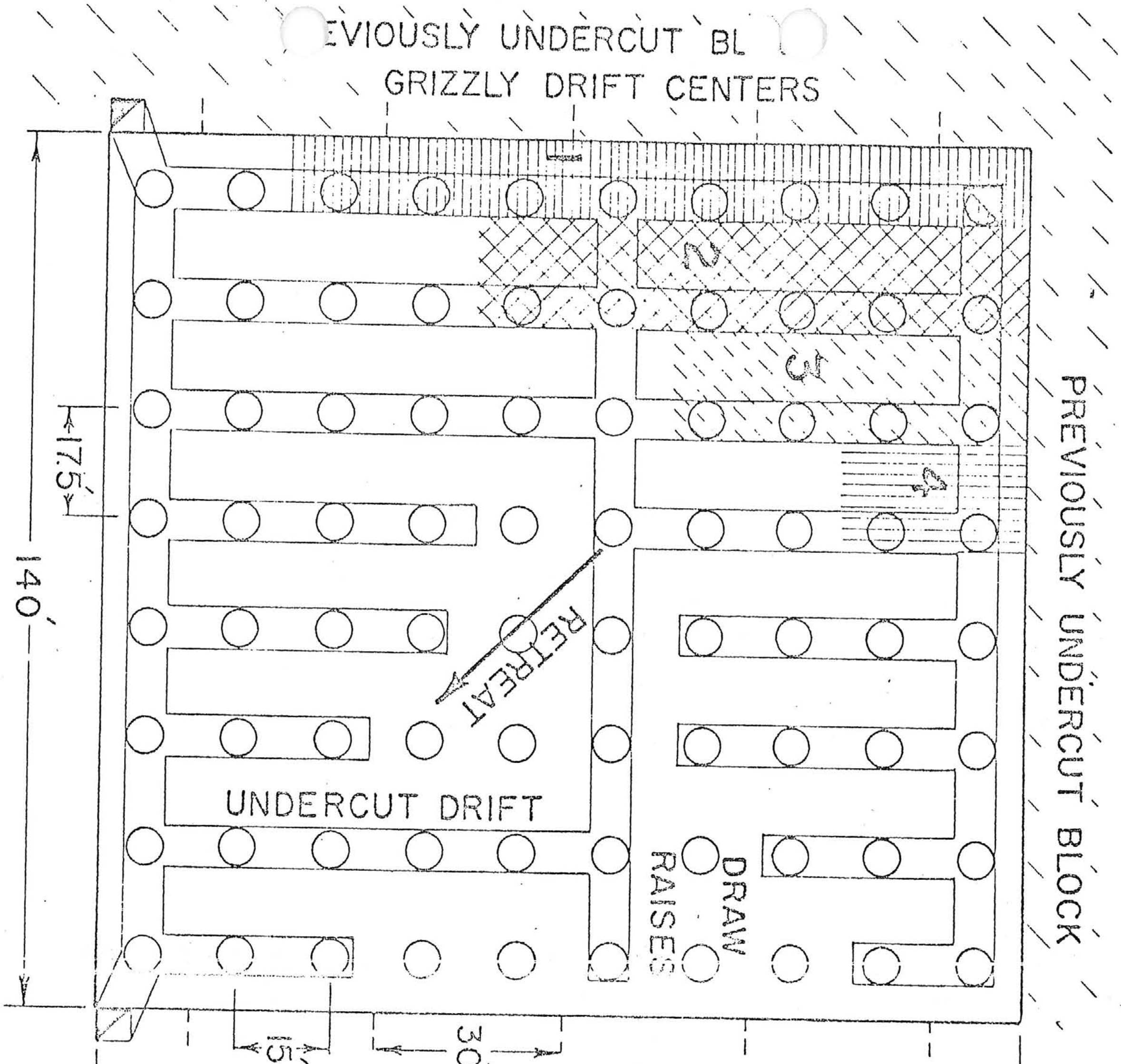
NORTH-SOUTH SECTION

[Alongside boundary.]

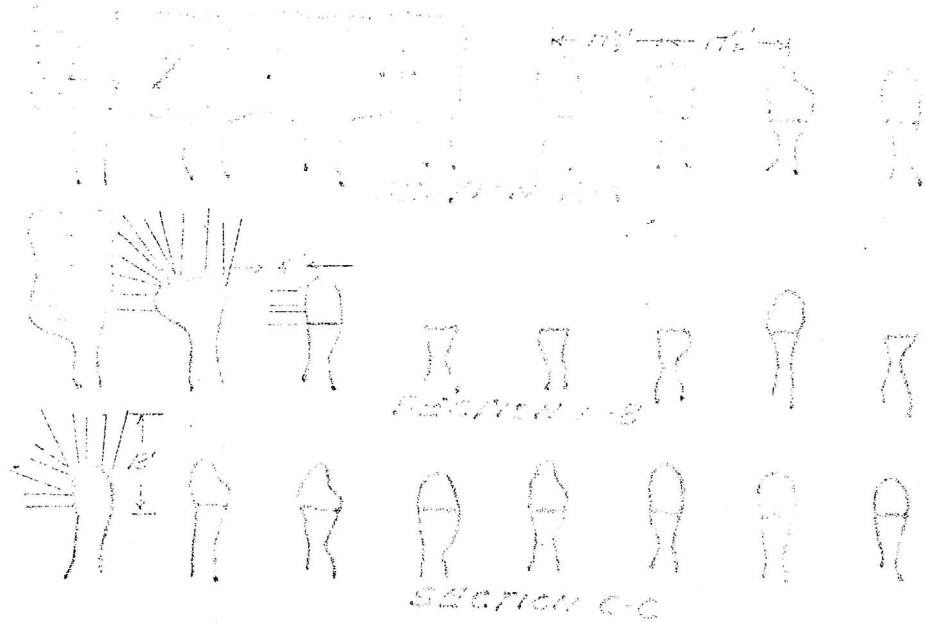


EAST-WEST SECTION

[Across any intermediate
line of draw raises.]



Plan



Sections

PROGRESSION OF UNDERCUTTING
Typical Block

UNDERCUT PROCEDURE

Undercut drifts are 5 x 7 timbered with 6" round posts and 6" x 8" caps. These drifts are driven over the tops of all the draw raises, 15 ft. above the grizzly drift floor at right angles to the grizzly drifts. Access undercut drifts are driven parallel to the grizzly drifts over the tops of the northernmost and southernmost draw raises, and in blocks with five or more grizzly lines, an additional access undercut drift is usually driven across the center of the block. Corner raises are driven unless the rock is relatively soft--or where the block joins an older block. Undercut pillar work usually begins before the drifts are all completed to prevent excessive drift repair and maintenance.

Undercutting can start at any position in the block, but usually is begun against an older caved block and retreats to a solid corner or corners.

Undercut pillar crews start the cave by drilling and blasting out a pillar between drifts or at a boundary of the block. The pillar crews retreat away from this initial cave, breaking the ground into the caved area. Before each pillar is blasted, the drift is widened on one side about four ft. and timbered if necessary. The remaining pillar, about eight ft. thick, is drilled out to a height of 13 ft. above the floor of the undercut, and the pillar and widened drift are shot. The timber is drilled with wood augers and shot with the undercut round. Generally, a 15-ft. section along the drift length is taken with each blast (from one draw raise to the next). Care is taken to insure that the pillar is completely broken by drawing off sufficient broken muck to observe the effect of the blast before the next adjoining pillar is shot. Millisecond delay electric caps are used in pillar blasting, and are wired in series - parallel, with not more than 25 primers in any one series. Circuits are tested with a galvanometer before being connected to the power source. All blasting lines go through an interrupter switch and from this switch to the main pillar blasting switch, which is a completely independent circuit.

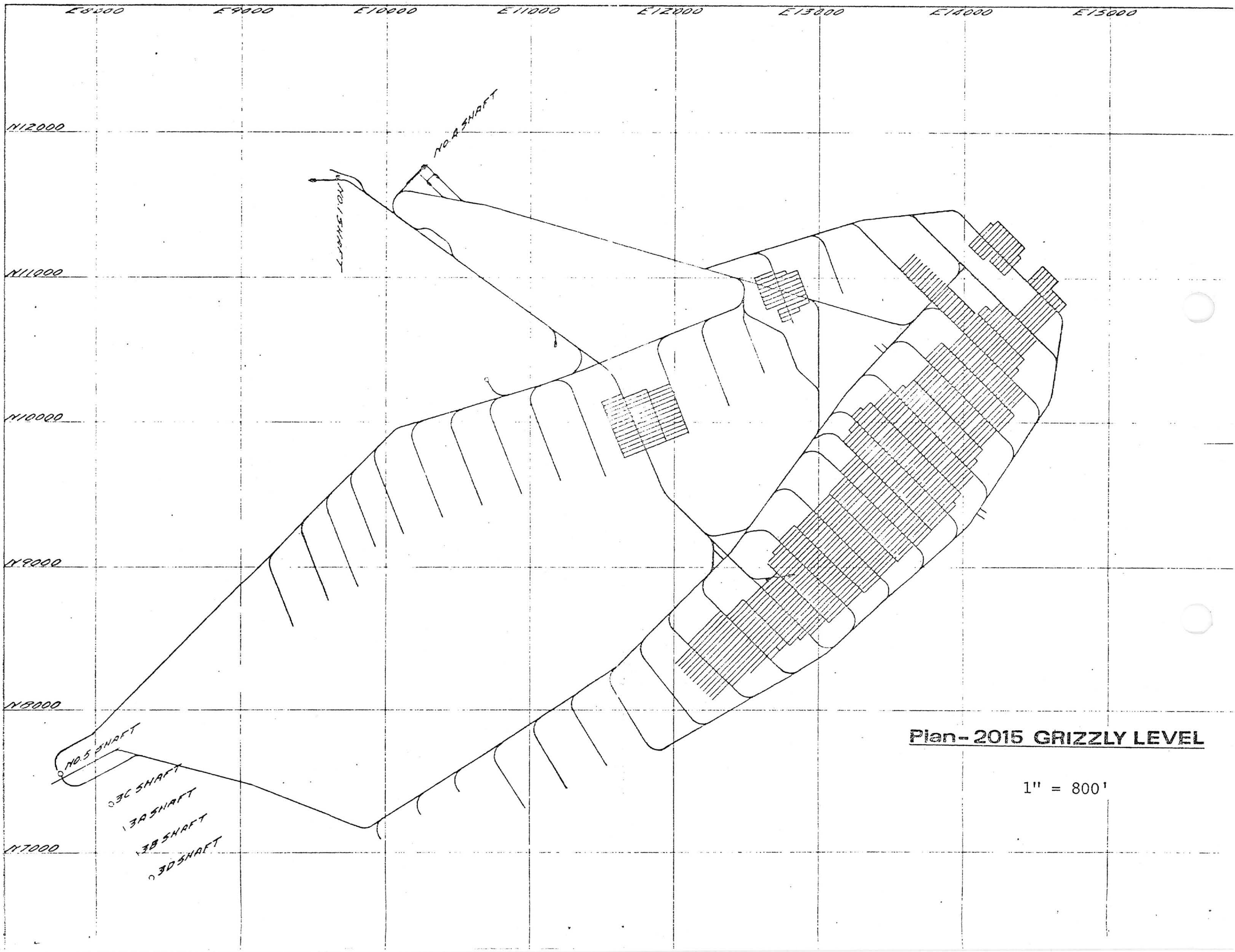
TIMBER:

Round posts and 6" x 8" caps with 6" x 8" stringers or sills over raise tops.
2" x 12" side and back lagging.
Raise tops temporarily covered with 2" lagging.

CREWS:

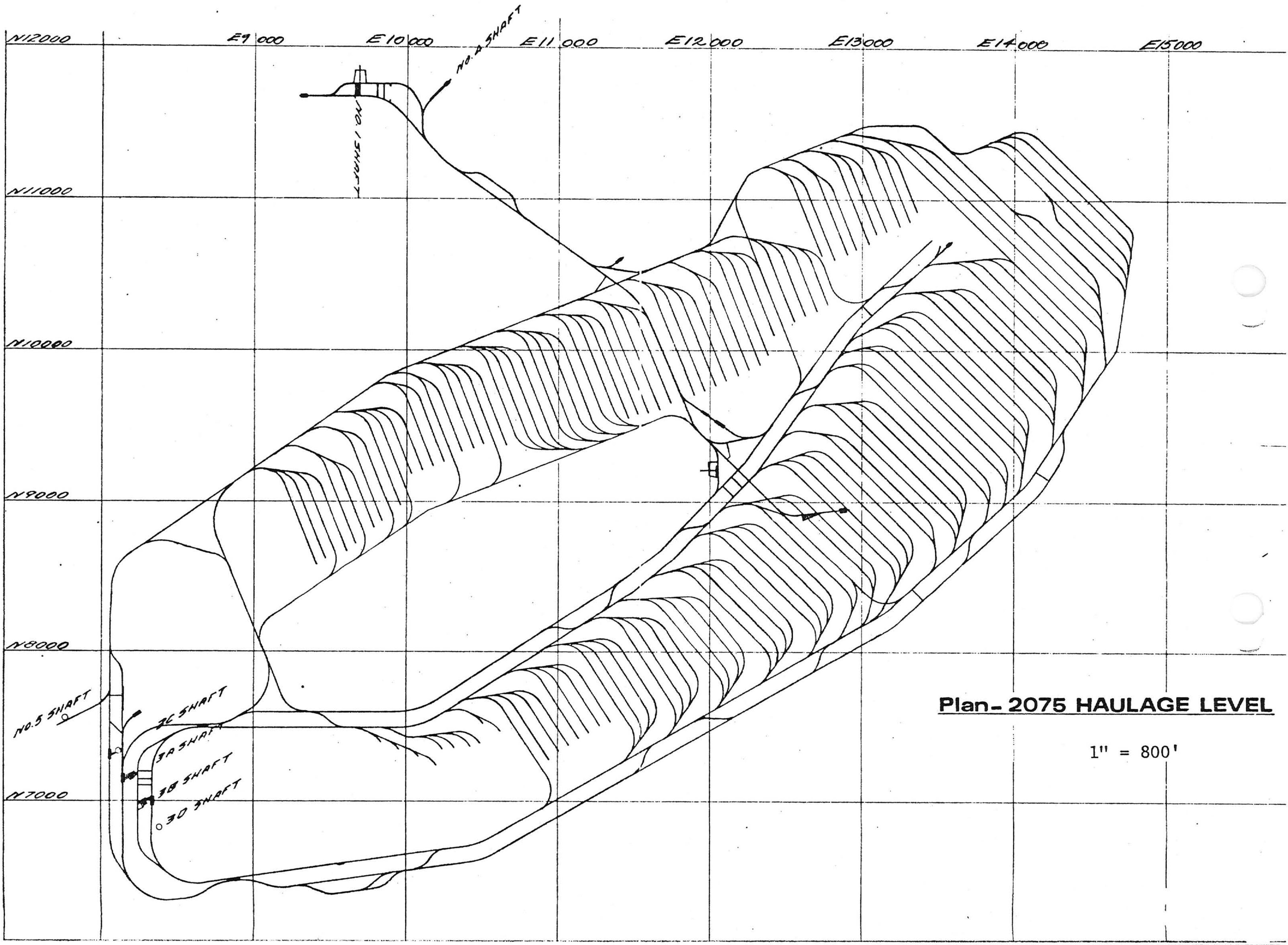
2-man crews drill 10 to 18 jackleg holes, and 20 to 30 stoper holes per shift, depending on draw raise spacing and ground condition. Undercutting proceeds on a 3-shift, 6-day per week basis.

45% N.G. powder or ammonium nitrate, primed with a stick of 60% Amogel is used with millisecond electric blasting.



Plan - 2015 GRIZZLY LEVEL

1" = 800'



Plan- 2075 HAULAGE LEVEL

1" = 800'

GENERAL MINE OPERATING DATA

Operating Shifts per Day	3
Operating Days per Week	7
Operating Days per Year (8 holidays)	357
Production per Day, Tons	62,500
Production per Year, Tons	22,312,500
Active Undercut Area, Sq. Ft.	446,000

Draw Point Spacing:
17.5' East-West x 15' North-South

Block Dimensional Data:	
Block Width	140'
Block Length	90' - 240'
Present Average Block	140' x 180'
Ore Height above Undercut Floor	100' - 600'
Distance Undercut Floor to Grizzly Level Floor	15'
Distance Floor Haulage to Floor Grizzly Level	60'
Grizzly Bar Spacing	14"
Grizzly Bar Material	Salvaged 90# Rail.

Draw Points (Average per Month):	
Active	1,350
High Pack	65
Held for Repair	80
Held for Grade	80
Total Draw Points	1,575

Types of Explosives Used:

Secondary Blasting -- 45% (No. 3) Amogel in 1-inch x 6" sticks or 60% (No. 1) Amogel in 1 - lb. bags. Initiation is with zero delay electric blasting caps. These are connected into a trunk line to a central underground location where blasting switches are located.

Primary Blasting - In the larger headings an ammonium nitrate explosive is used, except in wet holes. Initiation is by a 1-inch x 6" stick of 60% Amogel which, in turn, is ignited by a fuse cap.

An 8-ft. length of fuse with a spitter cord fuse igniter on the end opposite the cap is ignited with a hot wire fuse lighter.

All other development, except transfer raises, uses ammonium nitrate with regular delay electric blasting caps. Transfer raises load with 60% Amogel only.

All blasting is done at the middle and end of each shift.

Mining Equipment:

Big drifts use a San Manuel-made jumbo with three 8' booms and three 3" drifters with 6' feed shells. The booms are hydraulically operated and are powered by two 1-3/4 gpm, 11,000 psi hydropumps that feed into a hydraulic manifold.

Mucking is done by a rocker shovel with a steel flight conveyor dumping into 10 - ton bottom - dump development cars. The cars are switched by an 8-ton, 40-hp (or 9-ton, 40-hp) storage battery locomotive.

Smaller headings use feedleg drills with 2' or 4' single stage legs and stoper drills with 18" steel change.

Development slushing is handled by air-powered, double-drum slushers.

Underground Haulage Data:

Haulage locomotives are 23-ton, 4-wheel trolley type; each locomotive with two 125-hp, 275-volt DC motors. They haul fifteen 12- to 13-ton working load cars.

Ore cars are 300 cu. ft., 15-ton box-type with one stationary coupling and one rotating coupling.

Length center to center of coupling: 17'-6".
Width, overall: 6'-0".
Height above track: 5'-6".
Couplers: Rotary and non-rotary couplers equipped with rubber cushioned draft gear.
Track gauge: 36".
Car loading through air-operated guillotine undercut gates.

Car Dumps:

In the dumping cycle the motorman pulls through the dump and spots three cars in the dumper without uncoupling. Carstops rise and lock the train in position. The motorman activates the dumper which rotates 180° and returns to the upright position and the operation is repeated. Three cars are dumped in about one minute, or five minutes per train of 15 cars. Development cars are designed to fit the dumper, but because of their length and type of coupling, they must be uncoupled to dump.

VENTILATION

530,000 cfm goes down #4 and #1 Shafts from the surface.
180,000 cfm goes down #5 Shaft from the surface.
55,000 cfm Miscellaneous Intakes, in addition to shafts.

140,000 cfm - SubLevel (1715-1775 Levels).
570,000 cfm - 2nd Lift (2015-2075 Levels).

Sub-Levels - Three hi-speed Axivane mine fans: One MXC, 1715, 200-hp, 1160 rpm, 5' diameter; one MXC, 1775 200-hp, 1160 rpm, 5' diameter; and one MXC, 2315, 200-hp, 1160 rpm, 5' diameter.

2nd Lift - Four hi-speed Axivane mine fans: One MXC, 2075 450-hp, 1160 rpm, 6' diameter; one VXC, 2015, 450-hp 1160 rpm, 6' diameter; one VXC, 2015, 200-hp, 1160 rpm, 6' diameter.

Auxiliary ventilation air is directed through operating blocks by means of ventilation doors and Axivane 20-hp, large-volume, low-pressure, low-speed mine fans. Individual working places are ventilated by 10-hp and 20-hp high-pressure, high-speed fans through 24" ventilation pipe or with venturi type air movers. 60-hp fans are used to supplement the major fans in problem areas.

COMMUNICATIONS

On the haulage levels, haulage trains and supply trains are moved on direct orders from a dispatcher by the use of a radio phone system. In addition, radio phones are installed in repair shops and at vantage points for use by supervision.

Draw and haulage operations are coordinated through the dispatcher by an audio paging system.

A standard telephone system consisting of seven circuits aids in coordinating hoisting, maintenance, and service facilities between the surface and the mine underground.

DRAINAGE

Newly opened areas show an appreciable flow of water which is carried out of the working area by air-operated sump pumps, with a capacity of 150 gpm at 100' head. Mine underground was planned so that water drains either to No. 1 Shaft or to 3A and 3B Shafts.

No. 1 Shaft is pumping 3,200 gpm, principally from the 2nd Lift;
700 gpm, from 3B Shaft Bottom;
Present pumping is 3,900 gpm Total.

Mine water is pumped approximately eight miles to the Plant site for use as Mill water.

To prevent surface drainage from entering the mine, surface washes leading to the cave area are either diverted or dammed up to hold the water for evaporation.

SURFACE ORE TRANSPORTATION

Cars:	100 tons capacity, 35 to 40 per train.
Locomotive:	125-ton, 1,600-hp, diesel-electric.
Trackage:	132-lb. rail, 7-mile haul to receiving bin at reduction plant, level track.

CHANGE ROOM NO. 1

Accommodates 1,900 employees with lockers, showers, and toilet facilities. Heated with gas space heaters.

In the same building is the foreman's office, time office, mine survey office, safety engineer's office, dispensary, lamp room, and dust-counting laboratory.

CHANGE ROOM NO. 2

Accommodates 1,000 men and has office facilities similar to #1 change room. This change room is located in the #3 Shaft Area near #5 Shaft.

COMPRESSOR HOUSE

Five 3,500 cfm compressors, delivering at 100 lbs. air pressure, each equipped with 600-hp synchronous motor.
One 1,936 cfm - 350-hp synchronous motor.
One 1,596 cfm - 300-hp synchronous motor.
Total = 21,032 cfm, delivering at 110 lbs. air pressure for approximately 100 lbs. mine working pressure.
Four natural gas powered compressors each having a 3,200 cfm capacity. "Outdoor" type.
Two 7,000-cfm centrifugal compressors, each equipped with 1,500-hp motor, delivering at 110 lbs. air pressure.

NO. 1 and NO. 4 YARDS

Surface installations in this area include a machine shop (including car repairs), electric shop, drill repair shop, a blacksmith shop, truck maintenance shop, framing shed, carpenter shop, warehouse facilities, timber - treating plant, salvage area, pipe shop, cylinder repair shed, sand-blasting shed, paint shed, fire marshal shed, mine rescue training center, fuse and cap storage tunnel, batch plant, (including mix water cooling plant and additive storage tanks), standby power plant, compressor house, No. 1 and No. 4 hoisthouses, potable water treating plant, and changerooms.

There are storage areas for all material used in the mining operation.

Steel: Variety of structural shapes for shop fabrication jobs and mine ground support.

Timber: 12" x 12" drift timber.
2" and 3" lagging, various lengths.
6" and 8" cribbing, pre-framed.
Pole posts - Texas pine for undercut timber.

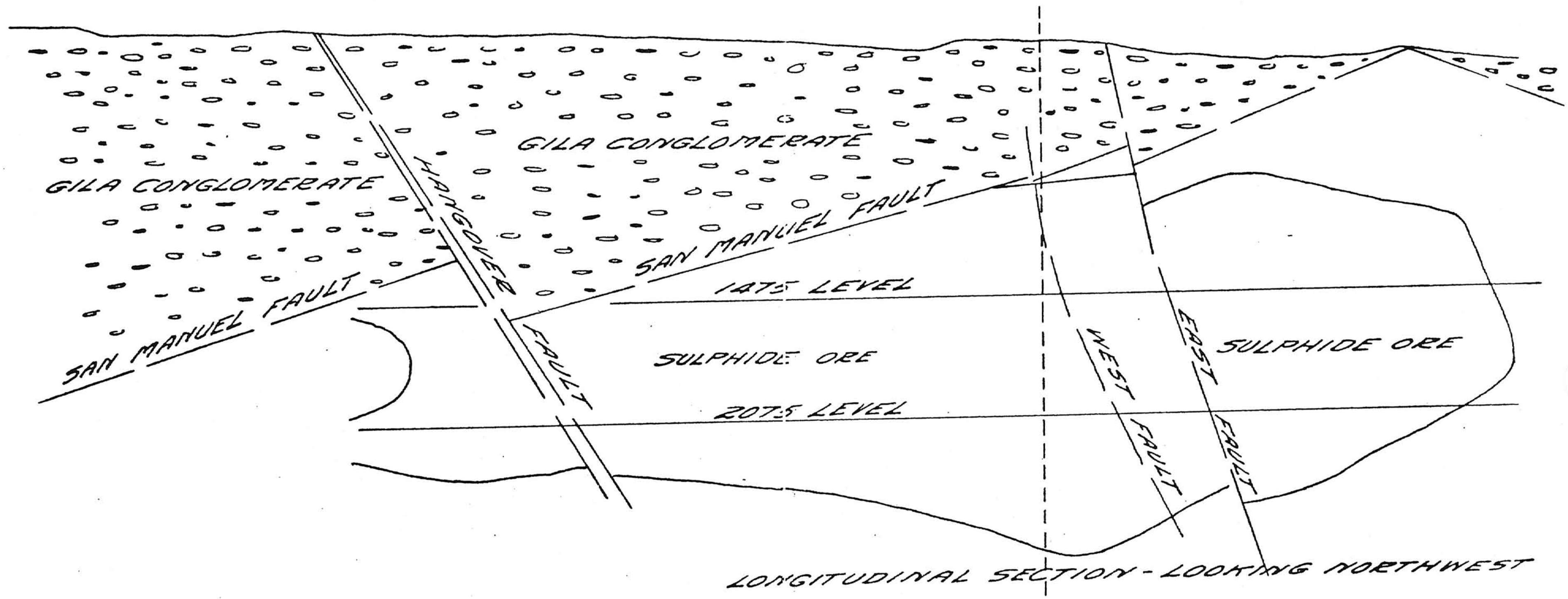
A large explosives magazine is maintained to supply the mining operation.

A planned maintenance and lubrication schedule is followed for all surface and underground operating equipment.

METAL PRODUCTION—San Manuel Only

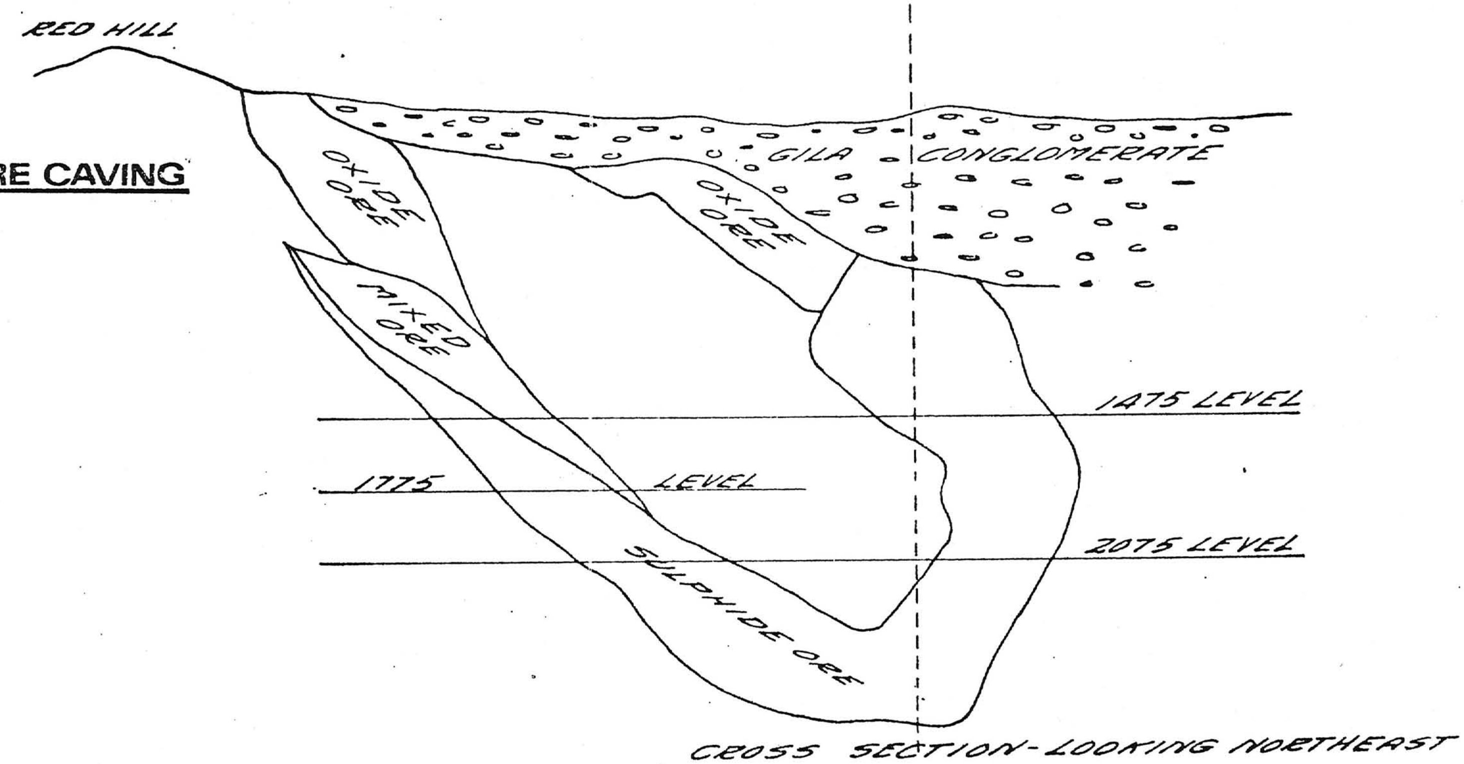
1971 - 12 months of operation.

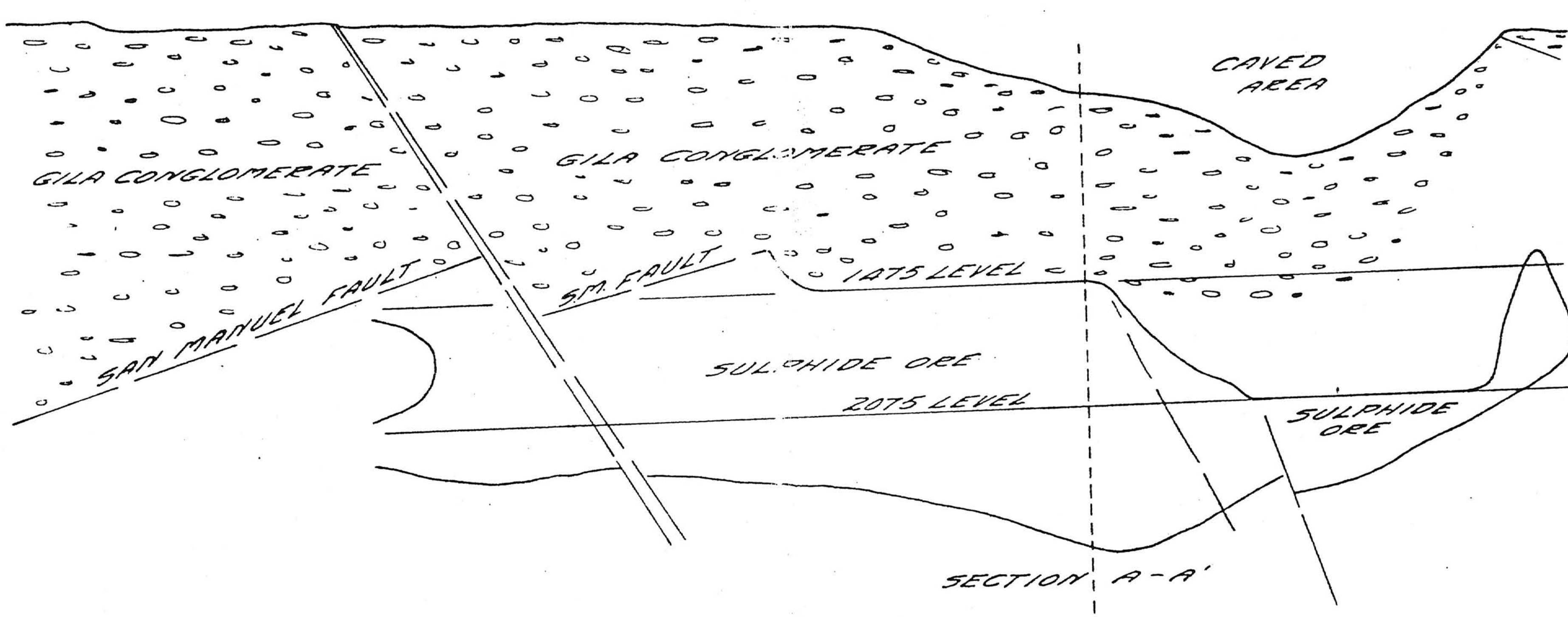
Average Daily Mine Production, Dry Tons	44,500
Copper Production, Tons	145,000
Molybdenum Sulfide Concentrate, Tons	4,150



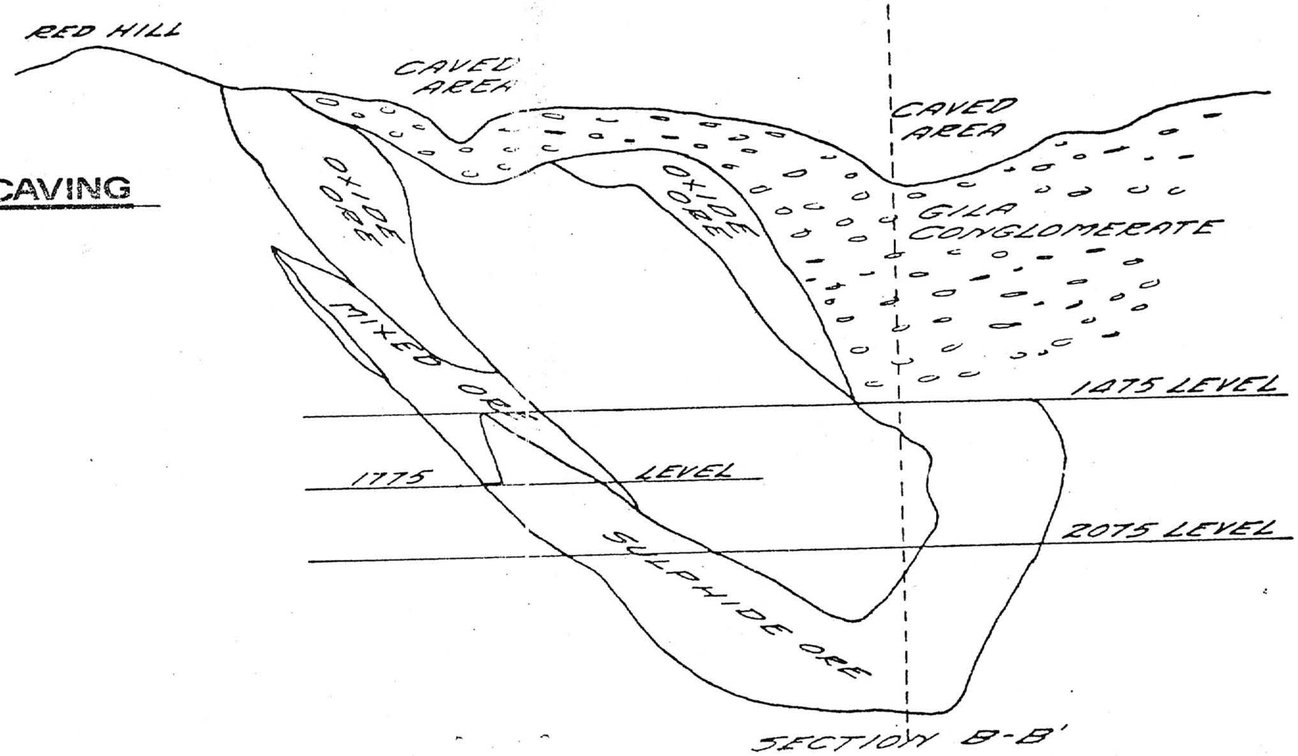
SECTIONS BEFORE CAVING

1" = 800'





SECTIONS AFTER CAVING



DRIFTING
At the
San Manuel Mine

By

DONALD A. CUMMING

Level Engineer
MAGMA COPPER COMPANY
San Manuel Division
San Manuel, Arizona

For

The Arizona Section

A. I. M. E

December 6, 1971

DRIFTING AT THE SAN MANUEL MINE

Contents

DRIFTING At the San Manuel Mine

General

The San Manuel orebody is composed of double levels or lifts at 300-ft. intervals in the north limb of the U-shaped orebody (Fig. 1). The south limb, or main orebody, is mined in 600-ft. lifts. Each lift consists of two levels, generally called the Draw or Caving level and the Haulage level. These levels are 60 ft. apart. All of the drifting is through either a porphyry or a quartz monzonite of disseminated mineralization, with occasional areas of diabase or rhyolite.

Since the first major mine development, this Company has continually looked for ways of improving its drifting methods. The presently planned increase in production to 60,000 tons per day, and the lack of trained personnel available, requires new development methods that will allow us to maintain this high production rate. In order to finish the drifting on the 2015 and 2075 levels, a total of 59,000 ft. is required, or two years at our present rate.

Ground Support Used

The ground support for drifting has been standardized for simplicity. In block areas where the ground will later be caved, or in areas of exceptionally bad ground, concrete is used as the final support; therefore, 4" or 6" H-beam steel arch sets are used as the initial support. (See Fig. 2.) The only exception to this would be on the haulage levels, where ground conditions dictate that 12" square timber sets may be used with little or no secondary timbering required. Drifts outside of the cave area are supported by either the 12" square timber sets (Fig. 3), 12" H-beam caps with 12" square timber posts, or 6" x 8" WF ventilation drift arch sets. (See Fig. 4.) Spacing of all sets is 5 ft., but varies to suit the ground conditions.

All turnouts are also standardized. These are 12" steel caps with 12" square timber posts. Turnouts on the haulage level have a radius curve of either 165 ft. or of 300 ft., while on the draw level they are all of a 105-ft. radius. (See Fig. 5.) Pre-concrete turnouts with 8" x 12" WF steel caps on 10" square timber posts may also be installed, should it be required on either level.

Equipment Used

1. The Drilling Jumbo

The first jumbo used was a 2-boom machine with manually operated booms and drifters mounted on screw feed shells. This machine is now inoperable, and we presently have:

a. Three 2-boom jumbos, one remote control and the other two semi-automatic. The semi-automatic are equipped with remote control of the booms, screw feed machines, and manually positioning shells. The other is completely remote controlled with chain feed machines, and capable of drilling vertical holes. One of these jumbos will drill an average round in about one hour and 15 minutes.

b. Five 4-boom jumbos. These are double-deck rigs and are all semi-automatic. They have screw feed machines with backhead controlled rockdrills mounted on shells that are manually positioned. Drilling out with these jumbos is a 2-man operation. They will drill a standard round in about 45 minutes.

c. Eight 3-boom, completely automatic jumbos, the majority of which were designed and fabricated for our methods through the cooperative effort of our operating, engineering, and mechanical departments. These jumbos use chain feed machines. The jumbos can drill vertical holes, and were designed to be used by one man. They will drill a standard round in about one hour.

The 17 jumbos described above provide each level a spare and one available on surface for overhaul. The term "semi-automatic" describes the jumbos that have hydraulic remote control positioning of the booms while the shells have manual horizontal and vertical positioning. Each of the jumbos has either a screw feed or chain feed machine. The screw feed are 3" bore machines with the automatic feed motor built into the drill itself, mounted on screw feed 8-ft. aluminum shells. The chain feed drifters are also 3" bore machines attached to mounting plates that slide on shells made from two reinforced channels welded together. There is an air-vane motor with right angle drive through worm gears at the front end of the shell driving the chain which is attached to the mounting plates. The drifter and feed are then controlled individually. We now have 73 drifters, with 50 in use and 23 as replacement units.

All drilling is with 1" Q.O. drill steel with tapered ends and rubber collared shanks. The bits are 1-5/8" diameter tungsten carbide. Our drill steel has a life average of about 300 ft., and at present we are experimenting with a smaller diameter drill steel and smaller bit size.

2. Blasting

Ammonium nitrate is used whenever possible. It is loaded into the hole using a Company-made loader-hopper. (See Fig. 7.) The hopper is a conical steel container coated with yellow epoxy enamel and holds 50 lbs. of the prills. It is joined from the bottom to a non-corrosive stainless steel ejector by a 12-ft. black semi-conductive hose. The ejector is hooked to an air outlet and blows the prills through semi-conductive hose into the hole. The whole unit is grounded while using.

3. The Mucking Machine

Originally the drift crews had been using Eimco 21 mucking machines which involved considerable hand mucking in our 12-ft. wide drifts. (Figs. 2 and 3.) A faster mucking machine was selected to eliminate this hand mucking and be transportable in our timbered drifts. Although other models met these requirements, the Eimco 40H was tried on a trial basis and proved even more successful than anticipated. This particular machine with its one-half cubic yard bucket allows our operators to load a 230 cu. ft. car in about five or six minutes. The steel flight conveyor was chosen over the belt conveyor because of its high capacities, reduced maintenance, and our type of muck. The Company does own one belt conveyor model.

There are now 17 of these machines on the property, with a spare on each level and one in the shop for reconditioning and overhaul.

4. The Drop Bottom Dump Cars

We presently have 108 of these cars. In selecting the cars to go with the bigger mucking machine, we took the following into consideration:

a. A car that would fit in the #4 Main Service Shaft. The cage had a width of 6'-9½" and a length of 13'-6".

b. A car that would fit the rotary ore dump, should it be required to dump there. The dump has an overall length of 52'-6" and will dump cars with a side height of about 5'-8".

c. A car that would allow maximum clearance in our timbered drift.

The car selected is 6'-0" wide, 5'-7" high, and 13'-2" long. It has a capacity of 230 cu. ft. To use this car, the #1 Shaft was designated as a waste hoisting shaft. A bottom dump pocket was cut in this shaft on each level, where dumping and closing wheels were installed. Each draw level also has an ore pass raised from the haulage level where the cars may be dumped. The ore dumped here is loaded into the 15-ton production cars and hauled to the rotary dump, thus providing two ways of dumping the development muck on each level.

The change in mucking machine, the large drop bottom dump cars, and the addition of remote control and semi-automatic jumbos improved our efficiency by more than 40%.

5. Battery Locomotives

The drift crews use 27 of the 4-ton, 2-motor storage battery locomotives. Each locomotive is assigned two 48-cell, 25-plate type 55X storage batteries with a voltage output of 96 volts. One of these batteries is placed on charge every shift, while the other is being used on the motor. The batteries have a weight of about five tons, giving the battery locomotive a combined weight of nine tons. With a 2,000 lb. pull, these locomotives are capable of a speed of 5 mph.

6. Fans

All drifts are ventilated with 20-hp or 10-hp series 1,000 Axivane high-pressure, high-speed fans, mounted horizontally in a top corner of the drift. These fans draw air from the main passageway and ventilate the heading through 24-in. diameter spiral lock vent pipe (10-hp fan gives 8,000 cfm and 20-hp fan gives 12,000 cfm). The drift crews are now using 82 of these fans, with 50 of them being 20-hp. These are usually installed by a construction crew.

All of the above equipment is an investment of over two million dollars, and therefore it would seem best that we coordinate our drift crews and systemize our method of drifting to increase our efficiency, rather than consider a complete change to the drift boring equipment.

Drift Working Cycle

In order to complete a cycle from and to a clean drilling face in our average timbered drift, the work would be as follows:

1. Drilling and Blasting

Drilling will be either a one-man or 2-man operation, depending on the jumbo used. The round is drilled using a "V" cut of two or four holes. These cut holes are deeper than the rest of the round. The relievers are also inclined slightly to ease the burden on the next holes. An average round would consist of 52 holes with ten lifters, eight or nine back-holes, and five rows of holes in between (Fig. 8).

Each hole is primed with a 1-1/4" x 6" stick of Amogel 60% powder and a White Sequoia safety fuse with a No. 6 cap, and loaded with ammonium nitrate. Generally, all lifters are loaded with Amogel only and, if the ground is wet, a 1-1/4" x 12" special gelatin 40% powder is used. The round is then timed and fired using igniter cord attached to the copper igniter on the fuse.

2. Booming the Cap

A flatcar containing the cap is pushed up against the muckpile and uncoupled from the motor. A roller (Fig. 9) is placed on the last cap in, and a 1/2" diameter cable 50-ft. long is placed over the roller with one end clamped to the cap to be installed and the other to the coupling on the battery locomotive. The locomotive is then used to raise the cap to the same height as the cap already in. The posts on either side of the last two sets have steel wrap around hip boom hangers (Fig. 6) on them. These hangers are secured in the rear of the post by a pin, and supported in the front by a 2" x 12" x 12" cleat nailed to the post. A 6" H-beam boom 12 ft. long is then pushed out under the open ground a required distance and blocked down. The crew then slides the cap out on top of these booms, spots it on centerline and grade, blocks it down, then tightly lags the opening in between with 3" x 6" x 5'-0" lagging.

3. Mucking

Due to the overhang of the conveyor on the mucking machine, it must be pushed in by separating it from the battery locomotive by a flatcar. The conveyor is raised hydraulically and set down to its proper level, resting on steel jacks. In order to load the long cars, the mucking machine coupling is attached to a sliding draw bar that provides three positions while loading. In order to keep the mucking machine close to the muckpile, channel type slide rails (Fig. 10) 10 ft. long are used, fitting directly over the rails and moved ahead with the mucking machine bucket. After mucking out, the mucker is thoroughly cleaned by the drift crew, and the loaded cars are transported to a carpool heading where they are picked up by tram crews, dumped, and returned. A standard round would involve loading about eight cars. With the setting up, barring down, switching of cars, and tearing down, it would take two men about two hours to completely muck out.

4. Putting up the Posts

The timber posts, though heavy at times, are relatively easy to install. They are placed up against a cleated saddle in the cap (Fig. 6), put on a 2-in. in one foot batter, and blocked up using our standard 3" and 2" blocks. They are then lagged up the sides using a 2" x 12" x 5'-0" lagging. A cleat is placed on each of the posts installed to hold the steel wrap-around hip hangers to be moved up before booming the next cap.

5. Miscellaneous Work

The drift crews advance by installing 10-ft. sections of 45-lb. rail. The 10-ft. slide rails are used for the intermediate distance in front of these sections. These rails are installed using standard steel ties. As the drift crew advances, the track crew keeps the permanent track to within 30 ft. of the face using the permanent piñon ties and crushed slag ballast. This permanent track is either 45 lb., 70 lb., or 90 lb., and the track crew would usually install 120 ft. at a time.

Ventilation is kept close to the working face by installing 24" diameter vent pipe in 20 ft. or 10 ft. sections. A rubber boot with stainless steel adjustable boot straps is used to insure airtight joints. The present trend is to have construction crews install this when the drift crew is not working the heading.

Drift Crews

The required quota of the drift crews has jumped from 2,400 ft. per month in September of 1968 to the 5,000 ft. per month quota for October of this year. (See Fig. 11.) Though we are now able to maintain our quota, we have not increased the combined fpm/s to where they were at the beginning of 1969 when we had 72 drift miners, or six crews. We now have 12 crews and 118 drift miners. Since the 1969 period, as crews were added to the drift force, the total footage, as well as the fpm/s, dropped slightly. This was due to the weakening of all crews by the addition of untrained individuals. To eliminate this problem and keep our efficiency at a respectable level, we began training individuals "on the job" and experimenting with the drift crew sizes. These individuals are classified as miner trainees.

The trainees work with the drift crews while not participating on their contract. When the individual has been trained in each phase of our drifting operation, and has been checked out by the supervisor, he is placed in a pool maintained to fill temporary and permanent vacancies. In March of this year the Company began experimenting with the drift crews in an effort to increase efficiency. The following was studied:

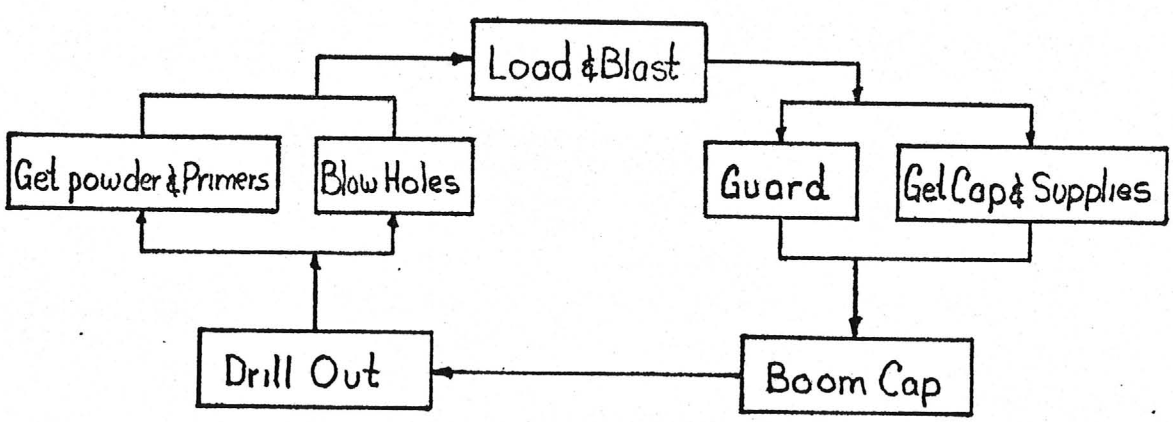
1. All drift crews up until then had their cars dumped from a "pool" by service crews. To increase efficiency would therefore mean adding men to obtain better service. It was decided to have a service crew dump the muck for the experimental crew only, and when not doing this, they would work as supply trammers for that crew.
2. A way to keep the experienced miner at the face as much as possible. This can be accomplished by having someone else do the less-skilled work, such as changing batteries on the locomotive, tramming supplies to and from the face, and moving the jumbo and mucking machine. The service crew could do this when not hauling muck cars.
3. To supply a trainee to this crew to be trained in our methods. This would partially help increase the crew footage while learning, and certainly help create miner material for future expansion.

The first effort made was to combine several of the crews to make a 9-man crew in five headings, working seven days a week and three shifts per day, with a trainee and supply crew. This would develop specialists in each phase of the cycle, with a 3-man drill-and-blast crew, a 2-man cap crew, a 2-man muck crew, and a 2-man post crew. Each phase of this operation depends on the other, and involves too large an area (five drift headings) for the special crews to cover. It did increase our efficiency, but was still not satisfactory in the overall performance. During this preliminary experimenting, very close communication was maintained with the miners and the supervisors involved. It was from this communication that the 5-man crew, 3-heading method we are now using was developed--and perfected to the satisfactory performance we now have. The average efficiency of this crew is constantly rising...the highest yet being a 1.71 fpm/s average of the three crews studied, which for the same period is much higher than the fpm/s of the 3- and 4-man crews. (See Fig. 12.)

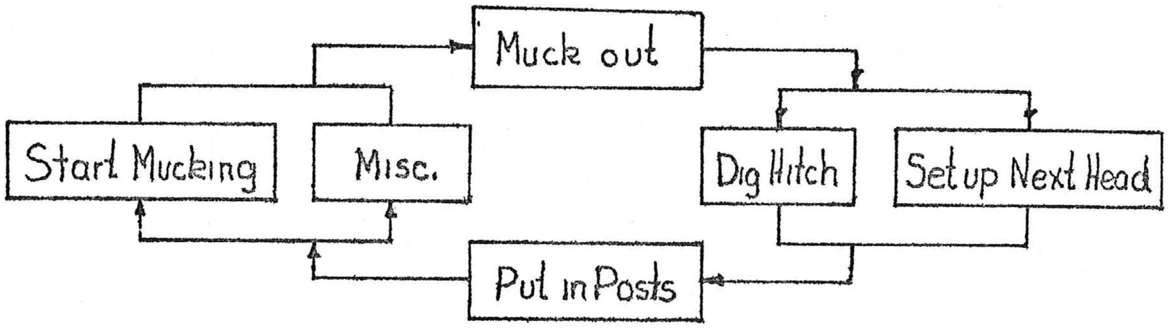
<u>The 5-Man Drift Crew</u>

This crew consists of five miners, two supply men, and a miner trainee. The supply men haul and dump all the loaded development cars and, as a result, can spend about one-half of their time helping the drift crew. The trainee in this crew is there primarily to learn, but does help increase the efficiency of the crew. The crew is given three headings to work. It uses one jumbo, one mucking machine, three motors, and 13 development cars. The men are divided into two groups and try to work the headings in the following manner.

<u>Group 1 -- Three miners, or two miners and a trainee:</u>



Group 2 -- Three miners, or two miners and a trainee:



Group 2 may not have the posts in and the face ready to drill by the time Group 1 is ready to start drilling. Group 1 would then move into the next heading and start whatever work is required in that heading, even if it is work usually performed by Group 2. With the very able supervision assigned to these crews, the work is coordinated very effectively to work each of the three headings equally over a period of a week. We now have four such crews working two shifts and involving 40 miners, eight trainees, and 16 supply trammers.

The 3-Man Drift Crew

This crew consists of a leadman and two miners, and is used in headings where speed is important, and in turnouts where the workload is increased due to the area to be excavated. This crew uses one jumbo, one mucking machine, one motor, and eight cars. When driving a single heading, a 1.67 fpm/s quota is expected of this crew, or one complete cycle per shift. The average 3-man crew can meet this quota. However, when driving a turnout, this efficiency will naturally decrease, hence the low fpm/s average as shown in Fig. 12. We now have three such crews working on three shifts (27 men).

The 4-Man Crew

This crew has two headings to work, and consists of a leadman and three miners. They are divided up into two men per heading with the work cycle coordinated by the leadman. The crew has one jumbo, one mucking machine, two battery locomotives, and eight cars. A weekly advance of 120 ft. is expected from this crew on three shifts. This is 1.67 fpm/s. The efficiency is highly dependent on the headings being in opposite phases of the cycle. Should one of the headings have trouble with bad ground or supplies, it usually throws both headings off. As shown in Fig. 12, a high efficiency is possible from such a crew, however the average over the period studied showed about 1.35 fpm/s. We are now working four of these crews on three shifts.

Crew Comparison

All three of these crews may be needed, depending on where and what is required at the time. In a situation where the drifts are available, the 5-man crew appears slightly better. (See Fig. 12.) The following compares crews on costs and performances.

Number of Men Per Crew	3	4	5
Total Equipment Cost	$105,000	$135,000	$175,000
Equipment Cost/Crew Size	$ 35,000	$ 33,750	$ 35,000
Footage per Manshift	1.1	1.3	1.5
Total Footage per Shift	3.3	5.2	7.5

--continued

	3	4	5
Manpower Costs:			
Leadman @ $36.56	36.56	36.56	
Miner @ $34.16	68.32	102.48	170.80
Supply @ $30.64			30.64
Total	$104.88	$139.04	$201.44
Labor Cost/Ft. per Shift	$ 31.90	$ 26.80	$ 26.80
Equipment Cost/Ft. per Shift	$31,800	$26,000	$23,300

The above figures indicate that on a labor cost per foot basis both 4- and 5-man crews are equal, however the lower equipment cost per foot indicates that future expansion may be possible without purchasing of new equipment with the increased use of the 5-man crew.

Supervision

All drifting at the mine is scheduled by the Planning Section of the Engineering Department. The execution of the schedule is under the jurisdiction of the General Mine Foreman. (Fig. 13.) The manpower on each shift is controlled by the Shift Foreman. Technical control of the drifting is by the Development Engineer on the level. The Level Foreman and the Foreman receive their orders from the Development Engineer through the Shift Foreman. The drift supervisor comes under the immediate jurisdiction of the Foreman. Three crews is the desired maximum for each supervisor, or about 15 men, since it is felt that efficiency will vary directly with the supervision provided.

Every week a work schedule (Fig. 14) is put out by the Development Engineer indicating the footage expected from each crew, the boss assigned, where this crew works, the number of men and the equipment to be used. On the bottom of this schedule is indicated the availability of all equipment and drifts for the pipefitters, mechanics, track crew, electrical, and ventilation crews. In this way, all equipment may be serviced, and the miscellaneous work in the heading brought up to date without interrupting the drift crews. Everyone concerned receives a copy of this at the end of the preceding week.

Conclusions

The only conclusion that can be reached is that we must always look for improvement. The experimenting mentioned above in both cases showed improvement over what we previously had. The important thing is to not sit back and be completely satisfied with the older methods. On a comparison to a drift borer, with the same investment in conventional equipment and excluding labor costs, our drift crews can come to within 75% of what the manufacturers predict of their equipment . . . and the drift crews are presently only coming to within 75% of what supervision expects of them. An advantage of the drift borer is that all of its footage would be in a single heading.

There is no doubt that drift boring with improvements is the method of the future. San Manuel, at least for the near future, is committed to the conventional methods. To then perfect this method to obtain the maximum from the equipment and manpower is the immediate goal. Several minor changes may be made in our equipment to achieve this goal. The crew experimenting has proved successful, but more important, indications are that there is still room for improvement.

A special "thank you" is extended for the completion of this paper to all Mine Departments at San Manuel.

CROSS SECTION THROUGH THE SAN MANUEL OREBODY

Section N 45° W from No.4 Shaft looking N.E.

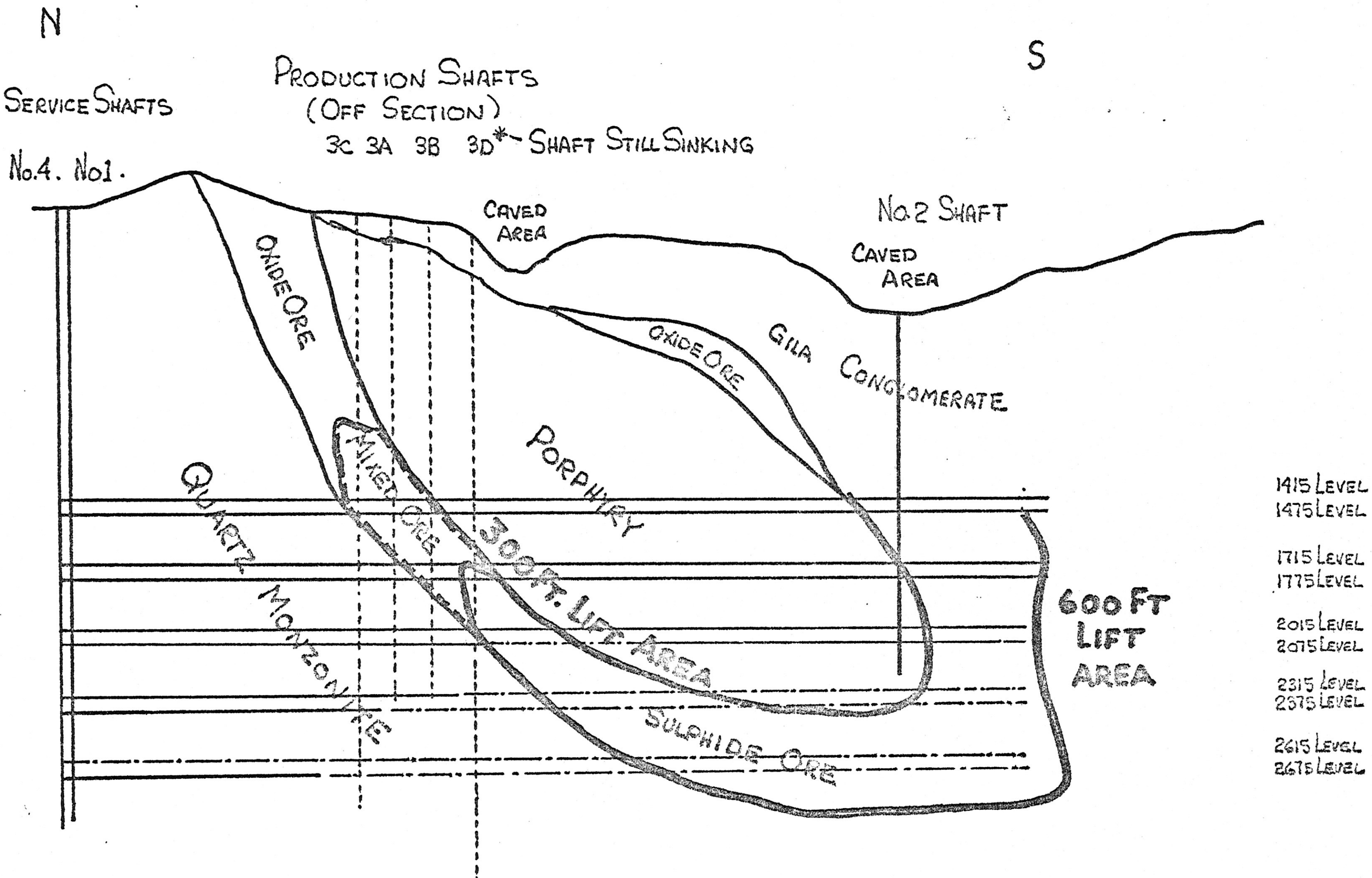


FIGURE 1

-·-·- Indicates drifts that we are to drive.

Generally Quartz Monzonite surrounds the orebody with Porphyry predominant in the core. Both however contain traces of diabase and rhyolite

Scale 1" = 800'

Section Showing Standard Pre-Concrete Set

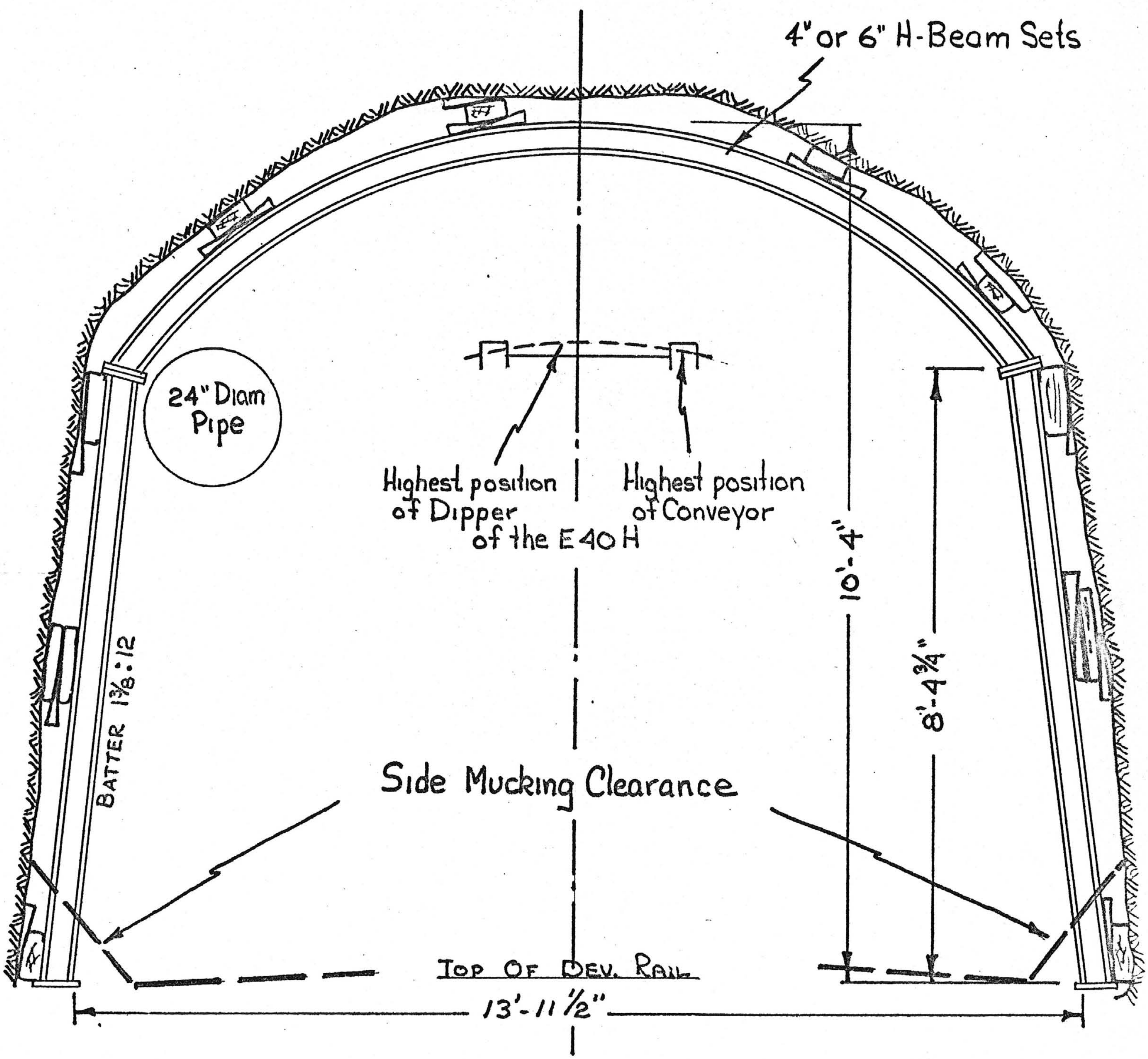


Figure 2

Section Showing Mucking Machine Clearances And Standard Set

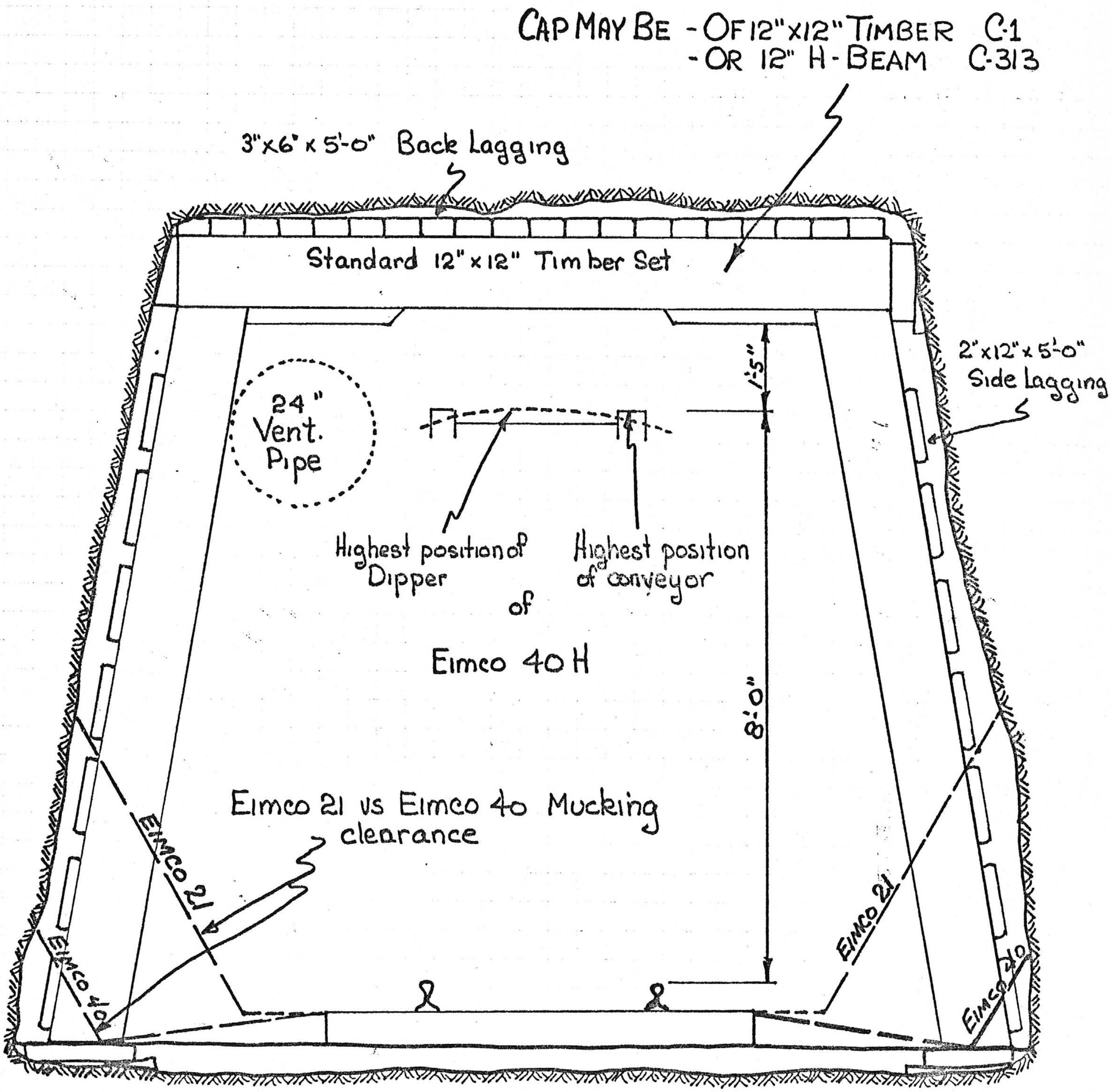


Figure 3

SECTION SHOWING STANDARD VENTILATION DRIFT SETS

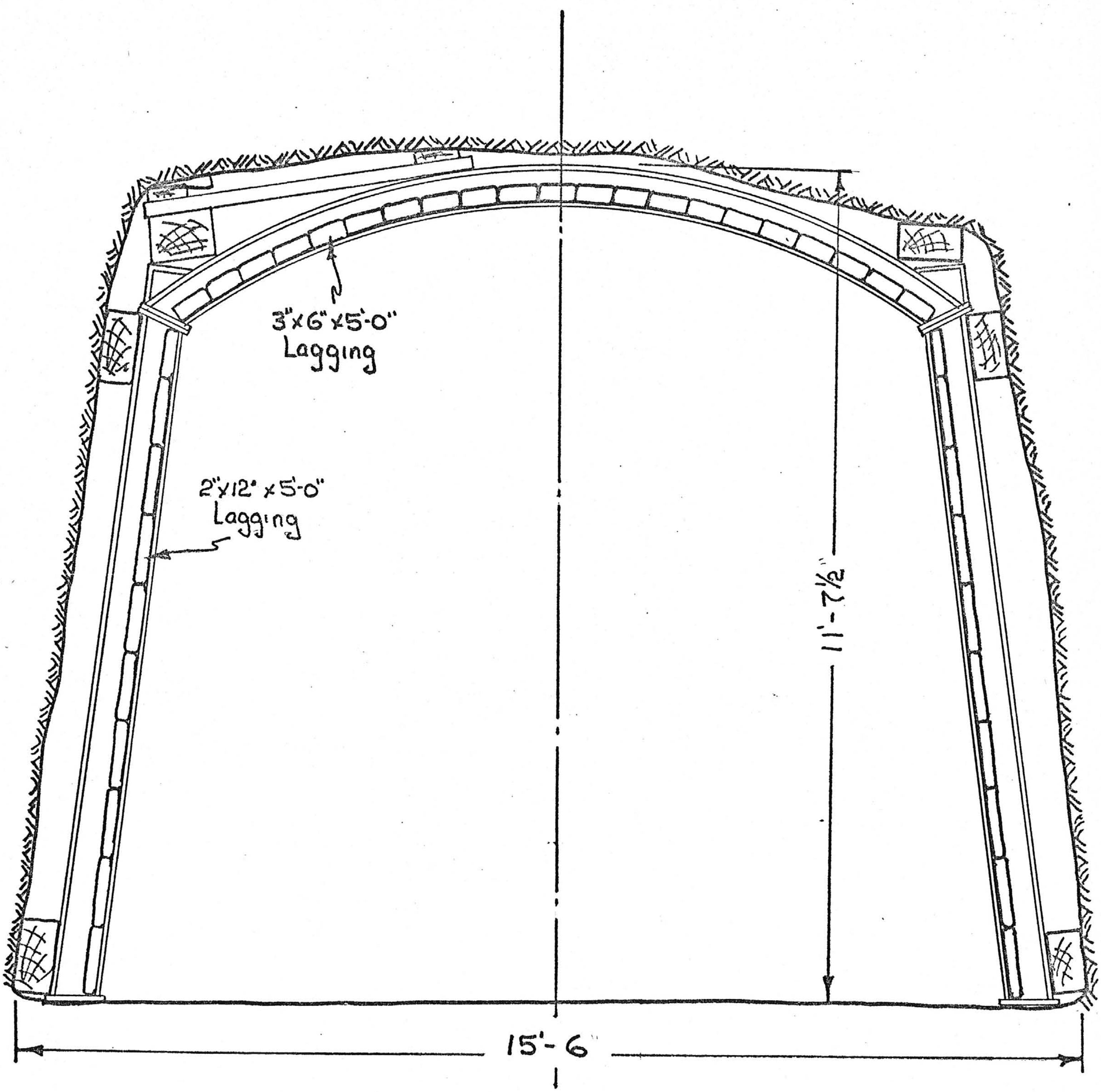
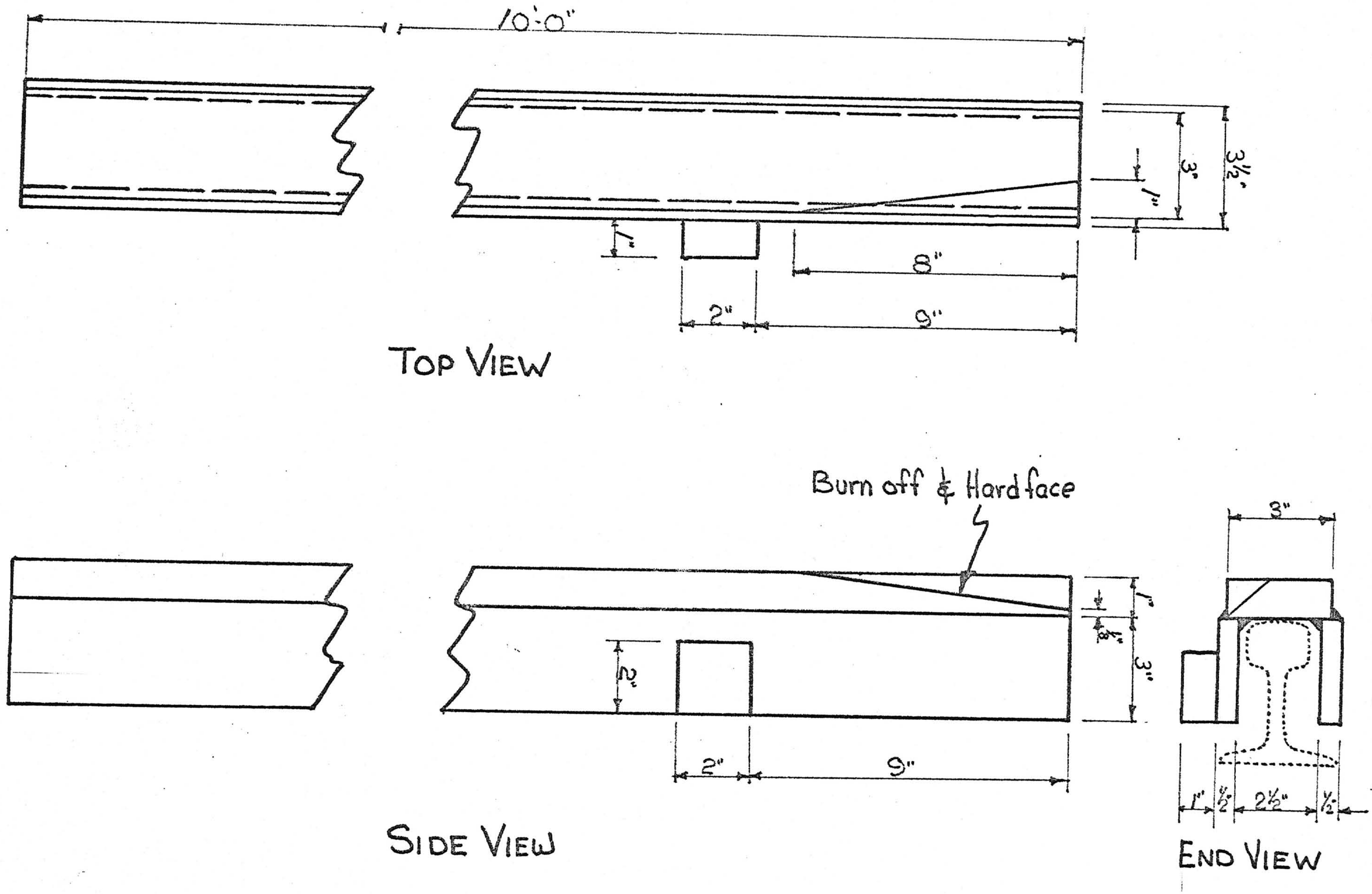


FIGURE 4



Slide Rails For 45# Rail

Scale 3"=1"

Fig. 10

Monthly Drift Requirements & Performance

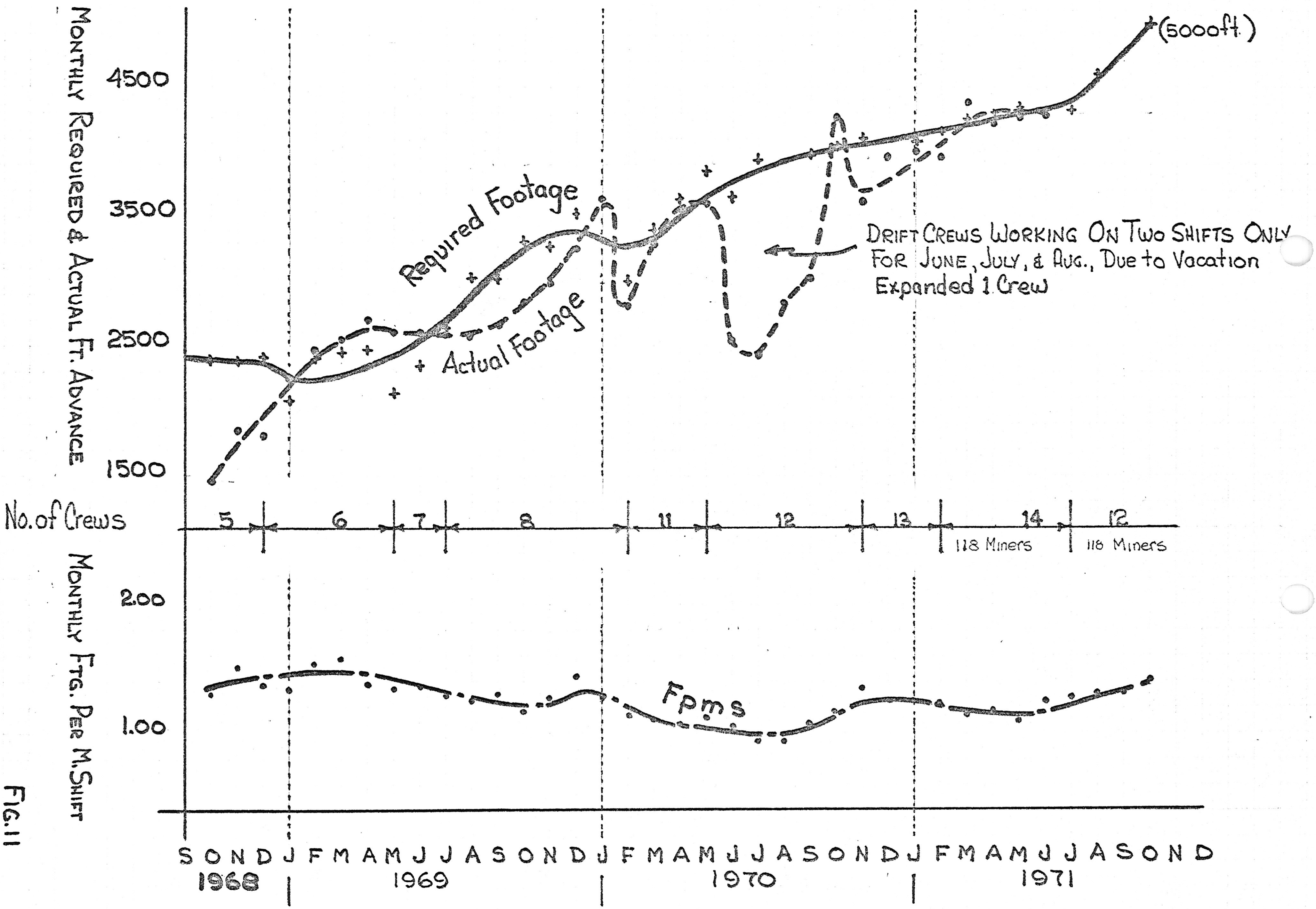


Fig. 11

CREW COMPARISON GRAPH ON WEEKLY & TOTAL FPMS

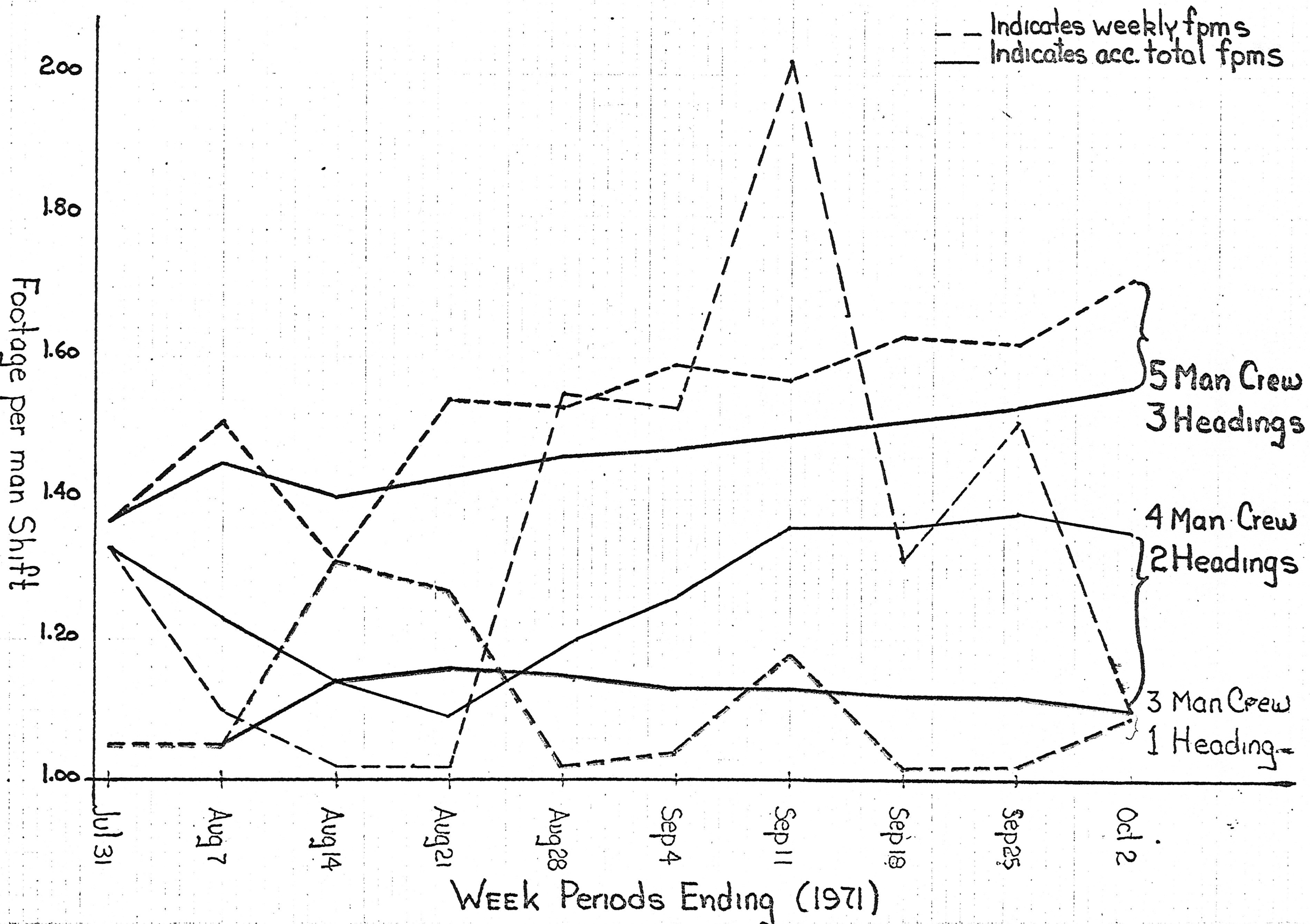


Fig 12

PRIMARY DEVELOPMENT ORGANIZATION CHART

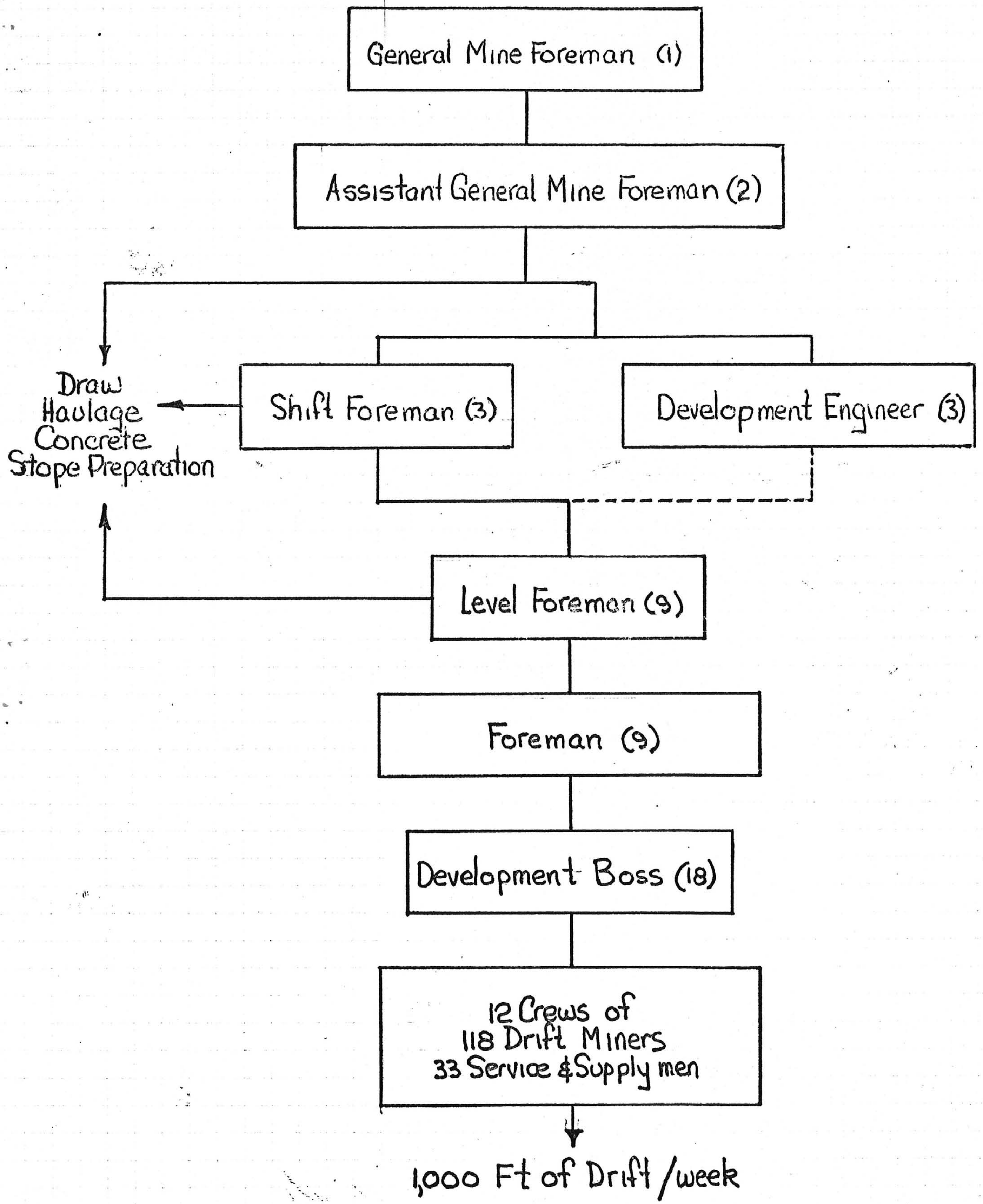


FIGURE 13

FIGURE 14

MINE OPERATING DIVISION

2075 Primary Development

Revised OCT. 17 /71

Shift	Rotat.	Crew	FOOTAGE Est.	FOOTAGE Act.	Boss	Working Place	EQUIPMENT Loco	Jumbo	Mucker	Cars	MEN Cont.	DP	Total	Contract Shifts	$ Bonus
1	C		60		KILPATRICK	P54B, P54C, P55A	2				5	3	8		
2	B	1	60		HERRERA	P54A, P54B, P55C	4	5	1	13	5	3	8		
3	A						5								
1	C						7								
2	B	3	60		HERRERA	P60B, P61A, P61B	33	4	17	13	5	3	8		
3	A		60		CHACON	P59B, P62A, P62B	39				5	3	8		
1	C		60		SWANSON	P24B, 26NLD, P26B	202				5	3	8		
2	B	4					203	19	2	10					
3	A		60		OCHOA	P24B, 26NLD, P26B	207				5	3	8		
1	C		30		KILPATRICK	P3C (N)					2		2		
2	B	6	30		PICKENS	P3C (N)	43	18	12	10	2		2		
3	A		30		IVERSON	P3C (N)					2		2		
1	C		60		ADKINSON	P66A, P66B, P68B	14				5	3	8		
2	B	7					27	20	9	12					
3	A		60		CHACON	P66A, P66B, P68B	204				5	3	8		
1															
2		9	CREW DRIVING RSE STATIONS				35	15	15	7					
3															
1	C		30		SWANSON	P28NLD					3		3		
2	B	11	30		MONFRED	P28NLD	206	17	18	5	3		3		
3	A		30		OCHOA	P28NLD					3		3		
1															
2		Tr.Rse.	RAISE CREWS				40								
3															
1	C		240		WILSON	FAIRCHILD Totals	16	7	7	70	55	24	79		
2	B	Comb.	180		MUSSER	TAFOYA									
3	A		240		QUEVEDO	MENDIOLA									

EQUIPMENT SERVICE

Locomotives				Jumbos				Muckers			
A	B	C	Sun	A	B	C	Sun	A	B	C	Sun
2	14	7	35	5	19	4	15	1	2	7	12
4	27	33	40		20		17		9		15
5	202	39	43				18				18
	203		206								
	204										
	207										

Spare Equipment:

Locomotives
- 2 - VENT.
- 36 - REPAIR
- 201 - BORE HOLE

Jumbos
- 8 - SPARE
- 11 - SURFACE

Muckers
- 11 - SPARE

Other

DRIFT SERVICE

A Mech. & Elec.
B Track
C Ventilation

A	B	C	Sun
P54	P24B	P54A	P3C
P55	P26	P55B	P28
P60B	P54B	P55C	
P61	P54C	P59B	
	P55A	P60B	
	P59B	P61	
	P62	P62	
	P66		
	P68		

DEV. ENG. N.T. GUTIERREZ

The transfer type reading

"AT THE SAN MANUEL MINE"

on the cover page also goes

on page 1.

Subsidence and Related Caving Phenomena

AT THE SAN MANUEL MINE

Fah

L. A. Thomas

Chief Geologist

MAGMA COPPER COMPANY

SAN MANUEL DIVISION

San Manuel, Arizona

January, 1971

SUBSIDENCE AND RELATED CAVING PHENOMENA AT THE SAN MANUEL MINE

INDEX

Plates [continued]

10

MAGMA COPPER COMPANY
SAN MANUEL DIVISION
SAN MANUEL, ARIZONA 85631

June 5, 1973

Harvey W. Smith
6016 North Kachina Lane
Scottsdale, Arizona 85253

Dear Harvey:

Enclosed are several copies of my complete subsidence paper as you requested in your letter of May 31, 1973. One copy is intended for the file which you are putting together and the others to fulfill the requests for papers which you mentioned. In about two or three months we will have the paper commercially printed and then copies will be more readily available. In the meantime, these Xerox copies seem to have reproduced very well.

I enjoyed the program which you people worked so hard to prepare and you may rest assured that all who attended appreciate all the work.

Sincerely,

MAGMA COPPER COMPANY
San Manuel Division

L. A. Thomas

L. A. Thomas
Chief Geologist

LAT:gs

Enclosure.

Subsidence & Related Caving Phenomena

SUBSIDENCE AND RELATED CAVING PHENOMENA

By Lloyd A. Thomas
January, 1971

Acknowledgements

Surface subsidence related to draw from the San Manuel mine has been under constant observation and study by a number of people from the beginning of production to the present writing. Magma Copper Company personnel have observed the cave area more continuously than any other group, and documentation on the progress of cave is recorded in Company files in the form of semi-monthly reports, monthly reports, special reports of unusual phenomena, ground photographs, aerial photographs, topographic maps, contour maps, transit surveys, etc., forming a substantial backlog of raw data of which this paper hopefully will arrive at a coherent summary. Allied to this surface observation are 15 years of basic production data which includes, among other things, data on rock fragmentation, heavy ground conditions, production efficiencies, powder consumption, repair, high pack, core recovery, etc., much of which is beyond the scope of this particular summary.

Previous summaries of early cave action and subsidence phenomena have been written by Magma personnel and they include at least two reports by J. D. Pelletier (1) (2) and one by L. A. Thomas (3). Additional early observations on the cave by outside workers include two master's theses under the auspices of the University of Arizona written by Griswold (4) and McLehaney (5). A later theses by Hatheway (6), also under the University of Arizona, discusses his observations on the mechanics of subsidence and the type of mass wastage units which form within the cave area. Many workers representing the United States Bureau of Mines have taken part in some phase of observing subsidence phenomena at San Manuel, and while most of their work is unpublished, one paper by Johnson and Soulé (7) describes a method used to measure surface subsidence during the years when the cave area was relatively small. A comprehensive program of mapping the fracture pattern exposed in the First Level drifts was carried out by E. D. Wilson, who reported his conclusions in two consecutive references cited (8) (9).

In this report, the sections dealing with the early part of First Level history which include "The Initial Failure of the Gila Conglomerate" and parts of "The Continued Growth of the South Orebody Cave Area" are the result of observations recorded at the time by J. D. Pelletier. The remainder of the report is largely a summation of previously unpublished observations and conclusions recorded in Company files by L. A. Thomas and M. A. Enright. No real attempt has been made to include the commentary of outside investigators, except insofar as certain raw data provided by the United States Bureau of Mines appeared pertinent.

No general discussion of the geology of the orebody is included here although the regional structural setting is introduced as being necessary for an understanding of the mechanics of caving as they are to be presented in this paper. The gross geologic setting is primarily the concept outlined by Lowell (10), but the physical characteristics attributed to the rocks comprising the orebody are the responsibility of Magma observers.

Introduction

Two routine techniques have been used over the years to study the development and growth of the cave area. Initially, subsidence pins were set in place over the orebody on a 100-ft. square grid system, with installation being accomplished by both the U. S. Bureau of Mines and Magma personnel. Over the years nearly 1,000 of these pins were set and each one surveyed for coordinate position and original elevation. A triangulation net was established for the purpose of sighting on each pin when the ground began to move and traverses were no longer safe. Regular transit or theodolite surveys of the pins were made until September, 1961, by which time the cave area had grown so large and so many pins had been destroyed that the system was abandoned. In the beginning, transit surveys were made every two weeks as detectable slump developed at the surface. Later, monthly surveys were initiated and as the cave area grew and deepened, quarterly and finally semi-annual surveys became the established routine. Some of the advantages of the pin grid/transit survey technique included: (1) horizontal component of movement was readily discerned and measured, (2) small sag on the order of 0.1 ft. was easily detectable, and (3) a fine control system was available for mapping cracks on foot, to almost any degree of detail which the observer might desire. The obvious disadvantages, of course, are the tedious job of triangulation and plotting hundreds of separate pin locations and the eventual destruction of pins by cave action which ultimately left the interior of the cave with almost no recognizable control points.

Therefore, coincident with the start of Second Level production, semi-annual flights of the cave area were begun and a topographic map of the area was produced by photogrammetric methods. The advantages of this method include fine resolution of the interior topography and an excellent view of the larger peripheral tension cracks. However, small cracks cannot be seen and horizontal components of movement are not readily measurable by this method.

THE STRUCTURAL SETTING

In order to explain the caving phenomena which have occurred due to production from the San Manuel mine, it is necessary to consider the characteristics of the rock types which are related to the orebody. Hence, a brief description of the gross geologic setting is pertinent.

As conceived at the present time, the San Manuel orebody is the lower half of what was originally an elliptical (in cross-section) cylinder some 8,000 ft. long with major and minor cross-sectional axes of 5,000 and 2,500 ft. The cylinder was comprised of a central core of monzonite porphyry rock which had been intruded into, and was therefore surrounded by quartz monzonite rock (Oracle granite) with the economic ore zone generally occupying the elliptical contact zone between these two rock types. After emplacement, the cylinder was diagonally sliced along its long dimension by the San Manuel fault, which separated it into halves, moving the upper half some 8,000 ft. down dip and bringing a wedge-shaped blanket of Tertiary conglomerates into position over the lower segment of the cylinder. Hence, the San Manuel orebody with which this report deals is seen to be the lower half of the bisected cylinder, with the igneous host rocks capped by a varying thickness of Gila conglomerate which lies above the major structural feature of the San Manuel fault. Thus the rock columns which are set in motion by draw are not homogenous masses of igneous rock reaching to the surface, but consist rather of both igneous and sedimentary components in varying proportions separated by the San Manuel fault. Since the physical characteristics of the igneous and sedimentary components of a given rock column are very different from each other, the pattern followed by caving and subsiding ground has reflected these physical differences and given rise to some of the phenomena which are to be described in this report.

The igneous rock masses in which the orebody occurs are structurally incompetent rocks, with the monzonite porphyry generally more incompetent than the quartz monzonite, Both have been closely fractured, strongly altered, and disrupted by several major post-ore fault zones. Structurally speaking, considering the size and shape

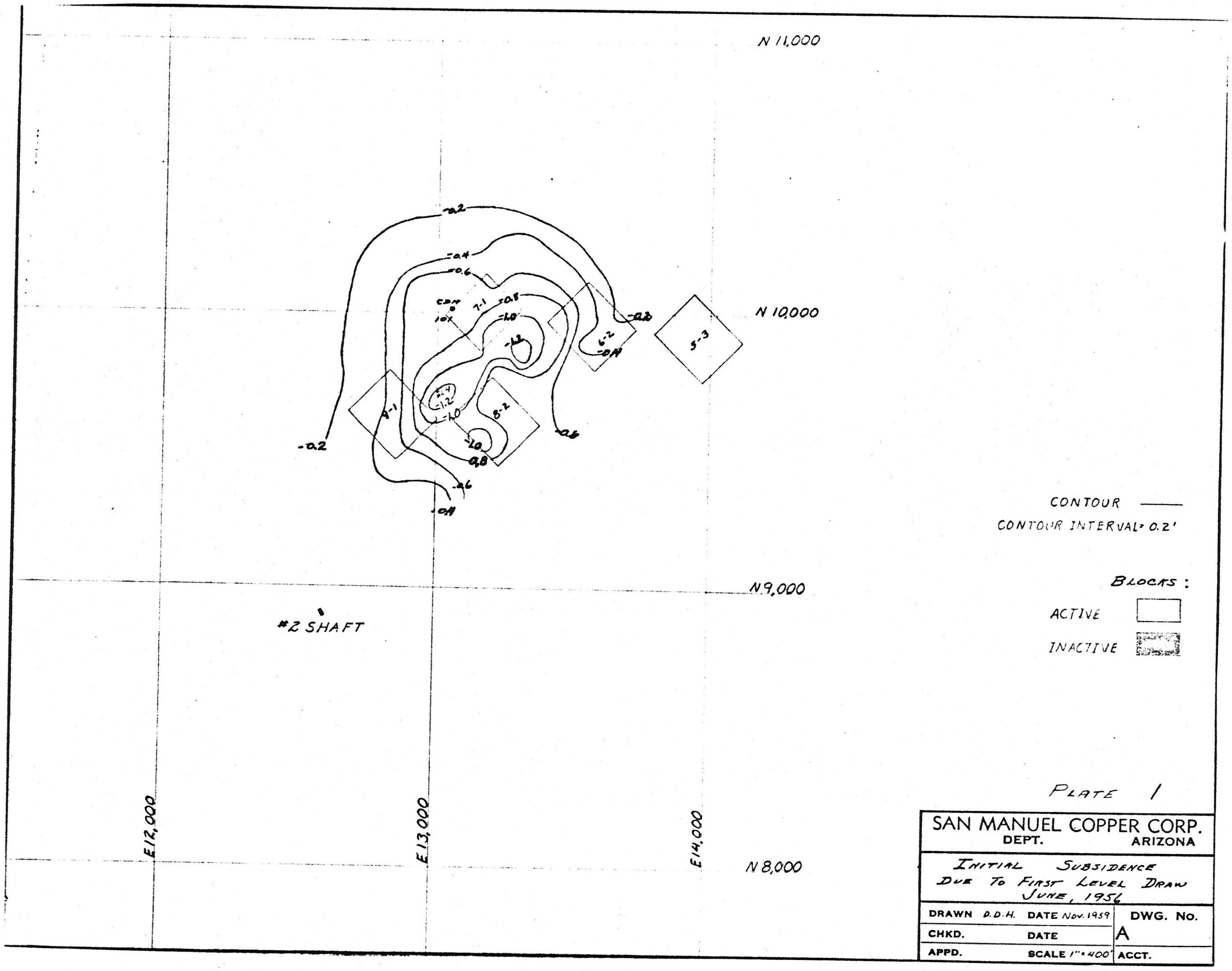
of mineralized ground and the degree of fracturing exhibited, the orebody and its inner porphyry core could be considered to be a macro-breccia comprised almost entirely of angular fragments (with no significant rotation) separated from each other by poorly bonded fracture planes which are usually coated with non-cementing alteration products such as sericite, chlorite, various argillaceous products, or fault gouge. In the monzonite porphyry rock, the size of the angular fragments ranges from $<3''$ to 24", or perhaps occasionally more. Quartz monzonite fragments, normally less angular than porphyry and less closely fractured, may range in common sizes from 3" to 6", to boulders several feet across. The compressive strength of such an environment cannot be great and those investigations to determine numerical values for it which have been carried out (normally by personnel from the U. S. Bureau of Mines) have shown some extremely low values. Pancake-type instrumentation installed on 1415 Level in Panel 8 Fringe Drift showed that rock in the drift was crushing and the timbered drift beginning to cave at only 1,500-1,600 psi. This is perhaps the absolute minimum value for compressive strength in monzonite porphyry but it serves to illustrate how little strength the porphyry exhibits in gross aspect when it is placed under compressive stresses. A reliable average value for stress required to cause failure in porphyry is probably not known, nor is it known for quartz monzonite.

In contrast to this igneous environment below the San Manuel fault plane, the sedimentary beds of Gila conglomerate which lie in the upper plate must be considered to be very competent rock, relatively unfractured despite some local weaknesses due to throughgoing post-Gila faults or tuff beds concordant with the attitude of the enclosing strata. In the course of 15 years of operating experience at the mine, it has been clearly demonstrated that the Gila is capable of standing in a vertical escarpment 300 ft. high for many years and that where several hundred feet thick, it is capable of arching over spans several hundred feet across.

SUBSIDENCE DUE TO FIRST LEVEL DRAW
SOUTH OREBODY

A. The Mining System. A "checkerboard" pattern of undercut blocks, 210' wide and a multiple of 30' or 35' in length, was chosen as the mining method by which to initiate production, for it was felt that such a system would give a maximum area in both directions from which caving action could withdraw support from under the massive Gila conglomerate. The east portion of the South orebody was chosen as the location for the first blocks because, among other advantages, it was cut by two major post-Gila faults which were believed, and later proved, to be helpful in creating an initial breakthrough in the conglomerate. Undercutting began with Block 7-1 on November 24, 1955 and progressed sequentially through Blocks 9-1, 6-2, 8-2, and 5-3 at an average rate of 1,055 sq. ft. per day. After the completion of Block 5-3 undercut on June 15, 1956, a sufficient number of draw raises were available for the desired production rate of 30,000 tpd and undercutting was temporarily halted. At this point the five undercut blocks had a total undercut area of 260,400 sq. ft., and an area within their undercut perimeter of 627,000 sq. ft. [The "undercut perimeter" is that line which circumscribes the entire area undercut. In the case of a checkerboard system, the area within the undercut perimeter must necessarily be much greater than the area actually undercut.] This area was deemed sufficient to cause failure in the Gila conglomerate capping and the initiation of a successful surface subsidence.

B. Initial Failure of the Gila Conglomerate. The first evidence of failure in the conglomerate occurred on April 7, 1956 when a large volume of air began to discharge out of the collar of Churn Drill Hole 101 which was located within the undercut area of Block 7-1 (Plate 1). The hole had been plugged since its completion at the San Manuel fault plane (350 ft. below the surface and 770 ft. above undercut level), and the sudden appearance of air discharging at the collar showed that the debris forming the plug had fallen away, probably into a small void created just under the fault plane. Since the undercut at this point had been completed about 2½ months earlier, the indicated average rate of progress of the cave upward through the monzonite porphyry rock column was approximately 10 ft. per day. At the same time, total draw from the draw raises beneath the churn drill hole averaged 75 vertical ft., giving an indicated average draw rate of about one ft. per day and an indicated average expansion in the porphyry of 11%. Daily plumbing of CDH 101 was started in an attempt to record the progress of cave upward through the conglomerate beds. The record of these probes shows that the conglomerate was periodically spalling off at the bottom, failing in tension along bedding planes. The first detectable surface cracking occurred on May 7, 1956 at a point 35 ft. outside the vertical limits of Block 7-1 when the average draw from the block had reached 8% of the total rock column. The crack occurred along a bedding plane in conglomerate although on the same day plumbing of CDH 101 showed that the actual cave had progressed only 33 ft. upward into the beds, leaving 319 ft. of conglomerate which had slightly sagged, but was not yet significantly broken. During May, 1956 bedding plane cracks continued to form at the surface and, in fact, bedding plane cracking related to the draw from Block 9-1 (the second block undercut) appeared on May 15th, 1956, 80 days after completion of undercut-



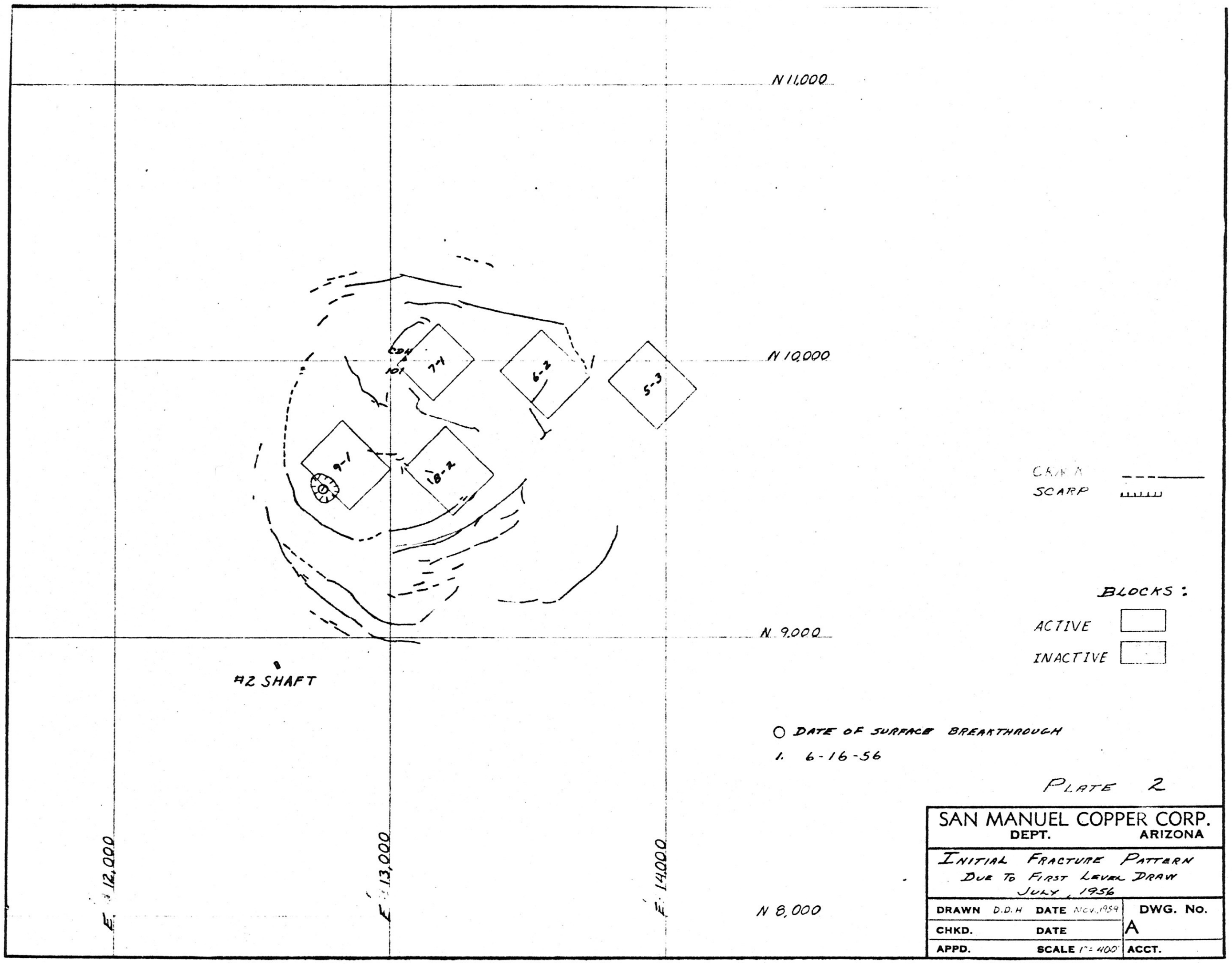
N 11,000
N 10,000
N 9,000
N 8,000
E 12,000
E 13,000
E 14,000
-0.2
-0.4
-0.6
-0.8
-1.0
5-3
#2 SHAFT
CONTOUR
CONTOUR INTERVAL= 0.2'
BLOCKS:
ACTIVE
INACTIVE
PLATE 1
SAN MANUEL COPPER CORP.
DEPT.
ARIZONA
INITIAL SUBSIDENCE
DUE TO FIRST LEVEL DRAW
JUNE, 1956
DRAWN D.D.H.
DATE Nov. 1959
DWG. No.
A
CHKD.
DATE
APPD.
SCALE 1"=400'
ACCT.

ting there when draw from that block had reached 12% of the total rock column. The higher percentage of extraction required for the sag to reach the surface over Block 9-1 may have been due to the fact that it had a greater thickness of conglomerate in its total column.

In late May and June, tension cracks concentric to the general center of draw began to form, and by the end of June they had expanded outward in all directions until an oval area 1,800 ft. long by 1,400 ft. wide had been defined whose outer margin was comprised of cracks dipping vertically to steeply outward from the mining area. These cracks were independent of structure in the conglomerate except where fault planes in the conglomerate inclined in nearly the same attitude as the cracks, the cracks followed the fault planes for some distance. (See Plate 2.)

A plumbing of CDH 101 on June 12, 1956 found the depth to cave at 225 ft. below the surface over Block 7-1. Thus, in the 65 days of elapsed time since the cave had reached the base of the conglomerate, it had progressed upward into the conglomerate ± 125 ft., or an indicated rate of two ft. per day.

Developing in conjunction with the surface pattern of cracking was the first measurable subsidence of ground. A broad, shallow slump developed over the general mining area and by May 18, 1956 (11 days after the first cracking) it had reached a maximum of 0.59 ft. in the area just north of Block 8-2. By May 28, 1956 it was 0.76 ft., with the area showing maximum readings expanding easterly toward the south edge of Block 7-1. June 12, 1956 recorded 1.44 ft. maximums in a pattern similar to that of May 28th, as shown in Plate 1. Note that the June 12 readings were taken at the same time as the last plumbing of CDH 101, and subsidence in the vicinity of the churn drill hole on this date measured 0.70 ft., even though 225 ft. of conglomerate remained essentially unbroken there. By mid-June it became apparent that the area showing detectable slump was bounded by the tension cracks concentric to the general center of draw, and that maximum values for the slump were also centered over the general mining area. An accelerated rate of change in the subsidence area set in during June, for by July 5 and July 26, maximum readings had reached 4.32 ft. and 5.01 ft. respectively, with the deepest parts of the subsidence area localized just south of Block 7-1. This area remained the deepest part until March, 1960 when the pattern of draw shifted it to the east over Block 6-3. Note the uniformity of action shown by the developing cave area as it formed, for the subsiding mass, with one exception, never showed any tendency to develop individual centers around individual blocks, but rather settled slowly as a unit within the area outlined by the concentric tension cracks.



N 11,000
N 10,000
N 9,000
N 8,000
E 12,000
E 13,000
E 14,000
7-1
6-2
5-3
9-1
8-2
#2 SHAFT
SCARP
BLOCKS:
ACTIVE
INACTIVE
○ DATE OF SURFACE BREAKTHROUGH
1. 6-16-56
PLATE 2
SAN MANUEL COPPER CORP.
DEPT.
ARIZONA
INITIAL FRACTURE PATTERN
DUE TO FIRST LEVEL DRAW
JULY, 1956
DRAWN D.D.H
DATE NOV. 1959
DWG. No.
A
CHKD.
DATE
APPD.
SCALE 1" = 400'
ACCT.

As subsidence progressed, certain of these outermost tension cracks became the limiting boundary of subsiding ground and eventually they developed into the prominent scarps which outlined much of the mature cave area.

The single exception to the pattern of broad general subsidence of the developing cave area occurred over the west edge of Block 9-1 where on June 16, 1956 an area about 100 ft. in diameter suddenly broke through to the surface with the "plug" dropping approximately 30 ft. (See Plate 2.) Concentric tension cracks soon formed around the rim of this isolated breakthrough, so that it resembled a small duplication of the subsiding area as a whole within which it itself occurred. This exceptional case occurred in the newly developing subsidence area for the simple reason that a pipe rose to the surface along the general plane of the East fault, one of the throughgoing post-Gila structures which was effective in inducing local weakness in the cohesion of the conglomerate beds. The fact that it so occurred points to the wisdom of designing the initial blocks in a caving system to utilize the known planes of weakness to induce an original collapse at the surface. The pipe rose through 670 ft. of igneous rock plus 450 ft. of conglomerate and collapse occurred when average draw from the block had reached 177 vertical ft., or 15.7% of the total rock column (9,000,000 cu. ft. of igneous rock withdrawn). Since the collapse absorbed only 30 vertical ft., the expansion in the porphyry column calculates to approximately 25%-30%, assuming insignificant expansion in the massive Gila beds. Block 9-1 had been under draw for ± 135 days at the time of collapse, so the indicated average rate of progress of the caving action upward is approximately 8.3 ft. per day. In view of the observations made by plumbing CDH 101, it is likely that this 8.3 ft. average actually consisted of a factor of several times that rate per day in the igneous column, a brief interruption of progress at the San Manuel fault plane, and an eventual continuation through the Gila beds at a much slower but continuous rate made possible by the weakening influence of the East fault.

Block 9-1 was also the location of some underground observations on cave angle and rates of progress of the early cave, which were made from the 1285 exploration level driven some 105 ft. above the First Lift undercut. These observations were necessarily all made early in the history of the mine as the exploration level was soon destroyed by draw. The plan map of Block 9-1 (Plate 3) shows the location of the block and the exploration drift with the drill holes which tested the early angle of cave from 100 to 400 ft. above undercut. Vertical sections through the drilling are included as Plates 4 and 5. Note that in its early stages the caved limit actually arched inward over the block, and

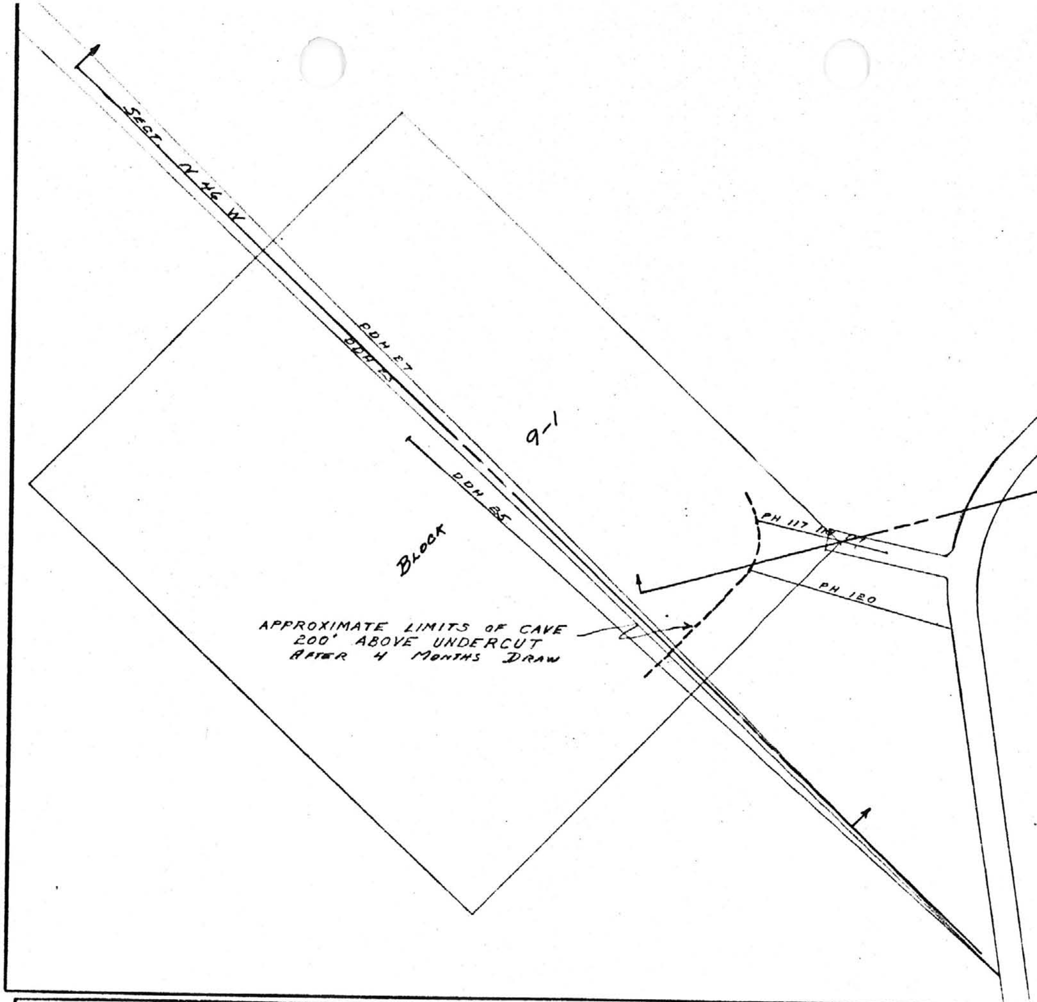


PLATE 3

SAN MANUEL COPPER CORP.
DEPT. ARIZONA

PLAN SHOWING DIAMOND DRILL PROBES INTO BLOCK 9-1 FIRST LEVEL

DRAWN J.D.P.	DATE	DWG. No.
CHKD.	DATE	S-
APPD.	SCALE 1"=50'	ACCT.

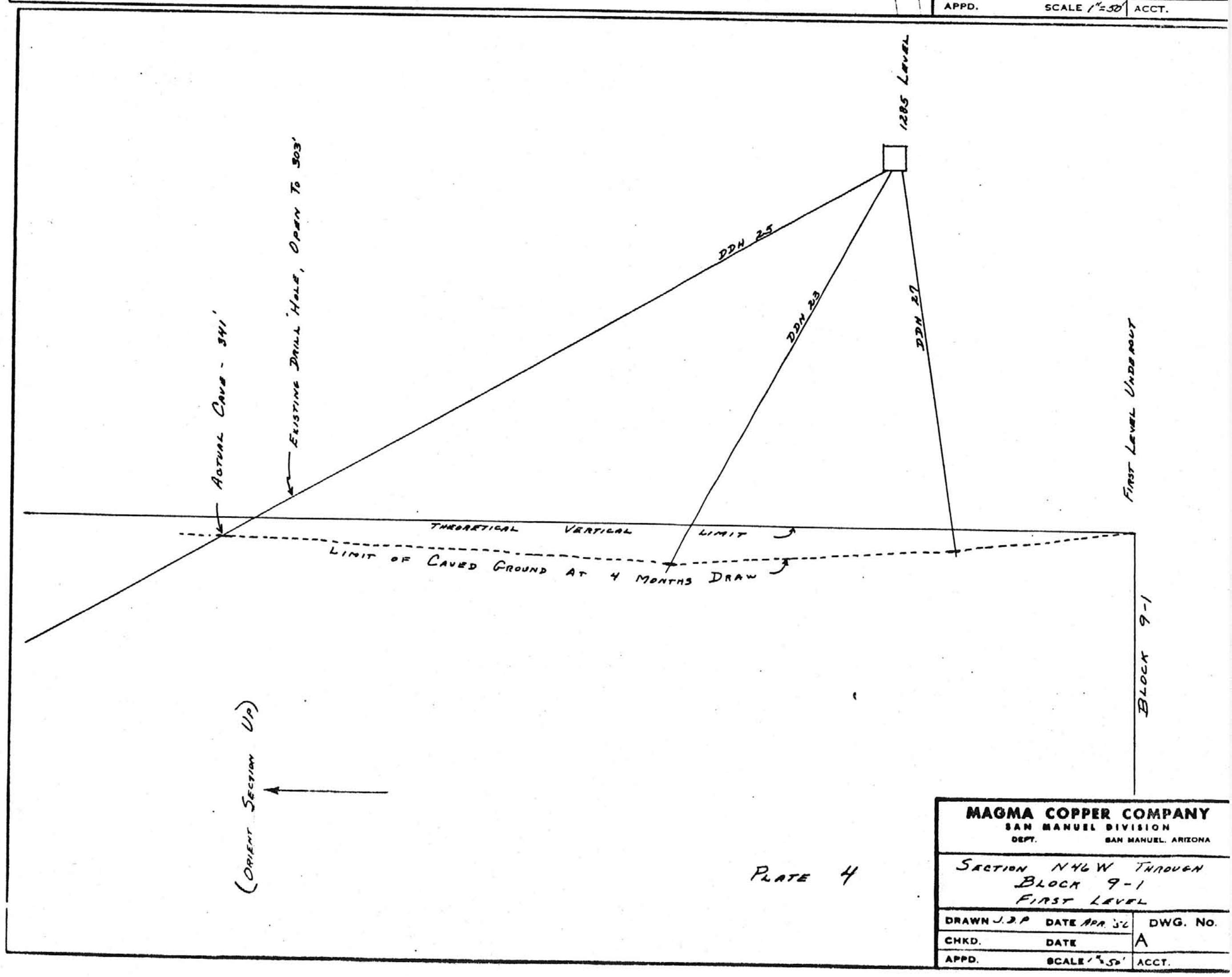
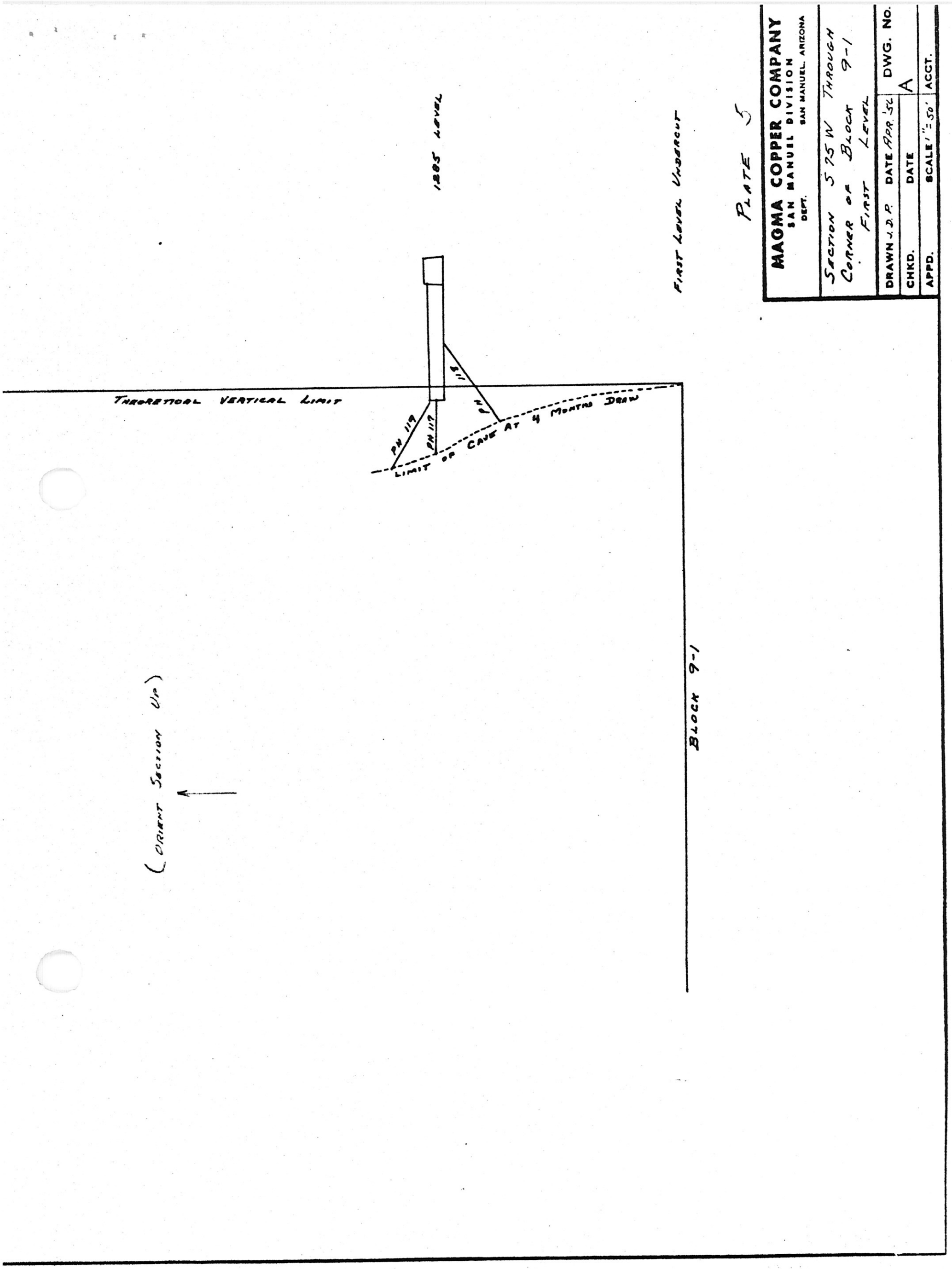


PLATE 4

MAGMA COPPER COMPANY
SAN MANUEL DIVISION
DEPT. SAN MANUEL, ARIZONA

SECTION N 46 W THROUGH BLOCK 9-1 FIRST LEVEL

DRAWN J.D.P.	DATE APR. '56	DWG. No.
CHKD.	DATE	A
APPD.	SCALE 1"=50'	ACCT.



that the square corners did not initially break out very far above undercut level. As a result, the rectangular shape of the active draw area became rounded at the corners and slightly more constricted with increasing height. The illustrations shown are applicable to the month of April, 1956 prior to the detection of any cracking or sag at the surface and hence presumably before any real pressure had been transmitted onto the porphyry by sag in the conglomerate mass. No further testing of the shape of the caved limit was possible, but subsequent observation of the behavior of the surface suggests that the arching effect was soon destroyed, for once the conglomerate began to sag, stresses were applied to the porphyry wall in excess of its compressive strength and it failed toward the block, resulting in an ultimate cave angle which lay outside the vertical limits of undercutting. This mechanism is believed to be the explanation for the very large tonnage overextractions which were obtained from the initial checkerboard blocks on the 1415 Level.

Before describing subsequent growth of the cave area, it might be well to summarize the caving mechanism as it had occurred to this point. In a paper prepared for presentation to the Tucson subsection of AIME, J. D. Pelletier wrote a clear summary of the action producing the initial breakthrough, parts of which are appropriately quoted here:

> "by piecing together information gained from a study of the draw and by observation of subsidence at the surface and weight patterns underground, the following caving action is thought to occur:
>
> Immediately after undercutting, caving progresses rapidly upward through the fractured porphyry of the ore zone. The speed of caving is much faster than the rate of draw and is limited only by the expansion of caved rock within the block. In one instance where we attempted to measure the rate of caving by probing a diamond drill hole which passed through an [active] block 100 ft. above undercut, it was found that the cave had advanced past the drill hole the first day after removing undercut pillars. Although small arches 30 to 40 ft. across are observed during undercutting, no arches are thought to persist in the porphyry once the block begins to cave.
>
> . . . Observations from the 1285 Exploration Level driven 105 ft. above undercut showed that during early stages of the draw the ground tended to cave nearly vertically to slightly inward from the block boundaries. The tendency to pull inward is

especially notable in the corners of the block and results in the cave assuming a [more] circular section in plan rather than continuing the rectangular shape of the undercut.

. . . When 10% of the porphyry column has been drawn, the cave has reached the bottom of the conglomerate and its upward progress is retarded. The conglomerate, because it is not a fractured rock, caves very slowly and is capable of standing in an arch for long periods of time and over areas much larger than our broadest blocks.

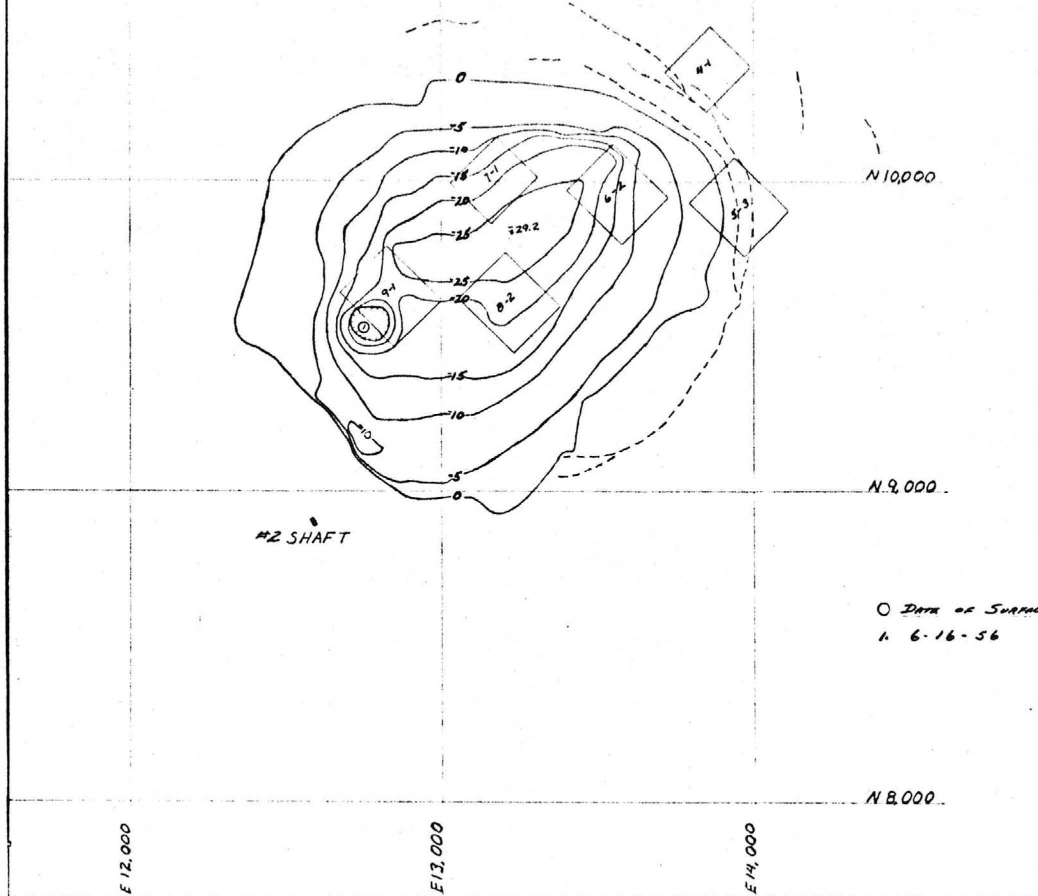
With continued draw from the blocks, a void forms in the lower part of the conglomerate column and the weight of the conglomerate above the arch bears on the pillars surrounding the block. The combination of this arch load plus the weight of the rock column itself, exceeds the compressive strength of the porphyry and the rock fails toward the block. ... Failure of the underlying porphyry allows the massive conglomerate capping to sag and, when the limits of strain are reached, the capping over the entire mining area shears off in a bending failure and subsides as a unit. Cracking of the surface expresses itself in two ways: first, in the central area where the surface is under lateral compression, the cracks are low angle breaks usually along bedding planes . . . and second, at the outer margin of the subsiding area, where the surface is under lateral tension, steep tension cracks develop a concentric ring pattern around the entire area. These cracks dip vertically to about 70 degrees away from the subsiding area and develop [by rotation] into fissures which sometimes attain great widths and depths of several hundreds of feet [i.e., to the base of the conglomerate column]. With continued subsidence the tension cracks become large scarps which outline the area beneath which the porphyry is failing in compression" . . .

C. The Continued Growth of the South Orebody Cave Area. By November, 1956, the maximum subsidence reached 29.2 ft. with the oval pattern well established and nearly all activity occurring within

the limits of the original concentric tension cracks. (Plate 6.) Block 4-1 was undercut in November and by then total subsidence represented 32.2% of total draw volume. June, 1957 recorded 86.6 ft. of subsidence south of Block 7-1 and, influenced by draw from Block 4-1 and the very weak Vent Raise fault zone coursing the northeast side of the mine, the cave area expanded to the north and tension cracks opened up further out over Block 4-1. Total subsidence now represented 55.7% of the total draw volume, and a new block (3-2) had been added to the producing blocks. On November 20, 1957 the surface over Block 3-2 collapsed over an area of 28,600 sq. ft. (compared with an undercut area of 36,750 sq. ft.) and completely outside the scarp line of the main cave area which existed at the time. Draw in 3-2 had reached an average of 159 ft., which represented 15.0% of the total rock column. The hole which formed (Plate 7, page 17) was located about half within the vertical limits of Block 3-2 and about half outside in a southwesterly direction toward Panel 4, and for several months it remained separated from the rest of the cave area. At breakthrough, the "plug" dropped perhaps 200 ft. vertically, indicating that a very considerable void had been drawn somewhere beneath. This block was located just under the lip of the conglomerate at the east end of the mine and it is probably the lack of any substantial thickness of conglomerate which accounts for its independent action at the surface and the obvious lack of "pressure caving" which resulted in a low tonnage extraction for a virgin block. Large tension cracks had formed still further out on both the north and south sides of the cave area in the interval since June, 1957, though the main subsidence was still located within the old perimeter and had reached a maximum of 134 ft. Total subsidence now represented 67.0% of total draw volume.

The final observations on the progress of cave as it rose through the 1285 Exploration Level were made in 1957. Note Plate 8 which shows the relationship between observed collapse of the exploration drift and draw from nearby blocks. On July 10, 1957 the drift was collapsed at the location illustrated, which is exactly on line between the two active blocks 9-1 and 8-2. The record does not show if this was the actual date of collapse, but it does serve to illustrate that after 16 months of production, the porphyry mass 100 ft. above undercut had failed well outside the vertical limits of the nearest active draw. The plate also illustrates the collapse of the drift over Block W9-4 which was undercut starting August 9, 1957. By August 14th the exploration drift had collapsed almost vertically over the block boundary and the collapsing ground continued to erode outward as long as observation continued.

Plate 8 also shows the location of Block E9-5 and the south scarp of the cave area. Block E9-5 was put into production starting



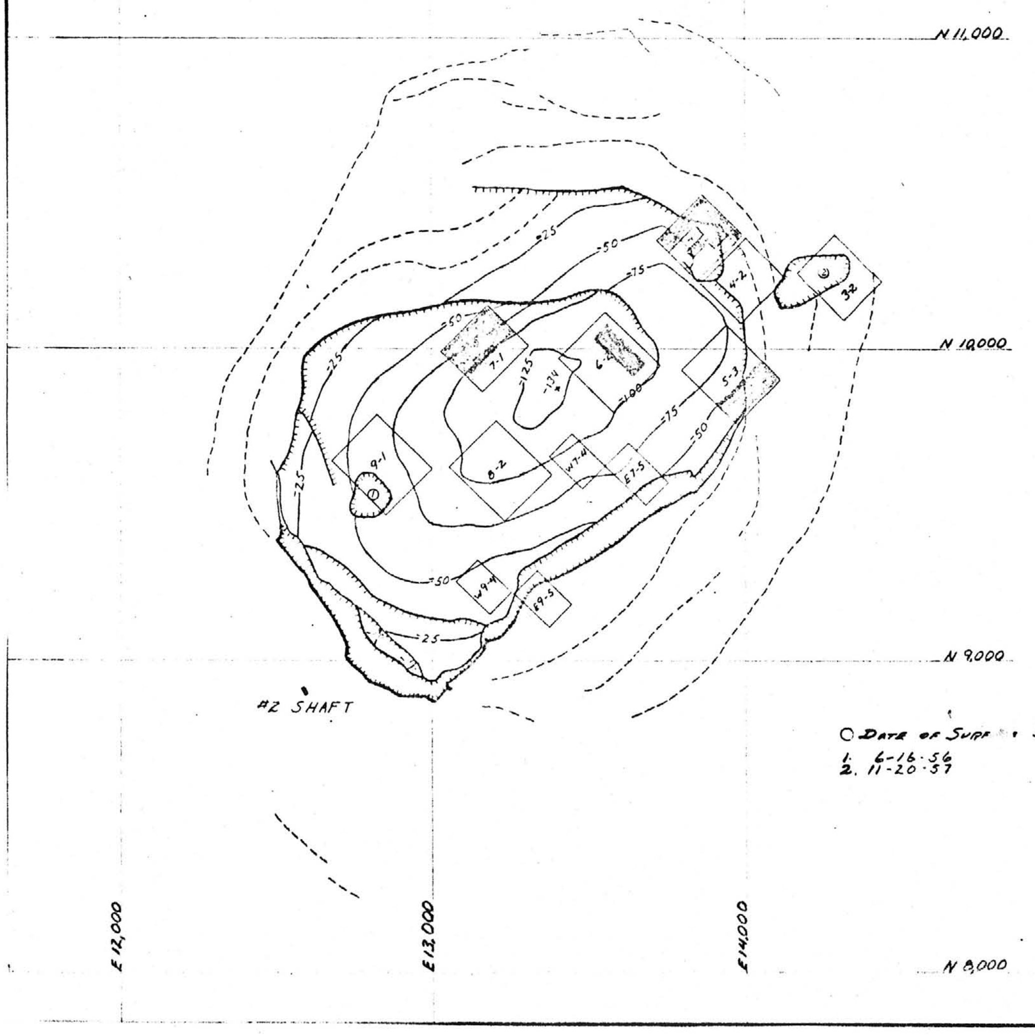
CRACK
SCARP
CONTOUR
CONTOUR INTERVAL = 5'
BLOCKS:
ACTIVE
INACTIVE

O DATE OF SURFACE BREAKTHROUGH
1. 6-16-56

PLATE 6

SAN MANUEL COPPER CORP.
DEPT. ARIZONA
SUBSIDENCE
DUE TO FIRST LEVEL DRAW
NOVEMBER, 1956

DRAWN D.D.H	DATE Nov. 1957	DWG. No.
CHKD.	DATE	A
APPD.	SCALE 1"=400'	ACCT.



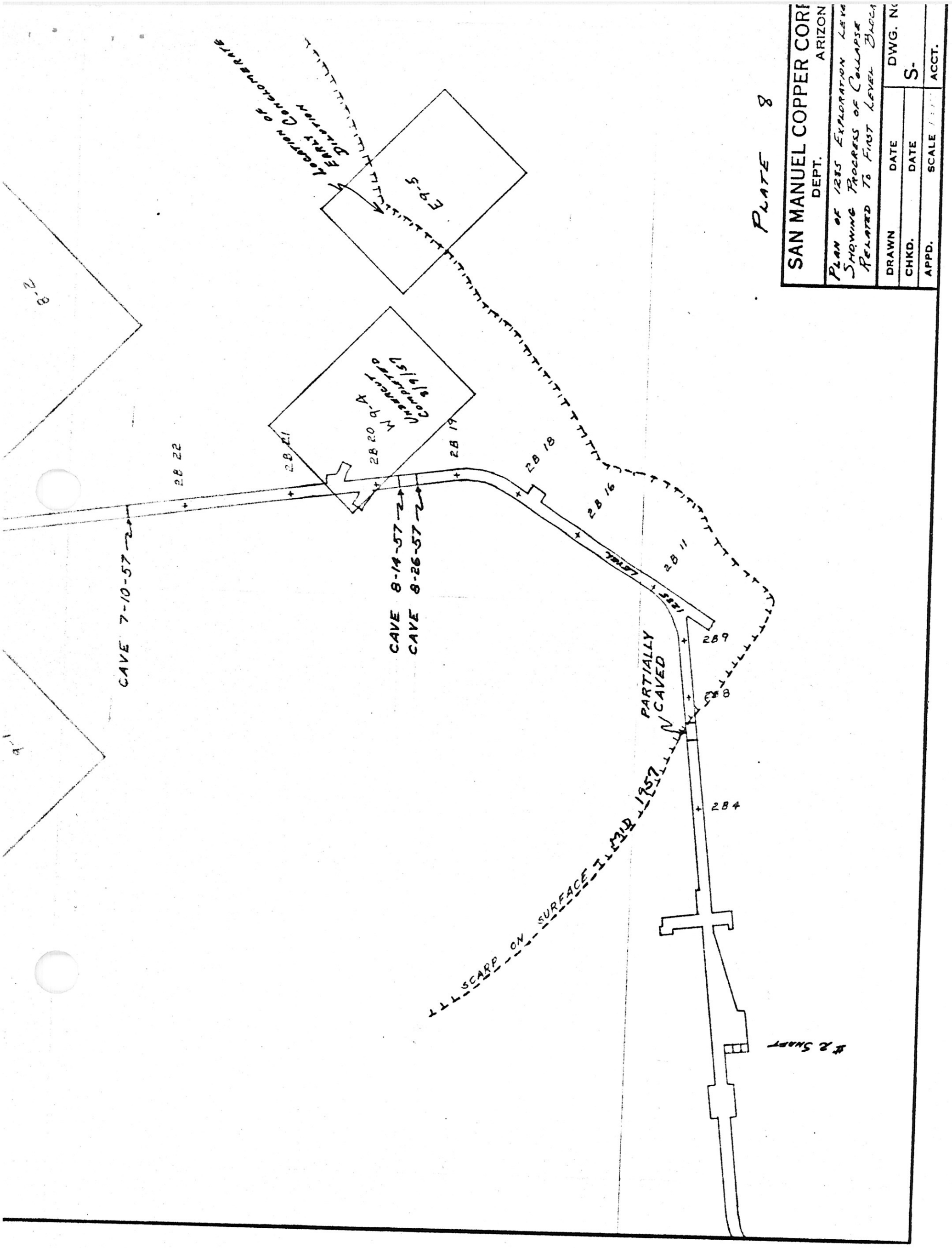
CRACK
SCARP
CONTOUR
CONTOUR INTERVAL = 25'
BLOCKS:
ACTIVE
INACTIVE

O DATE OF SURF[illegible] BREAKTHROUGH
1. 6-16-56
2. 11-20-57

PLATE 7

SAN MANUEL COPPER CORP
DEPT. ARIZONA
SUBSIDENCE
DUE TO FIRST LEVEL DRA[illegible]
DECEMBER, 1957

DRAWN D.D.H	DATE Nov. 1957	DWG. N
CHKD.	DATE	A
APPD.	SCALE 1"=400'	ACCT.



September 9, 1957 and by December conglomerate dilution appeared in draw raises located under the main scarp line. This occured because the tension cracks which formed the scarp line went deep enough to form channelways along which slabs of conglomerate could work their way down rapidly into the ore zone. The same pattern was repeated in Blocks 4-2, 4-3, and 11-3 where earliest dilution appeared in those draw points which were under prominent surface scarps.

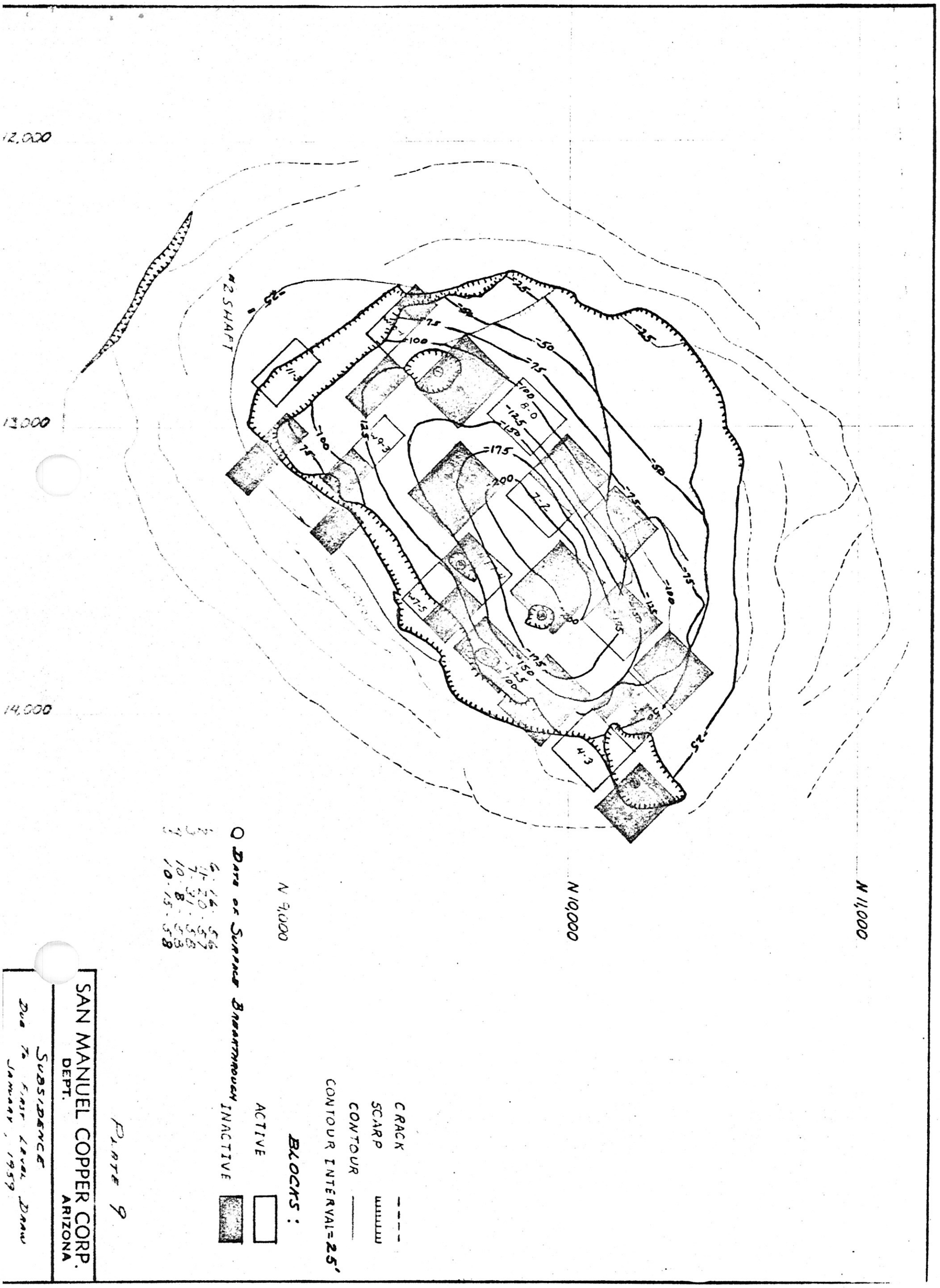
A tabulation showing the continued growth of the cave area through the period when routine volume calculations were being made follows:

Date	Maximum Subsidence, Ft.	Subsidence As A % of Draw
July, 1958	175	70.9
January, 1959	210	72.8
July, 1959	235	79.3
November, 1959	245	80.4
March, 1960	265	81.8
September, 1960	312	85.0
March, 1961	321	87.9
September, 1961	325	87.9

Plate 9 shows the subsidence pattern at January, 1959.

After September, 1961, volume calculations were discontinued because the area which had been the focal point of the studies was substantially depleted (on the First Level) and the cave area had become increasingly dormant (presumably approaching a state of equilibrium) as mining activity shifted to the west end and to the blocks of the North Orebody which are the subject of a subsequent section of this report. The cave area had grown in size as well as depth over the years and in September, 1961, it had a rectangular shape, slightly rounded at the corners and measuring 2,700 x 1,600 ft. inside the scarp line. Surface subsidence calculated to 500,526,000 cu. ft. against a draw volume of 569,156,000 cu. ft.

The consistency in both the pattern of subsidence and the location of the area of maximum subsidence over the years should be stressed here. In its early formative stages, note that the deepest parts of the cave area took the shape of an inverted cone centered over the middle of Panel 7 which was the center of draw



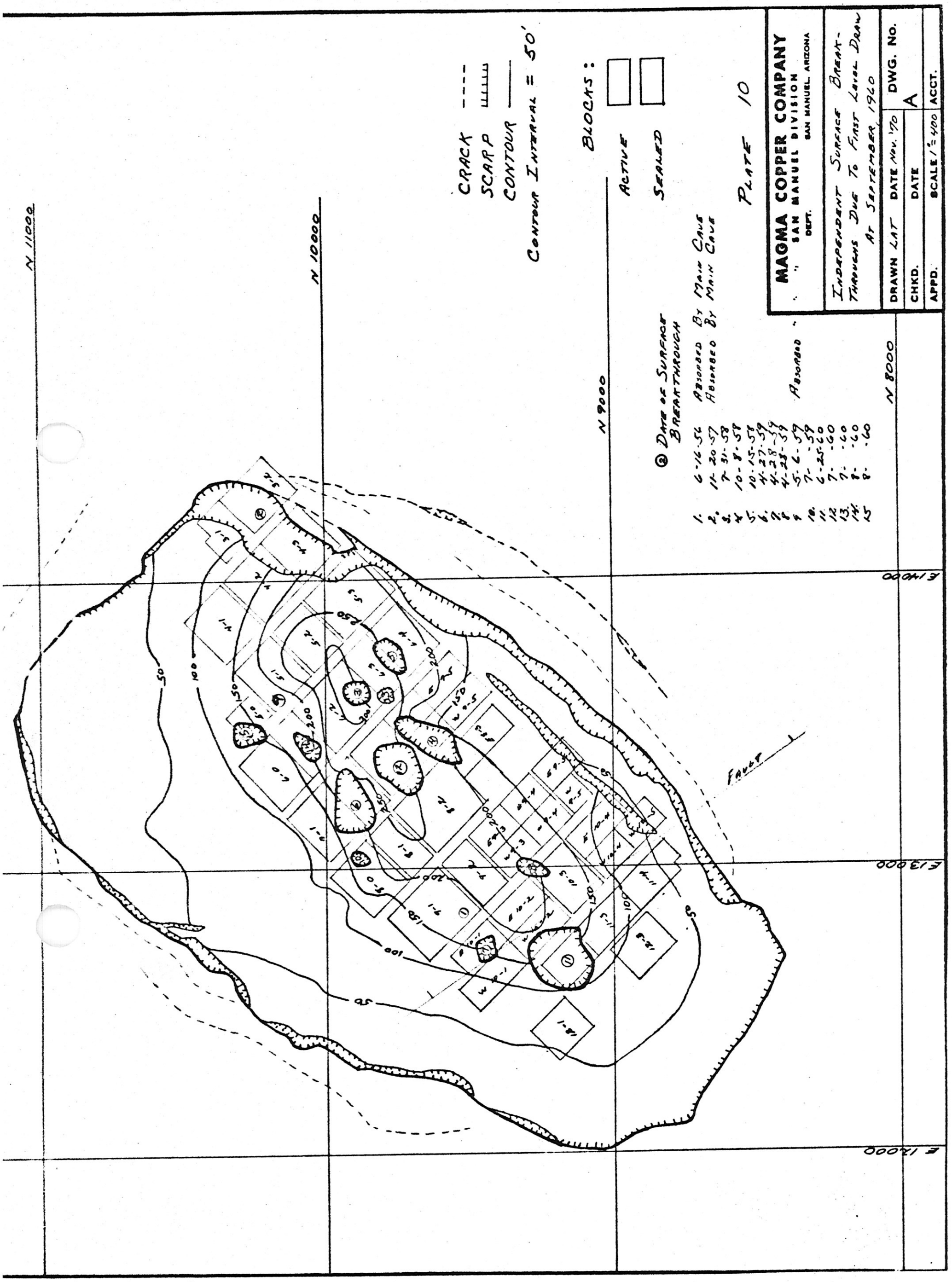
as well as the center of subsidence. As mining expanded longitudinally along the South Limb, the cone gradually grew into an elongate trough whose axis was likewise longitudinal with the orebody and which persisted generally over the axis of South Limb ore. In a gross way, a tendency developed for surface rock to move down and laterally in the direction of the trough. This was well shown by the series of subsidence pins which had been placed on the surface on a 100-ft. square grid prior to caving. Periodic surveys showed that these pins were gradually rotating toward the axis of the trough from all directions as the center deepened. This rotational convergence averaged about 100 ft., and in exceptional cases pins moved as much as 175-200 ft. laterally while slowly subsiding. Here again, the evidence shows the massive conglomerate acting as a unit over the entire area within the scarp line, for the convergence of the pins persisted and was always toward the trough whatever the pattern of draw from the blocks below.

A glance at a map of the cave area with the 1415 blocks superimposed on it, or any vertical cross-section drawn through the South Limb which shows the cave angle (i.e., the angle subtended between a horizontal line and a line connecting the undercut to the surface escarpment), shows immediately that the position of the mature cave area is decidedly asymmetrical with respect to the position of the undercut perimeter. This is equivalent to saying that the cave angles in the cross-sectional direction are not the same on opposite sides of the orebody. The reasons for this asymmetry are believed to be two, one dependent on the fundamental regional structural setting; the second of a more local nature. If one recalls the regional structural setting of the orebody as described earlier, it is remembered that the South Limb of ore wraps around an inner core of monzonite porphyry and that the ore zone in turn is surrounded by a large halo of pyritic rock in which quartz monzonite is often predominant. The compressive strengths of these rocks are quite different, with the porphyry being much the weaker. Hence, the stress field which arises around the undercut perimeter due to draw must have a concentric arrangement with the magnitude of stress contours decreasing outward. A stress capable of rupturing porphyry would be smaller, and therefore further out, than a value which would rupture pyritic quartz monzonite. Consequently, porphyry of the inner core would rupture at a greater lateral distance from the undercut perimeter, which, in effect, would produce a flatter cave angle in the porphyry core than in the pyritic halo. This is probably the most fundamental explanation for the asymmetry of the cave angles and the mature cave area did indeed show a 66° cave angle through the inner core and a much steeper 83° angle coming up through the pyritic halo. In addition, a local condition, i.e., the presence of a very weak longitudinally striking shear zone (the Vent Raise fault) which is inclined toward the workings and closely bounds the undercut perimeter, probably further weakened the porphyry and made it susceptible to rupture and flow. (See Plate 20.) [Additional data supporting this contention will be presented in sections following.] It follows from the above that if accurate average values for gross compressive strength of the igneous rock mass were known and the effect of loading by the conglomerate caps could be determined, it would be possible to accurately predict the location of the surface escarpment which would form from mining a given area, since this scarp would form directly over the perimeter within which igneous rock was failing in compression. This would not define the full extent of cracking, however, for tension cracks would continue to form well outside the escarpment, as the escarpment slowly rotated inward toward the center of cave. It also follows that in dealing with the problem of the creation of a scarp line or the outward expansion of an existing scarp line by new undercutting, that under the structural setting at San Manuel, one eventually runs into an area vs. height-of-conglomerate problem. Since the conglomerate cap over the San Manuel segment thickens from east to west from a feather edge to some 2,000 ft., it is logical to presume that the ability of the conglomerate to arch increases with increasing thickness and that therefore it would require larger and larger undercut areas to bring it down as its thickness increases. An interesting case history of this developing situation will be discussed in the section on North Orebody subsidence history.

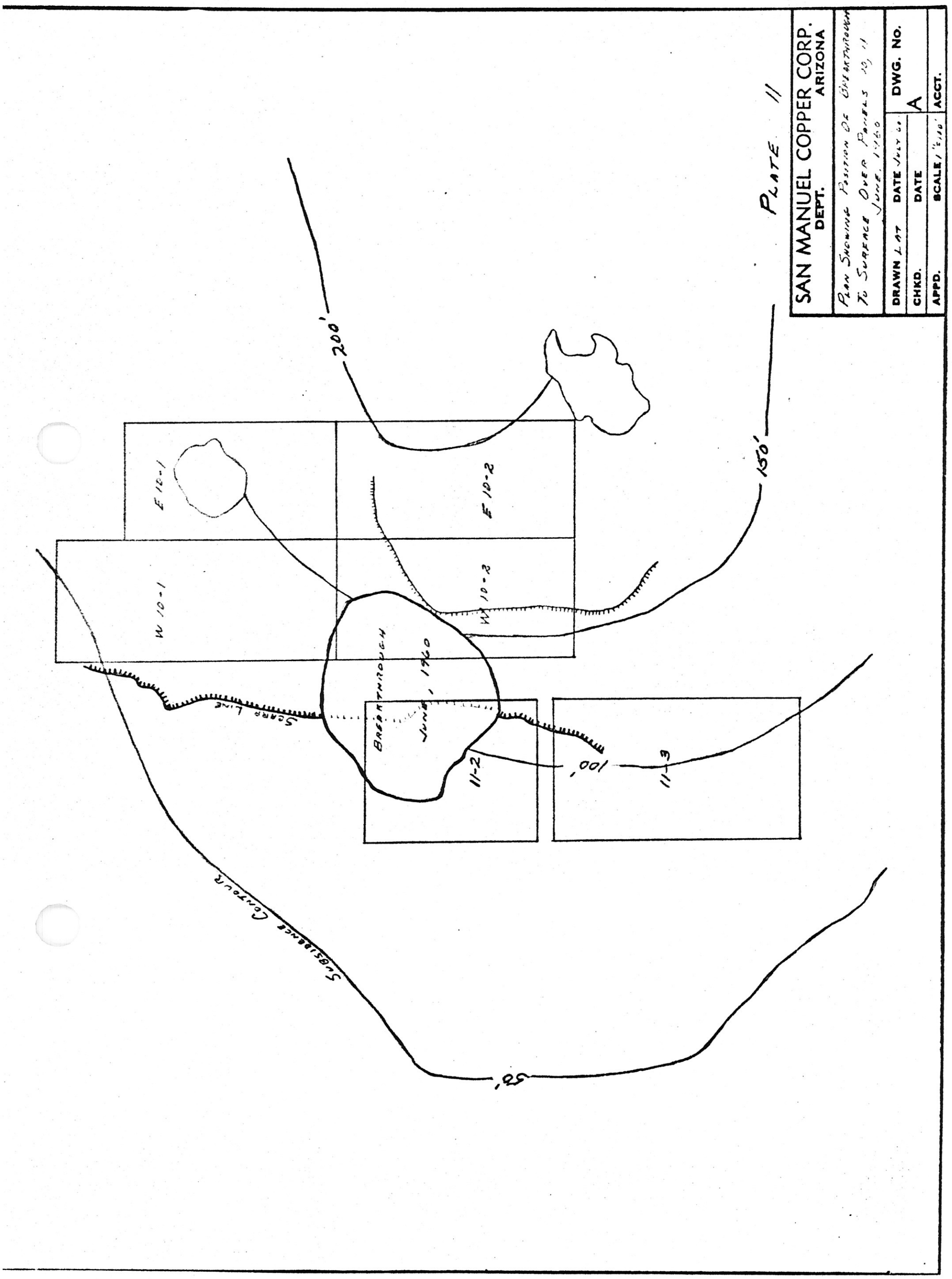
D. The Period of Independent Breakthroughs. While the general surface over the South Orebody was settling as a unit mass as detailed above, eventually some independent cave action related to the draw from individual blocks did occur, particularly in the period of time from August, 1958 until August, 1960. During this time at least 13 individual pipes broke through to the surface in an independent action which took place within the subsiding perimeter apparently without significantly affecting the overall rate of change of subsidence. [Two earlier breakthroughs and their relation to structure have been discussed above.] Note that the time period involved here is almost three to five years after the start of first undercutting and that general subsidence of 100 ft. to 260 ft. had already occurred at the point of breakthrough on the day the pipes holed through. The tabulation on the following page shows the sequence, and Plate 10 illustrates the conditions at September 12, 1960.

The Block 5-0 breakthrough was the last one to occur over the South Orebody, and by the date of its occurrence, most of the First Lift blocks under the active cave area were drawn and sealed. As production moved westerly into a restricted width of

Locat'n	Date Occur-red-	Total Subsidence, Vicinity, Prior to Breakthrough	Total Height, Original Rock Columns	Thickness of Conglomerate	Average Height, Igneous Column Withdrawn @ Breakthrough	Remarks
6-2	7-31-58	145'	1,100'	320'	647'	Block sealed out before breakthrough.
W7-4	10-8-58	125'	1,060"	420'	604'	Block sealed out before breakthrough.
6-4	10-15-58	100'	1,060'	340'	185'	Active block 60% drawn.
7-2	4-27-59	180'	1,110'	380'	370'	Active block 90% drawn.
E10-1	4-28-59	130'	1,140'	490'	155'	Active block 28% drawn.
W9-3	4-28-59	170'	1,125'	500'	285'	Active block 59% drawn.
5-1	5-6-59	180'	1,095'	220'	528'	Almost sealed, 133% drawn.
8-0	7-?-59	140'	1,140'	440'	414'	Almost sealed, 86% drawn.
11-2	6-25-60	100'	1,140'	550'	235'	Active block, 42% drawn.
7-3	7-?-60	250'	1,095'	400'	252'	Active pillar block, 45% drawn.
6-3	7-?-60	260'	1,100'	340'	204'	Active pillar block, 48% drawn.
6-1	8-?-60	180'	1,120'	320'	235'	Active pillar block, 55% drawn.
5-0	8-?-60	130'	1,140'	200'	175'	Active pillar block, 53% drawn.



ore and under a constantly thickening column of conglomerate, cave action diminished and eventually became dormant until draw from the Second Level renewed it. Since a total of 65 blocks were undercut on the 1415 South Orebody, and since 46 of these were included between Panels 3 and 11 where the pipes came through, the question naturally arises as to why the particular blocks listed holed through and others did not. Two can be related to throughgoing fault zones, namely E10-1 and W9-3 which, like 9-1, probably rose along the plane of the East fault. Two others, 5-0 and 5-1, are related to a small interior scarplet which crossed the surface there, showing the presence of a significant vertical tension crack produced by earlier draw. One, Block W7-4, apparently was triggered by undercutting Block E7-4, suggesting that a void was present and needed only a little more diameter to cause failure. And one, Block 11-2, almost certainly rose along the original scarp line which formed from a deep vertical peripheral tension crack initiated by the original cycle of undercutting in 1956. (See Plate 11.) For the others, the record shows no obvious relationship. However, there is enough evidence here to suggest the following theory of cave action, all of which reinforces what has been said previously: By mid-1958, when the period of independent piping began, the first 11 blocks to be undercut on the checkerboard system (total area 382,200 sq. ft.) had been drawn to completion and sealed, many with substantial overdraw; and nine new ones (242,025 sq. ft.) had been undercut in their place. As a result of this production, a broad, general settling of the surface had been underway for over two years, forming a basin-like depression covering 1,800 ft. in the longitudinal direction and 1,400 ft. in the cross-sectional dimension, with a maximum recorded depth of 175 ft. The volume of the subsidence was calculated on July 25, 1958 as 172,160,000 cu. ft., resulting from the withdrawal of 242,868,000 cu. ft. of igneous rock from beneath the conglomerate capping. Yet with all this production accomplished, there was [essentially] no departure from the persistent monthly settling of the oval mass within the scarp line, indicating that shattered porphyry below the conglomerate was failing in compression over a wide area and moving laterally toward active blocks with the conglomerate cap riding down on top of it and gradually breaking up as it was differentially stressed. This action has been locally termed "pressure caving". Finally, after two years of this type of slump, the conglomerate had been broken up enough with peripheral cracks and internal steep intersecting cracks to allow some separate and distinct pipes to rise to the surface along the loci of crack intersections, all the while the mass within the oval continuing its normal rate of slump. By the time the last pipe holed through, August 1960, surface subsidence was approximately 385,000,000 cu. ft. against a withdrawal of igneous rock of about 457,000,000 cu. ft. The subsiding area had grown



to dimensions of 2,600 ft. x 1,600 ft. and showed a maximum depth of 312 ft. With the passing of intervening years, all the pipes were absorbed into the general sinking mass around them and by January, 1965 none were visible any longer.

It might be instructive at this point to discuss some of the ideas regarding piping which appear in the literature, for they do not necessarily agree with the mechanism which has operated at San Manuel. That piping occurs in the igneous rocks, and rises rapidly to the plane of the San Manuel fault, there to be retarded by the competent conglomerate column above the fault plane, is not in doubt, and the process has been discussed in detail under the section "Initial Failure in the Gila Conglomerate". But the role of piping up through the conglomerate column to break through to the surface does not conform to classical concepts. For instance, it is evident from a consideration of the preceding discussion on the independent breakthroughs that:

(1) pipes through the conglomerate to the surface were not even necessary to initiate breakup of the conglomerate cap and induce its subsidence. In fact, the process worked the other way around. When general subsidence had progressed far enough to break the conglomerate cap by differential settling into great monoliths separated by vertical cracks reaching to the base of the conglomerate column, then piping was able to rise along intersecting cracks as spalling into open spaces beneath occurred.

(2) The idea that the newly formed pipe would then act as a funnel to transfer large quantities of surface waste to undercut level is not supported by evidence detailed here. In fact, probably no slab of near-surface conglomerate has ever reached undercut level or been transferred very far vertically down one of the pipes discussed above. The evidence against this type of process is substantial. Of the pipes recognized, note that two formed after the blocks that caused them were sealed completely; eight more were in their final stages of draw, and only three were in their mature producing period. Interestingly enough, these three were E10-1 and W9-3 where the piping was related to the East fault, and 11-2 where the pipe rose along a pre-existing scarp. Apparently it took such throughgoing planes of weakness to allow piping to rise into the conglomerate during the mature cycle of draw in a block. The way in which active blocks gradually sealed out also argues against a mechanism of rapid vertical transfer of surface conglomerate. No mature block has ever been flooded with masses of conglomerate waste which suddenly appeared at undercut level in such quantity as to require

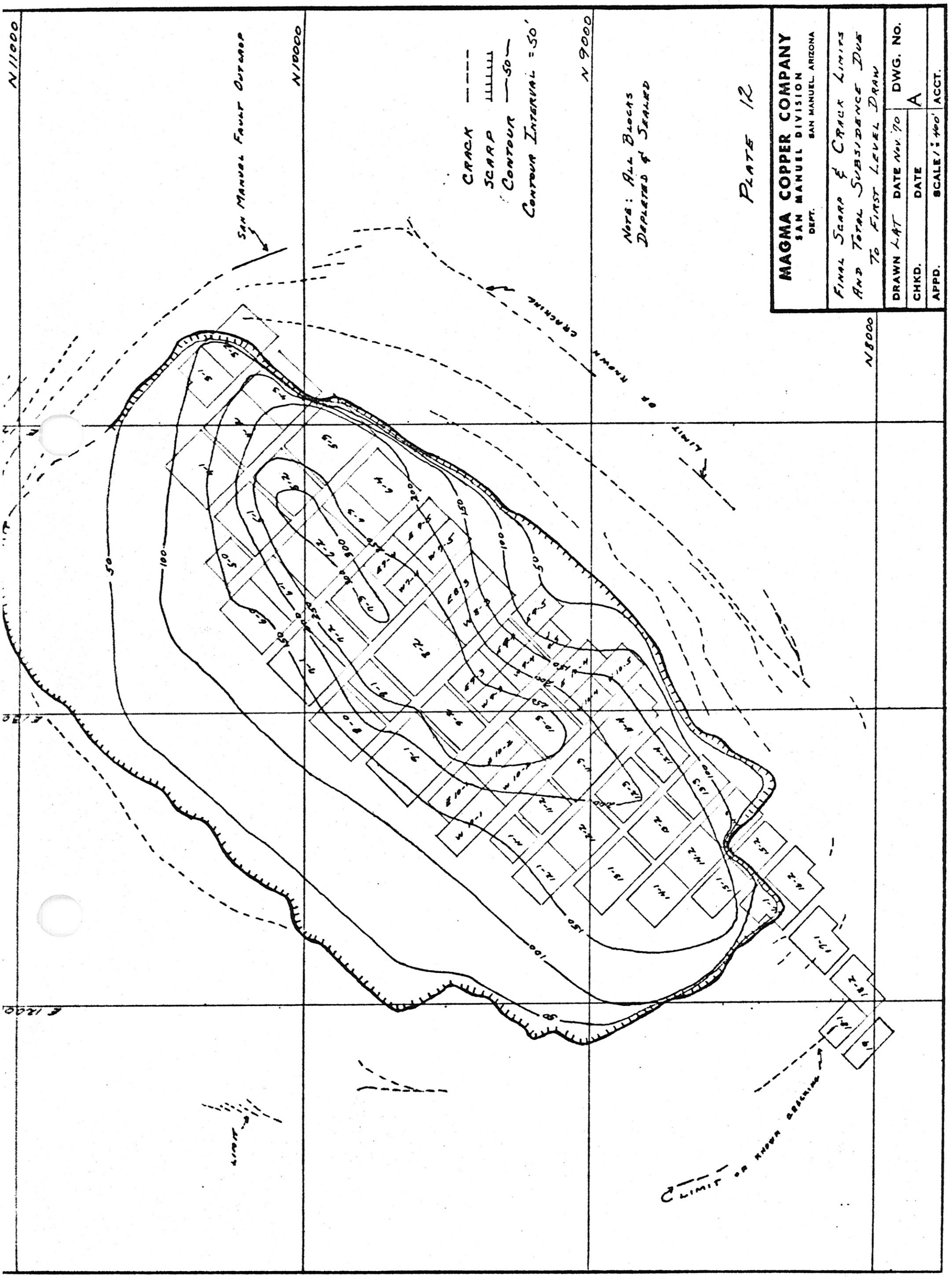
abrupt termination of draw. On the contrary, conglomerate has appeared first in small quantities in the draw points, and as time passes, the proportion of conglomerate to ore material increases until the overall product drops below economic grade and a seal is applied. The conglomerate dilution, therefore, probably came almost entirely from the bottom of the beds where slabs could break off along bedding planes and fall into the broken mass of igneous rock beneath, ultimately working their way to undercut level along with the surrounding igneous rock. [Also, we know the one special condition (discussed in Section C) where conglomerate slabs were rapidly transferred to depth down peripheral tension cracks which formed the main scarp line.]

E. The Advancing Scarp Line. The peripheral scarp which formed the southwest edge of the cave area is of particular interest since it lay in the direction of the undercut advance and hence would become the unstable edge, collapsing and re-forming further out as mining progressed along the strike of the South Limb. The original escarpment had formed along this side by mid-1957 (Plate 7) and it persisted as the outer limit of important subsidence until late 1958, despite the advance of undercutting into Panels 10 and 11. By late 1958, withdrawal of igneous rock from Panels 10 and 11 had progressed far enough that the original scarp was itself slumping and a newer one was forming about 600 ft. to the southwest over Panel 15. The new scarp line first manifested itself as a bedding plane crack in the conglomerate beds and with the passing of time it lengthened rapidly along the strike of the beds and began to slump inside the crack, with the motion including considerable bedding plane slip. By the end of 1958 the new scarp showed a height of about 10 ft., with the height of the original scarp now about 70 ft. (See Plate 9.) By mid-1959 the crack had lengthened commensurate with the width of the main cave area, and since it could lengthen no further, being supported at both ends by stable ground, conglomerate beds on the inside "tore" loose along tension cracks perpendicular to the strike of the beds, and the new scarp line arced around to feather into the main cave along these tension cracks which transected structure. By the end of 1959, with production still limited westerly at Panel 11, the new scarp line over Panel 15 showed perhaps 20 ft. of height, while the old interior one over Panel 11 was gradually being absorbed into the general cave and had dwindled to about a 50-ft. differential. These trends continued throughout 1960 and into early 1961, and the record of the cave area dated March, 1961 shows that the new outer scarp had an average height of approximately 35 ft., and that the old interior scarp had totally disappeared, being destroyed by heavy production from Panels 10,

11, and 12 which occurred during the interval. [Refer also to the section on independent breakthroughs which discusses the scarp-controlled pipe which holed through over Panel 11.]

Also during this interval, a third important bedding plane crack opened up, this one located over the middle of Panel 16 and variously 100 to 200 ft. further out than the second scarp now forming as discussed above. This third location proved to be the ultimate periphery of subsidence due to First Level draw, and its history of development is similar to that of the second scarp. In March, 1961 the crack was still a relatively small bedding plane break with no real slump, but which could be traced on the surface for nearly 1,000 ft. By September, 1961 significant withdrawal of igneous rock had been accomplished in Panels 13, 14, and 15, and the second escarpment had reached an average height of 60 ft. Along the third location, the crack had lengthened to 1,600 ft. and had left its bedding plane to arc around the north edge of the main cave area. It showed up to three ft. of slump on the cave side, and with the start of this amount of movement, ground still further to the southwest became inactive.

By November, 1961 significant draw had occurred in Panel 16 and undercutting was in progress in Panel 17. The third location, now seen to be partly fault-controlled, had taken over as the major perimeter of active subsidence and showed up to 15 ft. of displacement with ground inside the perimeter slumping at a general rate of one ft. per week. By the end of 1961 the vertical displacement had reached 30 ft., and subsidence within this perimeter was averaging two ft. per week. The scarp had also arced around to the south and was joined to the main cave area. The record does not detail the history of the intermediate second escarpment over Panel 15 at this period, but presumably it was being absorbed into the general cave as had happened to the first one over Panel 11. The fast development of this last scarp line as ground to the cave side of it rapidly subsided proceeded into 1962. In January, vertical displacement reached 40 ft., February showed 80 ft., and March, 1962 reached 100 ft. Heavy draw continued from Panels 13 through 18, and by June, 1962 the scarp's maximum displacement was estimated to be nearly 150 ft. After this rapid development, the scarp achieved a certain stability, and for the remainder of 1962 and early 1963 a broad but consistent settling of the mass within the scarp line characterized the cave action. In December, 1962 additional tension cracks formed outside the main scarp over Panel 17, the first real change noted there in 1½ years. By June, 1963, all draw from the First Lift South Orebody was complete and the entire west end of the cave achieved stability. Plate 12 shows the final location of the scarp line and the 1415 Level blocks which produced it. The final



position of the scarp line bears an interesting relationship to the tonnage extraction pattern from the underlying blocks. Those in the vicinity of the scarp but lying inside it showed an average tonnage extraction of 117%. Those lying outside the vertical projection of the scarp produced an average tonnage extraction of only 97%. This is a good illustration of the importance of "pressure caving" of the igneous rock under the conglomerate. Where the entire rock column was actively subsiding, the conglomerate exerted enough pressure on the underlying igneous rock to cause it to fail in compression and move toward active draw points, resulting in a fine overextraction from pillars, etc. But beyond the final scarp line, only the igneous rock under the San Manuel fault plane was subsiding, no "pressure caving" effect was present to aid in crushing the igneous rock and move it toward an active draw point. As a result, extraction was confined to the material within the vertical limits of each undercut block, where, acted on solely by gravity, rock moved generally vertically down toward some active draw raise. Once this loose column of igneous rock was withdrawn, there was no further pressure by which additional peripheral material could be forced toward the draw raise, and the raise was sealed due to the presence of slabs of conglomerate fallen from the immediate vicinity of the San Manuel fault.

It is immediately obvious from a glance at Plate 12 that there is a significant undercut area of the 1415 Level which lies outside the final scarp line, where little or no subsidence of the surface has ever occurred. This brings to mind the problem of causing the conglomerate column to collapse as its thickness increases and the area undercut decreases. No one knows at the present time whether the surface outside the scarp line did not collapse because it arched itself over a void or because the volume from which ore was withdrawn became filled with great slabs of conglomerate which now support a roof somewhere internally in the conglomerate beds. Based on our experience with thick sections of conglomerate over the North Orebody (q.v.), it is most likely that the westernmost blocks on 1415 South Limb have pulled a void which will eventually be triggered into a collapse by the advance of undercutting on the Second Level. The part of the undercut perimeter which juts out beyond the scarp line measures 104,000 sq. ft., and from this area 22,000,000 cu. ft. of rock was withdrawn; a figure approximating the ore reserve. This calculates to a theoretical 210 ft. high void in a 1,200 ft. total rock column, or removal of 17½%; insufficient to collapse the 950 ft. column of conglomerate.

SUBSIDENCE DUE TO SECOND LEVEL DRAW SOUTH OREBODY

A. The Mining System. A checkerboard pattern of undercut blocks was again chosen as the system for initiating production from the 2015 Level South Orebody. Undercutting began on April 26, 1962 in Block 8-4, and proceeded sequentially through Blocks 8-2, 7-3, 7-1, 9-3, and 9-1 until a total undercut area of 165,900 sq. ft. had been attained. With the completion of 9-1 undercut on September 29, 1962, undercutting ceased until August 1, 1963 as the area activated was deemed sufficient for the rate of production required. The average daily rate of undercutting was 1,270 sq. ft. per day; somewhat faster than for the First Level. Block widths were reduced to 140 ft. and block lengths were retained at a multiple of 30 ft. Note that two blocks were undercut in each panel on the first sequence. This presumably gave the advantage of allowing the entire panel to be mined in two sequences, limiting the total time the panel would be subject to maintenance and repair costs.

Since final production on 1415 South Limb did not occur until June, 1963, there was a time overlap of approximately 14 months when both levels were producing, although the closest blocks respectively were separated by 1,450 lateral feet.

The checkerboard system was continued only between Panels 4 to 10 inclusive and as undercutting advanced longitudinally down the strike of the South Limb, the system was gradually converted to a wedge pattern in which the lead undercut in each panel was made in the center of the panel and subsequent undercuts in that panel were effected by adding lines both north and south of the lead block. When fully developed, then, this system produced a wedge-shaped active undercut area which flared out from the center of the lead panel to the north and south edges of the trailing panels. Presumably the general center of each panel would seal out first since it was undercut first, and the outer perimeters would seal last, being undercut last. The advantages included a more uniform tonnage extraction than is possible with a checkerboard system and the possibility of abandoning completely the center of an older panel in cases of excessive repair costs. However, this system is difficult to keep in good balance since it requires periodic undercutting on both sides of the lead blocks and may tend to open up more active area than is really necessary. Furthermore, experience showed that it was almost impossible to get satisfactory draw rates or tonnage extraction from the last two or three lines adjacent to the pyritic zone on the south side when these lines were the last to be undercut. This adverse condition resulted from the hard, blocky nature of the ore along the pyritic contact. When undercut last in sequence, the ground did not get stressed sufficiently to cause it to break

up vertically over the undercut in time to prevent serious lateral dilution from flooding the draw points as more mobile waste moved in from adjacent sealed blocks.

Hence, the wedge system was effected only between Panels 11-15 inclusive and in the advance of undercutting starting with Panel 16, the lead block was taken at the south (pyritic) edge of the given panel, initiating a regular panel retreat system which attacks the hard, blocky ground first and adds lines as needed on the north side of the lead block, moving toward the increasingly incompetent porphyry mass of the inner core. This system, in use at the present time, has been very satisfactory so far from several points of view and may well be the optimum mining system which can be evolved in the wider parts of the San Manuel orebody. Among its advantages are:

1) the balance between undercutting and depletion is easier to maintain for at least two reasons:

   a. Each panel is an independent producing entity to which additional undercut area can be added as necessary. The new undercut can be taken in whatever trailing panel is most advantageous, or a new lead block can be added to the south end of the next panel.

   b. The point at which serious dilution appears in a block evens out considerably since each new undercut in a trailing panel always has two sides to virgin ground and two sides to active ground. This tends to allow good tonnage extraction from all blocks with less extreme variations than experienced on a checkerboard system.

2) the blocky, pyritic side of the orebody is always undercut first, which allows the full effect of changes in the stress pattern to bear on the hard rib as the undercut advances into virgin ground. Hence the hardest rock in each panel is always stressed first and longest and as a result the amount of breaking due to cave action is maximized, which improves both draw efficiency and tonnage extraction.

3) under the conditions of "pressure caving" which obtain in the San Manuel mine, there is always a certain amount of lateral flow of rock toward active blocks. Since the least competent rock moves more readily than the harder rock, the transfer is maximum in the direction from the inner core toward the pyritic rib. (i.e. north to south in this limb). Hence, taking the lead block in a panel on the south end actually facilitates the process of lateral transfer by removing the hardest rock first, which allows less competent rock to move in to replace

it. Each succeeding undercut in the given panel benefits from this process by receiving some overdraw of ore grade from the north and blocks along the northern perimeter show the greatest benefit, receiving large volumes of rock from outside the vertical limits of undercutting. Since this peripheral material being transferred is monzonite porphyry with marginal values in chalcopyrite and molybdenite mineralization, it is a welcome form of dilution, and large volumes of it can be absorbed in the draw.

4) since in the panel retreat system the area actually undercut and the area of the undercut perimeter are nearly the same, a given rate of production sets less total volume of ground in motion than a checkerboard system would do, and over a long period of time probably produces less maintenance and repair problems. Admittedly the panel retreat has not yet been used in this mine for starting a new level, and until it is, we have no experience as to how it will stand up under the worst pressure conditions to which the operation is subjected.

B. The Renewal of Action in the South Orebody Cave Area. Since draw from underground blocks is the only process which produces discernible activity of the cave area, the major part of the cave was dormant between late 1961 and late 1962 when production was in transition from the west end of the First Level to the east end of the Second Level. However, by August, 1962, about four months after the start of undercutting in Block 8-4 which signalled the beginning of Second Level draw, it was possible to see minor changes within the scarp line at the east end of the cave; effects which could only be due to Second Level draw. Our knowledge of this renewal is, of course, much more gross than the detail accumulated from First Level draw since the action took place in and around the old established cave. By September, 1962 a major new peripheral tension crack began to form around the entire north, east, and south sides of the cave area. In places it followed old breaks in the conglomerate which had parted due to First Level draw, and at the east end it was confined by the surface trace of the Cholla fault zone. Its location was generally 400-500 ft. outside the main scarp line. This crack, and others inside it, continued to develop steadily during the rest of the year and, in fact, some erosion outward of the scarp line occurred in the southeast corner when great vertical slabs of conglomerate broke loose from the scarp line and rotated gradually toward the center of cave. The topographic map of December 30, 1962 showed that since the inception of Second Level draw, the center of the cave had subsided 22 ft. with the maximum change vertically over 2015 Block 8-2. (Note that this is an additional value, for this area had already subsided about 300 ft. due to First Level draw.)

A major change in the configuration of the south scarp began in April, 1963 when one of the many large peripheral cracks to the south became the locus of movement which eventually created a whole new escarpment along the south side. This particular crack, located some 300 ft. outside the existing scarp, bounded an area 1,000 ft. long inside of which three to six ft. of slump occurred in the month of April. In May, rotation opened the crack to a 20 to 50 ft. width and the mass to the inside, now closely fractured by differential stresses, had subsided 20 ft. The location of this new scarp line was just about as far south of the original escarpment as the 2015 undercut perimeter was south of the 1415 undercut perimeter; hence both undercut and scarp changed location a commensurate distance without particularly affecting the cave angle. It was apparent that a renewed general pattern of movement had set in which could be described as a rapid sinking of the central mass of conglomerate both directly over as well as beyond the limits of the 2015 active undercut perimeter, which allowed peripheral parts of the mass to rotate inward toward the center of subsidence. Along the south side, the result was the formation of an ever widening and deepening chasm between the relatively stable (new) scarp line and the rapidly subsiding interior. The renewed subsidence progressed as shown:

Flight Date	6-Month Increase	Total Subsidence Due to Second Level Draw, Max.	Total Subsidence, Maximum	Height of New South Scarp
July 1, 1963	Up to 45'	50'	± 400'	25'
Jan. 1, 1964	Up to 35'	100'	425'	50'
July 1, 1964	Up to 40'		465'	100'
Jan. 1, 1965	Up to 50'	150'	500'	150'
July 1, 1965	Up to 70'	200'	575'	185'

C. Growth Due to Second Level Draw. As of November 1, 1970, production from the Second Level South Orebody amounted to 73,160,523 tons which is equivalent to some 915,000,000 cu. ft. Since all of this material was withdrawn from blocks located within the existing scarp line, most of the growth of the cave has been a broadening and deepening of the interior. The more interesting aspects of this growth are best illustrated by a series of isopach maps showing subsidence due to Second Level draw which isolates the effects of Second Level draw from total subsidence and allows a better understanding of the increments of growth. Such a series of isopach maps has been prepared and some of them are illustrated here as Plates 13 to 19, to which the reader is referred for a better understanding of the following discussion.

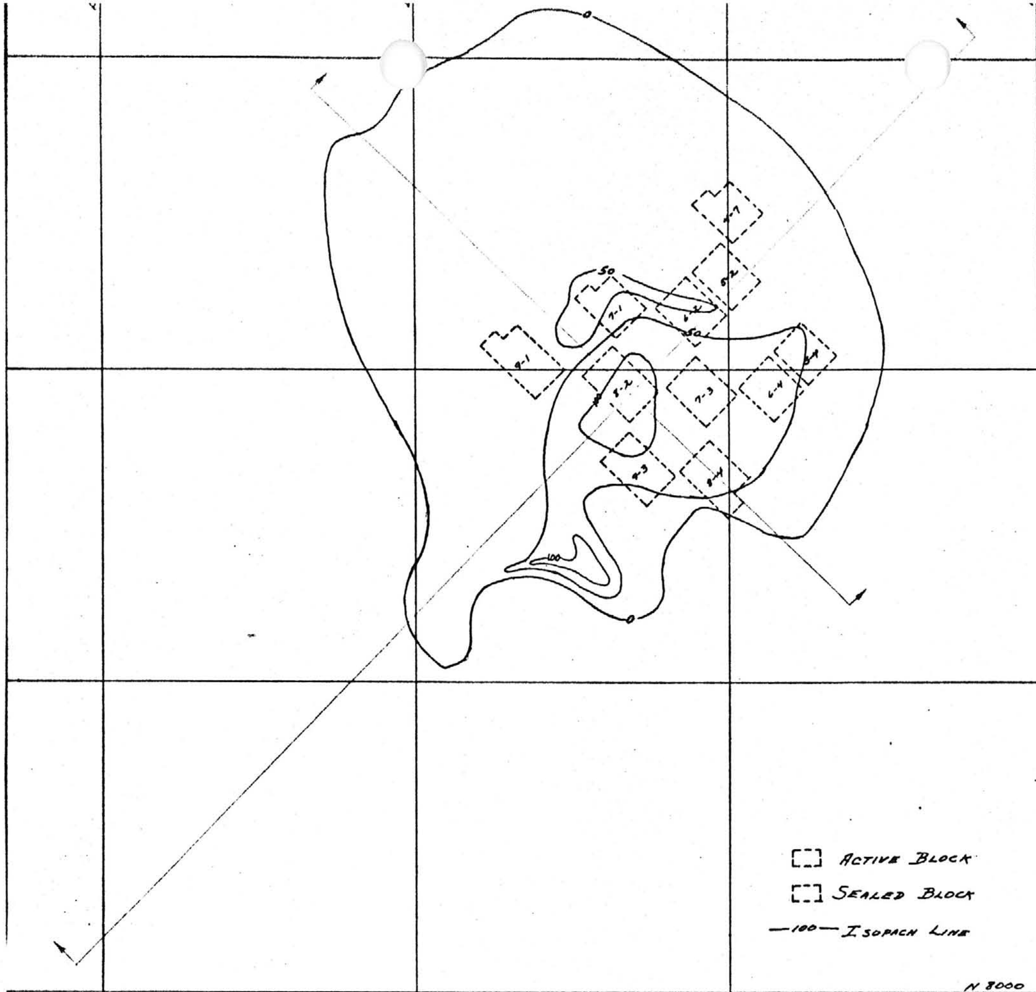


PLATE 13

MAGMA COPPER COMPANY
SAN MANUEL DIVISION
DEPT. SAN MANUEL, ARIZONA

SUBSIDENCE
DUE TO SECOND LIFT DRAW
1/1/64

DRAWN	DATE Nov. '70	DWG. No.
CHKD.	DATE	A
APPD.	SCALE 1" = 400'	ACCT.

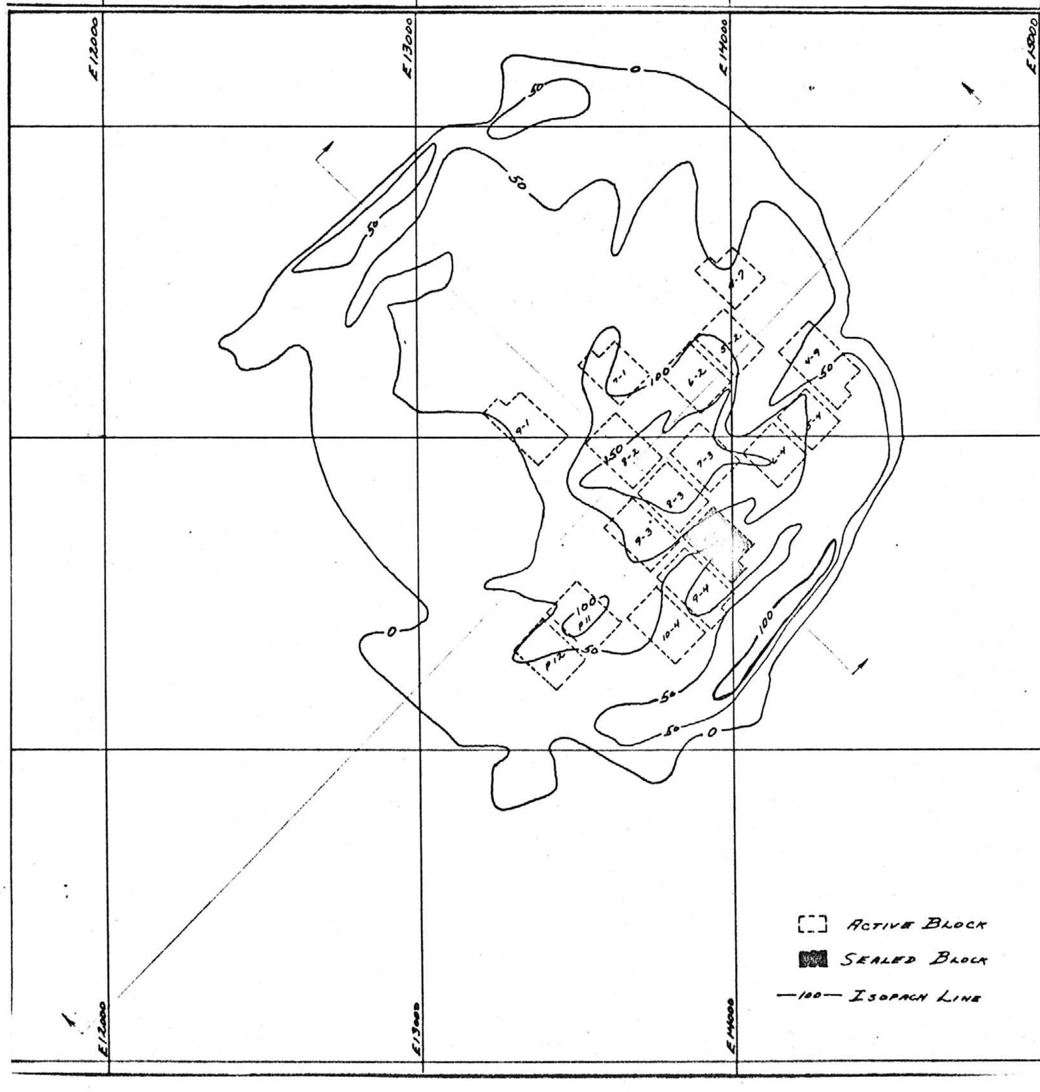


PLATE 14

MAGMA COPPER COMPANY
SAN MANUEL DIVISION
DEPT. SAN MANUEL, ARIZONA

SUBSIDENCE
DUE TO SECOND LIFT DRAW
1/1/65

DRAWN	DATE Nov. '70	DWG. No.
CHKD.	DATE	A
APPD.	SCALE 1" = 400'	ACCT.

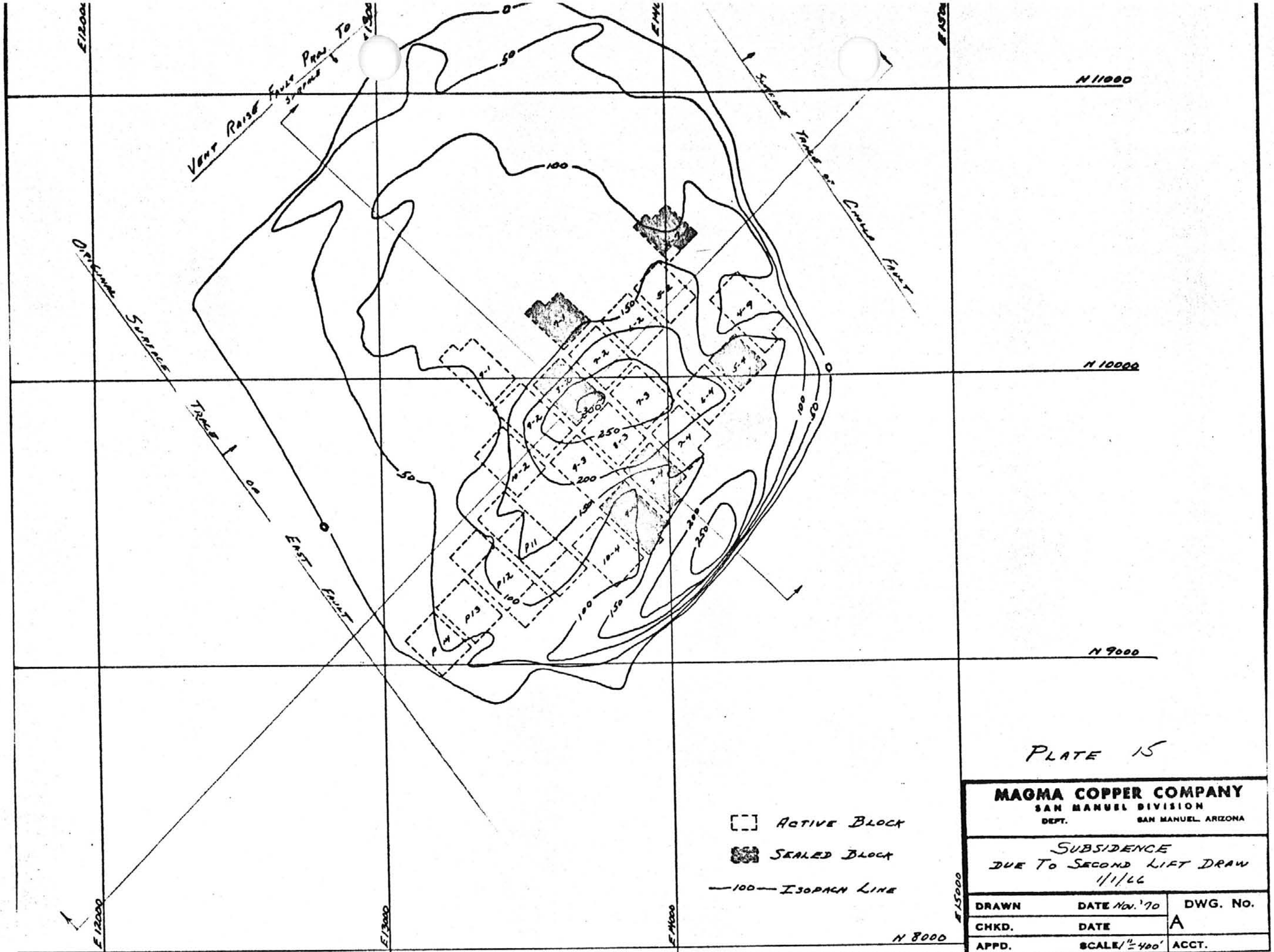


PLATE 15

MAGMA COPPER COMPANY
SAN MANUEL DIVISION
DEPT. SAN MANUEL, ARIZONA

SUBSIDENCE DUE TO SECOND LIFT DRAW 1/1/66

DRAWN	DATE Nov. '70	DWG. No.
CHKD.	DATE	A
APPD.	SCALE 1" = 400'	ACCT.

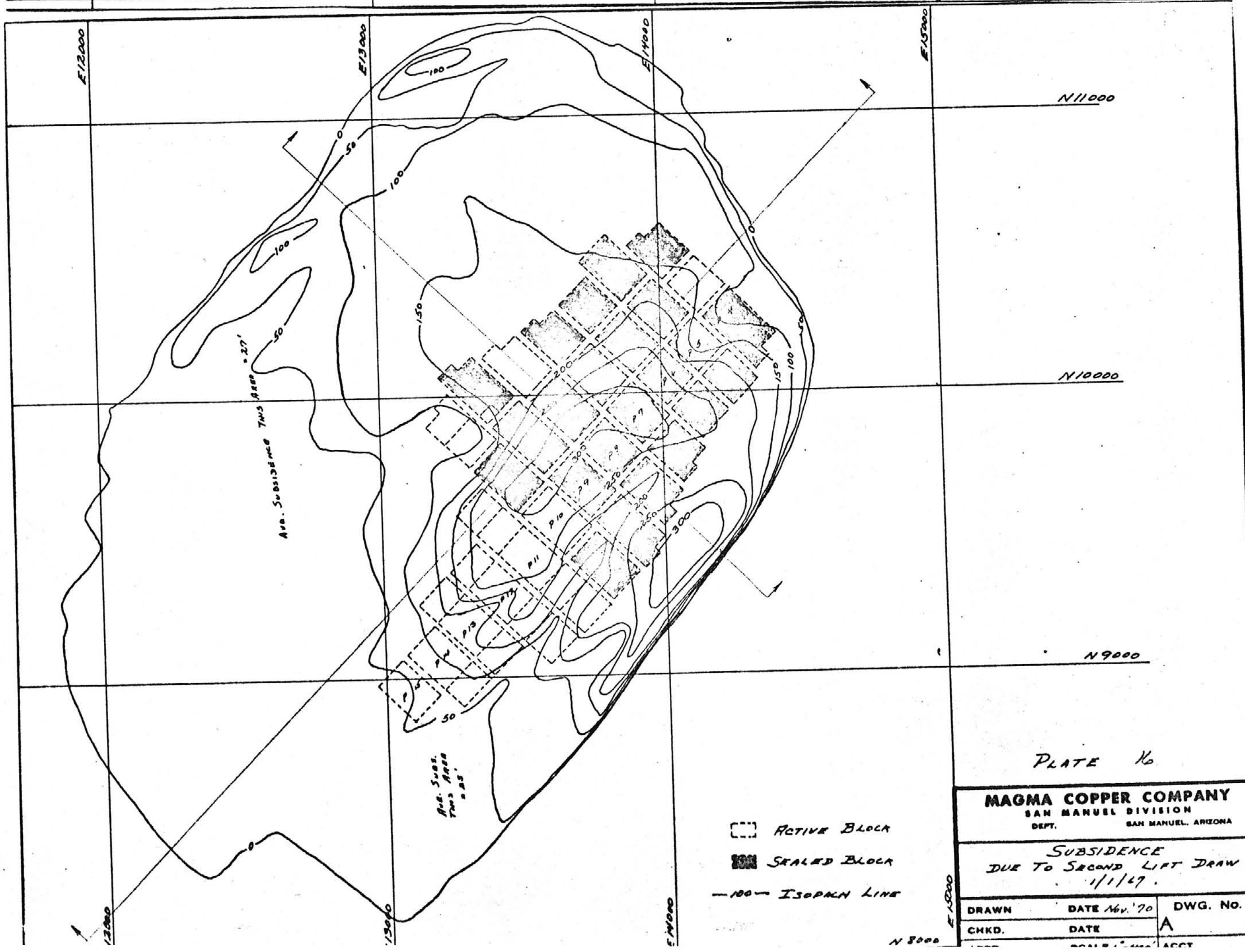


PLATE 16

MAGMA COPPER COMPANY
SAN MANUEL DIVISION
DEPT. SAN MANUEL, ARIZONA

SUBSIDENCE DUE TO SECOND LIFT DRAW 1/1/67.

DRAWN	DATE Nov. '70	DWG. No.
CHKD.	DATE	A

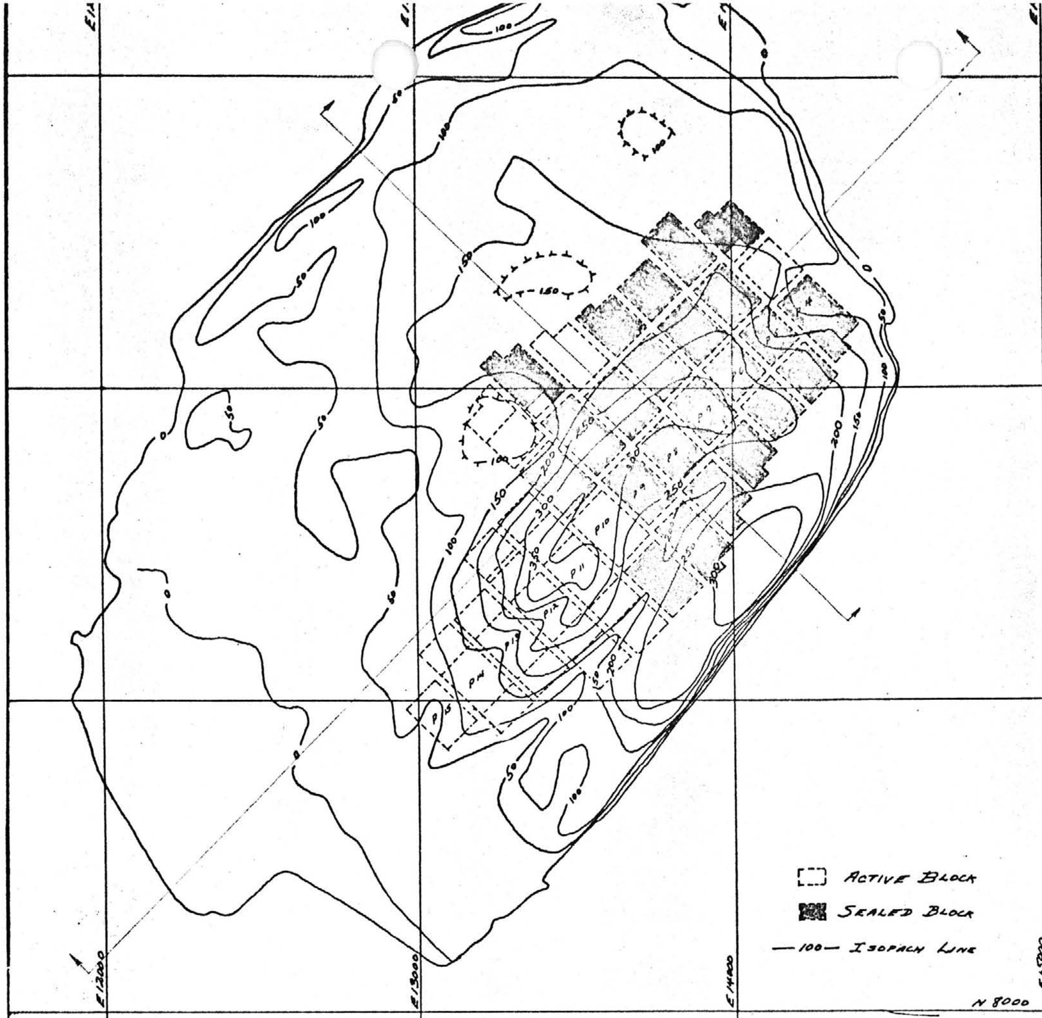


Plate 17

MAGMA COPPER COMPANY
SAN MANUEL DIVISION
DEPT. SAN MANUEL, ARIZONA

Subsidence
Due To Second Lift Draw
7/1/67

DRAWN	DATE Nov. '70	DWG. No.
CHKD.	DATE	A
APPD.	SCALE 1" = 400'	ACCT.

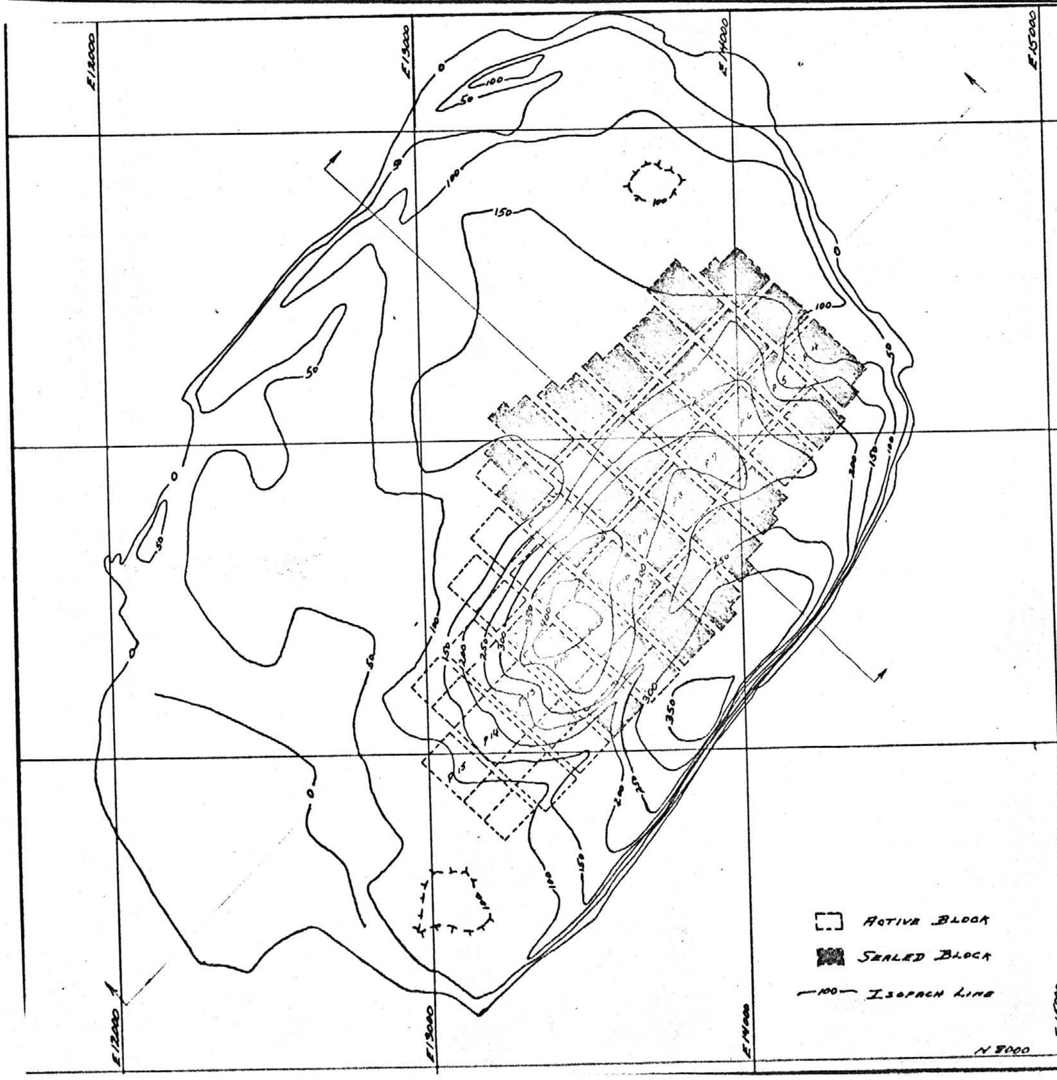
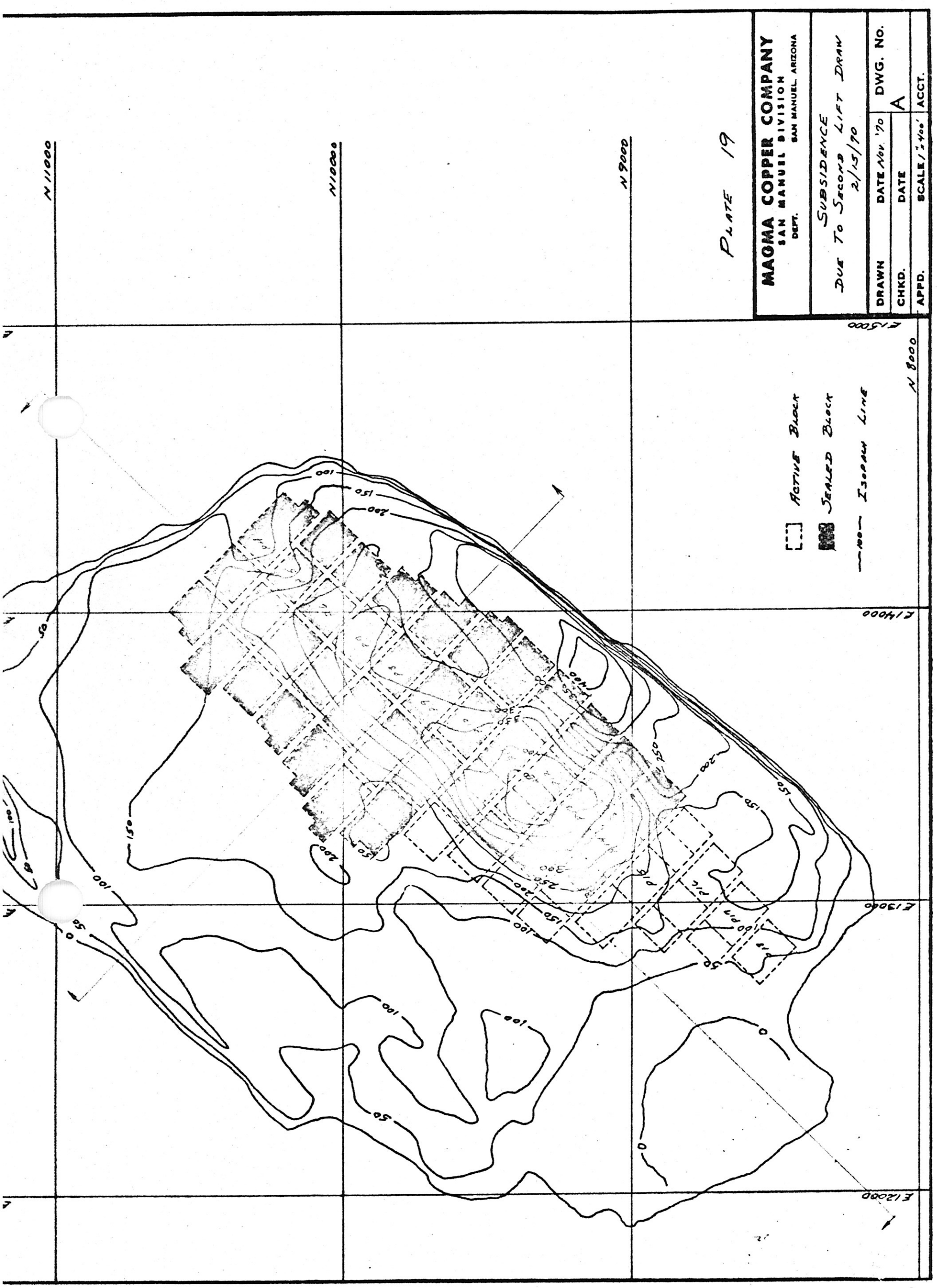


Plate 18

MAGMA COPPER COMPANY
SAN MANUEL DIVISION
DEPT. SAN MANUEL, ARIZONA

Subsidence
Due To Second Lift Draw
1/1/69

DRAWN	DATE Nov. '70	DWG. No.
CHKD.	DATE	A
APPD.	SCALE 1" = 400'	ACCT.



By early 1964 the crack along the trace of the Cholla fault could be followed for 1,700 ft. and a new outer crack, 500 ft. long, formed still 400 ft. further out, indicating the Cholla had not permanently limited the effects of slump. The isopach map of January 1, 1964, Plate 13, shows the very great surface area which had been set in motion by Second Level draw, i.e., 1,500 ft. longitudinally by 1,700 ft. in cross-section. Within this broad parallelogram, a limited funnel centered over Blocks 8-2 and 9-2 showed the maximum change of 100 ft. and a much larger area had slumped 50 ft. These isopach lines were spatially related to the general center of draw from the 2015 Level blocks as shown by the illustration. In April, 1964 two large peripheral masses of rock slumped into the cave area. One, at the northeast end involved a surface area of some 200,000 sq. ft.; the other occurred along the north scarp and involved nearly 100,000 sq. ft. of the surface. This latter slump was the more interesting in that it lay almost 800 ft. laterally outside the undercut perimeter and showed evidence of a very flat effective cave angle. By late 1964, the center of subsidence had shifted gradually to the southeast which accelerated the development of the new south scarp as tabulated above, and by mid-1965, the original scarp due to First Level draw had been absorbed into the expanding perimeter and soon disappeared.

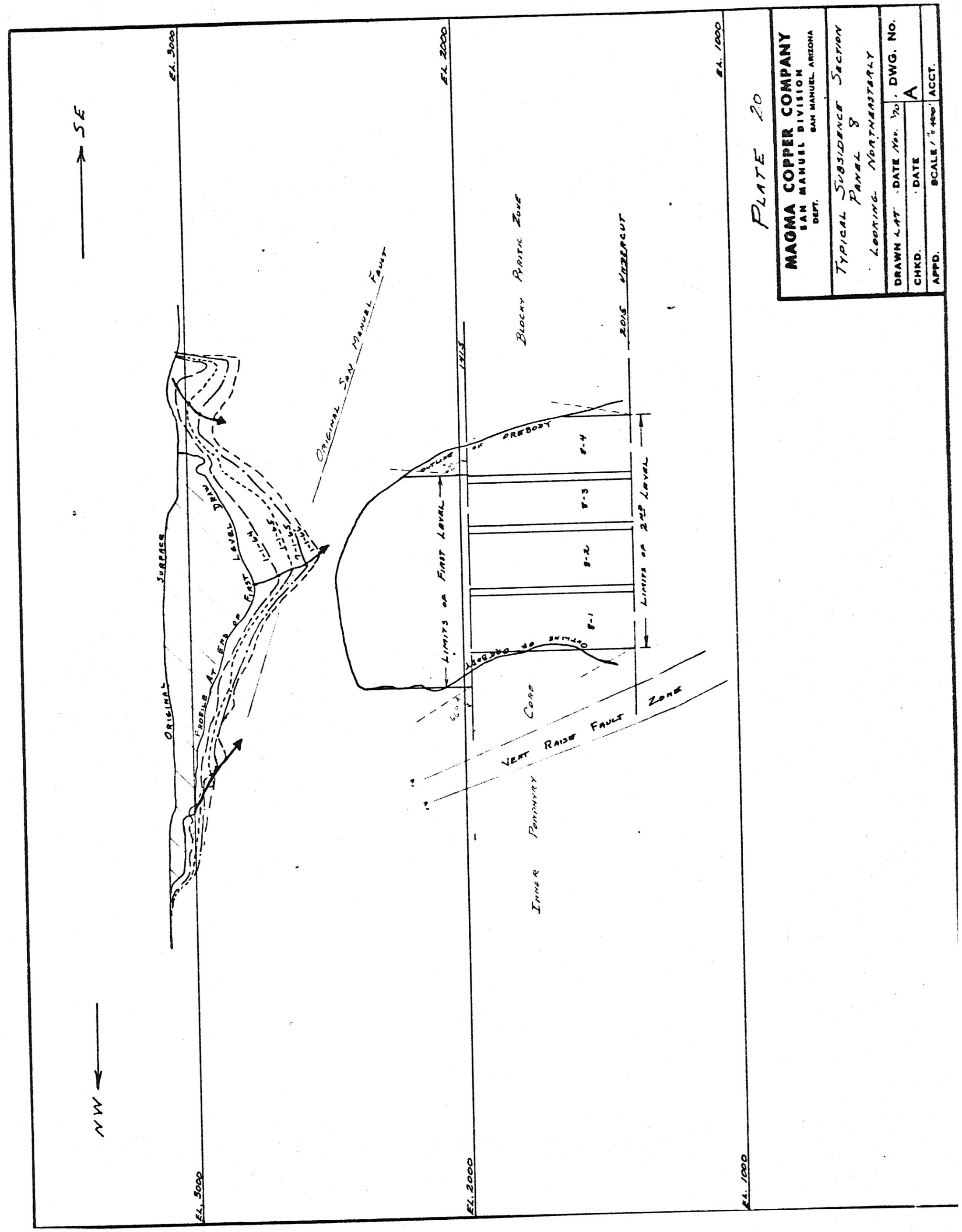
Plate 14, the isopach map for January 1, 1965, shows some interesting features. It retains the essential shape of the preceding year, but shows a substantial area over Panels 7 and 8 which had subsided 150 ft.; the 100-ft. subsidence line was greatly expanded and a truly large area asymetrically placed with respect to the undercut perimeter had dropped in excess of 50 ft. The extreme edge of the 50-ft. isopach line is 800 ft. outside the limits of undercutting. The westerly limit of detectable subsidence at this time was along the projected location of the East fault zone, although most of the fault plane was long since destroyed by First Level draw.

By July 1, 1965, 200 ft. of subsidence over Panels 7, 8, and 9 marked the maximum change and 50-100 ft. of vertical drop had occurred everywhere north of the undercut perimeter as far as the scarp line. This was such an unanticipated situation that a special report on the condition, written in July, 1965, reads, in part:

> ". . . active subsidence within the well established scarp line continues over the south orebody, with only minor activity occurring outside of this periphery. The condition of the cave on July 1, 1965 shows maximum total subsidence of approximately 575 ft. and a

maximum increase over the last six months of 70 ft. . . . The pattern which has developed from Second Level draw is an interesting one which is undergoing a constant broadening and deepening of established trends, an analysis of which may prove helpful in explaining part of the large overdraw which is being obtained from Second Level blocks. For instance, note the very considerable subsidence which has occurred to date several hundred feet northerly beyond the extreme limits of undercutting. Along an arc 750-800' outside the undercut limits, subsidence due to Second Level draw has already amounted to 50 feet, and the entire block of ground north of 4-7, 7-1, and 9-1 (an area 800 x 1200 ft.) has subsided an average of 75 ft. since the inception of Second Level draw, with a horizontal component of movement commensurate with the vertical change. Along the south side, the movement has been a combination of slump and rotation with relatively little lateral component. . . . It may be seen from these movements that the cave area is now acting as a gigantic asymmetric funnel in which draw from Second Level blocks has caused rapid downward movement of rock in the throat of a wide funnel at a point somewhat to the south of center (of the undercut perimeter). But the draw has also produced a considerable lateral component of movement in the mass of broken ground to the north allowing slump to occur far outside the undercut perimeter as ore withdrawn from the active blocks is replenished by new material moving in from areas which originally lay outside the undercut limits. . . . The volume of rock which is now in motion above the 2015 undercut level is so large and has such a significant lateral component of movement that even interior blocks should benefit greatly from replenishment. . . . To date the northerly limit of subsidence due to Second Level draw has been controlled by the position of the Vent Raise Fault zone; similarly the western edge has been limited by the East fault zone." . . . (Note Plate 15.)

Plate 15 is the isopach map of January 1, 1966 and Plate 20 (next page) a cross-section through Panel 8 showing rock movements to that date. Maximum total subsidence had reached 590 ft. (over the south half of 8-2) and the greatest change in the preceding six months had been 70 ft. over Panel 8, declining to 50 ft. over adjacent panels. Maximum subsidence due to second level draw had reached 300 ft. (also over the south 1/2 of 8-2) and



the parallelogram shape of actively moving ground was retained.

A major analysis made at the time concluded:

". . . the maps and sections illustrate a number of interesting features of the cave. Plate 20 shows that the funneling action continues, with a 40-ft. lateral shift of the throat to the south and a steepening of the direction of movement inside the south scarp to nearly vertical. . . . However, the directional arrow illustrating movement on the north side continues on in the same 45° plane which it has held for three years. Hence it may be assumed that on this side the lateral component of movement from north to south is as great as the vertical component (subsidence) and consequently the process of lateral replenishment of active blocks by material moving in from the north is still an important factor in the draw and ultimate tonnage extraction. Because of this "glacial" action within the cave area, it becomes impossible to predict accurately the ultimate extraction that will be obtained from active blocks . . . The scope of this lateral motion may perhaps best be appreciated by an examination of the isopach map, Plate 15. The 100-ft. line is a good example of how far-reaching the action is. This contour spans five panel widths and juts out 600 ft. beyond the undercut perimeter. The raw data from which this isopach map was prepared shows that subsidence along this arc is nearly as great as that directly over Blocks 7-1 and 9-1. The 50-ft. line is equally impressive, spanning nine panel widths and jutting north an average of 800 ft. beyond the undercut periphery. . . . Draw from recently undercut blocks in Panels 11 through 14 has not yet proceeded far enough to modify the fault-controlled parallelogram shape which was developed by older blocks to the east." . . .

As a result of this interior cave action, a true vertical escarpment had formed on the northern side by early 1966, replacing the general terracing which had previously existed in this area.

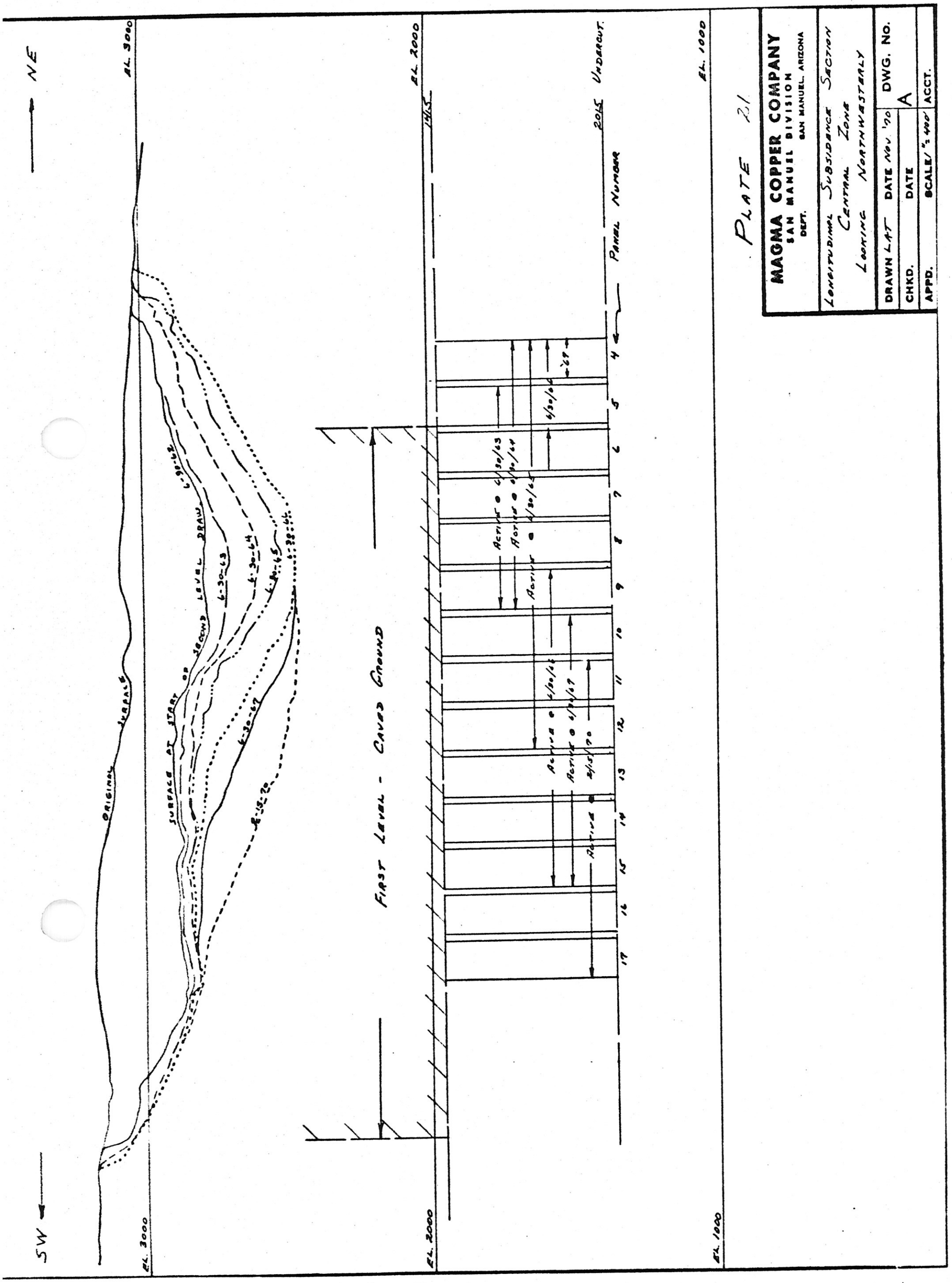
By July 1, 1966, the pattern of subsidence was beginning to change, reflecting the westerly progress of draw underground into the "wedge" system, and substantial sealing out of the older "checkerboard" blocks. The isopach map, of that date, shows that subsidence due to second level draw remained at 300 ft. (though the area increased) and the topographic map of the date

showed that maximum total subsidence also remained unchanged at 590 ft.

Other features illustrated by analysis of this flight included: 1) the surface over the wedge blocks [Panels 11 to 15 inclusive] was the most actively subsiding area during the 6-month flight interval, with Panels 10, 11, and 12 showing increments of 100 vertical ft. tapering to 50 ft. over Panels 13 and 14 and to 25 ft. or less over Panels 15 and 16. 2) Surface activity over the older checkerboard panels (4 through 10) had slowed, particularly over the sealed interior of Panels 7, 8, and 9. The deep, funnel-like center of subsidence over Block 8-2 did not change at all during the 6-month interval, but did become enlarged in diameter due to continued subsidence around its center.

By January 1, 1967 an increasingly greater proportion of daily production was coming from the wedge blocks and most of the original checkerboard area had been drawn to completion. These changes in draw were, as usual, reflected in proportional changes in surface subsidence. The greatest changes in the 6-month interval preceding January 1, 1967 occurred as: 1) a vertical subsidence directly over the wedge blocks which reached as much as 70 ft. over Panels 11 and 12 and tapered to no change over Panel 19; 2) during this period, slight renewed activity <u>outside</u> the western scarp line was noted for the first time since the final draw at the west end of the First Level. This action was primarily due to rotation and its subsequent enlarging of old cracks in the conglomerate; 3) vertical subsidence and rotation south of the wedge blocks caused considerable extension to the southwest of the high vertical scarp along the south side which by this date had reached a maximum height of 300 ft. and a general average height of 200 ft. over a length of 1,600 ft., a dramatic proof of the ability of the Gila conglomerate to stand in sheer vertical walls. Lateral flow from the north had slowed appreciably, but had not stopped completely north of the checkerboard area as these blocks were drawn to completion. Only minor changes in the 50- and 100-ft. isopach lines (Plate 16) occurred in this interval but an expansion was still detectable in the 150-ft. line. The overall changes, north of the checkerboard, were small however, compared to previous times, and neither had any substantial lateral flow developed north of the wedge blocks. Maximum subsidence due to Second Level draw remained at 300 ft., but the area within this line continued to expand.

Results of the July 1, 1967 flight over the cave area are illustrated with two plates which show conditions existing at that time. Plate 21 is a vertical longitudinal section through the central part of the orebody showing the surface profile at



the mid-year flight for each year since the 2nd Level went into production. The immediately obvious fact shown here is that nearly all subsidence between June 30, 1966 and June 30, 1967 occurred over the wedge blocks (including Panel 10), and that virtually no change in the surface profile occurred east of Panel 10. Thus this profile reflects very accurately the fact that most of the production came from Panels 10-15 and that Panels 4-9 were largely sealed.

Plate 17, the isopach map for July 1, 1967 shows several interesting features when compared with its counterpart of January 1, 1967. Both maps are almost identical from Panel 9 to the east and very different from Panel 10 to the west, illustrating again the close relation between production and subsidence. Another illustration of this relationship is seen by the appearance over Panels 11 and 12 of a 350-ft. isopach line, the maximum isopach which had occurred to date. In all previous analyses, the maximum isopach line was essentially coincident with the topographic low in the funnel of the cave over Panels 7, 8, and 9. Now, heavy draw from Panels 11 and 12 had caused a very substantial shift of the maximum isopach westerly, so that it no longer bore a rigorous spatial relationship to the topographic low.

Plate 17 and the individual cross-sections from which it was constructed also showed that in the 6-month interval ending July 1, 1967, the greatest changes within the cave area occurred as: 1) vertical subsidence directly over the wedge blocks of as much as 110 ft. over Panel 12 which declined westerly to about 30 ft. over Panel 15. A small subsidence of 10 ft. persisted even as far west as Panel 18; 2) a combination of slump and rotation away from the south scarp in the area between Panels 10 and 15. The scarp line proper showed significant development in this area south of the wedge blocks and in some cases, the vertical component of change was as great or greater than in areas directly over the wedge. Lateral flow from the north had not yet occurred extensively north of the wedge blocks, but rather, as shown by Plate 17, the isopach pattern had developed into an elongated trough over the wedge blocks rather than into the very broad, flat areas that developed north of the checkerboard area. This presumably reflected the fact that the wedge, being a smaller undercut perimeter than the checkerboard, did not set such a large volume of ground in motion.

A strike lasting from July 1967 to mid-March, 1968 shut down production from the mine and during this interval the cave area remained dormant, as is the normal case. With the resumption of production in March, 1968 cave action was again renewed and the first surface manifestations were visible by April, 1968, Subsidence of the interior over the wedge blocks and the continued development of the high south scarp marked the action at this point. The mid-year analysis of the flight maps which spanned four months of production and eight months of strike time showed that for the interval, maximum subsidence took place in a triangular shaped surface area (reflecting the underlying wedge) and that surface activity over inactive or newly developed areas was limited. The center of Panels 12, 13, and 14 increased in depth by 40-50 ft. and surrounding this a zone of 30 ft. increase showed over Panels 11 and 15 and the north end of Panel 14. Detectable changes tapered out in all directions from this area, reaching west to about Panel 17; north to 500 ft. outside the undercut perimeter, and east to Panel 10. The heavy draw from the wedge blocks over the preceding two years had changed the topographic low from an inverted cone to an elongated trough approximately parallel to the long dimension and lying over the center of the South Orebody. During the same time interval, very few changes were noted outside the main scarp line on the south or west sides.

The flight effective January 1, 1969 produced the isopach map shown in Plate 18 and results of this were compared to the map of July 1, 1967, an 18-month span which included ten months of production and eight months of shutdown time. In this interval a substantial part of mine production had been transferred to the 1715 North Orebody so activity over the South Limb reflected the lesser rate of production there. At both dates, Panel 15 represented the southwestern limit of undercutting, as the mining pattern was being shifted from the wedge to the present panel retreat system. Active subsidence during the 18-month period occurred everywhere west of Panel 9; both directly over active blocks and outside the undercut perimeter, particularly to the south and west as far as the existing scarp line. Maximum change occurred over the central and south parts of Panels 12, 13, and 14 where 70-75 ft. of additional subsidence was realized. Southwesterly from Panel 14 the increase tapered to 40 ft. over Panels 15 and 16; 30 ft. over Panel 17; 20 ft. over Panel 18 and negligible change beyond. Northeasterly from the center, the increase amounted to 45 ft. over Panel 11; 20 ft. over Panel 10, and negligible increase east of Panel 10.

Plate 18 shows the first appearance of the 400-ft. isopach line which is located over Panels 11 and 12 in the same position that the 350-ft. line had appeared at July 1, 1967, reflecting the continuation of established subsidence trends. Slow but persistent sag amounting to some 20 ft. continued to show as much as 700 ft. north of the undercut perimeter and spanning the north edge between Panels 12 to 16 inclusive. The action here was similar to, but not yet as extensive as that which had occurred earlier north of the checkerboard blocks.

In March, 1969 an undercut was taken in Panel 16 Lines (18-23) which marked the first advance southwesterly of the undercut perimeter in almost three years. It also was the first time that a lead undercut was taken on the south side of the panel and hence it marked the real beginning of the panel retreat system. By June, 1969 draw from this new undercut had caused renewed rotational activity outside the southwest escarpment.

The mid-1969 flight data showed routine changes over active blocks but also revealed one unusual item; i.e., the presence of a small cone some 90 ft. in diameter and 30 ft. deep at the apex, which was positioned vertically over Block 10-1 (1-4). This block had a cluster of draw raises which reached a tonnage extraction equivalent to a vertical column 1,200 ft. high from which draw proceeded at a rate of 43 inches per day during the final stages. The result was the formation of a small independent pipe which holed through to the surface within the main cave area; the only example to date of a pipe caused by Second Level draw forming within the main scarp line.

The remainder of 1969 and early 1970 produced the usual routine slumping of a large interior area over active blocks and more widespread rotation along the southwest scarp as the 2015 undercutting sequence approached closer to it. The lead block at the south end of Panel 17 was undercut during September, 1969 and in Panel 18 during February, 1970. This latter block advanced the undercut perimeter to within 150 ft. horizontally of the position of the long-established southwestern scarp line, and it still marks the leading edge of the undercutting at this writing.

The isopach map of February 15, 1970 is illustrated by Plate 19. Compared with the map of January 1, 1969 (a 13½-month interval), it shows the maximum increase in subsidence for the interval to be 125 ft. over the center of Panels 13 and 14. Southwesterly from this maximum, changes of 90 ft. occurred over the remainder of Panel 14, 60 ft. over Panels 16 and 17, and 30 ft. over Panel 18. North of the active panels, subsidence of 30 ft. was noted as far as 600 ft. beyond the undercut perimeter, an increase suggesting that lateral changes were becoming increasingly important here as the total area undercut in Panels 11-18 increased.

Continuity of established trends was reinforced again by the appearance of the 450-ft. isopach line over Panel 12 in the same location where the 350 ft. and 400 ft. lines had appeared in earlier times. In comparing the isopach maps with the topographic maps for each flight, it is seen that until January, 1967, the deepest topographic part of the cave approximately coincided with the maximum isopach contour, a reflection of the fact that much of the early production from the Second Level had come from vertically beneath important producing areas of the First Level. As Second Level production advanced westerly down the strike of the South Limb, the relative importance of Second Level subsidence compared to total subsidence increased, which shifted isopach contours westerly faster than topographic contours shifted. As a result, the isopach pattern anticipated the topographic pattern even though the topographic trough was steadily lengthening to reach out over Panel 12.

The undercut area of Panel 17 was extended in February, 1970 to include Lines 14-16 and again in October, 1970 to include Lines 11-13. Coupled with the draw from Panel 18 (16-21) this resulted in widespread peripheral activity around the southwest scarp, as these active blocks are variously only 150 ft. to 600 ft. laterally from the surface position of the escarpment. The first 200-ft. width of the peripheral area is breaking up along a series of sub-parallel tension cracks (partly fault controlled) which allows those masses closest to the cave to slowly rotate inward. This differential rotation produces great open fissures between which masses of conglomerate slump, forming local graben which then further fracture in random directions due to the stress of differential settling. This mechanism is operating at the leading edge of the escarpment at the present time.

NORTH OREBODY SUBSIDENCE DUE TO FIRST LEVEL DRAW

A. The Mining System. A panel and block system similar to that used in the South Limb was effected here. However, because the cross-sectional dimension of the North Limb rarely exceeded 400 ft., no panel was long enough to accommodate more than two blocks. Also, sulfide mineralization along the North Limb was interrupted by a deep-lying tongue of oxidation which was not mined, so two separate production areas resulted which have been termed the West Area and the East Area. These remained separated by a 600-ft. pillar, so two separate cave areas formed at the surface and are likewise referred to as the West Area cave and the East Area cave.

In the West Area, two panels centrally located along strike were chosen for the initial undercuts and from there, additional undercuts were taken as needed by advancing along strike in both directions. The initial location included Block 25-2 which began undercutting on May 13, 1959 and Block 24-1 which followed at June 17, 1959 for a total undercut area of 84,000 sq. ft. These were the only active blocks for nearly a year, and undercutting was resumed May 2, 1960 when Block 25-1 was activated, followed on June 11, 1960 by Block 24-2; on September 14, 1960 by Block 23-1; and on November 30, 1960 by Block 26-2, which added 132,300 sq. ft. of active area for a total of 216,000 sq. ft. undercut.

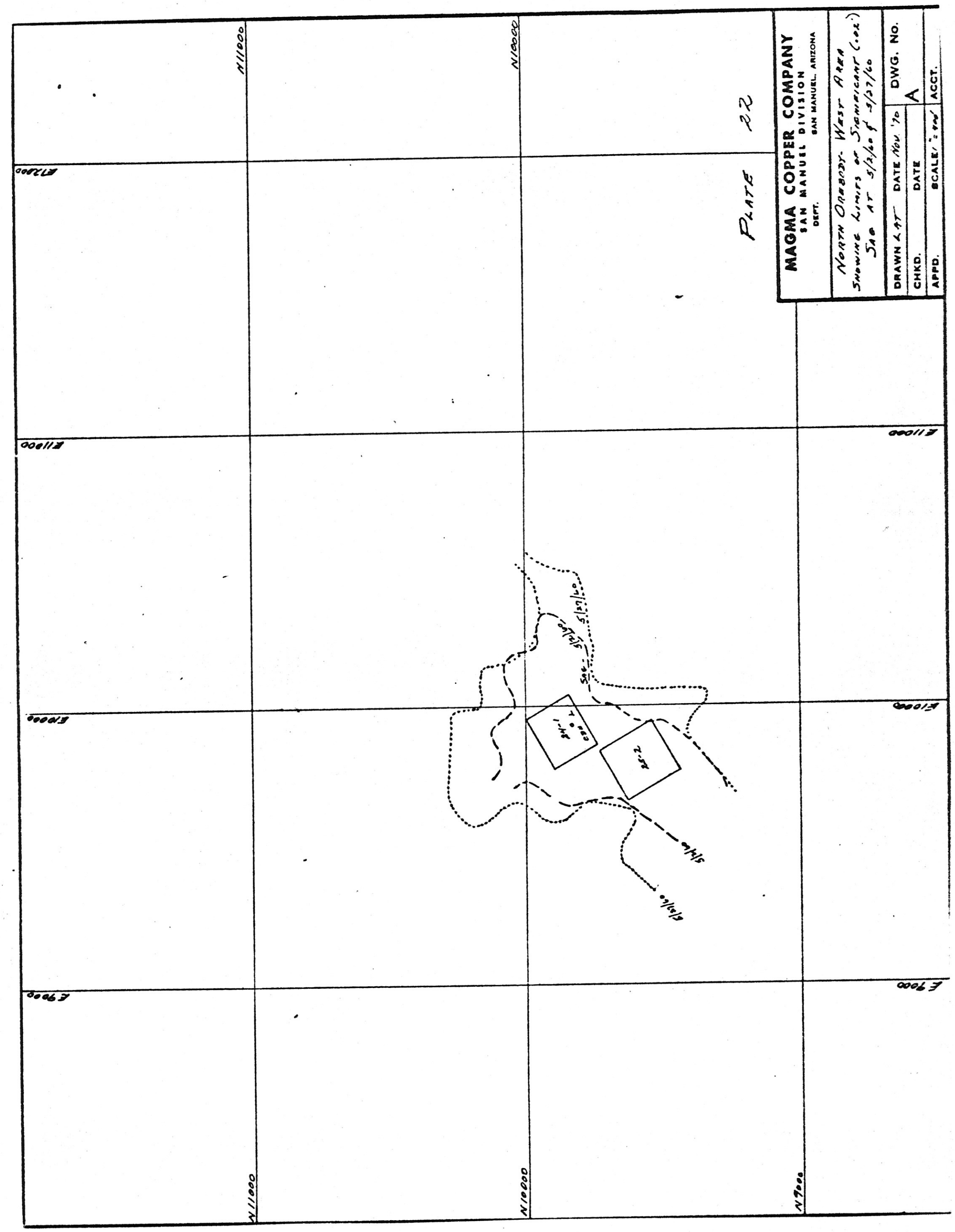
In a similar manner, in the East Area Block 34-2 began undercutting on December 28, 1961 and Block 33-1 began on January 18, 1962. These were shortly followed by 35-1 on April 4, 1962; Block 34-1 on July 16, 1962; and Block 33-2 on July 26, 1962, for a total block undercut area of 73,500 sq. ft. In addition, in the East Area, pillars between blocks were also undercut, giving a contiguous undercut area of 89,400 sq. ft.

The surface over the North Limb was studded with survey pins on a 100-ft. square grid in a manner similar to the South Orebody. Those pins over the West Area, where as much as 800 ft. of conglomerate existed, were surveyed repeatedly by personnel of the United States Bureau of Mines, headed by George H. Johnson. The survey data thus made available to Magma Copper Company was reduced by L. A. Thomas into a series of contour maps showing sag of the conglomerate surface prior to collapse. Some of these are illustrated in this report. Pins over the East Area were largely used for mapping fractures by ground reconnaissance and were not routinely surveyed, since no great thickness of conglomerate existed over those blocks, and no problem of breaking to the surface was thought to exist.

Also, over the West Area, a few churn drill holes in the vicinity of the active undercut area provided another opportunity to plumb through the conglomerate beds for changes in depth to plug or standing water.

B. The West Area - Initial Failure of the Gila Conglomerate. The first indication of change occurring in the rock column above undercut horizon was the loss in July, 1959 of water standing above the plug in Churn Drill Holes "L" and "T" in the vicinity of the active panels. At about the same time, July 23rd, a survey of subsidence pins by Bureau personnel showed a mixed pattern of very small increments of both rise and sag over the active area, indicating minute surface adjustments to a changing stress pattern. If we assume that these adjustments meant that cave action had progressed upward through the igneous column to the plane of the San Manuel fault, then we have a good approximation for its vertical rate of rise through igneous rock. The average igneous column was 600 ft. and production time amounted to ± 60 days, giving an indicated rate of 10 ft./day; a figure identical to that indicated by the initial draw from the South Limb (q.v.). Furthermore, the same data allow an estimate of expansion in the igneous column for the record shows that an average of approximately 52 vertical ft. of the 600-ft. column had been withdrawn, leaving perhaps 548 equivalent ft. remaining. This 548 ft. expanded to 600 ft. of broken rock, an expansion of ± 10%.

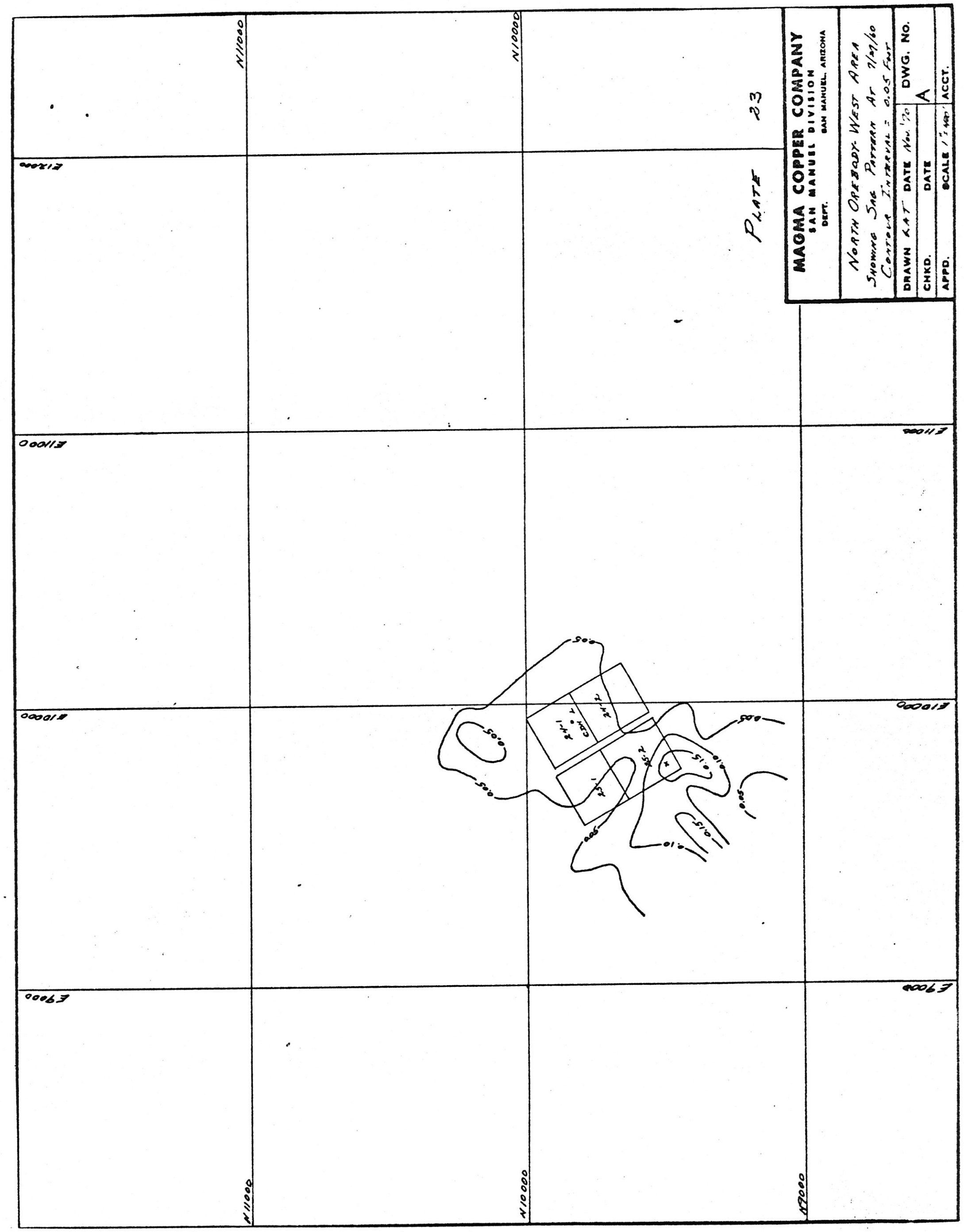
Production was shut down by a strike from August 8 to December 15, 1959 during which time no changes were noted in the surface over the North Orebody. However, by mid-September, 1959, seepage of water around the west side of Block 25-2 showed that some type of cracking had penetrated through the San Manuel fault plane to release water perched in the Gila beds above. Here again, the time lag suggests that the progress of cave was retarded at the plane of the San Manuel fault, even though a void must have been forming beneath the fault plane. Draw resumed in the blocks on December 15, 1959 with no further changes noted until February, 1960 when 300 cfm of warm, moist air from the mine underground began exhausting out the collar of Churn Drill Hole L which lay directly over Block 24-1. This hole had been plumbed regularly and had been plugged just below the San Manuel fault. With the plug fallen away, air exhausted up through the broken mass of igneous rock above undercut and used the churn drill hole as an escape to the surface. (Note Plate 22.) The condition implies that during February, 1960, no significant caving had yet occurred in the Gila beds over Block 24-1 even though average draw there had removed 146 equivalent ft. of igneous rock, or 23.4% of the igneous rock column. At the same time, draw from Block 25-2 amounted to 149 equivalent ft., or 27.0% of the igneous rock column; and total volume of igneous rock withdrawn from beneath the San Manuel fault amounted to 12,415,000 cubic ft.



Using these production figures and assuming that ± 10% is a reasonable expansion factor for the igneous rock, one is driven to the conclusion that a substantial void existed over the active blocks. Since the rock column withdrawn averaged 147 ft. of a 600-ft. column, then an average of about 450 ft. of equivalent rock remained. An expansion of ± 10% would swell this equivalent column by 45-50 ft. to give a broken rock column height of 500 ft., leaving a 100-ft. void between the top of the broken muck pile and the bottom of the San Manuel fault plane. Since the active blocks were located on a northeast trending diagonal, it is likely that the void now formed had an elliptical or hourglass shape elongated in the same direction. Note that this elongation is perpendicular to the attidue of Gila bedding. It is likely that from a rock mechanics standpoint, the two initial blocks in the North Orebody should have been taken on the opposite (northwest) diagonal so that any void forming below the fault plane would be elongated parallel to the direction of Gila bedding, allowing maximum chance for spall from the lower Gila column.

By the end of March, 1960, both Magma and Bureau of Mines engineers were recording a consistent, though very small, sag pattern at the surface. This pattern developed through April and May and reference to Plate 22 will show the reader the condition of sag as of May 2, 1960. Note that the pattern of sag reflects both the diagonal northeast position of the underlying blocks and the fact that the conglomerate cap thins easterly from Panel 24, where the sag bulges to a northwest trend. At the time of this survey, production from the active blocks had reached a volume of 18,569,000 cubic ft. Block 24-1 showed an average draw of 260 ft. which represents 38.0% of the igneous column and 19.5% of the total rock column. Block 25-1 was undercutting at this point and Block 25-2 had withdrawn an average of 180 ft., which represents 35.0% of the igneous column and 13.9% of the total rock column. Plate 22 also shows on the same map the sag outline of May 27, 1960, from which the growth of consistent sag in the preceding 25 days can be noted.

By July 27, 1960, the magnitude of change on pin elevations had reached significant enough values that it was possible to contour the pattern of sag on an interval of 0.05-ft. and Plate 23 shows the condition of that date. The maximum sag, over the southwest corner of Block 25-2, had reached a value of 0.203-ft. though no cracks had yet appeared at the surface, and average sag values were only on the order of 0.080-ft., or approximately one inch. Plumbing of Churn Drill Hole L at the end of July revealed that the hole was open only to 186 ft. below the collar, and was still blowing air. Presumably the Gila was still intact from the surface down to this point which might be considered the top of



caved ground. If this assumption be true, it shows that the impulse of caving action had been rising up through the Gila beds for the preceding year at an average rate of 1.4 ft./day, although the actual spalling action may be presumed to have been intermittent. By July 30, 1960 total draw from blocks had reached a volume of 26,373,375 cu. ft. Individual blocks showed:

Block	Ave. Draw, Ft.	% Igneous Col. Withdrawn	% Total Col. Withdrawn
24-1	331	48.2	24.8
24-2	47	8.0	3.6
25-1	89	14.5	6.7
25-2	186	36.2	14.3

On October 10, 1960, the first hairline surface cracks over the North Orebody appeared. At this point, draw had reached a volume of 36,638,000 cu. ft., representing 2,931,000 tons. By individual blocks draw had reached:

Block	Ave. Draw, Ft.	% Igneous Col. Withdrawn	% Total Col. Withdrawn
23-1	37	5.2	2.8
24-1	380	55.3	28.4
24-2	140	24.0	10.6
25-1	184	30.2	13.9
25-2	186	36.2	14.3 Sealed

The plan map showing the location of the cracks, which were separations along bedding planes in the conglomerate, is illustrated in Plate 24, which also shows the sag pattern at October 26, 1960. The maximum sag known at the time of discovery of the first cracks was 0.230 ft., or approximately 2-5/8 inches, although average sag over the undercut area was only on the order of 0.120 ft.

The first breakthrough to the surface over the North Orebody West Area occurred on December 14, 1960, when a nearly rectangular hole 100 ft. by 70 ft. came through over the southeast corner of Block 23-1, which location happened to be the topographic low of the area. See Plate 25, which also shows the sag pattern at December 1, 1960. It was possible to stand on a ridge across from the breakthrough and look directly into it. As a result of so doing, the following comments were recorded at the time: "The conglomerate is arched over at the surface, leaving a void underneath, and it appears that the main pipe comes up considerably west of the surface breakthrough, probably as far west as 24-1. It is interesting to note that the area of maximum sag (0.200 ft.) which existed over Block 24-1 seems to duplicate very well what we can now see of this void under the arch. The implication

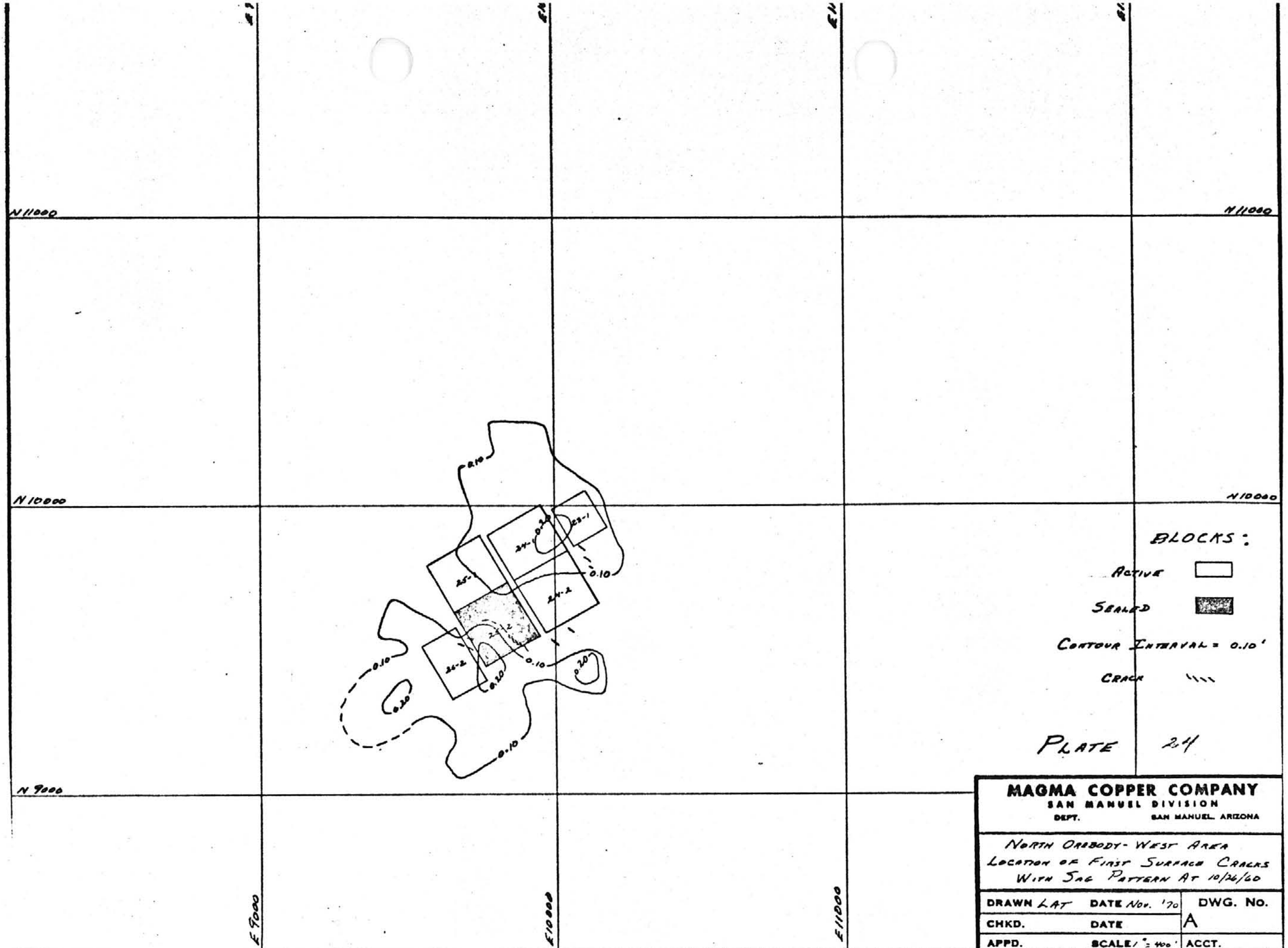


PLATE 24

MAGMA COPPER COMPANY SAN MANUEL DIVISION DEPT. SAN MANUEL, ARIZONA		
NORTH OREBODY - WEST AREA LOCATION OF FIRST SURFACE CRACKS WITH SAG PATTERN AT 10/26/60		
DRAWN LAT	DATE Nov. '70	DWG. No.
CHKD.	DATE	A
APPD.	SCALE 1" = 400'	ACCT.

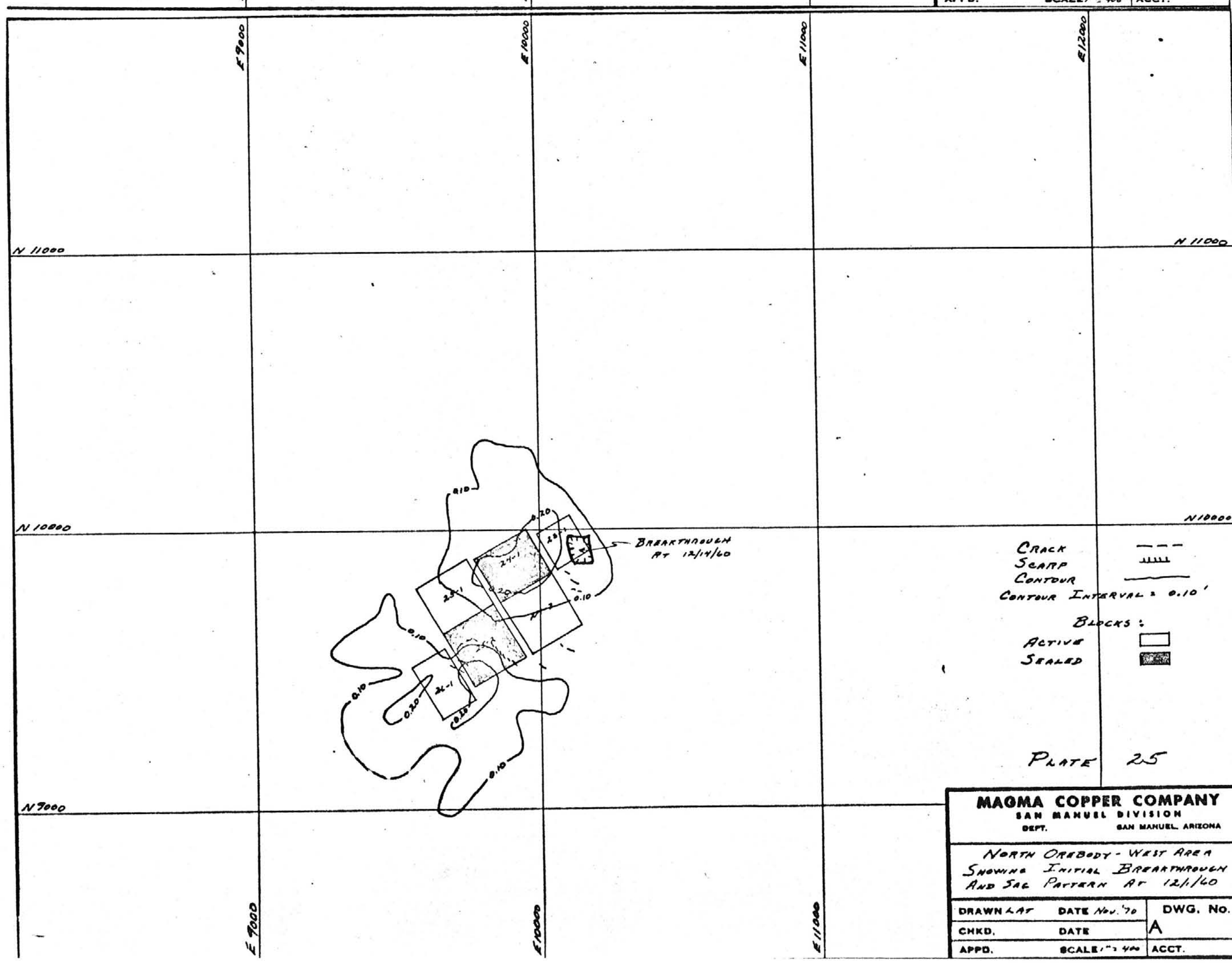


PLATE 25

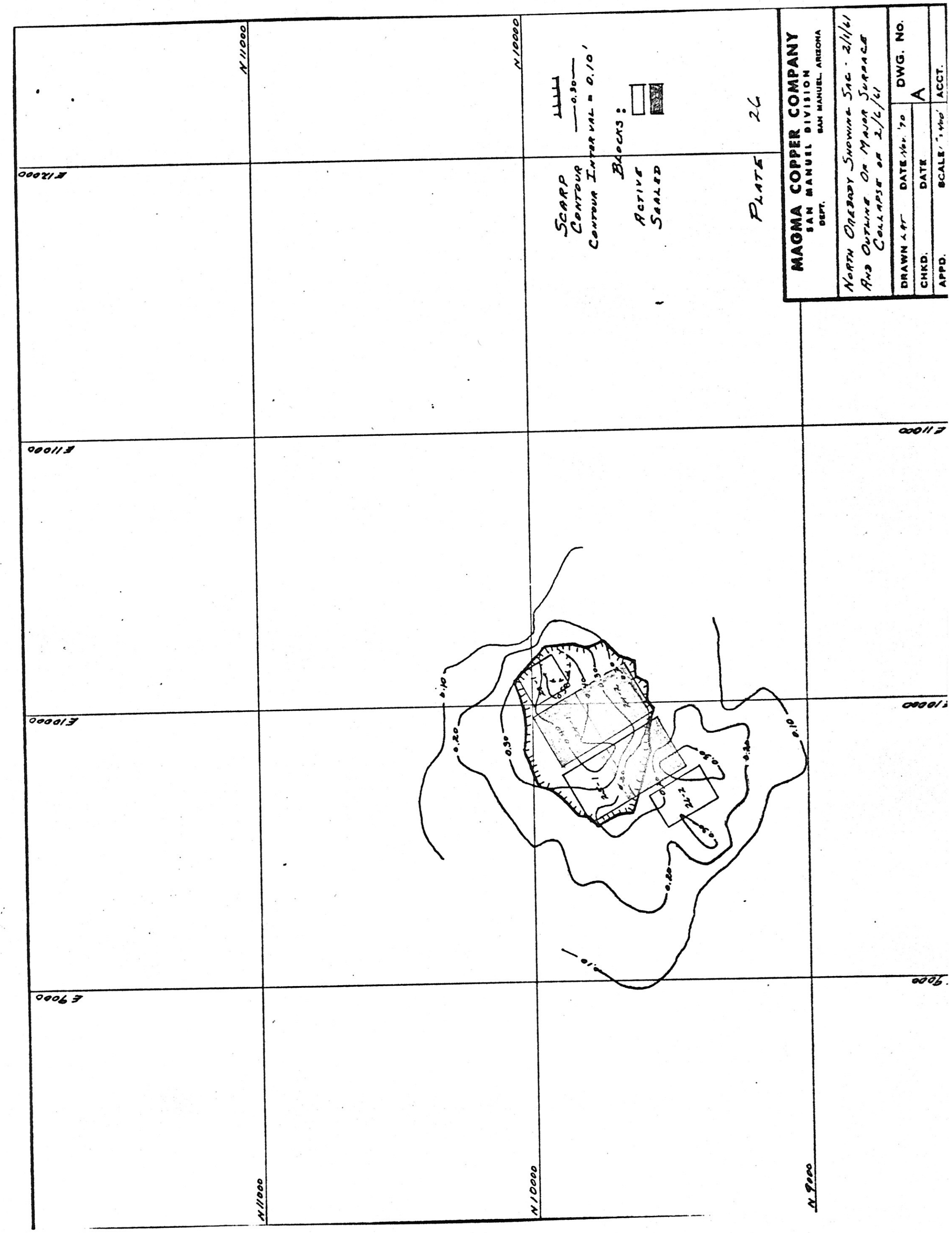
MAGMA COPPER COMPANY SAN MANUEL DIVISION DEPT. SAN MANUEL, ARIZONA		
NORTH OREBODY - WEST AREA SHOWING INITIAL BREAKTHROUGH AND SAG PATTERN AT 12/1/60		
DRAWN LAT	DATE Nov. '70	DWG. No.
CHKD.	DATE	A
APPD.	SCALE 1" = 400	ACCT.

of this is that the fairly extensive sag which exceeded 0.200 ft. was due to a near-surface void whose presence was previously unknown and which further means that the conglomerate had broken and caved to this near surface position just prior to breakthrough. There has been no particular development of cracks around the area beyond those which were forming before the breakthrough."

Another interesting deduction in connection with the rise of caving action through the conglomerate is possible now, for it may be recalled that as described in an earlier paragraph, the depth to cave was known to be about 186 ft. during late July. It took about 130 to 135 additional days for this 186 ft. to cave, indicating an average rise through the conglomerate beds on the order of 1.4 ft./day, the same average rate noted previously.

The small hole nearly doubled in size during January, 1961 as some pieces of conglomerate on the western edge fell away. However, the real collapse of the arch, one of the most spectacular occurrences in the history of subsidence at this property, came in the early morning hours of February 6, 1961, when the span, no longer able to support itself, fell into the void beneath, involving in the collapse a surface area of 233,500 sq. ft. Added to the 15,000 sq. ft. of the initial hole, the cave area suddenly was expanded to a total surface area of 248,500 sq. ft., with maximum dimension E-W of 600 ft. and N-S of 450 ft. The outline of the major breakthrough and its relation to the underlying blocks is illustrated by Plate 26, which also shows the sag pattern of February 1, 1961. It would appear that a surface sag of about 0.300 ft. was necessary to produce the stresses which broke the arch, as this contour closely follows the perimeter of collapsed ground. The shape of collapsed ground also bears a remarkable similarity to the shape of the undercut perimeter of the blocks causing it. Only the southwest half of Block 25-2 is anomalous, and the discrepancy there is probably due to the very poor tonnage extraction (only 186 ft. average draw from the block or 14.3% of the total rock column) which was realized.

The depth of the cave thus formed was estimated from vertical angle transit shots to be on the order of 125 ft. deep, giving a volume of subsidence estimated at 31,000,000 cu. ft., which represents 57% of the total draw volume of 54,600,000 cu. ft. With the collapse having occurred, it was again possible to estimate the average amount of swell in the broken rock. To arrive at this figure, both the area of and the tonnage removed from Block 26-2 have been excluded, as they may have played no important role in the breakthrough. Then, conceiving of the cave area



as an upside-down truncated cone with its base at undercut and its top at average ground elevation, we have the following data with which to calculate:

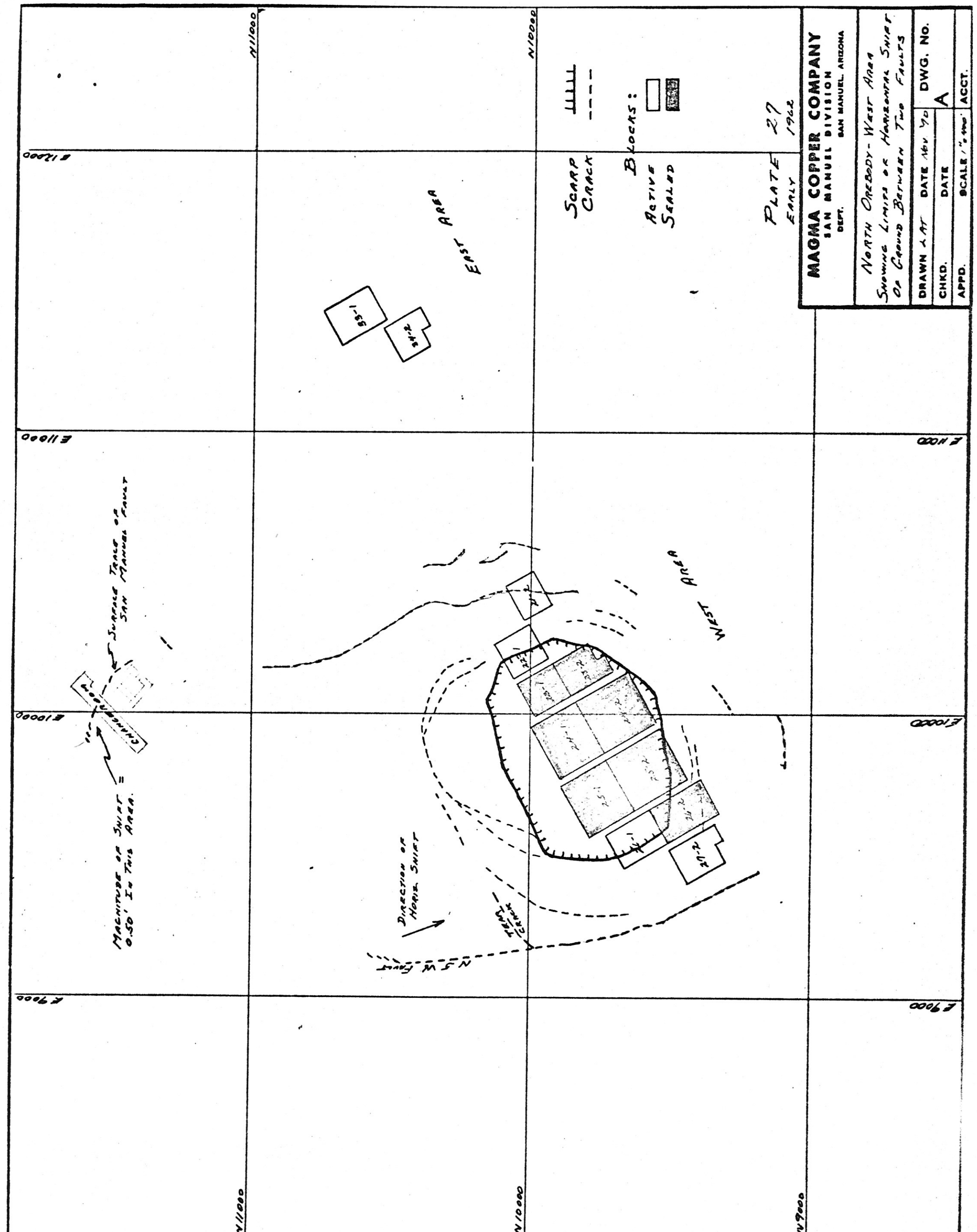
Total undercut area (excluding 26-2)	186,900 sq. ft.
Total surface breakthrough area	248,500 sq. ft.
Average of these two areas	217,700 sq. ft.
Original distance - undercut to surface	1,330 ft.
Present distance - undercut to surface	1,205 ft.

To compute the original volume:

Vol. occupied by broken rock = 217,700 x 1,205 =	262,000,000 cu. ft.
Vol. occupied by present hole = 248,500 x 125 =	31,000,000 cu. ft.
Original vol. of rock involved =	293,000,000 cu. ft.
Vol. of rock withdrawn (excl. 26-2) =	52,000,000 cu. ft.
Volume prior to collapse =	241,000,000 cu. ft.
Volume of rock after collapse =	262,000,000 cu. ft.
Amount of expansion between undercut and surface =	21,000,000 cu. ft.

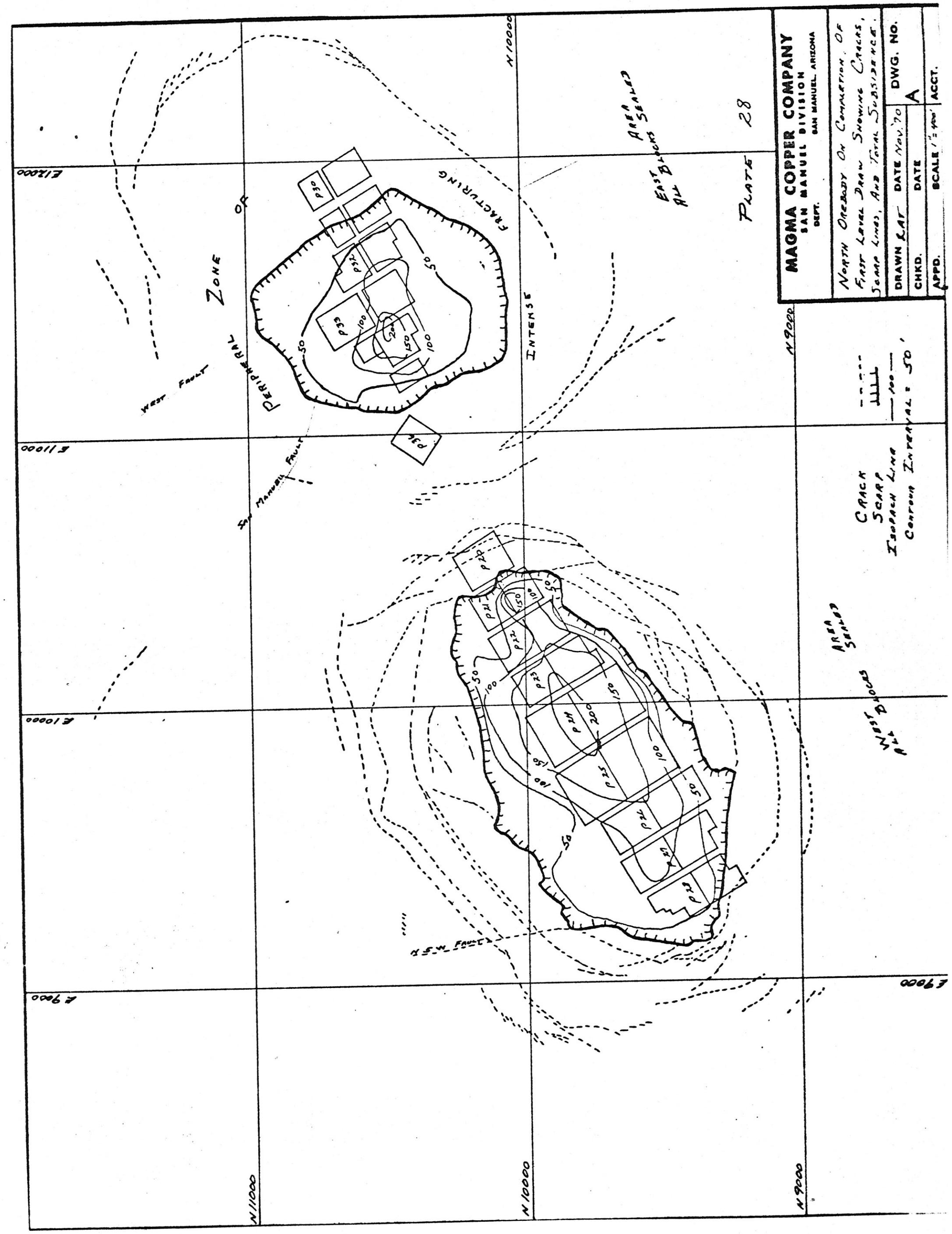
$$\% \text{ Expansion} = \frac{21,000,000}{241,000,000} = \text{approximately } 9\%.$$

C. The Southerly Shifting of a Large Mass of Gila Conglomerate. Plate 27 shows the condition of cracks and scarps early in 1962 when significant development of cracking had occurred along a N5W fault in the Gila conglomerate. In detail, this linear feature developed prominence because the mass of ground to the east of it shifted southerly during October and November of 1961. The result was a whole series of small en echelon tension cracks striking N 20°-25° E, but whose average trend lay N5W along the fault plane. The relative shift could be measured in November as 0.1 ft. horizontally in a direction S 20° E, with the horizontal magnitude gradually increasing as the lip of the scarp was approached. Tear cracks approximately perpendicular to the direction of horizontal shift developed too and are illustrated on the Plate. It was not possible in November to detect the full size of the area which was involved in this horizontal shift, but the limits became well-defined in March of 1962 when a whole series of tension cracks opened up 1,200 ft. further east along the surface trace of the San Manuel fault. It became immediately apparent that the horizontal shift was occurring because the whole wedge-shaped upper plate above the San Manuel fault outcrop and east of the N5W structure was being pulled SSE toward the center of subsidence. The resulting tensile stress which arose in the upper plate caused it to slowly slide down-dip along the fault plane. Being thus defined by a flat-lying fault, the motion per-



sisted far outside normal crack angle limits, and penetrated into the mine yard where it caused some damage to foundations or pipelines which happened to straddle the fault plane. The outer edge of detectable movement occurred about 1,200 ft. horizontally from the edge of the escarpment, or 1,500 ft. horizontally outside the undercut perimeter. The explanation of the motion is interesting and it occurred because the tensile strength of the Gila beds above the fault plane was great enough to overcome the frictional resistance along the San Manuel fault plane. When horizontal (i.e., rotational) stress was applied to the upper plate, failure occurred first along the fault plane, setting the whole upper plate in motion. Note that for a block of ground this size (1,200 ft. by 1,200 ft.) the area of the upper plate which was set in motion and, hence, in frictional constraint against the lower plate, was on the order of 1,500,000 sq. ft. The frictional constraint eventually had considerable effect, counteracting the tensile strength sufficiently to cause a series of steep peripheral tension cracks to open in the upper plate in a more normal manner, and once these cracks formed, they absorbed most of the rotational moment by opening up into wide fissures. However, the upper plate shift never entirely stopped until First Level draw was complete and all forces reached equilibrium. The ultimate magnitude of the shift decreased with increasing distance from the edge of the scarp, but by January, 1964 in the vicinity of the mine changeroom and service tracks where horizontal offsets could be accurately measured, it totaled 0.50 ft. Plate 27 shows the location of the various features discussed above.

D. Continued Development of the North Orebody Cave - West Area. In addition to, and occurring simultaneously with the shift described above, a more "normal" growth of the cave area took place. At any given point in time, the first 150-200 ft. peripheral to the existing scarp line was laced with large tension cracks, often fault-controlled. As undercut activity expanded, both northeast and southwest along the strike of the North Limb, the existing scarp periodically eroded outward in response, as cracks on each leading edge opened up and monoliths of conglomerate slumped into the cave. Plate 28 shows the final location of the scarp line due to First Level draw, the extent of mapped peripheral cracking at that time, and the magnitude of total subsidence due to First Level draw. The isopach contours over Panel 21 reflect the presence of an independent breakthrough which holed the surface in October, 1963, probably the result of withdrawal of igneous rock from both Panels 20 and 21. The location of the hole at the edge of the escarpment and the lack of vertical subsidence over Panel 20 suggest that draw from both of these panels actually coalesced and rose over Panel 21. Their combined withdrawal was therefore adequate to cause the independent piping. The tonnage extraction

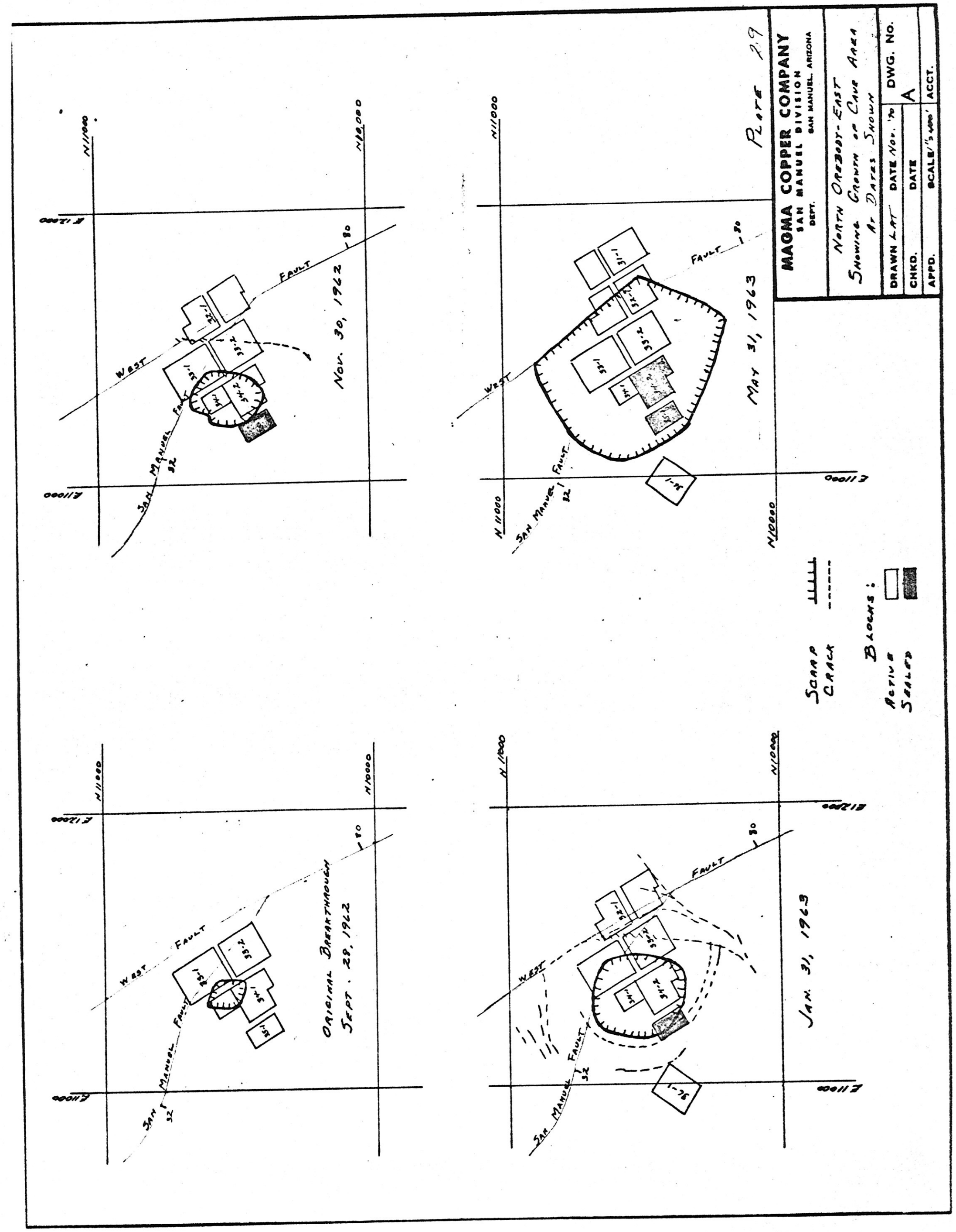


pattern also corroborates this view. Panel 21, drawn first and lying beneath ground already actively subsiding, produced 106% tonnage extraction. Panel 20, drawn after Panel 21 was sealed, and lying outside the limits of cave previously established, was not large enough to initiate substantial new breaking in the virgin ground above it and so it broke prematurely toward Panel 21, and drew down more rock from the slot over Panel 21. This meant premature dilution and the panel reached only 75% tonnage extraction. This example illustrates again the difficulty of getting good caving action started in quartz monzonite rock when the undercut sequence reaches these hard, blocky areas last.

Upon completion of production from the West Area, the volume of subsidence amounted to approximately 99,000,000 cu. ft. against a volume of igneous rock withdrawn of 147,000,000 cu. ft. Thus, for this area, average subsidence reached 67% of the volume of rock withdrawn.

E. The East Area - Initial Surface Breakthrough. The first noticeable surface activity over the East Area came in August, 1962 when surveyors detected a slight sag over the north end of Panel 35. No record exists on the development of this sag, for the breakthrough followed immediately. On September 28, 1962, an elliptical hole about 100 ft. by 150 ft. broke the surface over the eastern edge of Panel 34. The breakthrough was about 75 ft. southwest of the surface trace of the San Manuel fault, hence only a thin shell of conglomerate--perhaps 25 to 50 ft. thick--had to be penetrated by caving action. The elapsed time from the beginning of production to the detection of sag amounted to ± 150 days, giving an indicated rate of rise of caving action through the igneous column on the order of 8.5 ft./day. The average calculated height of 800,000 tons of igneous rock withdrawn at the September 28th breakthrough was 108 ft, or 8.6% of the total rock column, based on a total undercut area of 89,400 sq. ft. On the smaller actual undercut area (not including pillars), the average height was 132 ft., or 10.3% of the total rock column. In either case, breakthrough occurred very substantially earlier than in other areas, and this was probably because essentially the entire rock column was composed of igneous host rock. (For illustrations of the growth of the pipe, see Plate 29.)

Once formed to the surface, the pipe began rapidly enlarging in response to continued withdrawal of igneous rock at undercut level. By November, 1962 it expanded to 2-1/2 times its original size and peripheral cracks began to appear. By January, 1963 it had doubled again to nearly 300′ diameter and peripheral cracking was being detected as far as 700 ft. from the edge of the hole. By March, 1963 an intensely fractured zone from 100-300 ft. wide



surrounded the pipe, providing a good illustration of the contrast in mode of failure between the closely fractured igneous host rock and the massive bedded Gila conglomerate. The intensely fractured zone itself soon began to actively subside and the perimeter of the East Area scarp grew, particularly on the north and east sides. The mid-year flight of 1963 revealed the deepest part of the cave over Panel 34 to be 160 ft. deep and the dimensions within the escarpment to be nearly 800 ft. N-S and in excess of 500 ft. E-W. All East Area blocks were undercut by July of 1963, so the undercut perimeter had reached its full dimensions by then. Additional erosion along the eastern and southeast edges of the scarp in response to the remaining draw in Panels 30 and 31 brought the outline of the surface escarpment to its final dimensions of 900 ft. by 800 ft. by January 1, 1964. Further draw produced additional settling of the mass within the escarpment (which reached a maximum of 210 ft.) and many additional peripheral tension cracks on all sides, but no further growth of the diameter of the pipe. (See Plate 28.) Draw from Block 33-1 continued on a small scale until January 23, 1965 when the East Area was completely depleted as far as First Level production. A small cluster of draw raises in the northeast corner of Block 33-1 produced the largest individual tonnages ever pulled through one raise in the San Manuel mine. Three of these, adjacent to one another, reached average tonnage extractions expressed in equivalent vertical ft. of rock column on the order of 2,850 ft. Since no independent piping resulted from these withdrawals, which were 2.2 times the height of the entire rock column, it can only be assumed that considerable replenishment due to lateral flow was occurring here locally.

Note that Panel 36 (16,800 sq. ft. in area) withdrew an average of 186 ft. of igneous rock from a total igneous column of 1,035 ft., all lying beneath a 225' average conglomerate cap without breaking the surface.

THE SUB-SURFACE PATTERN OF PRESSURE
On Mine Extraction Openings

A. The First Level - South Orebody. All the initial blocks on 1415 Level were supported by timbered sets or relatively light 4-in. rigid steel rings, both of which give only 'point' support where they happen to be blocked against the surrounding ground. Both types proved inadequate to withstand the pressures initiated by draw, and the result was widespread failure of ground support throughout the area enclosed in the undercut perimeter. This led, in 1956, to the decision to line extraction openings with unreinforced concrete and thereby derive the benefits of an "area" support of ground surrounding the drift. No comprehensive installation of instruments to record pressure changes was attempted at the time, so raw data on the magnitude of pressures at the mining levels does not exist. Any attempt to define the pressure problems in quantitative terms must be done largely by inference.

It is well worth stressing at this point that the damaging pressures which rise in an incompetent host rock such as occurs at San Manuel are a direct result of the dynamic condition of undercutting and drawing ore; a static load of broken granitic rock is not sufficient, in itself, to produce crushing pressures at only 1,400 or 2,000 ft. of depth. Ample evidence of the latter fact has been demonstrated twice at San Manuel when lengthy labor disputes have shut down production for four months and eight months respectively. The result of the shutdown on underground drifts was a relaxation of pressure throughout the undercut perimeter, continuing until production resumed. The result in the cave area was a cessation of movement until production resumed; and it was this cessation of cave action between undercut level and the surface which allowed the relaxation of damaging pressures and the attainment of a static stress pattern on mine extraction openings. Upon resumption of production, the static stress pattern is immediately replaced with a dynamic stress pattern as cave action begins anew, and soon damaging pressures are felt within the undercut perimeter.

It is the function of the concrete lining to more nearly counteract these damaging pressures by placing support in the drifts which has greater compressive strength than the strength of either of the closely fractured host rocks. The concrete-lined grizzly level thus becomes a relatively stable horizontal grid immersed in a relatively plastic granitic medium.

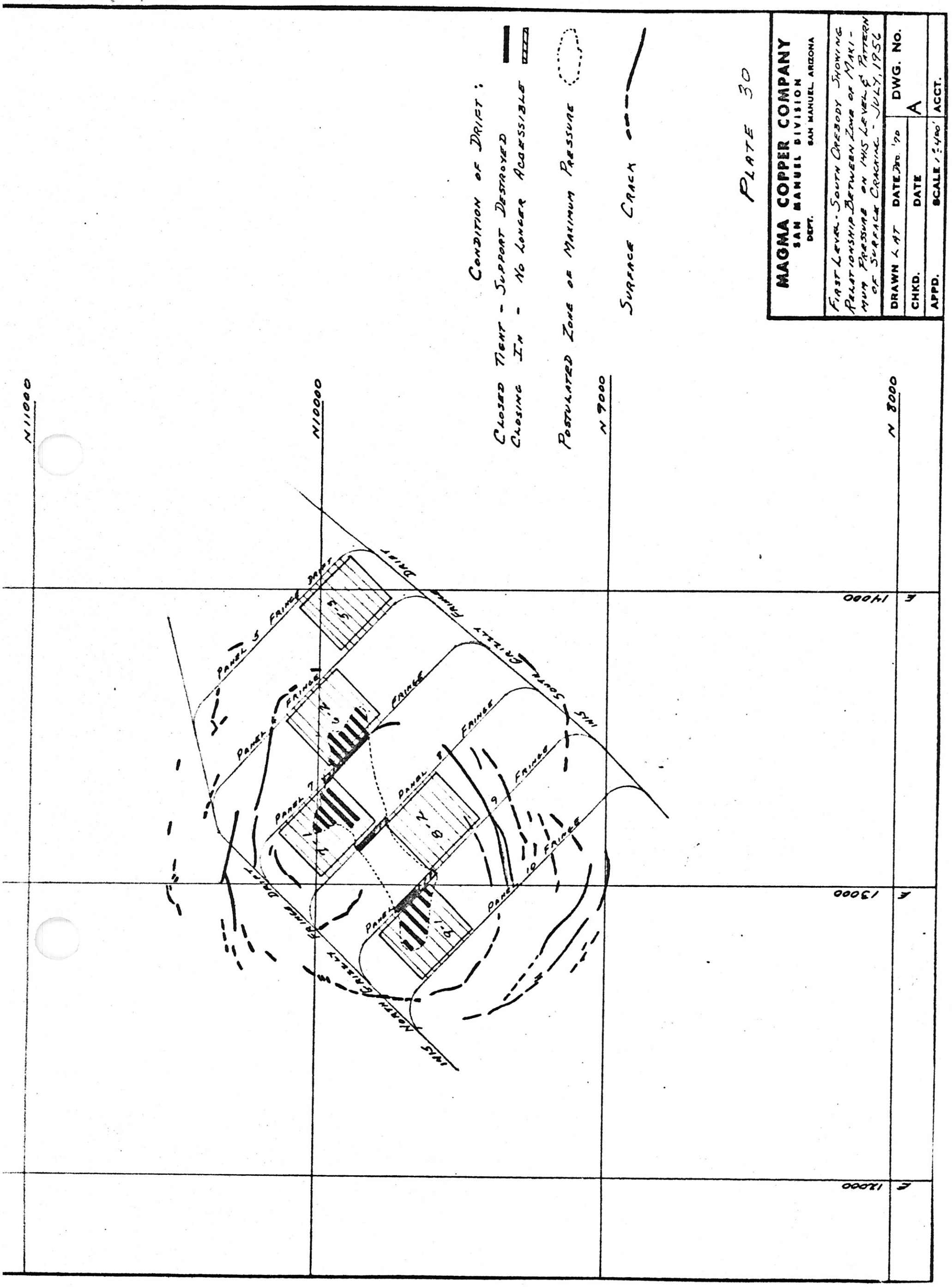
The first severe ground weight experienced in the mine began to affect the central part of Block 7-1 about three months after undercutting when the average draw from the block was about 35 equivalent ft., or only 3% of the total rock column. No cracking had been noted at the surface yet, but at the indicated rate of rise of caving action upward of 10 ft./day, the igneous column should have been broken all the way to the San Manuel fault. This probability, plus the central location within the block of damaging weight, suggests that excessive pressure due to sag of the conglomerate capping was being transmitted downward to the grizzly level at about the center of gravity of the draw through an igneous column which was thoroughly cracked but not yet broken enough to allow a rate of withdrawal which could relieve the pressure.

A similar condition developed in Block 9-1 between three and four months after undercutting when about 90 equivalent ft. of draw, or 8% of the total rock column, had been withdrawn, and the general central portion of the block began to fail under pressures which exceeded the strength of the timber support. At about the same length of time after undercutting, Block 6-2 began to experience similar failure of ground support, scattered throughout its central portion, but of a more spotty nature. At this point, all damaging pressures were located internally in the newly undercut blocks and peripheral fringe drifts showed no weight pattern. The uniformity of results in each of the three blocks argues in favor of a similar mechanism operating in each case, and that mechanism may well have been the sag of the conglomerate over each center of draw as the caving impulse reached the San Manuel fault plane. Note that in this early stage, each individual block temporarily did develop its own independent center of cave, else the initial weight pattern could not have been so consistently centrally located over each successive undercut with no effect on peripheral drifts.

However, the damaging pressures soon spread from their initial restricted central locations, and day-by-day they affected larger areas within each active block. By the time first surface cracking was noted in May, 1956, Block 7-1 (with 8% of the total rock column removed) showed exceptionally severe pressures concentrated on 20,000 sq. ft. of its south central area; Block 9-1 (with 12% of the rock column removed) showed similar failure over 11,000 sq. ft. of its undercut, and Block 6-2 (with about 6% of its rock column withdrawn) had been affected by persistent though spotty concentrations of weight over 20,000 sq. ft. of its area. To this point, adjacent fringe drifts remained relatively unaffected by excessive weight, but by June, 1956 the expanding wave of crushing pressures engulfed those parts of Panel 7, 8, and 9 Fringe Drifts which lay within the undercut perimeter, and caused their collapse, creating thereby a contiguous expanse of collapsed mine workings which stretched along the strike of the orebody for 800 ft. from Panel 6 to Panel 9 and showed an average width of

perhaps 200 ft. The virtual destruction of this much timber and steel ground support over the area indicated was complete at essentially the same time that large peripheral tension cracks opened at the surface, and the location of the smashed area was essentially both the center of gravity of the draw and the center of subsidence. See Plate 30.

Obviously the destruction of the underground extraction openings and the formation of the incipient cave area are related phenomenon, and an attempt to explain them and to assign quantitative values to the magnitude of the pressures at the grizzly level would be of considerable interest in future caving operations. A mechanism of caving action for the San Manuel mine has already been detailed in the section titled "Initial Failure of the Gila Conglomerate" (q.v.) and a consideration of the same theory from the standpoint of pressure generally corroborates what was stated there. It appears that in the immediate weeks after undercutting, these blocks, which were all on a checkerboard system, may have acted as an independent caving entities, with caving action rising rapidly in the igneous column to reach the San Manuel fault plane at an early date. Once the igneous base had been slightly withdrawn from beneath the fault plane, the conglomerate mass sagged enough to transmit severe pressures to the general center of each block. As withdrawal of igneous rock continued, these independent centers of conglomerate affected by sag rapidly expanded, (expanding likewise the area of severe pressure) until they coalesced into one large unit mass which soon showed enough total sag to cause it to shear loose along the great oval outline shown in Plates 2 and 30, thereby creating pressures severe enough to complete the destruction of most of the ground support on the grizzly level under the general center of subsidence. Thus the initial central location of pressure in the newly undercut blocks, the expansion of this pressure outward as draw progressed and the ultimate formation of a large contiguous area of heavy ground on the grizzly level, are all seen to be the result of a similar expanding sag pattern in the overlying conglomerate with the ultimate formation of a scarp line enclosing an area much greater than the area of the undercut perimeter. To arrive at this geometric condition only one vital thing is required, that being a very low overall compressive strength for large volumes of monzonite porphyry rock, and that element seems to be present in the host rock at San Manuel. As suggested in an earlier section of this paper, the monzonite porphyry contains so many physical defects, such as gouge zones, joints, mineralized veinlets, etc., that if columns of it several hundred ft. high are unconstrained on one or more sides, those columns cannot exhibit much resistance to crushing and must soon fail in compression toward some area of active draw. The value of this en



masse compressive strength will thus determine the diameter of the surface escarpment, for that escarpment will form vertically over the outer perimeter in which porphyry fails in compression. Hence the weaker the compressive strength of the rock, the further out the perimeter around which it fails and the greater the volume of ground which is set in motion by draw. In the case of the 1415 Level, the original undercut perimeter of the first four blocks enclosed an area of 430,000 sq. ft., but this set in motion a volume of ground which at the surface had expanded to include an area of 1,300,000 sq. ft. This great volume of ground, once broken loose and set in motion, transmitted much of its weight onto the restricted area enclosed within the undercut perimeter, where relief from the resulting pressures could only be obtained in those active grizzly drifts where the rate of draw was high enough to keep the pressure from thrusting onto the back or sides of a given drift. In a newly undercut level, not every draw raise can be drawn at this rate, and besides this, it must be remembered that it is the draw itself which is giving rise to the pressures, and so such places as the inside fringe drifts adjacent to checkerboard blocks had no chance to alleviate the pressure which stressed them. As a result of the settling of the great volume of ground within the incipient scarp line, the whole general interior of the undercut perimeter became a pressure field in which the average pressure easily exceeded the strength of the porphyry and also exceeded the breaking strength of light steel or timbered ground support. Consequently, when the rock itself failed early in the draw, the ground support soon followed and collapsed too.

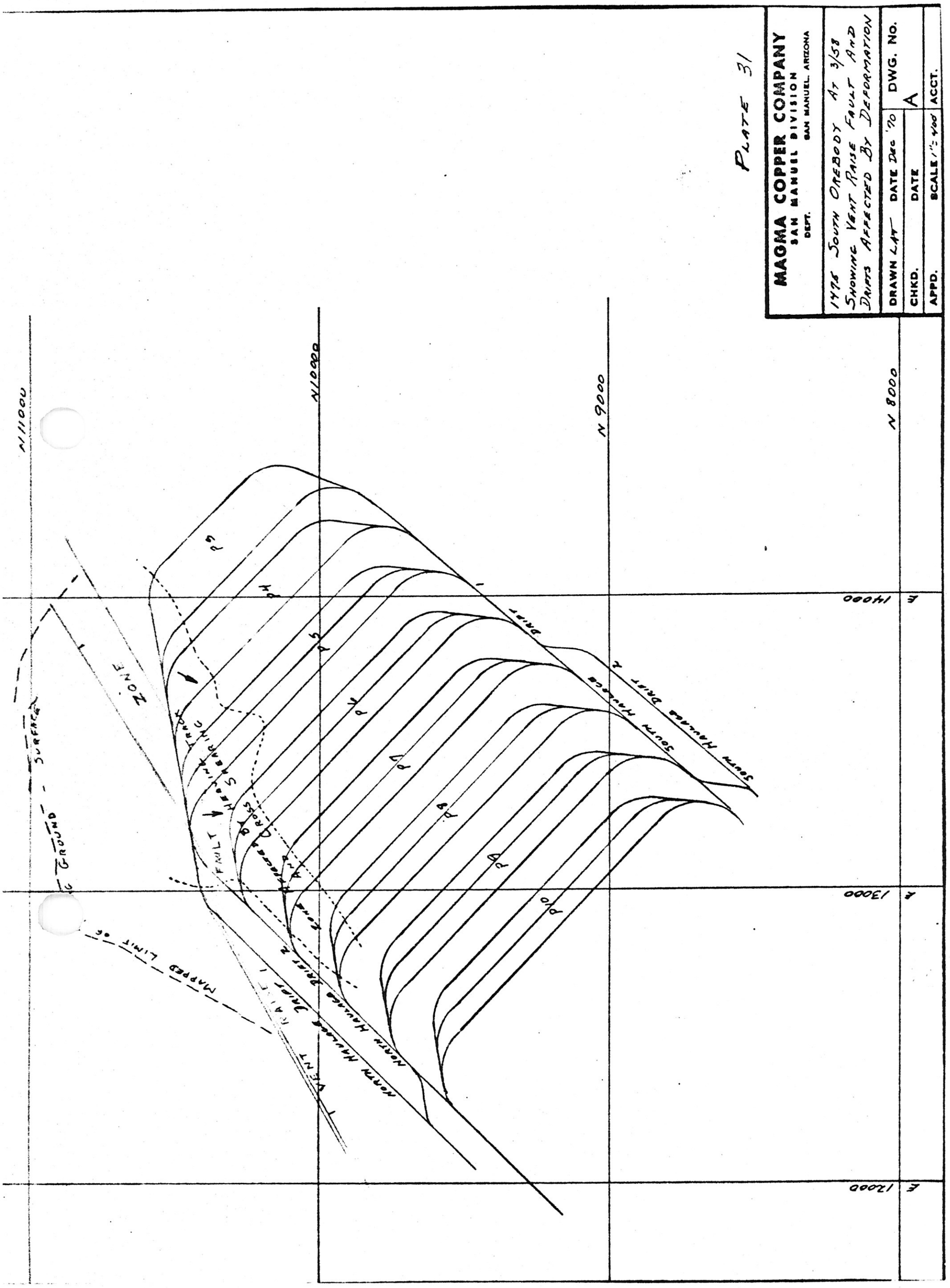
Since the volume of moving ground between undercut level and the surface in June, 1956 is known to have been at least 9.6 x 10^8 cu. ft. (the average area by 1,100 ft. vertical height); and since this large volume was thrusting its weight down on a restricted area estimated at only 290,000 sq. ft. (the whole undercut perimeter minus actively drawing areas); an order-of-magnitude value for the pressures occurring at grizzly level directly under the center of subsidence calculates to 3,900 psi. This value falls in a range far in excess of what normally would be encountered at that depth and also greatly in excess of the probable 1,500-1,600 psi breaking strength of the rock mass and provides a logical explanation of the failure of the ground support on such a widespread scale. Note that this is an average value and there is no reason why variations either above or below it could not have occurred to intensify or alleviate the weight problem in a local drift. Note also that the idea of a large frustum-shaped volume of ground bearing down on a smaller undercut perimeter does not, in itself, demand the presence of throughgoing structural zones to localize excessive pressures, though the presence of such features would (and did) intensify the ground weight problem where they intersect a drift. The notion merely requires the assumption of a very weak rock which eventually fails in compression well outside the vertical limits of undercutting as the impulse of caving action rises toward the surface, thereby creating a surface cave very much larger than the area of the undercut perimeter which effectively multiplies the pressures felt at the working level. In this light, wide shear zones or any area showing a dense concentration of fractures become features which weaken the strength of the igneous column even more, allowing failure in the rock column still further out beyond the vertical limits of undercutting which increases still more the volume of actively subsiding ground with its concommitant multiplication of forces.

The full effect of the weight pattern was thus felt on the 1415 grizzly level within eight months of the beginning of production and, once established as illustrated, it persisted in these centrally located areas for as long as significant draw continued there. It was obvious that timber or steel sets were inadequate as ground support, so in June, 1956, both development and repair procedures were re-oriented toward placing concrete lining in all drifts within the mining area and in known zones of weakness outside the undercut perimeter. Naturally such a re-orientation took considerable time to become fully effective, so repair operations had to continue at a high level for another year and a half. The concrete lining immediately improved repair conditions in those drifts where it had been placed, and stabilized ground conditions to a considerable degree. Occasionally a stretch of drift which had been lined would collapse again, and since test cylinders taken at the time showed that the best concrete poured tested in compression at approximately 3,700 psi, by inference, local pressure concentrations in excess of this value were still occurring. In many places, concrete lining was broken but did not collapse, suggesting that pressures being transmitted to the lining were approximately at the limit of the strength of the concrete. Since the average pour on the 1415 Level approached a strength of 2,700 psi, the inference again is that pressures on the level commonly fell near this range. From the above record of events, it appears that pressures on the grizzly level reached their maximum during the early formation of the surface escarpment, and that as new undercuts were added, pressure patterns shifted, in general declining to more manageable ranges. One reason which may account for this easing off lies in the fact that the additional undercuts still lay <u>inside</u> the periphery of the surface escarpment and hence while they added substantially to the area of the undercut perimeter, they added relatively less to the volume of moving ground, since draw from them did not advance the perimeter of the scarp. Consequently the <u>average</u> theoretical pressure on a square unit of grizzly drift declined. It is also

noted that some additional weight was transmitted when the undercut sequence moved far enough southwest to approach the original scarp line and renew sag and rotation in the conglomerate. Also, as the general center of gravity of draw shifted to the southwest, the location of the most persistently heavy ground tended to shift with it, keeping the general center of subsiding ground and the general center of draw vertically superimposed. This pattern continued until such time as the orebody narrowed to such an extent, and the conglomerate thickened to such an extent that surface collapse was not possible and general ground weight problems disappeared.

Ground weight problems on the 1475 Haulage Level were mostly related to zones of known structural weakness, particularly to the Vent Raise fault zone, and since this level was also timbered, much repair became necessary where drifts and structural zones were in close proximity. With the passing of time, these heavy zones increasingly restricted the flexibility of the haulage system and so from this experience a decision to line deeper haulage level drifts with concrete was made. Plate 31 illustrates the extent of heavy ground on the 1475 Haulage Level in March, 1958. The map is modified from Wilson (9) and may show the presence of two types of pressures. The deformation localized in the ladder drifts of Panels 6, 7, and 8 may be due to abutment pressures transmitted down from the undercut perimeter. The long stretch of haulage level and turnouts opposite Panels 3 to 6, which underwent considerable heaving of the track and a general south to southwest movement of the roof relative to the floor was likely deformed because of its proximity to the Vent Raise fault zone. The entire rock mass on the hanging wall side of the fault zone had been set in motion by draw from blocks to the south, and the resulting slump persistently damaged ground support in all those drifts which were intersected by the zone. The type of drift deformation experienced was exactly what would be expected from a relative slide down-dip of the hanging wall side of the shear zone. Similar deformation related to the West fault and Hangover fault systems have been noted in other parts of the mine, and its effects are quite predictable.

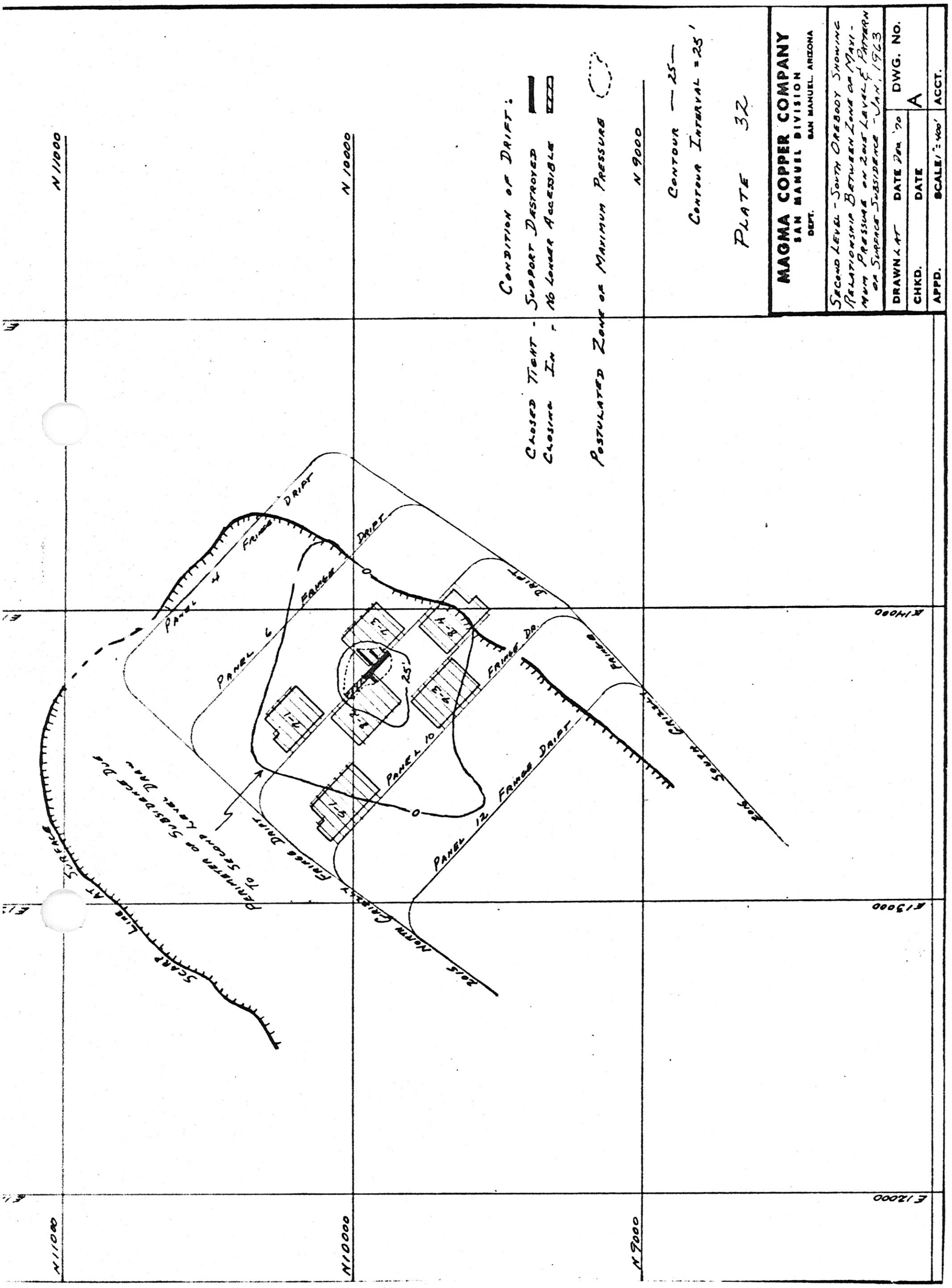
B. <u>The Second Level - South Orebody</u>. In view of the unfavorable experience on the First Level with timber or steel sets as ground support, the Second Level development program was oriented toward placing a minimum thickness of 18 inches of unreinforced concrete lining in all extraction openings in the vicinity of future mining areas or zones of known structural weakness. The only exceptions to the concrete program involved the 2015 South Grizzly Fringe Drift and the 2075 South Haulages and South Ladder Drifts which were developed with timbered ground support. These areas were not concreted because it was felt by analogy that they would



remain free of destructive pressures, and the passing of time has showed this judgment to be essentially correct. Cylinder tests taken on concrete poured in the initial mining area show an average 28-day compressive strength of over 3,400 psi, and adjustments in the mix in later years brought this up to nearly 4,000 psi. These efforts have largely eliminated the problem of widespread destruction of extraction openings with resultant inaccessibility to producing areas, and compared with First Level history, the loss of production on the Second Level due to drift collapse has been minimal.

The concrete lining, of course, frequently begins to break up as undercutting approaches a given stretch of drift, but experience has shown that the placing of steel liner sets inside the concrete often is sufficient to contain deformation until ore extraction is completed.

The weight pattern which thrust down on the 2015 Level was similar to that which occurred over the First Lift mining area. It was most severe early in the history of the level when a new cave pattern related to Second Level draw was being established, and it eased off to some extent when the new patterns of caving action were fully established. The heaviest zones were again spatially related to the general center of draw and to the center of subsidence. One persistently heavy zone in Panels 9 and 10 was obviously related to its proximity to the East Fault system and other local heavy areas with their presumed geologic control will be discussed below. The first serious drift deformation on the 2015 Level which could not be definitely correlated with known structures affected the central part of Panel 8 Fringe Dr. and the adjacent grizzly drifts in the north end of Block 7-3 and the south end of Block 8-2. These drifts were undergoing serious deformation or collapse by late 1962, after six months of production from 7-3 and 8-2, despite the fact that this area was located nearly directly beneath the deepest part of the pre-existing cave area and, hence, was pulling into 600 ft. of unbroken igneous rock topped by ± 900 additional ft. of rock broken by First Level draw. The length of time was sufficient to allow the impulse of caving to have risen all the way through the unbroken column and to be affecting the entire rock column. The location of the collapsed drifts and the pattern of subsidence at January 1, 1963 is illustrated by Plate 32, and it may readily be seen that the general center of subsidence, the general center of gravity of active blocks, and the location of maximum pressures are all superimposed. The magnitude of pressure which was affecting these drifts at January 1, 1963 is difficult to pin down, and no instrumentation was ever installed to give direct data. If one calculates the <u>average</u> pressure of the moving rock mass on pillar areas within the under-



cut perimeter, the value is ± 3,600 psi, which, while substantial, does not seem high enough to persistently destroy a concrete lining with auxiliary steel liner installed. There is a strong implication here then, that the weight of the moving rock mass gets concentrated in the zone under the center of subsidence and that pressures there are intensified. In any event, the coincidence of the center of subsidence with the areas of maximum ground pressure on both First and Second Levels shows that the mechanism of weight transfer was operating in the same manner on both levels and that it requires more than local structural conditions in a given drift to account for the persistent deformation of centrally located extraction openings during the "break-in" period of a new level.

Two interesting corollaries to the weight pattern in Panel 8 Fringe Drift became apparent with the passing of time. First, although Panel 8 Fringe itself remained persistently unmanageable over about 400 linear ft. of its central part for as long as significant draw occurred in Panels 7 or 8, the deformation never seriously affected the grizzly drifts adjacent, and crushing pressures rarely extended more than 15 ft. into a drift on either side. Perhaps this suggests that a satisfactory rate-of-draw was accomplished in the actual blocks themselves, thereby negating pressures in the block proper and intensifying them in the adjacent Fringe Drift. Also, during the time these panels were being drawn, leveling checks showed that the North and South Fringe Drifts were rising (as much as two inches), and were therefore in tension, balancing the general compression in the panel areas and demonstrating the existence of the classical subsidence strain pattern of tension outside the limits of moving ground and compression within.

Other areas of concentrated pressures have occurred locally on the 2015 Level, and an analysis of them is instructive for projecting trends on future levels. These special cases include:

1) A section of Panel 10 Fringe Drift and adjacent Block 9-1 where a major strand of the East fault system intersected the drifts and most of the active block lay in the hanging wall of the fault zone. In this case, the destruction of the ground support was so obviously related to structure that no doubt exists as to the origin of the stresses or proper means to cope with them. This area was not initially a location of concentrated pressure, though Block 9-1, which never drew at a satisfactory rate, did show persistent destruction of ground support. However, as draw progressed in those blocks in the hanging wall of the fault, enough differential subsidence occurred along the major strand to cause total collapse of

Panel 10 Fringe Drift and adjacent grizzly drifts at the Fault contact. Panel 10 Fringe Drift was never repaired and ore extraction was completed using the access drifts. In this case, the limits of moving ground which gave rise to the pressures are readily discernible and they would include a wedge-shaped mass of moving rock bounded on the west by the fault plane and on the east by the limit of active ground over Block 9-1. The wedge would this have its point downward, impinging on the grizzly level over an area of only a few thousand square ft. and its base would be at the surface enclosing an area projected up the dip of the fault plane and vertically over the block. Under these conditions, the weight of the moving rock mass, concentrated on the narrow point, could easily give rise to local pressure concentrations on the grizzly drifts in excess of 15,000 psi. In order to minimize the effects of this type of pressure, blocks in the footwall of the fault zone should be undercut and drawn before those in the hanging wall, and the number of drifts crossing the structure should be kept to a minimum until the footwall side has been substantially drawn out.

2) Nearly every panel fringe drift on the south side of the 2015 Level has taken crushing weight over a local stretch of 40 or 50 linear ft. where those fringe drifts lie vertically beneath the old 1415 undercut perimeter. The reason for this effect is best seen in a plan view of the two levels superimposed, where it may be seen that because of the cylindrical shape of the orebody at this depth, the perimeter of the 2015 blocks juts anywhere from 150 to 200 ft. outside the perimeter of the 1415 blocks, and hence every panel fringe drift and certain grizzly drifts on the 2015 Level must necessarily penetrate the ground beneath this old perimeter. The major pressures arise when the block to the west of the fringe drift is undercut and the unbroken mass of virgin ground above the level is released. Before it can be fully broken and under satisfactory draw, it transmits abutment-type pressures onto the drifts vertically beneath the old perimeter (probably by a tendency to rotate toward the center of subsidence [?]), giving rise again to temporary but very high stress conditions. These local areas can often be repaired and re-opened when the newly undercut block has been drawn enough to be well fragmented and no longer is acting as a monolithic unit. One operational solution to this problem might be the use locally of reinforcing steel in the concrete lining when the drift is originally developed.

3) There is a considerable amount of persistent pressure on the ground support around the 2075 Haulage Level in the

vicinity of Panel 18 South Ladder Drift. This area is cut by two rhyolite sills which dip southwesterly at a shallow angle. This means that at present, the up-dip extension of the sills is above the 2015 undercut and within the active draw perimeter. Since the rhyolite is a much harder and more elastic rock than the monzonite porphyry, there is a strong possibility that stresses produced by the draw are being transmitted down-dip along the sills to adversely affect the haulage level either by heave above a sill or by compression beneath one. The problem is not acute, but it does restrict the movement of trains on the haulage level from time to time. A possible solution is to concrete such rhyolite areas that lie down-dip from what will eventually be active blocks.

4) It has long been common knowledge at the San Manuel operation that in a given panel the advancing outside lines of a newly undercut block or the retreating inside lines of a sealing block will come under extensive horizontal pressures which will tend to close the drift tight by lateral movement of the outside wall in the direction of active draw. From a pressure standpoint, this occurs because draw from the block itself produces a volume of ground where pressures are negative while the surrounding rock has positive pressures (whether sealed or virgin). Hence the peripheral rock crushes toward the active line and tends to close it in the direction of negative pressure. The only solution in the case of a sealing area is to draw the remaining tonnage as fast as possible and with minimum repair. The solution in the case of an advancing undercut is more difficult to counteract, for while it might be possible to pour better (or even reinforced) concrete in a specific line which was presumed to be a future outside edge, if the mining sequence was altered for any reason, the reinforced concrete might be found to have been poured in the wrong place. As an alternate solution, if the next lines adjacent were undercut, the problem would be solved in one drift and transferred to another; and here again, it is not always possible to undercut the adjacent line upon demand.

5) An interesting condition has occurred during the undercutting and early draw stage of almost every block on the Second Level. This is the immediate but temporary appearance of stress in the ground support beneath the block during undercutting, the stress normally being severe enough to crack the concrete and occasionally demand the installation of steel liner sets. This phenomenon prob-

ably represents the readjustment of residual internal rock stresses which are released by the removal of material during the undercut stage. It normally eases off and disappears shortly after the completion of the undercutting cycle. Note that there is no way to eliminate this condition because it is an inherent part of the regional stress pattern which affects the entire rock mass in which the orebody occurs. As such, it will probably grow increasingly severe with increase in the depth of mining, and eventually the design of the concrete lining will have to be adjusted to counteract its effects.

Conclusions

This retrospective analysis of 15 years of subsidence phenomena associated with production from two levels of the San Manuel mine highlights what is probably the most unique aspect of the mine environment, namely the inherent overall weakness of the igneous host rock, even in the absence of known zones of throughgoing structure. If one recalls the cylindrical shape of the orebody as it surrounds a low grade porphyry core and is itself surrounded by a much more competent pyritic zone, a reasonable generalization could state that the entire volume inside the pyritic zone is normally a closely fractured, weakly bonded, reticulated rock mass which, when unconstrained, is capable of only the minimum amount of resistance to the stresses induced by draw. Consequently, rock failure on a widespread scale both above and outward from the undercut perimeter as well as within the extraction openings of a given level has been, and will remain, a dominant feature of production for the life of the mine. There are both advantages and disadvantages to operating in this structural environment, and presumably the future operation can endeavor to utilize the advantages to its benefit while attempting to minimize the disadvantages.

Some of the advantages of the environment which might be listed include favorable ratios for operational efficiencies such as tons per manshift and tons of ore per pound of powder. The possibility of obtaining very large tonnage extractions from within the undercut perimeter is an important factor related to the environment, and more indirect advantages, such as easier handling of muck through chutes and raises and lower crushing costs, also come to mind. Disadvantages of the structural environment include mainly the destruction of ground support on a working level by the stresses induced by draw and the widespread surface subsidence and peripheral rotation around the escarpment which produce an area of moving ground very much larger than the area undercut. Many of the plates included in this report illustrate the latter point very plainly.

Some specific conclusions and inferences which can be drawn from this study are:

1) The presence of at least eight important sets of fractures in the San Manuel igneous complex was demonstrated by Wilson (8, 9). From the standpoint of cavability, the density of distribution of total fracturing is probably more important than the presence or absence of given sets within a mining block.

2) There is a residual stress pattern in the igneous rock which must adjust locally during the undercutting stage. In adjusting, forces rise which are sufficient to crack concrete linings, cause drifts to heave, and deform transfer raises. Similar adjustments occur in the proximity of major shear zones when drifts are being developed. These forces are time-related and once a new equilibrium is achieved, relaxation occurs. As mining progresses to deeper levels, the severity of the adjustments is likely to increase.

3) The ultimate location of the surface escarpment around a mining area is primarily a function of the breaking strength of the unconstrained rock mass. The location of the scarp is the surface projection beneath which the igneous mass is failing and moving toward some area of active draw.

4) It follows from (3) above that if the rock mass is weak enough to fail well outside the undercut perimeter, then the scarp will form well outside also; and the weaker the rock mass, the farther out the scarp line can form. Pancake gauges and borehole pressure cells placed in the ore zone by the U. S. Bureau of Mines have shown that rock failure may occur at pressures as low as 1,500-2,200 psi.

5) From (4) it follows that if rocks of different strengths occur on opposite sides of the orebody, the perimeter of failure will be asymmetrical with respect to the axis of the orebody and hence the scarp line will also be asymmetrical. Such a condition obtains at San Manuel, and is further accentuated by the presence of the Vent Raise fault along the north edge of the 2015 Level undercut perimeter.

6) It also follows that once the rock mass peripheral to the edge of the active undercut has been broken and set in motion, lateral flow and replenishment of rock in active draw raises is inevitable. Such a mechanism accounts for the very great tonnage extraction which can be effected if the mechanism is controlled by proper undercut sequences (i.e., from hard rock toward soft).

7) It follows from (4) above that the extraction openings on a given grizzly level must support a frustum-shaped rock mass which is in motion all the way to the surface. Hence, with increasing depth the average pressure thrusting down on the workings increases also and ground support methods must be constantly improved as mining moves into the deeper levels. As a corollary to this observation, it has been demonstrated at San Manuel that during the early break-in period of a new level, the location of maximum pressure bears an almost exact spatial relationship to the center of subsidence and the center of draw. Forces within this hub become intensified (by rotation [?]) and rise to such magnitudes that no reasonable method of counteracting them has yet been determined. Once renewed cave action has been well-established and the undercut perimeter advanced, the unmanageable pressures ease off, but this may require two or three years to accomplish. With increase in depth of the mining operation, this particular problem looms larger and may require changes in undercut sequence to help combat it.

8) In view of (7) above, and experience on two levels, it is apparent that a checkerboard caving sequence is not the best way to approach the mining of an orebody in weak rock, despite the operational advantages which accrue. The major flaw in the checkerboard pattern, structurally speaking, is that it simply sets too much ground in motion for the rate of daily production required and the resulting pressures create such widespread deformation and collapse of mine workings that production is seriously restricted and increasingly costly. Inherent in this conclusion, then, is the suggestion that for a given rate of daily production the tonnage should come from as restricted an area as is compatible with haulage, and that there might be fruitful consideration given to rates of draw in excess of the San Manuel norm of 18 inches per day. Also inherent in this conclusion involving pressure vs. area is the idea that the orebody has an optimum daily tonnage which it is capable of producing without either opening up too much area or drawing active areas at too-high a daily rate. Of the two choices stated, opening up too much area for production would have by far the most serious consequences, as it could result in crushing pressures transmitted to significant areas of mine workings; while a too-high rate of draw would produce only an increase in the percentage of dilution of otherwise good ore. This consideration is very likely to become a major factor in the exploitation of the Kalamazoo segment, for it may well be that developing and undercutting a restricted area and utilizing a high rate of draw is the only way that the ore can be extracted at such depths. There is no evidence at the present time that high

rates of draw in the weaker parts of the orebody cannot be successful; in fact, any number of blocks along the northern periphery of the South Orebody have been drawn at rates up to 32 inches/day with no apparent harm to ultimate tonnage extraction. Since the structural conditions that obtained there should be similar to those that will be found on the Kalamazoo First Lift within the major part of the level, the comparison should be directly applicable.

9) It also follows that from the increase of pressure with increasing depth that tonnage extraction should improve constantly with each succeeding level, even if cut-off procedures remain unchanged. This conclusion follows from two causes:

   a) the grade of the dilution is always improving as one level vertically succeeds another with depth and hence more tons are drawn before average grade ore is diluted seriously enough to reach cutoff. But it is also very true that:

   b) lateral flow and replenishment will increase with increasing pressure giving rise to ever-higher tonnage extractions and this will eventually become a major factor in determining the rate of development and stope preparation and the manpower requirements therefor. From the current record it is worth noting that the First Level South Orebody achieved an overall tonnage extraction of 100.6% of the ore contained within the vertical limits of block boundaries on a 52,000,000 ton sample, while the Second Level South Orebody comparable figure is currently 135.0% for a 65,000,000 ton sample. Tonnage extraction is obviously going to continue to rise significantly on the deeper levels and the estimated rates for stope preparation should be adjusted accordingly.

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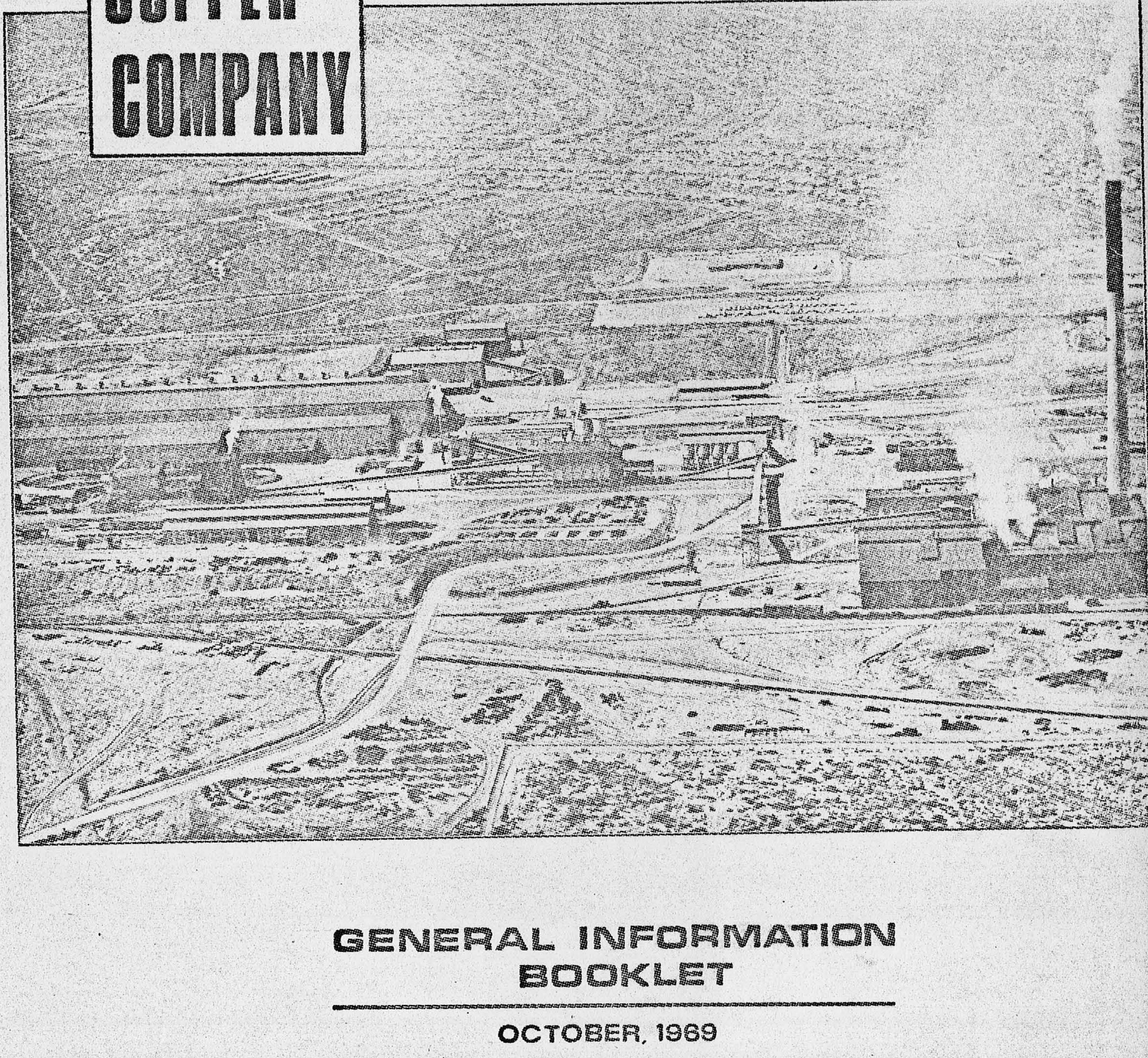
MAGMA COPPER COMPANY

SAN MANUEL DIVISION



GENERAL INFORMATION BOOKLET

OCTOBER, 1969

Address

Box M, San Manuel, Arizona 85631

Telephone

602-385-2201

THE HISTORY AND DEVELOPMENT OF SAN MANUEL

Magma Copper Company's San Manuel holdings are located in the southeast part of Pinal County, Arizona, about forty-five miles northeast of Tucson. The twin ore hoists and the dome-shaped Red Hill serve as landmarks for the mine area.

The concentrator, smelter, administration building, and other plant facilities are located some seven miles southeast of the mine area adjacent to the town of San Manuel.

The district was prospected prior to the Civil War, but there was little or no ore production until 1881. Until the advent of the San Manuel mine, the chief producers were the Mammoth and Mohawk mines located a mile farther north. Gold, lead, zinc, and some vanadium and molybdenum were the main recoverable metals at these properties. At least two exploratory churn drill holes were drilled in or near the San Manuel ore zone in 1917. The copper content indicated by these holes was not sufficient to encourage further exploration at that time.

In 1942, through the efforts of the property owners, James M. Douglas, R. B. Giffin, Victor Erickson, and Henry W. Nichols, all of Superior, Arizona, the Reconstruction Finance Corporation and War Production Board authorized the United States Geological Survey to investigate the property.

This survey confirmed the owners' original conception of the probable existence of important copper mineralization, and by its recommendation the Bureau of Mines was authorized to put down a limited number of churn drill holes. This test drilling started in November, 1943, and was continued until February, 1945 when seventeen holes had been drilled for a total of 15,844 feet.

Magma Copper Company obtained an option to purchase the property in 1944. In September, 1944, Magma exercised its purchase option, and additional adjoining claims held by the Apex Lead Vanadium Mining Corporation and the Quarelli family were purchased. Additional claims were located and in December of that same year Magma undertook additional exploration by churn drilling.

The San Manuel Copper Corporation was incorporated in August, 1945, and all of the property acquired by Magma Copper Company in the district was deeded to San Manuel. The corporate structure was changed in 1962 to include the San Manuel property as a Division of Magma Copper Company, rather than a wholly-owned subsidiary corporation.

Exploratory churn drilling was essentially completed in early 1948, and underground exploration and development was started in March of that year.

On July 10, 1952, the Reconstruction Finance Corporation authorized a loan of $94,000,000 to San Manuel for mine development and plant construction.

In early 1953, Utah Construction Company and The Stearns-Roger Manufacturing Company (a Joint Venture) were awarded a contract for the design and construction of the entire surface plant, including the concentrator, smelter, railroads, and auxiliaries. Principal subcontractors were San Xavier Rock and Sand Company, which furnished the concrete; Newbery Electric Corporation, which installed the electric control and transmission system; and Custodis Construction Company, which erected the smelter stack.

The concentrator was completed in September, 1955, and commenced trial runs on stockpiled and mine development ores. Smelter and remaining plant construction was completed in late 1955 except for minor cleanup work and the smelting of copper concentrates was started January 8, 1956. Five shafts had been sunk and over twenty miles of drifting had been completed to prepare the first lift for production. On January 23, 1956 the Mine was in production with the first stope undercut completed.

The 1475, or first level, was mined to completion early in 1965. The 2075, or second level, is now being mined, with development of the 1775, or intermediate level, also being carried on at this time.

THE TOWNSITE

To provide adequate permanent housing facilities for the construction period, as well as the future productive life of the mine, an agreement was made with the Del E. Webb Construction Company and M.O.W. Homes, Inc., under which they were to finance and build a town suitable for the accommodation of San Manuel's employes.

Active construction was started in mid-1953, and by late 1954, the town of San Manuel, consisting of 1,000 homes, shopping facilities, and hospital, was completed. Magma Copper Company acquired the town early in 1955. In 1957, an additional fifty homes were constructed, bringing the total number of houses to 1,050.

The Townsite of San Manuel, Arizona, is a cluster of modern homes and shops, interlaced with wide, surfaced boulevards and streets, on the sloping west side of the San Pedro Valley. To the east the Galiuro Mountains offer a spectacular view, while to the west the Santa Catalina Mountains serve as a magnificent backdrop to the residential area.

San Manuel was conceived and built for those who work for Magma Copper Company as well as for those in related activities--merchants, police officers, clergymen and others.

The main shopping center covers 32 landscaped acres off McNab Parkway. The shops located there offer San Manuel a wide variety of merchandise and services. This is supplemented by a second shopping center located in the lower part of the town.

Since April 1, 1955, the townsite properties have been managed by the San Manuel Townsite Division of Magma Copper Company.

The San Manuel Division Hospital, a half-million dollar institution which features not only the very latest equipment, but also the best in construction, serves the medical needs of San Manuel and vicinity.

Within the hospital there is a completely equipped surgery, nursery, two obstetrical rooms, emergency room, x-ray room, x-ray developing room, a patients' wing of thirty beds, doctors' offices, laboratories, therapy rooms, reception room, waiting room, offices, kitchen facilities and dining rooms.

The hospital is a member of the Southwest Regional Blood Bank of the Red Cross. This is a blood bank system which maintains stocks of blood of all types at all times and blood is available on call. In addition the hospital maintains a list of blood donors in the townsite, by type, who may be called in emergencies.

Just south of the hospital stands the nurses' home. Facilities exist for accommodating seven nurses in furnished, modern, completely air-conditioned quarters.

In this same area is located the Administration Building which houses the various management, accounting, engineering, exploration, purchasing, and personnel offices.

Three schools serve the educational needs of the elementary and high school students of San Manuel, Mammoth, and vicinity. San Manuel elementary schools and Mammoth elementary school comprise the Mammoth School District Number Eight, and San Manuel High School serves Mammoth High School District, which includes students from San Manuel, Mammoth, and vicinity.

The three schools were completed at a total cost of approximately $1,000,000 for buildings and equipment.

A 10,000-book public library serves the town of San Manuel and supplements the school libraries. The library has gradually expanded from a small one-room area in the Community Center building to its own quarters with over 1,500 square feet of floor space. This is a community library made possible by Magma Copper Company and supported by the citizens of the community, many club organizations, and the Townsite Division. Most of the work is done by volunteer workers who offer their time and services.

T H E   M I N E

The San Manuel orebody is part of a mass of mineralized rock, chiefly a granitic-appearing monzonite and a similar, though finer-textured, monzonite porphyry. This large zone of mineralization is covered for the most part by unmineralized conglomerate, a younger rock than those comprising the mineralized zone. The orebody, or portion of the general mineralized mass containing appreciable copper sulfide minerals in addition to iron sulfides, covers an area over one mile long by one-half mile wide. The known depth of ore extends about 2,600 feet below the surface. The control as to size and shape of the orebody is an arbitrary cutoff based on copper content of the mineralized rock. Therefore, that portion considered economically feasible to mine appears in the more northerly portion as a tabular mass up to 400 feet thick with its long dimension bearing northeast and lying at an angle of 55° from horizontal to the southeast. This attitude persists down dip for about 2,400 feet where it flattens and then rolls upward to form a cross-sectional fishhook shape. Within this part of the orebody there is a pronounced thickening, and it is the upper one-third of this southeast portion, starting some 1,100 feet below the surface, that was selected for initial mining operations. Of this 1,100-foot thickness from the first mining level to the surface, there is an average of about 430 feet of ore and 670 feet of waste overburden.

The thickness of the overburden and shape and size of the orebody combine to make open pit mining impractical. For these reasons the underground block caving method of mining was selected. The monzonite in which the ore occurs is well fractured, caves readily and crushes to a size that is easy to transport.

Block caving entails the undercutting or removal of a horizontal slice of ore of sufficient area (stope block) so that the unbroken ore above will not support itself, but will cave and slough into the undercut. As the broken ore is drawn, removing support from the ore above, caving extends to the surface, the overburden or waste rock following the ore down. When the waste rock reaches the undercut horizon, drawing is stopped and the stope block is finished.

The caved ore from the undercut horizon is drawn on the grizzly level through a series of closely spaced draw raises. The grizzly level which is the control level, is 20 to 25 feet below the undercut. On the grizzly level the ore passes through the grizzlies which consist of rails spaced 12 to 15 inches apart over the top of each transfer raise.

The transfer raise system funnels the ore from eight draw raises to one common loading station on the haulage level which is about 60 feet below the grizzly level. A loading station serves two transfer raises, each of which, when full of ore, holds 65 tons. The ore stored in the raises is transported by an underground electric railroad system to the ore hoisting shafts.

Loading operations from the transfer raises to the ore cars are controlled through steel chutes and air-operated chute gates. The ore cars have a capacity of about 12.5 tons, and each train is made up of 15 to 18 cars, or 185 to 225 tons per train, pulled by a 23-ton, 250-horsepower trolley locomotive. One of these trains is dumped every seven minutes. The electrical power system supplying the trolley is 275 volt DC with rectifier stations so situated as to maintain full voltage throughout the haulage system.

The underground track for the haulage system is 36-inch gauge with 70-pound rail through the panels. On the main lines between the mining area and the hoisting shaft, 90-pound rail is used to accommodate the heavy traffic and higher speeds.

At each of the two identical ore hoisting shafts, 3-A and 3-B, the trains pass through a rotary tipple on the haulage level which dumps three cars at a time. The cars, equipped with rotary couplings, do not have to be disconnected as the tipple turns 180° and then rights itself to dump the ore into the 1500-ton pocket or underground storage bin adjacent to each shaft.

The ore is drawn from the bottom of the pocket into a measuring pocket hopper which in turn discharges into skips for hoisting to the surface. The bottom dump ore skips, which hold 21 tons of ore, are hoisted to the surface and discharged into two 5000-ton surface storage bins, which in turn discharge into pan feeders that carry the ore to the two gyratory crushers located nearby. Discharge from the crushers is moved by conveyor belts to two 10,000-ton surface storage bins awaiting transportation to the Plant.

Each of the two ore hoisting shafts is equipped with a Nordberg hoist with 15-foot diameter drums and powered by two 3,000 horsepower electric motors. These hoists can be manually or automatically controlled. The operating hoisting speed is 3,000 feet per minute. The hoisting cable is $2\frac{1}{4}$ inches in diameter.

Nos. 1 and 2 shafts were sunk early in the program, and from these shafts the first mining lift was developed. No. 1 shaft, steel and reinforced concrete lined, now serves for downcast ventilation and hoisting of development waste rock. No. 2 shaft, sunk for exploration and quick development, has now been abandoned.

No. 4 shaft, steel and concrete lined, serves as a downcast ventilation shaft and as a service shaft for men and supplies. Men are lowered and hoisted at the rate of 120 men per trip; and timber, powder, and other supplies necessary for the mining operation are lowered to grizzly and haulage levels through this shaft. The 3-A and 3-B shafts, in addition to hoisting of ore, serve for upcast, or exhaust ventilation.

Other facilities at the mine include mechanical and electrical shops, modern timber framing shed, timber treating plant, warehouse, and change rooms. Mine air compressors provide 28,000 cubic feet per minute of compressed air for rock drills and other air-driven tools underground and on the surface.

Limestone and high grade silica for metallurgical use are mined from quarry sites 17 miles north of the Plant. These products are hauled by the San Manuel Arizona Railroad to the flux crushing plant in 50-ton, bottom-dump cars.

THE MILL

Ore transportation from the Mine to the Plant is by rail shuttle service in 100-ton capacity bottom-dump railroad cars. The 40 to 44 car train is pulled by a 1600 horsepower, 125-ton diesel-electric locomotive. The seven mile ore transportation track is standard gauge, 132-pound rail and was constructed with liberal curves and no grade.

At the 10,000-ton coarse ore receiving bin at the Mill, the train is run over the bin and four cars are dumped at a time through bottom-dump air-operated car gates with compressed air furnished by the locomotive.

From beneath the receiving bins, ore is fed by eight manganese steel pan feeders and two belt conveyors into two seven-foot standard Symons cone crushers at the rate of 1000 tons per hour to each crusher. Magnets are suspended at the head of each conveyor to remove tramp iron. Crusher feed passes over grizzlies where undersize material is bypassed. The crushed ore from the primary crushers is conveyed and distributed to four secondary seven-foot Symons cone crushers, each preceded by mechanical screens to bypass the undersize material.

The crushing plant is designed with all crushers on the same level. A panelboard on the operating floor and an intercommunication system provide complete control from one point. A 30-ton overhead crane with a 5-ton auxiliary hood services the crusher floor.

The final product from the crushing plant, all less than one inch in diameter, is delivered by belt conveyor at the rate of 2000 tons per hour to a 54-inch wide tripper conveyor. The tripper conveyor runs over the top of the 45,000-ton capacity fine ore bin in the concentrator building. The tripper travels the length of the bin distributing the ore at an even rate. Zipper fastening belts cover the slots through which the ore is discharged, and the zippers open and close as the tripper travels, thus preventing the dust being spread throughout the concentrator.

The ore is drawn from the bottom of the fine ore bin by belt conveyors onto a gathering conveyor which feeds each rod mill at the rate of 4,000 tons per day. A weightometer both registers and controls tonnage to each rod mill.

The concentrator is divided into ten sections, each section consisting of one 10-foot by 12-foot rod mill and two 10-foot by 10-foot ball mills. Ball mills are operated in closed circuit with 20-inch cyclone classifiers. Oversize material from classification is returned to the ball mills for additional grinding.

Each section of grinding equipment is operated from a control panel. Reagents and lime water pumps are controlled from the same control panel. The grinding bay is serviced by a 175-ton crane which is capable of taking out a fully charged rod or ball mill for repairs. A 10-ton crane serves for lighter, faster service. Both cranes are capable of travelling the full 1200-foot length of the concentrator building.

The classifier overflow goes to distribution boxes where, with reagents added, it is distributed to 48-inch rougher flotation cells, totalling 480 in number. The copper-molybdenum minerals float to the surface of the pulp in each cell and are gathered in froth launders. The material not floated in these cells is called tailings and is piped by gravity to the tailing thickeners where approximately 12,000 g.p.m. of reclaimed overflow water is returned to the mill for reuse. The thickened underflow is discharged to the tailings pond.

The mineral concentrate is pumped from the rougher flotation cell launders through cone classifiers in closed circuit with four 8-foot by 12-foot regrind ball mills, then distributed to 144 48-inch cleaner flotation cells. The tailings from this flotation are returned to the ball mill circuit. The final copper-molybdenum concentrate is pumped to another thickener with the thickened concentrate going to the molybdenum recovery plant.

Molybdenum sulfide is recovered by flotation in another series of flotation cells, after which the concentrate is pumped into a final thickener, filtered, dried, and conveyed by belt conveyor to the molybdenum concentrate bins. The material not floated in the molybdenum plant is the final copper sulfide concentrate. It is thickened, filtered, dried and conveyed to the concentrate storage bins in the smelter building. All overflow water from the thickeners joins the return water to the mill circuit.

Gold and silver are recovered by a precipitation process. The precipitates are fed to the smelter holding furnace and blended into the copper anodes.

THE SMELTER

The copper concentrate, amounting to approximately 1000 tons per day, averages about 28% copper. The concentrate is drawn from the storage bins in the smelter building by conveyor belts and is fed to one of two 100-foot long reverberatory furnaces through hoppers located along each sidewall of the furnace. The concentrate is smelted in the furnace at a temperature of approximately 2700° F, using natural gas for fuel.

Gases from the reverberatory furnaces operate four waste heat boilers which furnish steam at 475 psig to the powerhouse. A 10,000 kw turbo generator and two 30,000 cfm turbo blowers form the main steam-driven equipment in the powerhouse.

All gases from the reverberatory furnaces and the converters pass through electrostatic precipitators prior to entering the 515-foot high stack. Practically all solid matter is removed from the smoke. This dust has a high copper content and is returned to the smelting process.

Slag is skimmed into railroad car slag pots of 200-cubic foot capacity. The slag pots are then hauled to the slag dump. Matte, which is chiefly copper, sulfur, and iron, is tapped into 200-cubic foot ladles and transferred by crane to 13-foot by 30-foot Pierce-Smith type converters. There are three converters and two 60-ton overhead cranes.

After the matte has been poured into the converters, a flux with a high silica content is added. This flux combines with the iron to form slag which is skimmed and returned to the reverberatory furnace. Further blowing oxidizes the sulfur to sulfur dioxide and converts copper sulfide to metallic copper (blister copper). The molten copper is transferred by ladle to one of two holding furnaces adjacent to the anode casting section of the smelter.

In the holding furnace, excess oxygen is removed through the injection of reformed gas. The copper is poured into anode molds located on a 34-foot diameter casting wheel. The finished anodes, weighing 735 pounds each, are cooled in water in a bosh tank. The anodes are then removed by overhead crane and stacked on the storage floor where they are later inspected and loaded for rail shipment to electrolytic refineries.

OTHER FACILITIES

The flux preparation plant is located between the smelter and concentrator buildings and includes receiving bins and crushers for handling limestone and silica flux. A lime kiln for calcining limestone and a slaker provide metallurgical lime for the concentrator.

Other plant facilities include a machine shop with locomotive service and repair pit, carpenter and auto shops, warehouse, time office, and change house.

The San Manuel Arizona Railroad Company operates on thirty miles of standard gauge railroad from San Manuel to connect with the Southern Pacific Railroad at Hayden, Arizona.

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STATISTICAL DATA

Ore Deposit

Type	Disseminated copper
Gangue rock	Quartz monzonite porphyry
Metallic minerals	Pyrite, chalcopyrite, chalcocite, molybdenite, silver and gold
Copper content	About 0.75%
Molybdenum content	About 0.025%
Gold and silver contents	Minor amounts

Orebody

Outcrop	Triangular area, 300 x 400 feet; used for smelter flux
Orebody	The control as to size and shape of the orebody is an arbitrary cutoff based on copper content of the mineralized rock. Therefore, that portion considered economically feasible to mine appears in the more northerly portion as a tabular mass up to 400 feet thick with its long dimension bearing northeast and lying at an angle of 55° from horizontal to the southeast. This attitude persists down dip for about 2,400 feet where it flattens and then rolls upward to form a cross-sectional fishhook shape.
Fracture pattern	Intricate, three-dimensional network
Mineral occurrence	Disseminated through the gangue rock

Overburden or Capping

Description	Gila conglomerate and weakly mineralized monzonite
Thickness	Averages 670 feet

Mine Openings

Support	All ground requires support, either timber, steel or concrete.
Water	Newly-opened areas may show appreciable flow. Orebody drains rapidly.
Temperatures	Moderate

Mine Production

Capacity	40,000 to 43,000 tons daily
Mining method	Block caving
Underground haulage	Electric trolley locomotives; ore car capacity, 12.5 tons; cars per train, 15 to 18; trains per day, 180.
Hoisting	Hoisted through two identical vertical shafts
	First hoisting level 1475 feet (now completely mined out)
	Intermediate hoisting level, 1775 feet
	Second hoisting level, 2075 feet
	Hoists--2 6000 hp, double-drum
	Hoisting speed--3000 fpm
	Skips dumped per hour--50 each hoist
	Capacity of skip--21 tons
	Run-of-mine ore--up to 15 inches

Ore Crushing (Mine)

Bin capacity	Ore receiving bins hold 10,000 tons
Primary crushers	Two standard gyratory crushers
Capacity	2000 tons per hour

Ore Transportation--Mine to Mill

Storage	At mine loading point--two 10,000-ton bins
Ore moved	Shuttle service railroad
Railway construction	Seven miles, standard gauge, 132-pound rail, level, minimum of curves with liberal radius.
Type of cars	Bottom-dump, air-operated
Capacity of car	100 tons
Cars per train	40 to 44

Ore Crushing (Plant)

Bin capacity	10,000 tons
Two-stage crushing	Two 7-foot standard Symons; four 7-foot short-head Symons
Capacity	2000 tons per hour
Fine ore bin	45,000 tons capacity

Concentration of Ore

Concentrator	43,000 tons per day capacity
Rod mills	Ten 10' x 12' rod mills; primary grind in open circuit
Secondary grind	In closed circuit with 20-inch cyclone classifiers; twenty 10' x 10' ball mills with twenty sets (80) 20-inch cyclone classifiers
Flotation	940 48" mechanical cells
Concentrate regrind	Four 8' x 12' ball mills
Cleaner concentrate	900 to 1000 tons of concentrate to molybdenum plant for recovery of molybdenum
Molybdenum plant	Products--92% $MoS2$ concentrate ready to market and 28% copper concentrate to smelter concentrate bins
Gold plant	Products--Precipitate with gold assay of more than 1000 ounces per ton, and silver assay of more than 1500 ounces per ton; fed to smelter holding furnace, and blended with anode copper

Smelting of Copper Concentrate

Copper concentrate	28% final copper concentrate to one of two natural-gas fired, side-feed, reverberatory furnaces, 32' x 100'
Reverberatory products	Matte at 32% to 35% copper Slag, to slag dump Waste gases--About 50% of contained heat recovered by two waste-heat boilers each furnace. Flue dust recovered from gases by electrostatic precipitator before entering 20' x 515' stack.
Matte to converters	Three 13' x 30' Pierce-Smith type converters
Converter products	Slag, return to reverberatory furnace Waste gases, join reverberatory waste gases to electrostatic precipitator and stack Blister copper, delivered to anode holding furnace where it is blown and poled prior to casting into 735-pound anodes for shipment to electrolytic refineries.