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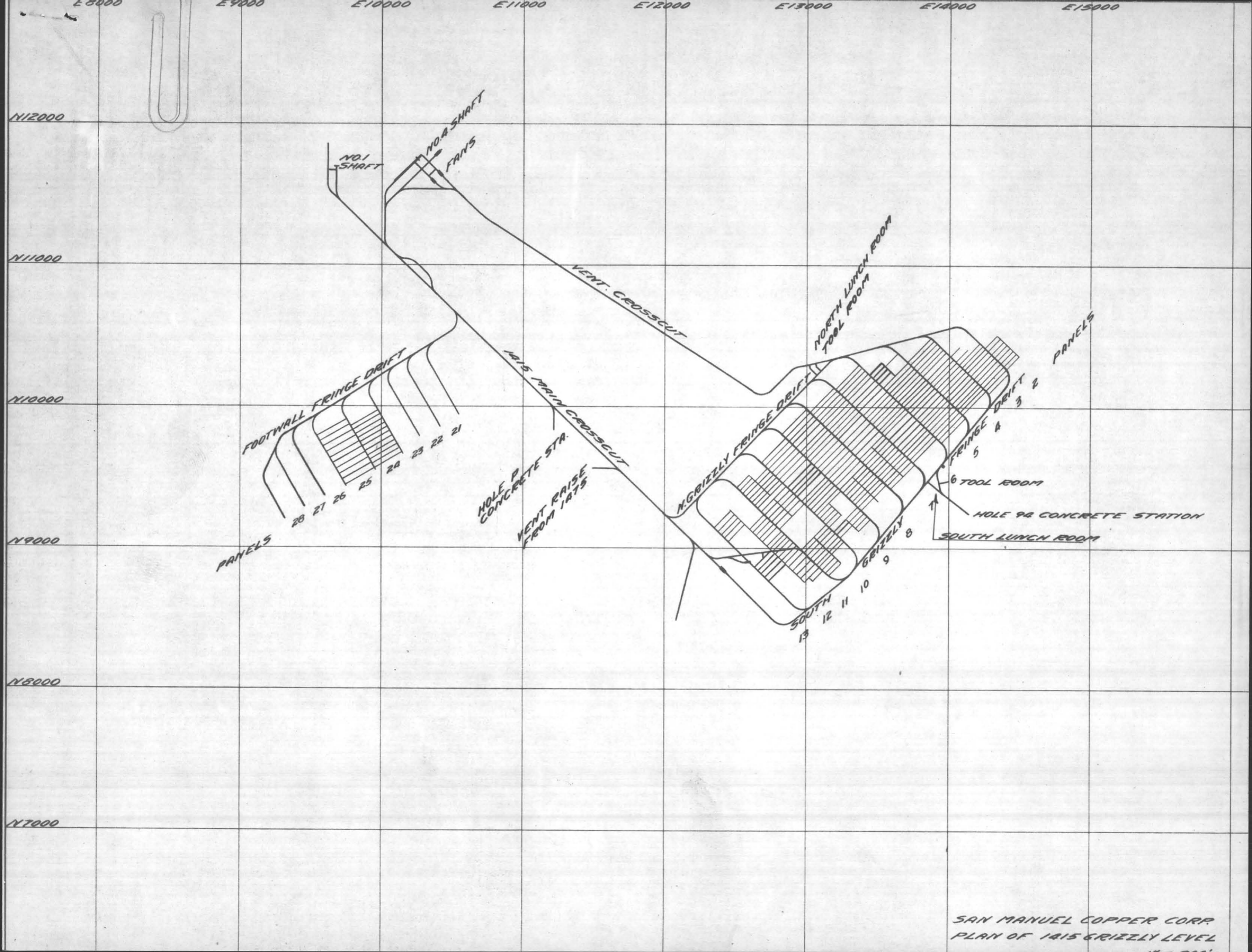
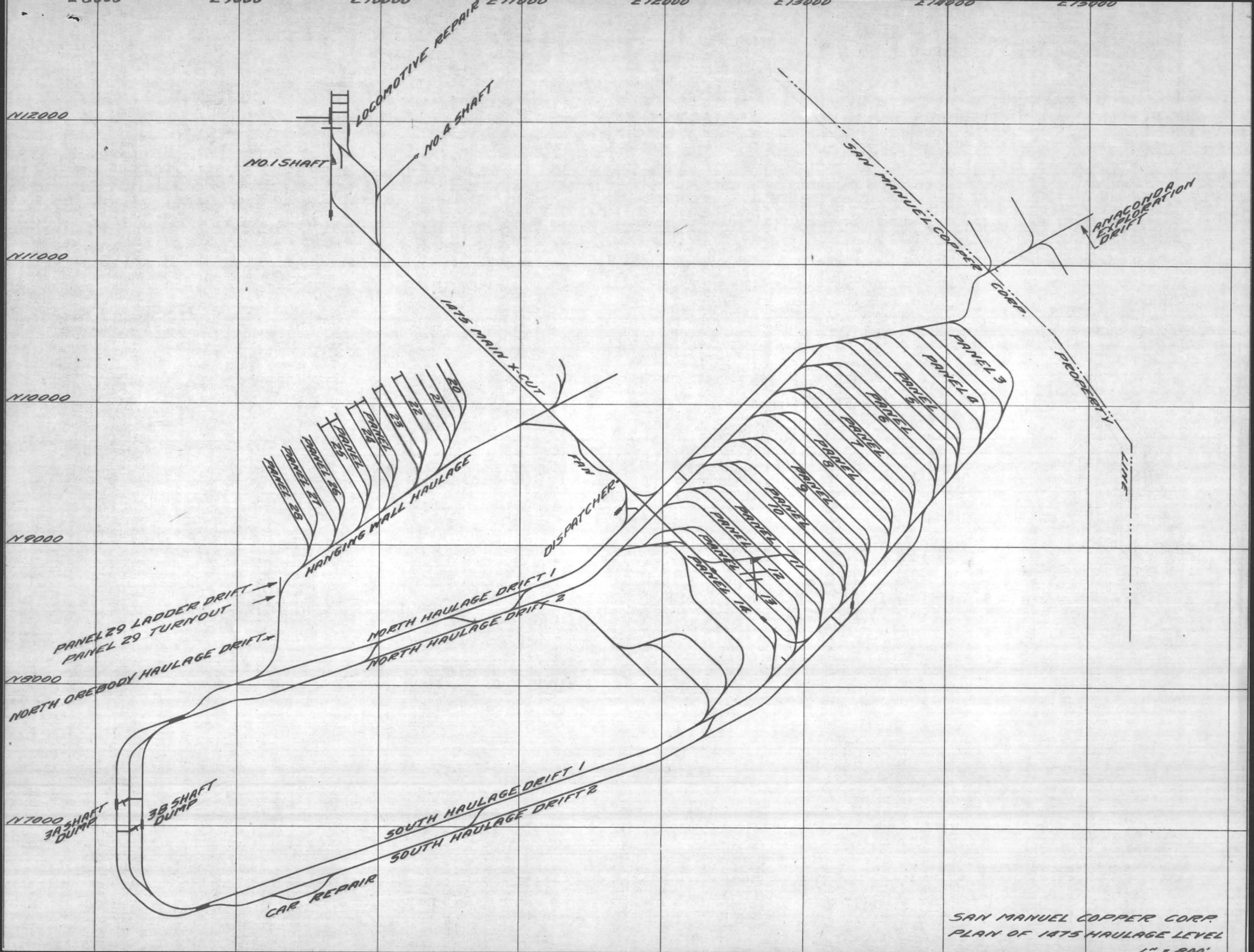
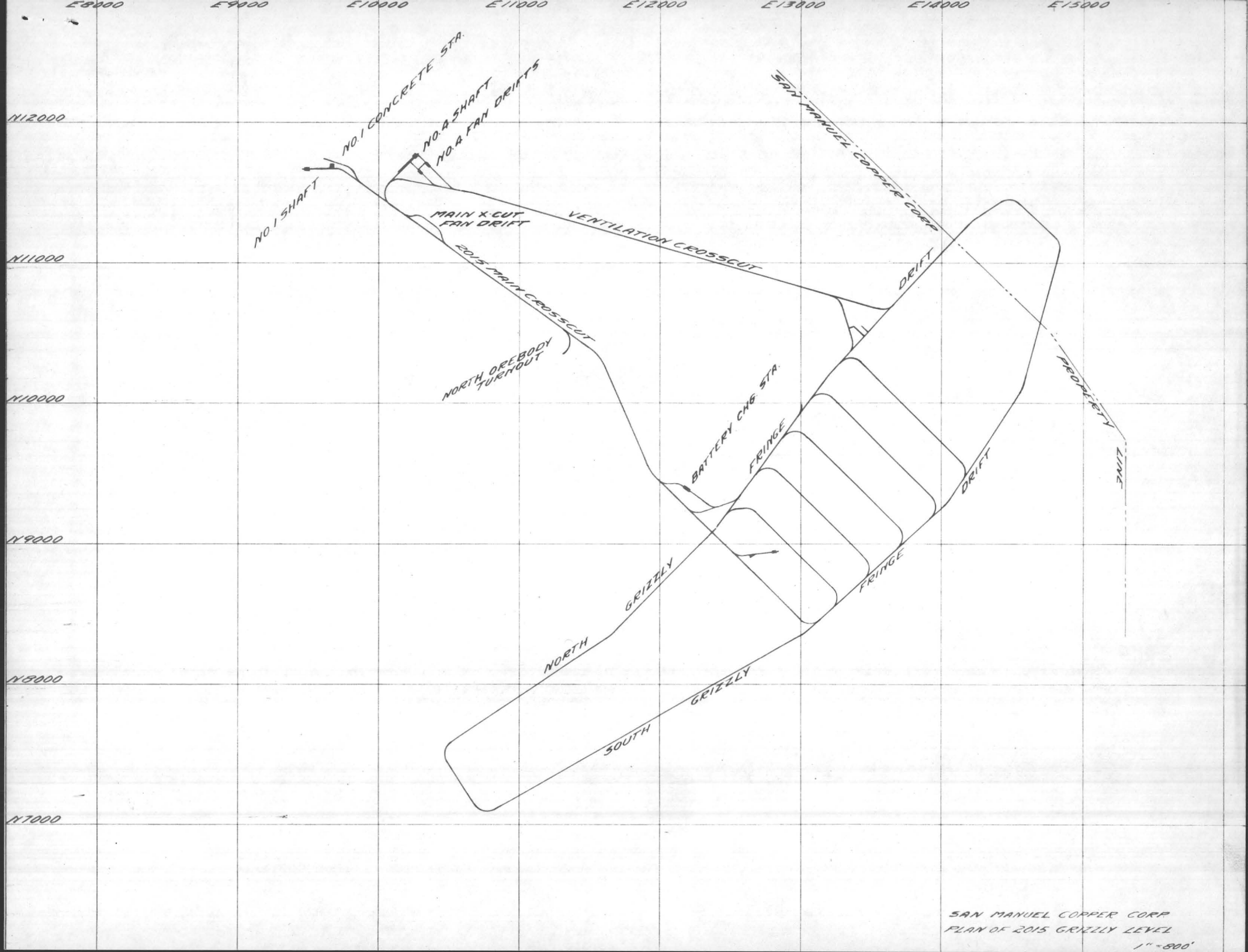


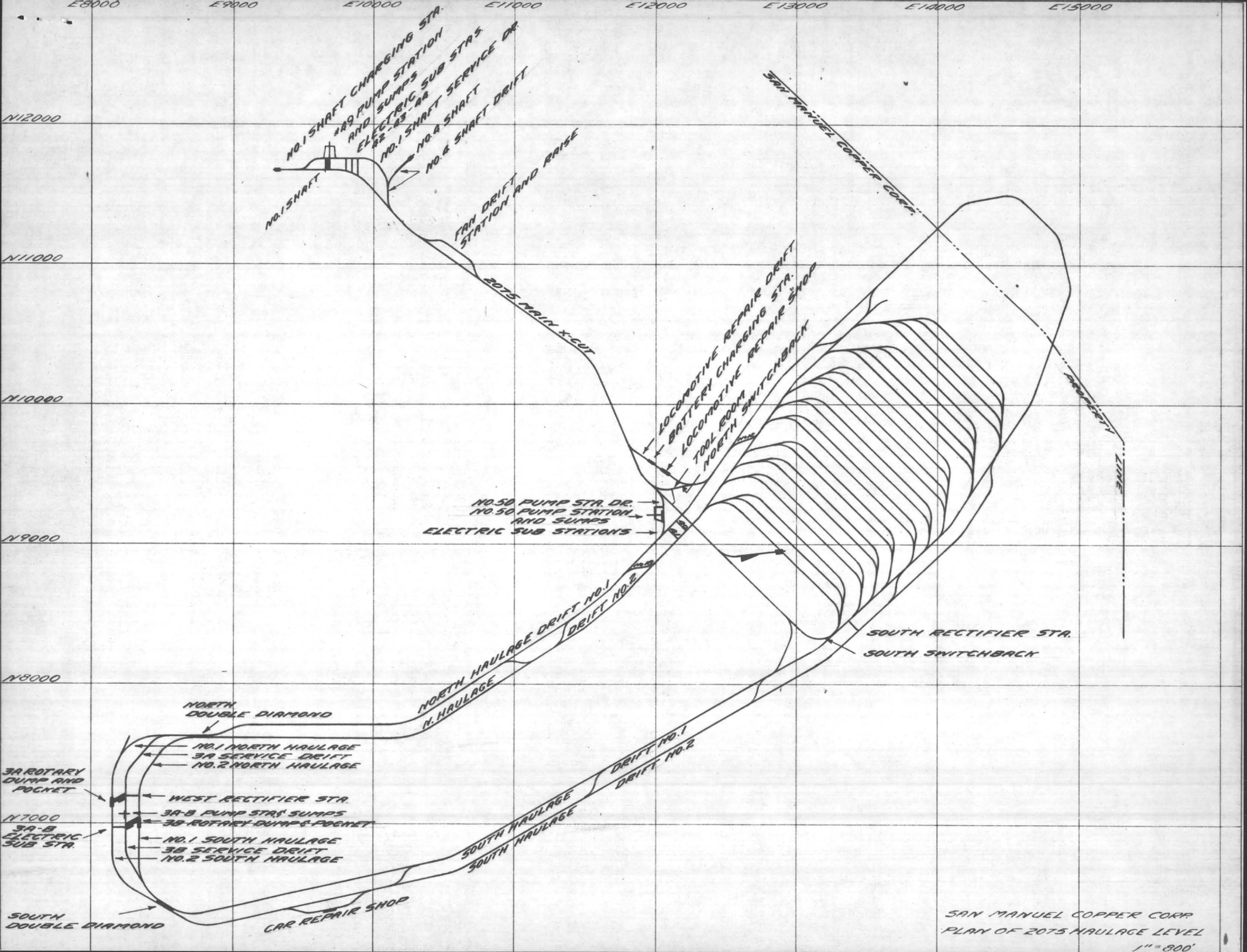
SAN MANUEL COPPER CORP.
PLAN OF 1415 GRIZZLY LEVEL



SAN MANUEL COPPER CORP.
PLAN OF 1475 HAULAGE LEVEL
1" = 200'



SAN MANUEL COPPER CORP.
PLAN OF 2015 GRIZZLY LEVEL
1" = 800'


E8000
E9000
E10000
E11000
E12000
E13000
E14000
E15000
N12000
N11000
N10000
N9000
N8000
N7000
NO.1 SHAFT CHARGING STA.
#49 PUMP STATION AND SUMPS
ELECTRIC SUB STAS.
#2-43-44
NO.1 SHAFT SERVICE DR.
NO.4 SHAFT
NO.4 SHAFT DRIFT
NO.1 SHAFT
FAN DRIFT STATION AND RAISE
2075 MAIN X-CUT
SAN MANUEL COPPER CORP.
PROPERTY LINE
LOCOMOTIVE REPAIR DRIFT
BATTERY CHARGING STA.
LOCOMOTIVE REPAIR SHOP
TOOL ROOM
NORTH SWITCHBACK
NO.50 PUMP STA. DR.
NO.50 PUMP STATION AND SUMPS
ELECTRIC SUB STATIONS
NORTH HAULAGE DRIFT NO.1
DRIFT NO.2
SOUTH RECTIFIER STA.
SOUTH SWITCHBACK
NORTH HAULAGE
N. HAULAGE
NORTH DOUBLE DIAMOND
NO.1 NORTH HAULAGE
3A SERVICE DRIFT
NO.2 NORTH HAULAGE
3A ROTARY DUMP AND POCKET
WEST RECTIFIER STA.
3A-B PUMP STAS. SUMPS
3B ROTARY DUMP & POCKET
3A-B ELECTRIC SUB STA.
NO.1 SOUTH HAULAGE
3B SERVICE DRIFT
NO.2 SOUTH HAULAGE
DRIFT NO.1
DRIFT NO.2
SOUTH HAULAGE
SOUTH HAULAGE
SOUTH DOUBLE DIAMOND
CAR REPAIR SHOP
SAN MANUEL COPPER CORP.
PLAN OF 2075 HAULAGE LEVEL
1" = 800'

April 2, 1958

Mr. F. H. Buchella, Gen. Manager
San Manuel Copper Corp.
San Manuel, Arizona

Dear Mr. Buchella:

On behalf of all of the fellows of the Anaconda Exploration and Geological Departments who were at San Manuel recently, I want to thank you and your staff for your very kind help and complete cooperation with us. We left San Manuel with a great respect for the engineering and geological work that has been done, and for the painstaking ore reserve calculations that have been made. We all appreciate the big job that has been accomplished to bring the San Manuel deposit into production, and we hope there will soon be an improvement in the price for copper which will give all of us some needed encouragement.

With sincere regards,

Yours very truly,

Roland B. Mulchay

RBM:S
bcc-Mr. J. L. Kelly

April 2, 1958

Mr. H. J. Steele, Chief Engineer
San Manuel Copper Corp.
San Manuel, Arizona

Dear Hugh:

We want to express our great appreciation for your kindness and the complete cooperation you and your staff gave us during our recent work at San Manuel. Certainly our task was made much easier by the painstaking and detailed work you have directed there, and we finished with a very high respect for your ore reserve calculations. We are very pleased to have had them made available to us.

I was sorry not to have been able to see you last week when I was in Tucson. I finished with only a few days there when I had planned on more than a week. Perhaps we can get together for dinner the next time I am in the Southwest.

Kelly, Wilson, Thompson and the rest of the fellows join me in sending our thanks for all the nice things you did for us.

With sincere regards,

Yours very truly,

Roland B. Mulchay

RBM:S

bcc: Mr. J. L. Kelly

April 2, 1958

Mr. J. D. Pelletier, Chief Geologist
San Manuel Copper Corp.
San Manuel, Arizona

Dear Jack:

All of us who worked on the San Manuel-Houghton examination want to thank you for the great help you gave us, and to congratulate you and your associates on the fine engineering and geological work you have done. We were most fortunate to have had your ore calculation figures made available to us.

Please convey our thanks to Thomas and Gibbs, to Richards, and to the rest of the fellows who helped to make our stay there so pleasant. I hope we can get together on one of my trips to the Southwest, and I will expect to have you call us if you wander into this vicinity.

With sincere regards,

Yours very truly,

RBM:S

Roland B. Mulchay

April 2, 1958

Mr. C. L. Pillar, Mine Supt.
San Manuel Copper Corp.
San Manuel, Arizona

Dear Chuck:

We certainly appreciated your help and kindness to us during our recent work at San Manuel. Our job was made much easier by the complete cooperation we obtained from you and your staff, and we want to congratulate you on the work being done to bring the tremendous San Manuel deposit into full production.

I hope you will be sure to call me if you are in this part of the world, and we can review socially some of the events since you were in Cananea!

With best regards,

Yours very truly,

REM:S

Roland B. Mulchay

FORM 1168

MEMO

To BARBER

Date 6/15/61 Time 9:40 AM

Phone from

To

Place

Subject Haughton

Present

Expect to have 1570 el. figures by 10:30 AM and will call

11:10 AM. Barber called.

		Total	ox. Cu
1570 main min	9,741,000	0.87	0.6% cut off

11:20 AM Figures given to VDP in N.Y. by telephone.

VDP

At Hall 201 xc good place to take a sample.

SOT to Anaconda.

Carl Taylor

FORM 1168

MEMO

To Barber

Date 5/17/61 Time 4 PM

Phone from RBM

To

Place

Subject Haughton

Present

Will have Haughton work out by end of week. Will send maps for work next week.

We can visit Lakeshore on May 29 — he will make trip there next week. Mrs West says original price of 3 million still stands, but period for investigation of mt., drilling, mapping, would be granted.

FORM 1168 6-8-54 5M

MEMO

To Barber

Date 3/7/61 Time 10:30 AM

Phone from

To

Place

Subject Houghton, Copper Butte.

Present

Call Kelly

Min Inspector Mar 14 – 15 – 16 – 17 –

— —

occupation tax

— —

Net proceeds tax

Assessment on state leases – get started as work must be done by May 21 and July lease dates.

Isabelle.
Missing MPRs

He will inquire about 4 wheel drive Chev and GMC.

He will call Reed and offer to supply drill records on Copper Butte holes.

BARBER BY PHONE JAN. 1961

E-W SECTIONS HOUGHTON

Total

16,230,257 TONS @ 0.82% Cu with 0.07 ox Cu.

FORM 1168 6-8-54 5M

MEMO

To RBH

Date 12/27/60 Time 10:40 AM

Phone from Harlin

To

Place

Subject Haughton, Mt. Wheeler

Present

Has reviewed Haughton data – believes no further underground work needed. – Will send letter to S.H. to that effect and terminate contract. Has given Harlin three undercut levels to figure ore reserves, 2075, 75 ft. above 2075 and 1775 – also to use both 0.5 and 0.6% Cu cut offs. He will have Hoffman and other review mining figures.

Centennial letter – believes their estimate reasonable – leave determination to Tony, Raymond, – myself

Write up contract + spell out equip. to be provided by ACM – Specify entitled to be charged by Centennial, to Anaconda, for use of this equipment, as well as any charged for their own equipment. Thus, charges will reflect cost of use of all equipment – with Centennial being charged by Anaconda on books for use of A. equipment, and corresponding charge back to A. by Centennial.

Harlin in Tucson until January 4 – Will be in S.A. three weeks – Betty to remain in Tucson for two days longer.

FORM 1168 6-8-54 5M

MEMO

To Barber

Date 12/16/60 Time 2:45PM

Phone from RBM

To

Place

Subject Houghton

Present

They will have maps and sections and one estimates ready for review by Nicolin at Christmas time. Have discovered evidence of fault parallel to Vent Raise about at N. limit of one zone.

Talked to Thomasson about work on Bed after first of year. He will be available and wants to come up.

Barber will try to communicate with Jim Mulchay about bringing truck up about Dec. 21.

HOUGHTON 12/7/60

SEC. 1100 – 1700 S.

15,690,000 @ 0.79% Cu

SEC. 3000 – 3600 E

15,665,000 @ 0.80% Cu.

NORMAL UNDERCUT ABOVE 2075

86' ELEV. 1383'

UNDERCUT 77' HIGHER AT 1460' MIGHT BE FEASIBLE.

ESTIMATES MADE IN

TUCSON 12/6 – 12/7/60

cost — estimate 12/7/60

1.15%

250,000,000

75,000,000 ✓

30¢ ✓

28¢

0.7% ✓

.2 ✓

.6.

.55

28¢ ✓

$10.00

4%

28% cons.

560

56

500

5

25.00

5

.78

13.5

5

.675

15

90

13.50

13.5 lbs

92

270

1215

12.420

.70

12.5# over all.

.28

5¢ milling

3 F+R

5¢ smelt.

2¢

15¢

13¢

13.5

13

1.75

.50

.75

1.25

2600.

200

800,000

500,000

100,000

1,400,000

700

10,000

7

FORM 1168 6-8-54 5M

MEMO

To RBM

Date 11/22/60 Time 2:15PM

Phone from Perry

To

Place

Subject Houghton drilling

Present

Houghton —
Tacked of XC extension over hole 41 — told him I did not believe advisable.
N of hole 11.

He agreed that underground work should be stopped.

FORM 1168 5M

MEMO

To RBM

Date 11/14/60 Time 4:30 PM

Phone from BARBER

To

Place

Subject Houghton

Present

48

593' drilled 1' today —

Prof. paper ~~Bull~~ 400

502 – 510 alt g mass FeS_2 sec. $Cu_2S^{(-)}$

– 532 oxid. no Cu_2S – shearing.

– 563 weak $Cu_2S^{(-)}$ on FeS_2 less than 0.5%

592' oxid. again.

N.L.
433-63-84

FORM 1168 6-8-54 5M

MEMO

11/28/60 8:30 AM

To RBM

Date 11/18/60 Time PM

Phone from Barber

To

Place

Subject Haughton

Present

#48 657' oxidized from 562

alt. clay from 562'

631 one inch gouge —

643' 2' broken material + gouge. 60°

all limonite — no FeS_2 or copper.

6.5 – 83.8	77.3	Total 1.22	0.48
83.8 – 127.0	43.2	1.16	0.05
– 152	25.0	0.219	.01
– 211.8	59.8	0.886	.03
– 518.5	306.7	0.229	.01

(43' today + tomorrow)

11/28/6 #48 at 696'

Through last strand of fault at 660'

From 660 - 661 little $CuFeS_2$ on FeS_2

661 – 696 Fresh 9' monz. with little sec. and clay alteration, few seams and dis of FeS_2 — no $CuFeS_2$

Can drill hole at – 25°

Hole at + 10° would cut above present hole ± 240 ft. on HW side of last fault — that is above FeS_2 min.

Houghton Drilling
Sept. '60

1324 ft – $7,568.61

$5.72 / ft.

FORM 1168 6-8-54 5M

MEMO

To RBM

Date 11/11/60 Time 3 PM

Phone from Barber

To

Place

Subject Houghton

Present

48

Assays to 191. ft.

		ox
0 – 6.5	7.16	.485
6.5 – 83.8	1.409	1.216
83.8 – 191.8	.84	low ox.

same Cu_2 with 0.3 zones.

191.8 – 402 } – same Cu_2S – average less than .85
418 } Fault

402 short sect. of

532' this morning – Cu_2 coatings to 502' – 0.6%

510 – 518.5 oxidized
no clays recovered.

502 – 510 no copper, just FeS_2

FORM 1168 6-8-54 5M

MEMO

To RBM SLC Date 11/10/60 Time 10:20 AM.

Phone from Barber To

Place Subject

Present

Haughton

48 485' 402 - 417' fault. zone, no good dips

into qm at 402 alt. little CuS_2 on FeS_2 veck dis.

402 - 485 less CuS_2 some FeS_2 increased sericite.

Corresponds with #46. - few short sections of mon. porphyry.

S.M.

Develop cut off grade 0.6% Cu or 0.5%.

FORM 1168 6-8-54 5M

MEMO

To RBM
Date 11/4/60 Time 3 pm.
Phone from Barber
To
Place
Subject Houghton.
Present

#48 345' −14° 338' start of fault zone
Spotty Cu_2S up to 297'; after that lean sulph. min, little oxidation.
strong oxide D 153; weak oxide past that.

Hole at Barrett Prospect by Jim Cazier
150' W of shaft. (vert. from surface)
−45° due E, seems to be parallel bedding

1300 + 1700

Runkel- prospect near Tyrone 10 mi NW.
feldspar - porphyry with ox. - sulph.

Amer. Metal Climax dropped claims S. of Tucson
3 holes 1100' bedrock. - Now held
by Bear Creek.

Pierce D. Parker - 7049 E. Luana Drive.

Duval has orebody at Mineral Park
Bear Creek at Copper Creek drilling

FORM 1168 6-8-54 5M

MEMO

To RB Mulchay | Date 10/11/60 Time 2:PM
Phone from Barber | To
Place | Subject
Present

#47 — 128' this morning N75°E + 3°

alt. iron ox st mong. por. — all oxidized.

43-51' fault zone.

He will lay out hole to SE of DDH 46, to go into HW of Cholla, and will check on size of station. Will call me tomorrow afternoon.

No trips at GSA meeting; told him to attend for 1st two days if he wished.

Has additional infor. on Cornell gallium property. Will make copies — of no interest to us by Perry + Newlin.

Imes equipment will be returned to them — has not been charged out.

Call from Barber 2:30PM 10/28/60

Sunday afternoon + return Wednesday.

Coords 11,271.8 N
14,868.5 E
El. 1901.06
S 74° 17' E
Incl. ~~-34°~~ ~~wrong~~ (15°)

#48 130.7' alt mong. por. 0-87 ft. seams Chrys. alt por. with FeOx,
87-107 - Cu_2S better than in sect. 0-87
107-115 - heavily ox. with small clays 55° with core.

#47 0-147 0.07% Cu
below 147- ±.02

FORM 1168 6-8-54 5M

MEMO

To RBM

Date 10/19/60 Time 11 AM.

Phone from Barber

To

Place

Subject Houghton

Present

47 slam 251' fault 212 to ± 235' - in second zone. alt mag per with FeO_x - no sulphides.

Tomorrow to SM - Friday Ajo - Aut Silver Bell

Borchardt, Stewart, Hoffman.

FORM 1168 6-8-54 5M

MEMO

To Barber
Date 10/8 Time 11 AM
Phone from RBM
To
Place
Subject
Present

Finished 46 at 502', Through fault dark mag.
in fault zone to 405
405-437 g-m with pyrite weak
Cu_2S weak
FeO_x some cone'
480 mag weak FeS_2 trace ox.
tr. Cu_2S on FeS_2
484 gm FeS_2 ox. to FeS_2
502 gm pyritic min,

47 New hole N75°E. flat up.

Hoffman arriving even of Sept 26, there
until 22nd — at symposium and trips to open pits.
Then to Cananea.

Pickard starting salary — $500? Talk
to Kelly.

Kenn. to drill at Copper Creek.
Told him to investigate Huston prospect N. of
Cane Creek if drilling abandoned.
U.S. drilling hole at Fierro near old Hanover shaft.

FORM 1168 6-8-54 5M

MEMO

To RBM

Date 10/3/60 Time 3:25 PM

Phone from Barber

To

Place

Subject Houghton

Present

May get out core barrel in DDH 46.

Geology — oxid. — from 357' to lower fault — very little FeS_2 left. —

379 1/2 8" brec. rock with clay + dip 55° with core.

386 2" zone of clay + gouge

391 – 94

396 1/4" clay – 55°-60° with core.

398 1/2 2-1/8 clays make contact with monz. per. highly altered – purple red iron oxide color.

401 – 2 6" brecc. monz. per.

FORM 1168 6-8-54 5M

MEMO

To Barber | Date 10/13/60 Time 4 PM

Phone from RBH | To

Place | Subject Houghton

Present

DDH 47 at 162' - hole caving and will be reamed.
Told him to continue well into HW of HW strand of Chollar.

DDH 46.

0 - 183	0.178	0.094 ok
183 - 201.5	1.869	0.060 "
201.5 - 465	0.097	0.013 "

Told him hole 375 at -45° from DD Sta 7. would be OK -- engineers have already set lines.

FORM 1168 6-8-54 5M

MEMO

To RBM

Date 10/12/60 Time 11:10 AM.

Phone from Bowden

To

Place

Subject Houghton

Present

S70°E + 5° to −20°

at −20° would be about 130 ft below level.

360' to intercept main strand.

± N 11270 E 14870

#47 152 ft. still in mossy por with weak FeOx
no sulphide.

N75°E + 2° sup. El. 1904

10/13 162 ft. in AM.

#46 183 − 201 ± 1½ − 2% Cu-S.

S75°E 400' to fault at
120'
−18°
205' S. of #46

FORM 1168 6-8-54 5M

MEMO

To RBM
Date 10/1/60 Time 1:40PM
Phone from Barber
To
Place
Subject Haughton
Present

91

N 11367.86 S88° E
E 15021.81 -44°

0-167 m.p. oxidized

DDH 46 366' Fault 167-183' good strong fault El 1900.73

½" clay ¼ dips 40° with line of hole

all in qtz. monz. past 10'-15' alt qtz. mon with fair Cu_2S.

195-366' leached alt qtz. monz. with 20' sections of iron oxide, remainder strong FeS_2 with sericitic alteration.

will drill today

		Tot Cu	
DDH 45	0-38'	0.189	.100
	195	1.119	.022
	595	0.22	

10/2/60 #46 410' 405 core to last Cu_2S at 357 but very weak

Fault zone at 390' strong zone with breccia and gouge - oxid. - In fault zone q.m. with iron oxide.
qtz. monz. with FeS_2 - mostly oxidized, no $CuFeS_2$ or Cu_2S.

traces of $CuFeS_2$ to 260'

S30°E - 50 500' at present station.

in DDH 35 short sections - flat to S.

DAY
Lo 3-2924
Finer
546'

(35)

S60E
-10°
500'

No-RBM

FORM 1168 6-8-54 5M

MEMO

To RBM

Date 9/26 Time 8:35 AM

Phone from Barber.

To

Place

Subject Houghton

Present

DDH 43	0-265	0.094	0.035
	428	.706	.026
	505	.231	.012
DDH 44	0-67.3	0.752	0.010
	470	.374	0.011

up to 20' of .6+

45 1% - 180'

0-38	mon p.	oxide - Cu_2S
79	" "	no oxide - Cu_2S
168.5	g. m.	spotty Cu_2S
378	m p	$CuFe_{3-}$ - $Cu_2S^{(--)}$
674	m p.	little ox. weak pen. min.

to 360 ± 0.4 - 0.5

46 Collared due E. - 45°.

S. Manuel

Aug 60

38	T31		
	132		
		263 NX	
		141 BX	
			45
			404
39	NX		64
			5
40	NC		66
	NX		123
	BX		
			4
41	NC		235
	NX		246
	BX		13
42	NX		149
	BX		10
43	NC		231
	NX		154
	BX		
			1704

44

HC	10
NX	180
BX	66
	256
	1704
	1960

$ 5.56 / ft.

1960 | 10906.41
9800
11064
9800
12641

FORM 1168 6-8-54 5M

MEMO

To RBM
Date 9/13/60 Time 2 PM
Phone from Barber
To
Place
Subject Haughton
Present

#43 – to 395' assays

265 – 367.5 .767 Total 0.025 ox.
less than 50% core

395 lower grade.

Finished at 505'

Recd. for new hole – S 80° E – 45° new hole from end of main XCNE

#45 looks good to 210'. Finish next week.

For hole #46 (new rec.) will need to enlarge drill station – he will write letter to San Manuel.

Asked him to send Barrett pulps (N.M. Prospect) and sketch map with assays for me to show to Perry last week in Sept. Also short description of property.

505

Carl Lockers Westinghouse
EL-5-3413

9/3/60 Barber will recommend hole to E -45° from end end SW min XC. 500'

Possible extension DDH 29 after completion Rec. #22

Dependent on results hole Due E from E. end, plan hole to SE.

Make one estimate on section both ways.

HOUGHTON PROJECT

AUG 24 1980

DDH No.	Coordinates North	Coordinates East	Collar Elev.	Bearing	Inclination	Final Depth
36	11,509.25	14,449.70	1899.83	N 35° 29' E	-59° 34'	565'
37	10,940.13	14,864.80	1900.05	N 25° 46' W	-73° 45'	500'
38	11,329.51	14,543.86	1898.26	—	-90°	523'
39	10,925.88	14,868.66	1900.10	S 32° 36' E	-50° 11'	248'
40	10,924.37	14,869.26	1903.99	S 27° 36' E	+10° 21'	194'
41	11,391.60	15,029.54	1901.61	N 16° 38' E	-40° 23'	485'
42	10,934.11	14,863.75	1923.82	—	+90°	
43	APPROX 11,392	15,027	± 1901.6	± N 30° W	± -50°	

DDH 41

0-146'- Altered monz. por.; iron oxide, local $CuSiO_3$

146'-168.9'- Highly altered, crushed, sheared qtz. monz. & monz. por.

168.9'-207.5'- Highly altered, fine grained monz. por. (?); iron-oxide stain

207.5'-317'- Altered qtz. monz., short sections monz. por. strong hematite stain; sheared at 280'-300'

317'-418'- Altered qtz. monz.; abundant pyrite, irregular chalcocite coatings on pyrite; some hematite; sericite alteration

418'-485' - Altered qtz. monz.; feldspars alt. to sericite, clay; local strong hematite; fault zone from ± 436'-439'

AUG 24 1960

HOUGHTON PROJECT

	Interval (ft)	Feet	Inspiration Total % Cu	Inspiration % Oxide Cu	San Manuel Total % Cu	San Manuel % Oxide Cu
DDH 28	0 - 45.1	45.1	0.427	0.0128	0.427	0.020
	45.1 - 121.8	76.7	0.681	0.0214	0.647	0.020
	121.8 - 148.0	26.2	0.412	0.010	0.521	0.029
	148.0 - 253.0	105.0	0.677	0.0126	0.655	0.0198
	253.0 - 321.5	68.5	0.707	0.0170	0.657	0.031
	321.5 - 363.0	41.5	0.512	0.010	0.533	0.082
	363.0 - 395.0	32.0	0.378	0.013	0.433	0.045
	395.0 - 440.0	45.0	0.052	0.010		
DDH 37	0 - 115.4	115.4	0.428	0.010		
	115.4 - 170.0	54.6	0.612	0.010		
	170.0 - 238.9	68.9	0.428	0.010		
	238.9 - 280.4	41.5	0.672	0.010		
	280.4 - 369.3	88.9	0.388	0.010		
	369.3 - 500.3	131.0	0.139	0.010		
	0 - 280.4	280.4	0.500	0.010		
DDH 38	0 - 156.2	156.2	0.919	0.012		
	156.2 - 246.8	90.6	0.617	0.010		
	0 - 246.8	246.8	0.807	0.011		

(assays intermittent to 276.0' are between 1.00 + 1.88 % Cu)

FORM 1168 6-8-54 5M

MEMO

To RBM

Date 8/24/60 Time 3:30 PM

Phone from Barber

To

Place

Subject Houghton assays

Present

# 38	0 - 313.5	0.826 % total	0.01 % ox Cu
	523.0	0.241 total	0.010 ox. "
# 39	0 - 241.2	0.307	0.010 ox

43 at 241' in gr. some Cu_2S on FeS_2 spotty Cu ox not encouraging.

44 + 20° up from 1st station.

42 162' completed does not look good

41 cut Vent Rainc fault at 153 ft. whole [illegible] weak

43 " " " " 134-136 ft and 140-142

some Cu_2S on far side of fault.

45 from end 2nd station - new rec. # 22.

Some kernel oxide assays - high

FORM 1168 6-8-54 5M

MEMO

To Barber

Date 8/22/60 Time 3:45PM

Phone from RBM

To

Place

Subject Houghton drilling

Present

41 — completed at 485' fault zone at 430 in straight oxide, no Cu.

Probably nothing to represent section in 9

Rec. # 22

He will send assays on 37 + 38, call me on Wednesday with further results.

Houghton June 1960

28	63
30	27
31	588
32	526
33	431
34	288
	1923

$5.99

11519.49
9615
19044
17307
17379

FORM 1168 6-8-54 5M

MEMO

To Kelly
Date 7/7/60 Time 2:40 PM
Phone from RBM
To
Place
Subject Houghton
Present

Suggested he review Houghton Rec. #14, possibly steeper to 55° + to keep in FW of Cholla Fault, as projected from XC through probable position at level elevation in CDH 15.

He will have Barber work on Sect. 3400 NE and long section through main crosscut.

Drilling going OK — told him to keep after drillers to keep core recovery good — in DDH 32 from 306-407 only 60% and DDH 33 0-118 ft. only 59%. He has brought this to attention of Mazzero, and Barber + Thomsen will continue to emphasize.

Told him to tell Byrd that we cannot go into any campaign to acquire ground in N. part of Hanover Dist. If he and Burney have anything definite to present, we will consider. U.S. Smelting holds dominant position, and probably impossible to deal with them on reasonable basis at this time.

He will leave tomorrow, be here Sunday night.

FORM 1168 6-8-54 5M

MEMO

To RBM
Date 6/6/60 Time 9:10 AM
Phone from Kelly
To
Place
Subject
Present

Room 221 151 So Tucson Blvd.

Tel Ea 5-6067.

30 20° NW XC #1 365 ft. mixed oxide + Cu_2S.

28 NE drift AX drilling slowly down to 395'

31 vert down at ste in mid. of XCNW #1 just started.

AD Thompson — familiar offset of Miami fault. says he used to work for ACM — Kelly referred ~~to~~ him to Inspiration.

Amer Nat Climax 3 holes

1 — overburden 1130'

2 — bed rock ("dike" rock) 1100 ft

3 — overburden 1100' still going —

Vernon Smith Bayless driller.

750-800 cons. sand, and back into Cg.

DDH 26 Houghton

59"	0.57	34.81
36	1.05	37.80
55	0.66	36.30
60	0.57	34.20
210		143.11

$$
\begin{array}{r}
.681 \\
210 \overline{)143.11} \\
1260 \\
\hline
1711 \\
1680 \\
\hline
310
\end{array}
$$

Kelly — Sl. assays #25

322 — 700.

FORM 1168 6-8-54 5M

MEMO

To Kelly

Date 5/19/66 Time 10: AM

Phone from RBM

To

Place

Subject General

Present

Drilling going good

28 278 looks pretty good.

29 -55° 129

27 -90 515

Average 6-7' to 256'

into sil. at 50'

out at 450' ±

Pilot hole - 169' will be drilled to 250.

Drill F at 122-125'

From surf to inter. ±61° E.

Have talked to Williams - hesitant to take job - Kelly believes he wants to continue teaching.

Barber has sent letter on Bear Creek and other properties with map.

FORM 1168 6-8-54 5M

MEMO

To Barber

Date 5/2/60 Time 3.40

Phone from RBM

To

Place

Subject Houghton

Present

Barber believes that with one more man, they could handle 2 machines at Houghton without Kelly.

Told him M + H would want to review Three R area maps, — Barber says Clark still working for Bear Creek.

Bear Creek has taken up Anglin property and large area to N, extending to E. of Twin Butte road.

FORM 1168 6-8-54 5M

MEMO

To RBM
Date 4/22/60 Time 2: PM.
Phone from Kelly
To
Place
Subject
Present

DDH 3 700' 485'

-15°

Low grade 200-350' fair to good.
Sludge to 196. 0.5 +
To 478 77%.

2nd machine hole above #3

Advised him to move to N end of 1st NW crosscut.
Had water in hole, did not use mud.

Discussed with Perry on mar. 1.

Got approval for drilling with 2 machines.

Pickard — 400-$450 Raise.
Write Perry.

Patent proceedings Mohawk Ext. vein shaft.

Ext. claim 280' deep. capped with concrete

Reid sent notes to SLC.

Forwarded to Kelly 4/22/60

Talk to Dwyer about total Haughton costs.

Dwyer — money for Haughton work

March 30th Pickard

FORM 1168 6-8-54 5M

MEMO

To RBM | Date 4/4/60 Time 3 PM

Phone from Kelly | To

Place | Subject Houghton

Present

Finished 2nd hole at ± 700. ft. last 70-80 ft. very low grade. One section above should be good.

Are ready to start -15° hole - Told him to start taking sludge, will revise when we hear from Penny. We sent letter from here today.

Asked him to contact Parker about their - I will return shortly.

Told him to look around for sampler if we need one for sludge.

FORM 1168 6-8-54 5M

MEMO

To RBH

Date 3/31/60 Time 2:45 PM

Phone from Park - Bayles Bros

To

Place

Subject Houghton Drilling

Present

Told him Kelly would recommend mud drilling for next hole; but we will have to get approval of NY and Sun Houston.

Cement time excessive, and sludge recovery poor in any case.

FORM 1168 6-8-54 5M

MEMO

To RBM
Date 3/30 Time 1:45PM
Phone from Kelly Tucson
To
Place
Subject Haughton
Present

482'

Lots of cement time, $1,400 in cement time

Core recovery in lower part 177-352 83%

better than that to present bottom.

0 - 80 0.63

80 - 105 .35

105 157 .57

157 - 252 .33

Advised him to write up problem

to send to N.Y.

$50 raise for Paul Pickard. Ohio State

has geol. degree.

Calculator for Tucson office.

FORM 1168 6-8-54 5M

MEMO

To Kelly | Date | Time 3:30 PM

Phone from RBM | To

Place | Subject

Present

DDH 1 600' stopped in very low grade.

DDH 2 177' at 45° to SE. going well NX.

Reid there, getting along OK.

Told him to give Barber suggestion on land report maps.

Discussed Evans-Byrd prospect in Coeur d'Alene district. Told him to tell Evans we would make reconnaissance, but he should submit map of location of prospect and report.

2/25/60

HOUGHTON

MAKE 1"=20' MAP OF LEVEL ASSAYS. PUT ON AVERAGES IF SAN MANUEL WISHES THEM

50 SCALE LEVEL MAPS AND SECTIONS WITH GEOLOGY. PUT ON AVERAGES OF 0.5 % CU AND ABOVE.

SEND COMPLETED DRILL LOGS TO S.L.C. TO BE INKED. ORIGINAL AND TWO COPIES TO BE RETURNED TO TUCSON, COPIES SENT FROM S.L.C. TO PERRY AND NEWLIN, COPY KEPT THERE.

SEND COPIES OF ASSAY MAPS AND SECTIONS TO PERRY, NEWLIN, S.L.C. ABOUT EVERY TWO MONTHS. HAVE GEOLOGY MAPS RETURNED TO TUCSON FOR POSTING ABOUT EVERY TWO MONTHS.

SEND OUT COPIES OF ASSAY CHECKS WITH SAN. MANUEL.

ASSESS. WORK AT HOUGHTON - NW GROUP - STATE CLAIMS.

letter written to Perry 3/3/60

FORM 1168 6-8-54 5M

MEMO

To Kelly | Date 2/29/60 Time 1:30 PM

Phone from RBM | To

Place | Subject Houghton

Present

Read Park letter and mine on Houghton drilling to Kelly — OK with him.

Hole 509 ft. — going well.

61.29
2.00

Told him Dodd would give M & H answer on Mar. 2.

FORM 1168 6-8-54 5M

MEMO

To Kelly

Date 2/22/60 Time 11 AM

Phone from RBH

To

Place

Subject Houghton, Miss.

Present

Houghton 1st hole
350' 70% ore. going alright.

Sam Mancel.
Buckeman - will assay MoS_2 samples.

Nothing new on Saginaw,

He will tell Garrow my plans.

I will call him from LA, and will expect to arrive in Tucson Wednesday night.

Kelly — assay chgs
from Inter to Butte
cut off or 20th of
month?

1 copy direct to
Butte — reach there
by 3 or 4th of month

give Kelly 3 copies
keep 1
send 2 here, one
approved copy for
Butte.

Call to Kelly PM 1/9/60

Truck driver
2 more copies of
rec. 150 holes do
not correspond to
~~to~~ June 5 letter.

FORM 1168 6-8-54 5M

MEMO

To Kelly
Date 1/4/60 Time 10:30 AM
Phone from RBM
To
Place
Subject Houghton
Present

$400 for sampler & trunk - on Boyles payroll.

I will talk to Stephens about Leigh, and let Kelly know - should bring car.

Thomason rent $75 plus part of utilities - told him to pay any part of utilities that would be fair.

sent He will check on "Red Hill" Pima Dist drill logs to Cooper.

Velasco had nothing on Saginaw.

Date of reinvestigations on outside of report.

Discussed Dick Williams - Williams to get Ph D in about one month - may want job. Kelly will talk to him, and ask Williams to write me.

I will discuss with Perry.

He will work out something to find out about Questa on find.

FORM 1168 6-8-54 5M

MEMO

To Kelly
Date 2/15/60 Time 3:40 PM
Phone from RBM
To
Place
Subject Houghton
Present

Hill Still in trouble - if not through tomorrow he will recommend drilling new hole. May drill with mud to present depth to test core recovery as compared with normal drilling. NW face about far enough. Below 231 ft. assays pick up to 0.8% Cu. increased $CuFeS_2$ - good checks with S.M. in upper part of hole.

Described Saginaw situation - told him we may have to go to Clark.

He will make reservation for Garnoe, and will talk to Buchella about MoS_2 samples. Will communicate with him about his letter.

Kelly 2/10/60

Trouble with hole 252' lost bit, lost top. have 2 bits.

Will try to drill ~~through~~ cemented section.

core

Assays to 184' — 0.29 % Cu.

sludge

to 100' check very closely.

Cu. to HF now 0.6 % oxide Cu.

Bear Creek taken up large block in Dragoon.

pts. — someplace near Tombstone.

Pickard. — truck driver — samples at Houghton

FORM 1168 6-8-54 5M

MEMO

To RBM
Date 1/25/60 Time 9 PM.
Phone from Kelly
To
Place
Subject
Present

Drilling Tot. afternoon 80' 75% locally 100%
low 33%

Difficult to keep up with sludge through autoclaves.
Nossig about 50-55% core recovery.
Will cut SS. station to handle sludge.
Mozzino and Ray Nelson drillers.

Thomasson mapped flat holes - Challe fault
at 530 ft.

NW XC should be fin. Mar. 1. Expects to
be fin. by time 3 holes drilled.

Bills to Kelly by 25th, will send one copy to
Butte directly, 2 copies here.

Smith has conferred with S.M. on oxide assays.

20767

Drill Holes 6/5

Assaying.

Samples

Assaying costs 20¢.
1 copy direct to Butte
2 copies here 1 + 1

Call to Kelly
1/9/60

FORM 1168 6-8-54 5M

MEMO

To Kelly

Date 1/20/60 Time 300 PM

Phone from RBM

To

Place

Subject General

Present

Increase approved.

Machine set up at Houghton yesterday, start today - problem to get core and sludge out of mine.

Pickard on Bayless payroll - start 16th at $400 /month.

FORM 1168 6-8-54 5M

MEMO

To Kelly

Date 12/17 Time 10 AM

Phone from RBH

To

Place

Subject Houghton

Present

Told him to negotiate for Houghton bldg at price of $150-250

Told him to talk to Reed, and tell him he could review reports of interest at Tucson. No written communication.

Asked him to determine ownership of child Aldridge and Old Reliable pipe for possible AEC reference

FORM 1168 6-8-54 5M

MEMO

To Kelly - TUCSON
Date 5/4/59 Time 3 PM.
Phone from RBM
To
Place S.L.C.
Subject
Present

Kaolin deposit in N.M.

40' shafts 800 Acres prospected

35% Al_2O_3 1% FeO_x 35-40% SiO_2

low $CaCO_3$.

Kelly will make reconnaissance after June 18th
Bagdad Trip 16 & 17 June.
Thomsen can go to coast any time.

Barber will decide about Canada trip by end of week.

Kelly will try to send Safford publicity to Perry.

He will talk to C. Smith about Houghton oxide co assaying.

San Manuel drilling scout hole for water ahead of face - no news on equipment yet according to Pelletier 5/3.

FORM 1168 6-8-54 5M

MEMO

To Kelly Tucson | Date 3/20 Time 8:45 AM.

Phone from RBM SLC | To

Place | Subject Houghton - rain

Present

Kelly has talked to all drill contractors - one bid received from Mettler Bros.

Advised him to get Smith and S.M. managers together to discuss outside assaying - possibly S.M. will be restricted to their former methods which apparently have approval of government agencies involved.

Told him we would send compressor if needed; he believes they may set up filtering station underground.

Asked him to get hotel reservation for Wednesday night, Apr. 1.

Asked for test forms.

Finished Park Hill mapping - no change in conclusions. Drill hole by Kern Co. under way no information being given out. Kern has taken up + recorded ± 300 claims to S and W.

Asked him to check on RBM field trip reservation to Pine Dist. for GSA meeting.

Tom Speaght. Supt Pr 4-2621

FORM 1168 6-8-54 5M

MEMO

To Kelly
Date 2/26/59 Time 10 AM
Phone from RBM SLC
To
Place
Subject Haughton etc
Present

ACM drift 175' from turn. — Will not start until contract signed.

Look up letter Jan. 8.

Bayles
Sp + Hurwood
Longyear
Metler
Jay
Ram Nosack.
Odgers.

Kelly will send out letter to drill contractors this week. Will ask for bids before Apr. 1. Does not expect drilling to start before 4/15.

He will determine whether or not he must check bills from S.M. — told him I believed he would only check advance for period of bill.

Told him to contact Cyprus on Old Dick samples, then dispose of them.

Told him not to give geologic information to Kern Co. Land. Mapping of Park Hill cores in progress.

Swenson + Cheriton left today for Canada. Visited San Manuel yesterday.

FORM 1168 6-8-54 5M

MEMO

To Kelly
Date 6/4/58 Time 11:30
Phone from RBM
To
Place
Subject San Manuel - Houghton.
Present

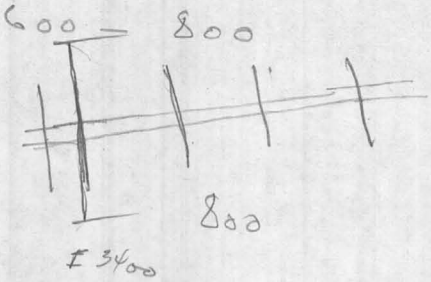
5-6 month before work will be started.
Says Newlin insists on driving XCs from main NE XC to drill steep holes thru one zone. Does not want to drill steep holes or in HW of Cholla fault.

No discussion of drill contract - Ramon Pampesch? San Manuel driller

Map pilot holes? Newlin plan requires twice as much XCing as original plan. @ t$70/ft.
Suggested we present info to Perry, and Perry give to Newlin.

Laundry lease on Houghton group. Ask Dwyer - Told Kelly to inform Lawyer we would need release from Mrs. Murray before lease was cancelled.

Says he contacted Bissett, who was leaving for Mexico. Says Bissett not impressed with P.C. offer, and said he had had better ones.



N 60°E on S 1700 Section

(1) 3700 E in HW of Cholla

(2) N 30 W on 3400 E 400 ft. drill down holes, and holes in FW of cholla. ± 200 ft W of Cholla. Fault at 1475.

3200 E well drilled out by Chum drill holes except

± 3000 E

FORM 1168 6-8-54 5M

MEMO

To RBM | Date 5/19/58 Time
Phone from Kelly | To
Place | Subject
Present

Plan work on 1475 level as large XCs will be necessary on either the 1475 or 1415.

Kelly plans XC N 60°E on sect. S1700 to about coord. 3700E in HW of Challa Fault. At this point work should be in rock and holes can be drilled to prospect possible mineralized zone in HW of Fault. Only one hole definitely in HW of fault and in ore.

He would XC N30W on coord 3400 E. for 400 ft. then drill down holes. Also could drill to E. to prospect ore against Challa Fault in FW. This sect. (3400E) would be about 200 ft. W of Challa on 1475, 300' W on 2075.

3200 E. sect. well drilled out from surf. although boundaries not definitely established.

3000 E would be W. of SH line.

Kelly will send rough sketch of planned work.
Suggested that he estimate Mo content of h. ore and recoveries.

FORM 1168 6-8-54 5M

MEMO

To Kelley

Date 1/29/59 Time 11:10AM

Phone from RBM

To

Place

Subject Haughton

Present

Haughton
Drift. 2 wks ago just sidesets.

Assays on soluble Cu – check difficult with San Manuel.
Should take up with Newlin

Samples at Inspiration – not in too bad shape.
Discard Old Dick rejects?

Could not find Browning prospect south of
Superior

FORM 1168 6-8-54 5M

MEMO

To Kelly -- Date 1/13/59 Time 9:25 AM

Phone from RBM To

Place Subject Houghton

Present

Talked to Jodie Eckstein Kelly at Superior.

Suggested we get samples on both sides of working, and grab samples of muck.

Arrange sample and mapping personnel to care for two shift's work.

Specify size of station - that is ten or twenty ft. pull of rods.

Asked him to call me if he has letter from Hardin on drill bids.

1285 L. E 12000 verticle.

OXIDE ORE OUTLINE?

LABELS ON OREBODIES?

SECTION LABELS

2 - 5 - 2

LONG SECT. LOCATIONS

1475 ORE OUTLINE

S. ORE AT S.M. Fault?

Heavy ground between 6+7?

2015
2075 ACM ore outline

Why blue for basic dikes,

E 4400

W2600 107 Section 2330-2375

E200 Proj. most of 2075 L?

Long Section 40° and 45° lines?

Stress rock covered by concrete and timber on upper levels and by lagging on lower levels.

Quote "Conclusion" in transmitting letter.

FORM 1168 6-8-54 5M

MEMO

To Kelly Tucson | Date 3/4 Time 2:50 PM

Phone from RBM | To

Place | Subject

Present

Kelly says Wilson left this morning will stop at Hall prospect.

Rest of crew will leave Tucson, probably the end of this week - maps should be finished. Humphrey's family there, so cannot leave Thompson's car.

Eastland mapping on boo at San Xavier; must be back at Christmas on Monday Feb. 10. Work can be finished by Tucson office personnel.

Told him I would try to get down first part of next week

FORM 1168 6-8-54 5M

MEMO

To RBM

Date 2/28/58 Time 4:15 PM.

Phone from Kelly

To

Place

Subject Houghton.

Present

Says they will need at least one week to complete coloring maps.

Told him to include 1285 DDH sections but no outline on those sections; include copy of J.M. one reserve sheet but only in letter to accompany report.

I will call Carroll Weed, and ask him if there is anything more he can suggest to go into report.

Eastlick still working at San Xavier.

FORM 1168 6-8-54 5M

MEMO

To Kelly — Tucson | Date 2/24/58 Time 10:30 AM.

Phone from RBM | To

Place | Subject Haughton, various

Present

Kelly says letter and maps almost completed; told him to emphasize figures were confidential, and only send letter to Weed, Perry, myself. Maps sent with letter to be included in report later

Told him Wilson has called, 2/23/58 7PM and that I had suggested approximate estimate of oxide ore available for pit operations.

Eastlick has been working at San Xavier for 10 days, helped by Brown. Has completed 840 level.

Asked him to give home on time reports.

Says Humphrey, Parker, Thompson reports have been sent.

Asked to have Fortuna tracings sent up.

rregu-
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Failures Decline

NEW YORK, Feb. 6 (AP)—Busi-

Magma Has Loss

Liquor

TORON

welfa
pu

E THE

Houghton - oxide Cu reserve.
MoS_2 distribution.
Water.

- B. Mary - status

1/15/58

FROM Tucson

Call ~~to~~ from Dwyer —

He says he will send Barker

$400 advance for Houghton assessment

work immediately. Barker should send

statement of payments made each

two weeks which will be reimbursed

to maintain advance at $400

RBM.

200 SCALE PLAN MAPS WELL ALONG - WILL PROJECT WELL KNOWN FAULTS AS WELL AS POSSIBLE.

USING 100 SCALE SECTIONS TO MAKE TOTAL ORE ESTIMATE WITH 1415 UNDERCUT, 2015 UNDERCUT AND ADDITIONAL ORE BELOW.

THREE BLOCKS CHECKED GENERALLY - SHOW FIGURES.

HAVE WATER DATA

HAVE MOST OF PROPERTY DATA.

ASK IF WATTS, ETC., HAVE OBTAINED ALL INFORMATION ON. 13 BLOCKS BEING DRAWN.

KELLY WILL SUBMIT PRELIMINARY LETTER AS SOON AS POSSIBLE AFTER ORE ESTIMATES ARE MADE. GUESS AT AVAILABLE OPEN PIT OXIDE. MoS_2 CONTENT WILL NOT BE ACCURATE.

TAKE HOUGHTON TONNAGE CALCULATION. ABOVE 2075 L., ALMOST NONE ABOVE 1475.

TAKE 1957 CU AND MoS_2 PRODUCTION FIGURES.

			% Cu Assay	Sol. ox. Cu.
1950	THOMPSON. – DEV AND PROBABLE 0.5% CUT-OFF			
	OXIDE.	6, 021, 600T	0.921	0.756
	SULPHIDE	19, 332, 000T	0.84	0.12

INFERRED 2000' to E.

OXIDE	14, 400, 000 T.
SUL.	45, 000, 000
TOTAL	84, 800, 000 ??

1948	GILL. DEV. AND PROBABLE 0.5% CUT OFF			
	OXIDE	1, 844, 800	0.958	0.751
	SULPHIDE	18, 635, 200	0.876	0.123.

Present grade ± Total 0.80

oxide content .07

.72 ± Sulph.

MoS_2 ± 0.018% ?

ORE MINED 14,000,000 TONS.

Samples 5# per 100 Tons thru drawpoints.

More if grade low or block nearly drawn.

30000

300

9,000,000

56,000,000 T above 1475 horizon

+ 12,000,000± in N. Orebody.

Check on water supply

Oxide orebody - tons + metallurgy

1/13/58

50 Scale assay maps

1285 Down - drilling in fan rings - vertical.

1415 Grizzly - sampled on fringes.

1475 Harlequin - completely sampled

2015 Grizzly

2075 Harlequin

Geologic maps

on 200 and 50 scale.

MoS_2 from composite samples.

1/15/58

From Tucson

Call to Carroll Weed –

He says Eastlick may be available on part-time basis. Will talk to Reed.

Has had no conversations about cut off grades. Believes we should make calculation above 1475, between 1475 and 2075 and other.

He will be at S. Manuel tomorrow night after 4 PM, and stay Friday. Hannifin leaving Friday morning, Rood & Watts Friday evening. Told him I would see him tomorrow evening and Friday.

Use their assays check?

Use their ore estimates unless change in cut-off needed. They used 0.5%

calculate MoS_2

At least use their estimate to start, calculate other.

Water.

~~Area.~~

Metallurgy of oxide ores.

1/17 – Carroll Weed wants calc. of how much ore SN must leave if no deal, calculated on 40° slope.

Get 50 scale diam drill plans. –

" MoS_2 assays on DDH's.

" map showing accurate location of stope blocks and tonnages + grade from each.

FORM 1168 6-8-54 5M

MEMO

To RBM.

Date 1/27/58 Time 4:30 PM

Phone from Jim Wilson

To

Place

Subject Houghton Project.

Present

Discussed Houghton project — said he would be ready to go to work anytime.

Told him I would call him toward end of week.

FORM 1168 6-8-54 5M

40'W. 120' E.

MEMO

To RBL - SLC

Date 1/22/58 Time 7:30 PM

Phone from Barber

To

Place Tucson

Subject Haughton

Present

Barber says underground work should be finished about Jan 28th. Earthink is helping at S.M.

Thompson staying at Flamingo — Ellen Steele arrived. Kelly not yet in N.Y. but still expected Jan 26, in Tucson.

FORM 1168 6-8-54 5M

MEMO

To CEW

Date 1/3/57 Time 12 Noon

Phone from ~~CEW~~ RBH

To

Place

Subject Arizona job

Present

Send Barber, Thomsen and Brown to S.M. on Tues. I will go down the end of the week

CEW says so far no dilution (RBH?)

He estimates 6 weeks for job — (probably more RBH)

CALL FROM J.L. KELLY 9/26/57 Tucson - S.L.C.
TO RBM

Houghton

Amount of casing left in churn drill holes

Size of CDHs

To be used to establish approx. direct cost of hole.

Asked Bob Dwyer to send money to Stephens, surveyor, for advance on Houghton patent survey, as per letter of Sept. ?

Barber 5-6 mi SW of Wickenburg biotite gran. covered to W by val. flows.

Advised him MacAlister could probably give little help by geo-physical work.

reported 20 such sales for August.

San Manuel Attracts Interest of Anaconda

FLORENCE, Sept. 1.—(AP)—An

7/17/60

Kenn. to increase capacity within 6 months at domestic properties

3 areas prospected at Tintic with favorable showings

Winze work at Bergin under Michaelson – not Moon. Costs can be charged off currently – will only do enough work to check few points in orebody.

Advisers indicate trouble in Africa may last 10 years.

RBH – CDM

Leaching Mixed Oxide and Sulphide ~~Ore~~ Copper in Inspiration,

H. W. Aldrich and W. G. Scott.

E + M J Vol. 128 No. 16 P 612 - 619
17 P 652 - 657

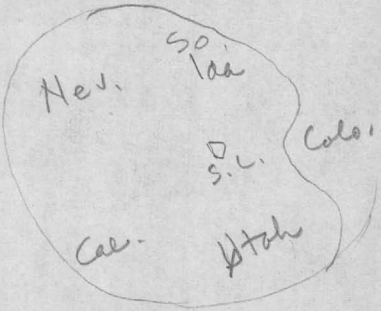
T. SMELTER 9/24/56

8600 T Lead
Smith
15,000 cap.

5800 T. Zinc
[illegible]
[illegible]

6500 T. Mill

35,000 T cap.



KENNECOTT ANNUAL REPORT

	TONS 1957	TONS 1956
CU		
Total Sales TONS	552,944	495,219
40% Foreign 60% US in 1957		20% Foreign 80% US.
PHELPS DODGE PRODUCTION CU	246,418	270,181
A.S.&R. PRODUCTION (OWN AND ON TOLL) CU	422,580	434,699
A.S.&R. REF. ZN.	202,450	197,553

INSP. NEW PLANT BY NOV 1 1956

TREAT 16,000 T. thru conc,

18,000 Tons leaching plant ± 2000 T. slime plant

Stripping now 3.1 - 1, will be 1.2 - 1.4 to 1 total. stripping continue at present rate (50,000 T/day ore + waste) for 4 years.

Will make up to 85,000,000#/yr with new plant although grade will drop + sulph. increase.

At smelter will make all anodes to produce straight elect. cathode copper at Insp. ~~prep.~~ elect. plant - estimate 0.5 ¢/lb saving by this method.

Have found 100,000 T on Thornton 3700 B. at 1% Cu in area expected to be waste - may contain more ore at depth.

Jackson - Douglas
aircompt
Clinton
Payson.

Swansea

Lidke -

361 998 Nacco.
Bal #3 Q.C. #4

[illegible] Bottom

AJO

8/4/56

20 [illegible], Top -

1. Fr. monz on S side, NW of Ark Mt.
2. 1260 E side cr. monz.
3. Rhyolite below Ark. Mt. - near monz cont. S. side of pit
4. Diorite(?) with FeS2 seams N. side - some chlorite.
5. Peanut porphyry? near rhy. - Loc. fam. contact
6. p.m. andesite dikes.

Monz. on N. side cr. and contorted fractures little more alt. than S. side.

10 yd. mean shovel

Stopped truck haulage recently

Close down 2 weeks start Aug 25th

Barr Mgr. O'Neil Chief Eng
15 yrs at Ajo.

Do ~~take to painter~~

Glass -
Patagonia U3O8
Expense acct - [illegible]
~~Quigley - Betty [illegible]~~
~~Stone Pa[illegible]~~

WHITE

~~Peggy Terrones~~
~~Dorham~~

Golden West
C 7250
Chromspun 10.90
9.00

Balboa
Nylon.

Doreline 4.90

~~Todfuts.~~

LUNCH WITH FINK — 11/29/55

Says ASER Lord went to England to obtain lease on Mazapil Cu Co. holdings — At same time Steele of Amer. Metals — Peñoles bought 30% stock interest in Mazapil, and blocked lease —

Peñoles will step up activities in Concaal Oro — Mazapil area, and will buy all equipment at Namaquipa (Venturosa Co.)

11/30/55 LUNCH WITH Geo. ORDONEZ — CAMPILLO.

ORDONEZ GOING TO SA. AMERICA DEC. 4, for 2 weeks — has been flying over ACM Guina gold deposit.

Kenn has placer col. tantalum operation in Nigeria producing 10 tons/day, conc. Also are prospecting large mass of "carbonatite" in Canada (supposed to be igneous $CaCO_3$ mass) for same metals, but very low grade — is now metallurgical problem.

He believes copper is pricing itself out of the market !! only Napus increased demand will take up slack — Also believes U_3O_8 price will break in 1962.

MISC. DATA Feb. 1956

MAGMA NEW ORE W. of fault on O'CARROLL HOR.

SAN MANUEL STARTED MoS_2 PLANT 2/11

Conc. operating on ~10,000 tons/day from 3 blocks in Mine 245 x 270
245 x 210
210 x ? (smaller)

up to 600 ft high —

MoS_2 grade 0.04% Mo.

BYRD 2/20 DUVAL

DRILLED OUT 30,000,000 tons of commercial grade - if correct in wide scout holes, 130,000,000 tons grade probably very low!!

~~Apr.~~ ±12 1956 L. A. TIMES
UNIV. OIL

DRILLING ON FOX LOT AT 8680 FT

GROSS INCOME FROM LEASE $8500-9000/DAY

LEASE 280 A. PAY 20TH CEN- FOX 20% GROSS ROYALTY, 50% of net profits after investment + dev. costs recovered. Still must recover $2,500,000 of $3,000,000 spent before production.

Selling oil at $2.56/bbl 26° gravity

4,700,000 a ft. gas/day.

8 wells completed.

Frisco Firm Given Contract For New Kennecott Smelter

RAY, May 28—(Special)—Western Knapp Engineering Co. of

years.
She

SLEEP WELL TONITE
S.W.T. SLEEPING

L.A. TIMES
6/13/56

OIL NEWS

Kern Test Scheduled by Shell

AJO 6/16

At 1200 or 1300 level 200' in pit across 2 benches averages 2% Cu - chalcopyrite in monzonite.

Treating 35,000 T./day, total 75,000 T/day.

Grade for July estimated at 1.1% Cu.

Costs RBH - estimated at 16-18 ¢ on 0.8 grade
13-14 ¢ on 1.0% "

0.8 grade

25¢ Ton ore	¢ 1¼-1. use	58
25¢ " waste		75
Mill		90
• Smelting		40
Misc.		
		2.63 ¢.

CITIZEN 6/22/56

THE CANANEA CONSOLIDATED COPPER COMPANY, S.A.

DATOS ESTADISTICOS

Año	Kilos de cobre contenidos en una tonelada métrica de metal extraído	Beneficios económicos para la República Mexicana derivados de la Compañía: (Sueldos, Impuestos, Materiales y Mineral comprado en la República, Etc.) Moneda Nacional
1938	38.4	$ 9,328,269.78
1939	52.4	18,967,115.49
1940	52.4	11,622,556.86
1941	50.0	18,740,340.21
1942	46.1	24,459,977.79
1943	32.6	21,121,588.99
1944	26.4	15,682,014.69
1945	24.3	18,455,048.05
1946	23.0	19,340,391.12
1947	12.8	27,617,492.21
1948	10.5	43,617,353.62
1949	9.1	53,266,648.76
1950	8.3	71,045,897.87
1951	8.8	95,366,975.71
1952	8.1	112,226,270.51
1953	8.2	97,971,536.33
1954	7.8	178,438,971.67
1955	7.1	281,007,417.51

Cananea, Sonora,
25 de abril de 1956.

SAN MANUEL

Trouble in primary crusher with iron + timber
5000 times down / month?

Recovery down from 92 to 89% – some oxide in
ore

Water (from outside?) in ore, makes trouble in
bins + crusher.

Production 18,000 T / day.

Concrete (mixed on surface, transported thru CDH)
8 blocks caving, (not yet reduced in size)
or concreted
new ones concreted.

De Cellas to leave for Export-Import bank (Tocquepala?)
on Oct 15.

EBU 6/29/56
From Jack Dugan

SAN MANUEL PROPERTY
Pinal County, Arizona

ORE BODY:

Disseminated mineralization in monzonite porphyry and quartz monzonite.

Principle Sulphide minerals: Chalcopyrite, Chalcocite, Pyrite & Molybdenite.

Ore Reserves:

Sulphide: 370,000,000 @ 0.785% Cu.
Oxide: 112,000,000 @ 0.717% Cu.
0.02 to 0.05% MoS2
Appreciable recoverable Ag and Au

Overburden: 0 to 1,000 feet thick
Average over S.E. Ore body: 670 feet
Ground Support: All ground requires timber, steel or concrete support
Water: Newly opened areas show appreciable flow; drains quickly.
Pumping 2,800 to 3,000 g.p.m.

No. 1 & No. 4 YARDS

Magor Mine Cars (red) - 310 cu. ft. capacity) 12.5 tons
Allison Mine Cars (green) - 280 Cu.ft. capac.) capacity (approx.)
Length center to center of couplings: 17'-6"
Width overall: 6'-0"
Height above track: 5'-6"
Trucks: National Malleable Mine Car truck; 14" cast iron wheels
Couplers: Nat'l Malleable Willison Rotary and non-rotary couplers
Rubber draft gear for coupling shock absorption
Mine Flat Cars: 11'-0-1/2" center to center couplers,
9'-5-1/2" x 4'-10-1/2" bed

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

Timber: 12" x 12" drift timber
3" lagging x 5' long
6" x 8" cribbing, pre-framed
Pole lagging and posts - Texas pine, for grizzly drift lagging and U.C. timber

No. 4 SHAFT - 1,577' deep

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

2 Cage compartments each 14' x 8', rounded end compartments for manway, pipe, electric cable and ventilation

Man & Supply Shaft and downcast ventilation
210,000 c.f.m. now
300,000 c.f.m. anticipated
Cages: 2 decks, 50 men per deck, inside dimensions 6'- 9-1/2" x 13' - 6"

Rated capacities: 16,000 lbs supplies/deck
12,000 lbs men/deck
cage weight 20,000 lbs. (approx.)

Headframe: 109 feet to ℄ sheaves
2 - 14' cast steel sheaves on timken pillow block

Hoist: Nordberg double drum
15' diam., 90" face drums
2- 1/4" hoisting rope
Maximum hoisting speed - 1,500 fpm
Single reduction drive, 2 700-h.p. D.C. motors equipped with M.G. set consisting of 1 1750-h.p. A.C. motor and 2 600-KW D.C. generators. Hoisthouse equipped with 30-ton crane with 5-ton auxilliary.

Compressor House: 2 - C.P. 3,500 cfm compressors delivering at 100 lb. air pressure, each equipped with 600-h.p. synchronous motor.
1 - C.P. 1936 cfm - 350 h.p. synchronous motor
1 - C.P. 1596 cfm - 300 h.p. synchronous motor

Total - 10,536 cfm delivering at 100 lbs. air pressure. Provisions for addition of future compressor.

No. 1 SHAFT - 1,643' deep

Dimensions: 25'-5 7/8" x 6' -4 compartments:
2 hoisting 6'-5 3/8" x 6'-0"
1 manway 5' x 6'
1 svc. hoist 5' x 6'
Structural Steel Sets & Reinforced Concrete

Headframe: 100'-6" to ℄ sheaves
2 12' diam. steel sheaves

Wellman-Seaver Hoist, 250-h.p., rope speed 900 f.p.m.
1 1/8" hoisting rope

Downcast ventilation, auxilliary hoisting; will be sunk to develop 2nd mining lift.

CHANGE ROOM

Accomodates 1,640 men with lockers, showers and toilet facilities.
Heated with gas space heaters.
Shower water temperatures automatically regulated.
In the same building in Foreman's Office, Time Office, Safety Engineer's Office, Dispensary, and Lamp Room.

No. 3A SHAFT - 1708' deep
No. 3B SHAFT - 1707' deep

Twin ore hoisting shafts, 195' apart.

Dimensions: 29' x 7' inside concrete

4 Compartments, each 6'-6" x 7'-0"

Structural Steel Sets buried in concrete for smooth lining, concrete curtain walls between each compartment with 3'-7 1/2" x 4' windows in each set.

Steel guides supported every 3'.

Headframe: 181' to ℄ sheaves, 14' diam. cast steel sheaves

Ore bin: Diameter: 60'
Height: 66'-6", top of bin 92' above collar
Capacity: 5,000 tons live load
Loading Gates: 4 on each track, air operated

Ore Cars and Transportation
Cars: 100 ton capacity, 35 to 40 per train
Locomotive: 125 ton, 1600 hp deisel-electric
Trackage: 132-lb. rail, 7 mile haul to receiving bin, level track

Skips: Bottom dump, Lake Shore Jeto Skips
18-ton capacity
Dimensions: 33'-5" long, 6'-1 1/2" wide, 6'-6" deep
Weight: 26,000 lbs. (approx.)

Hoist: Nordberg Double Drum, automatic or manual operation
15' diam, 116" face drums with 3,700 feet of 2 1/4" rope capacity in 2 layers
Drum shaft, 27" diam. through drums
Hoisting speed, 3,000 fpm
2 3,000-h.p. D.C. motors equipped with H.G. set consisting of 1 4,000-h.p. motor and 2 2,500 KW D.C. generators.
Steel plate flywheel.
Approx. weight - 44 tons.

6/23/56

June 18, 1956

AIR MAIL

Mr. R. S. Newlin, President
Inspiration Consolidated Copper Company
25 Broadway
New York 4, N. Y.

Dear Dick:

I wish to acknowledge receipt of your letter of June 14th, with which you enclosed a copy of the first draft of the proposed agreement with Callahan Zinc-Lead Company, and covering the 79 Mine.

I think you have worked out a very good and very fair arrangement with Callahan on the 79 Mine. I note specifically the option payment is $375,000 and there is no down payment. We will have two years to examine the property before the first minimum royalty payment becomes due. The subsequent rate of royalty payments is moderate enough if we find a mine.

I have no suggestions to make concerning the arrangement.

Yours very truly,

P. D. I. Honeyman
Vice President and
General Manager

PDIH:CW

cc: Mr. R. B. Mulchay

Esleeck
Fidelity Onion Skin
MADE IN U.S.A.

6/19/56

June 18, 1956

AIR MAIL

Mr. R. S. Newlin, President
Inspiration Consolidated Copper Company
25 Broadway
New York 4, N. Y.

Dear Dick:

This week we investigated the possibility of our obtaining control on a lease and option basis of the Chilito group of claims lying between the 79 and the New Year group. We find the situation complicated and involved.

It seems the owners of the Chilito group, the Velasco family of Winkelman and two other outsiders in April of 1953, gave a 10-year lease and option on the property to Tex Murray and Gordon Wainwright. The option price was $150,000. The option to purchase only ran to April, 1956, when a cash payment of $10,000 and future payments of $2,000 per month were called for to hold the purchase option in effect. Murray and Wainwright did not take up the option, but instead obtained a three-month extension to July 23, 1956. By that date, to hold the option, they must make payments which they say will amount to approximately $18,000. It will probably be impossible for Murray and Wainwright to meet these terms. Two things are thus apparent:

(1) The owners say they will not renew the option to purchase to Murray and Wainwright unless all conditions are met. They expect to be able to deal directly with Inspiration, since they know we are interested. Undoubtedly, their option price will be at least doubled to $300,000 and they would insist on a $10,000 payment plus the $2,000 a month arrangement.

(2) It would do no good for Inspiration to deal directly with the owners on any terms because Murray has the 10-year lease with the sole right of entry to the property. Mr. Ladendorff advises that under the circumstances Murray could forbid Inspiration to enter the property at all and so no examination work could be done. Incidentally, Murray is currently mining and shipping to Hayden some low-grade oxidized copper ore from a surface showing on the property. This showing is in the quartzite which, due to faulting of neighboring blocks and erosion, is found on the surface in this particular spot.

To deal now with Murray and Wainwright, Inspiration would have to take their obligation under the option off their hands, and this would amount to some $18,000 cash plus whatever consideration they would insist on

for stepping out. Actually, they talk of some cash to themselves plus an increase in the option price to $400,000.

In passing, when Miami Copper drilled the Chilito group, they had an arrangement with Murray for an option price of $200,000 and the right of entry for examination for 10 months with no cash payment of any kind. This, of course, was prior to April, 1956, when the cash payments to the owners became due. Miami's results were negative. Also, this property was drilled by Kennecott. Actually, Chilito has little to sell. Under the circumstances, as far as we can see (Messrs. Ladendorff, Mulchay and Reed concurring), this makes the whole deal a most undesirable one.

In so far as Kullman-McCool is concerned, we do not feel that any cash payment prior to examination of the property is justified. As I told you, a not unreasonable long-term deal could be made, but so far they have insisted on a cash payment of $15,000 as a starting point. I have not commited myself to this. Actually, Kullman-McCool is not too important at this stage, anyway, as it is out on what might be the fringe and the showings are meager. I would suggest that I further contact them and offer them a deal without a cash payment. They just might go for it, and if they do, it might later help us with Chilito.

Further, it should be pointed out that as yet we have not found much in the way of encouraging results from the drilling so far done on the New Year group which might indicate any continuous zone of ore mineralization between Christmas and the 79 group.

I have just completed reading the deal you worked out with Callahan on the 79 group. This deal is much more reasonable than those the owners of these other properties would agree to. I suggest we go ahead with the 79 and let time work for us in persuading these other people to agree to more reasonable arrangements on their respective properties.

Yours very truly,

P. D. I. Honeyman

P. D. I. Honeyman
Vice President and
General Manager

PDIH:CW
cc: Mr. R. B. Mulchay
Mr. G. H. Ladendorff
Mr. E. F. Reed

Call from Honeyman 5/31/56

Villesvik arriving in Inspiration today - will try to make option

18 months, 2000' drilling

Afterward $10,000 min royalty to be doubled after 5 yrs, all payments to apply on purchase price.

Total price $200,000

May make some concession.

R. B. MULCHAY
FEB. 1954

DRIFT PRACTICE AT SAN MANUEL

American Institute of Mining and Metallurgical Engineers

Arizona Section

December 7, 1953 Tucson, Arizona

H. I. Ashby, General Foreman

SAN MANUEL COPPER CORPORATION

DRIFT PRACTICE AT SAN MANUEL

The drifting program at San Manuel is for the purpose of bringing into production the ore above the first haulage level in the South ore body. This ore body is a mass of hydrothermally altered, well fractured monzonite. It is cut by some post mineral rhyolite dikes and by at least two major faults. It is about thirty-five hundred feet long, about a thousand feet wide in the middle and tapering towards each end, and is eighteen hundred feet thick. The strike of its axis is roughly Northeast and Southwest. The ore minerals are chalcopyrite and molybdenite. The capping overlaying the monzonite is Gila Conglomerate.

The first haulage level is planned so that roughly the upper third of the orebody will lay above it. This is the upper third in vertical dimension, not tonnage. The second haulage level will probably develop the greatest tonnage.

A grizzly level is being driven sixty feet above the haulage level. The grizzly level connects between the mining area and the service shafts, so that supplies and men can come down to the grizzly and proceed into the working places without cluttering up the haulage level or having to climb or be hoisted back to the grizzly. This connection is also a vital part of our final ventilation plan. In this the service shafts will be downcast and the ore shafts will be upcast with the bulk of the air coming across on the grizzly crosscuts, proceeding down thru the transfer raises, and going out the haulage fringes to the ore shafts.

The haulage level is being advanced along both the north and south fringes and connections are being driven to the three shafts which are yet to be sunk.

All of our driving is being done from either No. 1 or No. 2 shafts. We have pockets and skip hoisting at each shaft. We have crosscuts connected between them on both levels so we are able to dispatch muck to either shaft when necessary and can get supplies down either one. We always change equipment like loaders, cars, and locomotives at No. 1 as the compartments are larger and the cage is bigger. Another reason is that No. 1 shaft is near our shops and tracked equipment can be hauled on track from the collar to the shop. If we bring equipment out at No. 2 shaft it must be loaded on a truck and hauled about a mile to the shop.

At present we have only eight drift crews working. We planned to have more going by now but have been held up by equipment deliveries. Five of these crews are driving haulage drifts and three are driving grizzly drifts. During the month of October these eight crews made twenty-six hundred and fifteen feet of advance. This is an average 327 feet per crew or nine-tenths feet per man shift.

Each crew consists of four men. One is the leadman and the other three are usually two miners and a mucker. As we work seven days a week each man has a separate day off and his place is filled by a replacement from the labor gang. The men work on a bonus system. They are required to drill, blast, muck, timber, lay their own track, and hang their own flexible vent tube. Air and water lines as well as

rigid vent pipe are installed by the labor gang. The bonus system is set up so that a four man crew which makes a complete cycle of operations in a shift will make between six and seven dollars per shift bonus. From the price per foot for both excavation and timbering is deducted the straight time wages. No overtime is deducted. The difference is bonus and is split on the same ratio as the day shift wage variation between leadmen, miners, and muckers. Thus a leadman gets 20% more than a mucker and a miner gets 10% more than a mucker. After the bonus is figured overtime is paid, if worked, on both the wages and bonus. Turnouts are paid on a scheduled price for each set. As the sets get larger and more difficult to put in the price goes up. Irregular openings are paid by cubic content and car transfer stations are paid as so many equivalent feet of drift.

Each drift crew has an eight-ton battery locomotive and eight five-ton, a bottom, gondolas, and one flat car. Each crew has an air powered, rocker arm type loader with a deck height sufficient to load our cars and still clear a cap eight and one half feet above the tie. They also have a jumbo consisting of two booms mounted on a car made in our shops. This car is counterweighted so that no roof jacks are required to hold it in position. After it is spotted in the face ordinary wooden wedges are placed behind the wheels and it stays in place while the round is being drilled. The booms are actuated vertically by a feed screw turned by an air vane motor. The horizontal movement is manual but can be locked in position with an air brake. Each boom carries a three inch auto feed drifter mounted on an aluminum shell, with a feed screw capable of changing six foot steel.

The drill steel section used is inch quarter octagon. Early in our drifting program we found that steel breakage was only about half as frequent with this section as it was with one and one-eighth inch round steel. We therefore have all of our drifters equipped with jackhammer type front heads and steel pullers. Our practice is to make new steel into drill rods seven feet and eight inches long with a regular ringed shank. This is an overall measurement after upsetting the ring. Most of our breakage occurs near the shank or actually across the ring. Therefore, this piece of steel can be broken and reshanked twice and still be long enough for our drifters. After the third breakage it can be either made into jackhammer steel or stoper steel. If we followed the common practice of using one section of steel for drifters and another for jackhammers and stopers, we would soon accumulate a large stock of drifter steel, too short to be used on the long feed machines and not the right cross section for the drills that require shorter lengths of steel.

We use a one pass friction type of bit and are quite satisfied with the results it gives us. The size most used in the drifts is inch and five-eights.

As we operate on a three shift basis and the crews blast whenever they are ready, good ventilation is essential to both the health of the men and rapid progress. Prior to the time that we had a connection between the two shafts we used the regular sinking fans for ventilation and added seven and one-half horsepower axial flow fans in series along the ventline to maintain adequate air flow at the faces. We use 24", thin wall, metal pipe for ventilation duct and the booster fans are modified in the shop so that they can be coupled into the line with

regular pipe couplings. They are hung on a simple bracket between the caps and offer no more obstruction in the drift than the pipe.

We had a fifty horsepower fan on the collar of No. 2 shaft, as the greatest length of drift to hole thru was driven from there. We had a twenty horsepower fan on the collar of No. 1 shaft.

Shortly after we holed thru between the two shafts we drove a run around fan drift in the main crosscut. There is now installed in this drift a two hundred horsepower axial flow fan which blows 72,000 cubic feet per minute of air thru the main crosscut. This makes No. 1 shaft downcast and No. 2 shaft upcast. We have an airlock with electrically controlled, air operated doors in the haulage level main crosscut and are now installing a similar arrangement in the grizzly level main crosscut. By means of doors, the main airstream is made to flow thru one of the panel drifts and back along one of the south haulage fringes to No. 2 shaft. Thru a raise from the fan drift to the grizzly level part of the air is led back along the grizzly main crosscut to both No. 1 and No. 2 shafts. From this main airstream booster fans carry the air into the operating faces.

All drifts are timbered and side lagged. Our ground is so fractured and altered that it is necessary to put booms ahead and put up the cap and lag the back before mucking out. If the ground is particularly weak, the back is caught immediately after the blast, but generally four or five cars can be mucked before booming the cap and catching the back. Then the rest of the round is mucked out and the posts stood. While two men are blocking the set, the other two men on the crew go out with their train and bring the jumbo to the face. While two men are drilling out, the other two catch up the side lagging and make up the primers.

Electric primers are used in wet places and fused primers in dry places. We use 40% straight gelatine powder in all headings.

For electric blasting a 440 v. circuit is carried from the nearest transformer substation to a locked switch which is placed in a convenient location at least five hundred feet from the face. From the locked switch a circuit is run a short distance to a double throw, three pole switch. This switch has a short length of three conductor cable terminating in a three pronged plug wired to the blast side. The cold or off side of the switch has a ground and short wired to a three pronged plug using wire with bright red insulation. This switch is so wired that when it is in the off position the circuit between it and the locked switch is both shorted and grounded. To reach from this switch to the face we use a reel wound with several hundred feet of a pair of #14 house wires. These are taped together every five feet or so to facilitate winding on the reel. The reel has a female receptacle in the side connected to the wires so as to make contact with the plugs mentioned earlier. The circuit is always left with the locked switch locked and the reel switch thrown to the off position, and the plug on the red wire plugged into the reel. In this position all the segments of the blasting circuit past the locked switch are shorted and grounded, clear up to the ends of the reel wire. The ends of the reel wire are made up to the lead wires of the round. Incidentally, we always wire up in series parallel. Blasting procedure is to unplug the plug with the red wire from the reel and plug in the plug on the three conductor black wire. Next the switch is thrown from off to blast, then the blaster goes to the loaked switch, unlocks it, and throws it to blast. The shots go off immediately and he then reverses the procedure clear back to changing the plugs in the blasting reel. As the face progresses

the second switch and reel are moved ahead at intervals. Usually the locked switch is left in its original position until another drift is started between it and the face.

Our fused primer procedure is probably the same as other mines. We spit with hot wire spitters. We tried spittercord but found that it did not work well when a fogging device was used. Immediately after the round is spit we turn on a device in which compressed air and water are blown in a fog towards the face. This fog reduces the dust and gas in the return ventilation air considerably.

Our timber interval is five feet and no attempt is made to break past the next set. With the long feed drifters it is not necessary to change steel except for worn out bits. This speeds the drilling considerably but limits the kind of cut that can be drilled. Our ground is generally not hard enough to require a burn cut. We use a horizontal vee cut in the two rows of center holes.

We do not try to advance more than four hundred feet past a switch or a horizontal car transfer. Our choice of a horizontal car transfer over a vertical cherry picker was dictated by the fact that we feel a horizontal car transfer is safer and easier to operate, requiring no power, and we would rather maintain a wide opening than a narrow high opening. Also we are able to use the abandoned car transfer stations for timber storage, transformer substations, and mine sanitation stations. We have found that a good train crew can leave an empty car at the loader, run back four hundred feet to the switch or car transfer, spot an empty, and return to the face by the time the loader operator has loaded the empty. Any longer run than that causes the loader to wait.

When advancing into water bearing ground, or ground that we suspect may be water bearing, we drill an AX diamond drill hole for a pilot hole from the car transfer station. This hole is usually parallel to the drift centerline and gives us the information we want without slowing down the drift crew or causing them to abandon the face while drilling is going on. Our ground drains quite readily and in many instances we have been able to drain ground ahead of us with pilot holes so that when we drove we had only water in the lifters and our progress was much faster than it would have been had the water table been at the top of the drift instead of the bottom.

Since we changed to dry transformers instead of the oil cooled type we can park them in abandoned car transfer stations with no additional expense for fire proofing. We use a 300 KVA dry transformer and load center that is never removed from a flat car. When the time comes to move it to a new location, the electricians need only detach the plug connections and it can be rolled onto the track.

We are at present driving three types of drifts: Haulage drifts, Grizzly fringe drifts, and Grizzly panel drifts.

Haulage drifts are timbered with a ten foot cap and two ten foot, five inch posts. Post batter is an inch and a half per foot. Most haulage drift is driven with twelve by twelve timber but in some of the less altered monzonite and in the Gila Conglomerate we are using ten by twelve timber with the twelve inch dimension at right angles to the line of the drift.

All track is 36" gauge and we are driving with 45# rail. Ties are six feet long, six by eight, treated fir.

The track carried by the drift crew is put in with the bottom of the tie even with the bottom of the post. This speeds the driving as no time is lost digging hitches for foot blocks, which is a very time consuming operation where much water is encountered. The track is later raised about six inches and the ties placed so that they are on twenty inch centers. As the track is raised by the track crew it is moved six inches to one side of the drift centerline to provide a ditch. The muck left on the sides is about sufficient for ballast leaving the space from the end of the tie to the post for the ditch. This still leaves the track one foot below finished grade. We plan to make this final lift when we change to heavier rail and add ballast to make a better roadbed for our production trains. The hydrothermally altered monzonite that we are driving thru does not make good track ballast.

Several of our haulage drifts are being driven down grade into water bearing ground. This necessitates carrying a pump discharge line in the drift and maintaining a pump that has a suction close to the face. We usually have a small sump for this suction twenty to fifty feet from the face and scavenge the water from the face to this sump with an air driven centrifugal pump. This is light and readily portable, and if the main suction is close enough needs only be turned on when working on track. Six to eight inches of water in the face does not seriously impede mucking, timbering, or drilling.

Grizzly fringe drifts are timbered with a ten foot cap and two nine foot, five inch posts. All posts have the same batter. The track carried by this drift crew is only a few inches below grade so that the track crew can raise it sufficiently to put in all the ties

and just come to grade by doing so. This track is carried on drift centerline. No provision is made for a ditch because in the rare instances when we do hit water on the grizzly level we can always diamond drill a short hole to a haulage drift below and allow the water to drain.

Grizzly panel drifts are timbered with nine foot caps. In all other details they are the same as grizzly fringe drifts.

Both grizzly fringe drifts and grizzly panel drifts are timbered with twelve by twelve timber throughout.

In the haulage pattern all turnouts are on one hundred and sixty-five foot radius curves. No haulage curves are less than one hundred and fifty foot radius.

In the grizzly pattern all turnouts are on one hundred foot radius curves and this is the minimum radius of any grizzly drift curve.

Turnout caps are made from ten by twelve wide flange H. beam. These have a bridge cap above them of ten by twelve timber. The steel rests on regular twelve by twelve drift posts.

We drive these turnouts just as they come in the drift pattern. If we come to them small end first we muck the tangent with straight track and lay a jump switch to muck the curve, using the same mucker. When driving from the large end first, we muck all we can get from the track and then set up an air driven slusher and side slush the rest of the muck to the track. The steel cap is raised by running a cable sling over the last cap and attaching one end around the middle of the steel cap and the other end to the locomotive coupler. The locomotive then raises the cap until the booms can be run out under it.

I wish to thank Mr. J. F. Buchanan and Mr. C. L. Pillar for their helpful criticism in preparing this paper.

Trip to San Manuel January 11, 1951 with

R. H. Sales, A. W. Ruff

With Ashby visited exploration level at 1285 ~~(or 1285)~~ about 250 ft. opened up at right angles to shaft #2, probably drift to east with tail track to west. Rocks badly broken with many small flat slips with flat west dip, considerable chloritic alteration; few narrow quartz seams. Mineralization pyrite and chalcopyrite, almost entirely as disseminations. On surface dump pieces show late pyrite seams to cut early glassy quartz seams. Ore on both 1285 and 1475 levels reported to assay plus 0.8% copper.

E.F. Reed 3/19/52 xc 1500' to E on 1275 lev.

Visited 1475 level whichwill be first haulage level. Both levels being driven as about ten by ten headings with false caps against back to take up heavy ground pressure and facilitate repairs. Using some steel I-beams above caps on 1475. Some felsite or rhyolite near face on 1285. Diamond drill hole being drilled easterly on 1475. AX core, recovery seems good although core badly broken.

Shaft below 1700 ft., but temporarily stopped. will be advamced to about 2100. Another haulage planned at about 1875.? Pumping about 1200 gallons per minute from shaft, occasionally to 3500. Are below Tiger Mine, but do not seem to affect water level there. Tiger operation, Mohawk shaft sunk to 1125, have about 100 ft. of crosscutting to reach end of Collins vein in footwall of fault. Lowering water table about two feet per month, pumping up to 3500 gal. min. usually 2500 gal/min

In #2 shaft stopped to see contact of Gila Cg with monzonite porphyry. Could not see actual contact but many pices of red gougy material seen behind lagging outside shaft, and fault is probably there alright. Could not see direction or dip.

SAN MANUEL PROSPECT

H. J. Steele and G. R. Rubly

This paper describes the exploration by churn drilling of the San Manuel disseminated copper deposit in Arizona. The discussion covers the generalized geology and mineralogy, tonnage and grade developed to date, and the drilling and sampling operations.

SAN MANUEL PROSPECT

LOCATION AND HISTORY

The San Manuel property is located in townships 8-south and 9-south, range 16-east, Gila and Salt River Base and Meridan, State of Arizona. This area is in the Old Hat mining district, southern Pinal County, and is about 3-1/2 miles southwest of the town of Mammoth on the San Pedro River and one mile south of the Mammoth Mine at Tiger, Arizona. The property consists of over 150 contiguous lode claims.

The Old Hat district was prospected prior to the Civil War, but there was little or no production until 1881[1]. The chief producers have been the Mammoth and Mohawk Mines, now consolidated and operating as the St. Anthony Mining Development Company, Ltd. Production to 1936 was $5,200,000, of which 83% was gold[2]. Since 1940 the production has been chiefly lead and zinc, and some molybdenum and vanadium. There are a few small properties in the vicinity that have produced very minor tonnages of copper ore.

In the San Manuel group there are claims, located in 1906, that have been held continuously to the present time, and at least two churn drill holes were put down on or near the outcrop in 1917. No logs for these holes are available, but it is assumed that the grade of copper encountered was not sufficient to encourage further exploration.

A small amount of ore has been shipped from open cuts on the San Manuel outcrop area, but there is no record of the tonnage or grade. It is estimated that not more than 50 tons were mined. These cuts, plus a few shallow shafts and some short adits indi-

cate limited and unenthusiastic prospecting at San Manuel.

In 1942, through the efforts of James M. Douglas, R. B. Giffin, Victor Erickson, and Henry W. Nichols, owners of the San Manuel group, the Reconstruction Finance Corporation and War Production Board authorized the United States Geological Survey to investigate the property, and by their recommendation, the United States Bureau of Mines was authorized to put down a limited number of churn drill holes. The Bureau of Mines drilling carried on from November, 1943 to February, 1945. Seventeen holes were drilled, varying from 305 to 1990 ft. in depth.

The property was brought to the attention of Magma Copper Company in June, 1944, and an option agreement was made with the owners. Subsequently, a group of claims lying north of the original group and held by the Apex Lead Vanadium Mining Corporation, and a group to the east held by the Quarelli family, were optioned. A drilling program was laid out to conform to the Bureau of Mines pattern, and drilling started in December of 1944. San Manuel Copper Corporation was incorporated in September of 1945, and has carried on the exploration program to the present time.

GEOLOGY

General

The mineralized outcrop, chief copper mineral of which is chrysocolla, is a triangular area with a northeast-southwest base of 380 ft. and with the apex of the triangle 400 ft. to the southeast. Northwest of the base is Red Hill, so called because of the prominent gossan coloration. With a few exceptions, conglomerate and alluvium cover the rest of the property. The elevation varies from 2900 ft. to 3500 ft. above sea level. The area slopes gently

to the east, with the exception of a small rugged and precipitous portion around the south side of Red Hill, and is cut by numerous dry washes with easterly courses toward the San Pedro River. The cover is sparse brush and cactii characteristic of this portion of the state.

Rocks

Relatively few rock types are found in surface exposures or are encountered in drilling. These formations were mapped by Dr. G. M. Schwartz of the United States Geological Survey, and the logging of all drill holes has conformed to his classification[3]. The rocks are quartz monzonite, monzonite porphyry, diabase, felsite, and felsite breccia, Gila conglomerate and recent alluvium.

The quartz monzonite found in the drilling area is granitic in texture, showing varying degrees of hydrothermal alteration. Quartz is abundant; the feldspars and biotite are extremely altered in most cases. Rutile is common, and the rock in general is sericitized and kaolinized.

The monzonite porphyry has a fine groundmass composed largely of a mosaic of quartz and feldspars. There is an abundance of biotite but less quartz than in the quartz monzonite. The feldspar phenocrysts are usually well altered, and the biotite is often bleached or chloritized. Secondary quartz flooding is fairly common, especially near the ore zone. A distinct phase of dark grey, fine-grained porphyry, containing an abundance of biotite in the form of phenocrysts recrystallized to aggregates of grains and much disseminated secondary hydrobiotite in the groundmass, is found in the hanging wall of the ore body to the south of the outcrop area.

Alteration products found in this section in both the quartz

monzonite and monzonite porphyry, as reported by G. M. Schwartz, include the following minerals: the hydromicas, allophane, epidote, leucoxene, montomorillonite, dickite, chlorite, potash clays, and calcite.

Diabase occurs as dikes cutting both the quartz monzonite and monzonite porphyry. The rock is well altered, grading from fine to coarse diabasic texture, and is mineralized with chalcopyrite and pyrite, or with chrysocolla and iron oxides in the oxidized zone.

The felsite, which is unmineralized and which cuts the diabase and the monzonites, is light grey to light cream in color, fine-grained, with occasional small quartz and feldspar phenocrysts in a finely crystalline groundmass.

The diabase occurs infrequently, and the felsite, while more abundant than diabase, is relatively unimportant in the overall rock distribution.

The Gila conglomerate consists of alluvial material interbedded with lava flows, breccias, and tuffs. The coarse conglomerate on San Manuel is composed of boulders of acid and basic volcanics cemented with clay. A phase of the Gila conglomerate, designated as granitic conglomerate and composed largely of fragments of relatively unaltered quartz monzonite, has been found in drilling to the southwest.

Structure

The structure at San Manuel, interpreted largely through drilling results, can only be generalized at the present time.

The Gila conglomerate, which covers the major part of San Manuel, strikes northwest and dips northeast at 20° to 45°. The San Manuel fault, striking northwest, more or less parallel to the strike of the conglomerate, and dipping southwest at 25° to 35°, forms the

contact of the monzonite and the conglomerate. The San Manuel fault is in turn cut by two 60° to 75° northeast dipping faults striking northwest. The most westerly of these faults is designated as the west fault, and the most easterly as the east fault. (Fig. 1). This step faulting of the conglomerate is illustrated in Fig. 2, and has been substantiated by drilling to the northeast. Minor outcrops of hematite-stained, altered monzonite porphyry occur at intervals along the surface exposure of the most northeasterly segment of the San Manuel fault. Drilling northeast of these exposures indicates that the contact of the conglomerate with the old erosion surface dips northeasterly at 20° to 25°. The indicated vertical displacement of the west fault is in the neighborhood of 400 ft., and of the east fault, 200 ft. The east fault has been considered to be the southward extension of the Mammoth fault as mapped by Peterson[4] and others. A more complex fault system may be expected and is indicated by the drilling, but correlation of evidence of faulting is not possible from the churn drill results.

The outcrop area, as shown in plan in Fig. 1, is bounded on the southwest leg by the San Manuel fault and on the northeast leg by the west fault. The base, as indicated in cross section in Fig. 3, is controlled by decreasing copper mineralization, and is therefore, an assay boundary. A diabase dike, striking northeast and nearly coinciding with this cutoff of copper mineralization, was originally thought to be a diabase filled pre-Gila fault which abruptly cut off the northwestern extension of the ore body. Drilling northwest of this line has indicated a gradational decrease of copper mineralization in the projected ore zone rather than faulting.

The persistency of the San Manuel fault on both strike and dip is unusual, and it is difficult to visualize a low angle normal fault of this magnitude. The most feasable explanation, with the limited data at hand, can best be presented by this possible sequence of events:

1. Intrusion of the monzonite, and mineralization of the intrusive masses.
2. Oxidation and secondary enrichment.
3. Deposition of the Gila conglomerate. (Events 2 and 3 may be contemporaneous).
4. San Manuel fault formed, dipping 60° to 65° to the southwest.
5. Regional tilt of 25° to 35° to the northeast and development of the fault system, which strikes northwest, and dips to the east.

The San Manuel ore body has been described as a tabular mass, striking north 60° east, and dipping southeast at 50° to 60°; and is shown this way in Figs. 2 and 3. It must be remembered, however, that this strike and dip is largely due to an assay boundary. Any arbitrary cutoff figure selected would change the attitude of the ore body. The strong pyritic mineralization below the outlined ore body has not been bottomed by any drilling to date. The upper portion of the hanging wall rock is very lightly mineralized; alteration and mineralization increases at depth until the zone of better copper mineralization is encountered.

Copper mineralization has been proved by drilling for 5000 ft. in the northeast-southwest direction, and 1400 ft. in the northwest-southeast direction. Pyritic mineralization is known for 2200 ft.

northwest of the ore zone.

Mineralogy

In the oxidized zone the chief ore mineral is chrysocolla; cuprite is present locally; malachite and azurite are rare. The chrysocolla is generally found in joints and fracture planes.

Chalcocite is the predominate copper mineral in the secondary zone. Residual primary chalcopyrite, occurring chiefly as intergrowths with the secondary minerals, and secondary chalcopyrite are found. Some native copper is found as well as minor amounts of bornite and covellite. There is very conclusive evidence of one or more stages of partial oxidation of the secondary zone in parts of the ore body. This may be due to variations of the water table and regional tilting.

Chalcopyrite and minor amounts of chalcocite are the chief copper minerals of the primary zone. The proportion of chalcopyrite to pyrite determines the footwall of the ore body. Slight oxidation is found relatively deep in the primary zone, and is thought to coincide with fault planes. The sulphide minerals are well disseminated throughout the monzonite, as well as occurring as veinlets. Chalcopyrite and chalcocite are frequently found as films on pyrite. Molybdenite is present in sufficient amount to be of economic importance, and can be seen megascopically in a large proportion of samples in the primary zone. Gold and silver are found in small quantities throughout the deposit, but sufficient work has not been done toward determination of the gold and silver content to state the average assay at this time. Rutile is fairly abundant throughout, and is thought to be present as an alteration product of biotite.

Polished sections of the concentrates, selected at intervals

in the sulphide zone, are prepared and examined by G. M. Schwartz of the United States Geological Survey.

CHURN DRILLING

General

A grid co-ordinate system, bearing 29°-50' west of true north, controls the churn drilling pattern. The orientation was set by the Bureau of Mines, and was based on the strike of the apparent northwesterly limit of copper mineralization of the outcrop. The drilling done by the Bureau of Mines and the early drilling by San Manuel Copper Corporation was spaced rather widely over the area to establish probable continuity of the ore body and to determine the required spacing of holes for future drilling. When intermediate holes were drilled and the general attitude of the ore zone became apparent, the present drilling pattern evolved. This pattern is based on a 400-ft. interval in the direction of strike, which is from east to west on the drilling grid, and a 200-ft. interval in the north-south direction. All holes since the beginning of exploration have been controlled by the grid co-ordinate system. The few exceptions are due to rugged terrain making the correct location impractical.

The drilling grid is extended from permanent reference points by transit and tape traverse; elevations are carried to the collar of each hole by a line of levels from established reference points. Elevation datum is from the United States Coast and Geodetic Survey bench marks. The drilling grid is tied to the San Manuel co-ordinate system, which is based on a true north meridian. Holes are spudded in as near the intersection of grid co-ordinate lines as practical, and the error in hole location is rarely more than one

foot in either direction.

Statistics

At the present time 67 holes have been completed, and 11 are in progress for a total of 110,854 ft. of drilling since exploration of the property began. The Bureau of Mines drilled 15,824 ft., and San Manuel Copper Corporation has drilled 95,030 ft. The deepest hole drilled to date has been 2600 ft. All drilling is done on contract, with the contractor furnishing all equipment and tools.

The average drilling conditions at San Manuel are illustrated by the following tabulation of statistics for a three-month period. Drilling shifts refer to time actually spent in drilling; total shifts worked refer to drilling shifts plus the time spent moving and setting up on new locations, repairs, fishing, and miscellaneous lost time:

	Average	Best Drill	Poorest Drill
Elapsed Time	91 days	91 days	91 days
No. of Drills Operating	6	1	1
Total Footage Drilled	13296 ft.	3439 ft.	71 ft.
Drilling Shifts	698.6	116.5	13.8
Advance per Drilling Shift	19.0 ft.	29.5 ft.	5.1 ft.
Total Shifts Worked	825	133.8	23.7
Advance per Shift Worked	16.2 ft.	25.7 ft.	3.0 ft.

The average time required to move to a new location was 4.6 shifts. This includes pulling casing from the completed hole, moving, and rigging up preparatory to spudding in.

Equipment

The selection of churn drills for exploring this property was based on the record of accurate tonnage and grade estimates computed

from data obtained by churn drilling at other well-known disseminated copper deposits. Exploration by diamond drill was rejected because of probable low core recovery if the formations proved to be highly fractured.

Several types of churn drills are operating on the property; Bucyrus-Erie machines being in predominance. Models 24-W, 24-L, 29-W, 29-T, and 36-L, with drilling capacities ranging from 1500 ft. to 3600 ft. are represented. These are all-steel machines, very compactly built, and ideal for working in rough terrain. The space required for a set up is comparatively small, and the machines can be handled over fairly steep grades. There are four Ft. Worth Spudders in operation: Models Super D, F, Jumbo H, and Super J; having drilling capacities ranging from 2500 ft. to 5000 ft. These are larger machines of wood construction, and require considerably more room for a set up, as the engine is usually set back of the rig some 20 to 30 ft. Power transmission is by flat belt drive. They are more awkward to move, and take longer to set up and tear down than the Bucyrus-Erie machines.

The drilling contractors, operating under similar contracts, hire all help required to run the rigs. A crew usually consists of a driller, a tool dresser, and welder. Since all bits are built up by electric welding, the tool dresser is more of a general helper than his name implies. Some contractors work on a three 8-hr. shift basis; others on two 12-hr. shifts. All drills are now operating 24-hr. per day, and 7 days a week.

DRILLING

As the early drilling was comparatively shallow, holes 8-in.

in diameter were started, reduction to 6 and 4-in. holes then being possible. When it became evident that deeper drilling would be required, with the possibility of having to case and reduce size of hole several times before reaching bottom, the starting hole was increased to 10-in. and finally to 12-in. diameter. This permits running 10-in. casing followed by 8-in., 6-in., and 4-in. casing when necessary.

There has been considerable discussion regarding the advisability of starting holes with 12-in. bits. It was thought that the larger diameter hole and the greater shock produced by heavier tools would increase caving in fractured ground. The drill contractors preferred the 8-in. bit because a larger bit would probably slow the drilling speed in the upper portion of the hole, where contract prices per foot are relatively low. It has been our experience, however, that the larger sized hole does not decrease speed of drilling greatly, and that caving would be encountered at about the same place in the hole regardless of size of bit used. The larger hole gives one better assurance of being able to drill to the depth required. The 12-in. hole is never carried to more than 1200 ft., and casing is usually required before this depth is reached. The 10-in. hole is carried to 1500 or 1600 ft. if possible, allowing the hole to be completed with either an 8-in. or 6-in. bit. The deepest hole, which was 2600 ft., was finished with a 4-in. bit after having set a 400 ft. liner of 4 in. pipe at 2400 ft. Some of the ground seems to stand well, as three holes, the deepest of which was 2200 ft., have been drilled without the use of any casing.

Drill locations and access roads are constructed to meet the requirements of the type of machine that will occupy the site. The

Bucyrus-Erie machines dump the sludge on the left side, facing the machine, while the Ft. Worths dump on the right side; thus necessitating a different type of location. All roads and locations have been made by bulldozer and scarifier, on contract; no drilling and blasting have been required.

Fishing for lost tools is taken care of by the contractors, with the exception of some long difficult jobs below 1000 ft., in which case the company has assumed part of the burden of cost. This situation has not come up often.

Unless casing is ordered by the field engineer to shut off possible caving and the resultant contamination of samples, especially in or near the ore zone, its use is usually left to the discretion of the contractors. The contractor is required to make every effort to recover all of the casing, but there are times when several joints have to be cut off the bottom in order to salvage any. This is accomplished by setting off a bomb, fired electrically at a coupling near the bottom, or by using a "collar buster"; a device lowered on the string of tools to a predetermined depth, which punctures and slits the pipe until a collar is reached and cut. The bombs are not too satisfactory because they tend to swell the casing in one or more places, making withdrawal of the pipe difficult.

The use of any drilling muds or Aquajell is, in general, prohibited. The exceptions to this rule are to permit the use of Aquajell, plastered on casing couplings when being run into the hole to make later withdrawal easier, and during fishing operations in caving ground. In crooked holes it is occasionally necessary to drop surface rocks in the hole and redrill in order to straighten it. In any of these operations the driller is required to thoroughly

clean the hole before sampling operations are resumed.

In order to accurately establish the depth from which samples are coming, the holes are measured frequently with a special measuring device consisting of a reel of 3/32 in. galvanized aircord marked with lead buttons every 10 ft. The 100 ft. marks are identified by a code of elongated buttons. The measuring cable was calibrated by attaching a 7-lb. lead weight to the zero end; lowering it down a vertical shaft; and marking off the units on surface. When measuring, a very definite jerk is felt when the weight is pulled off the bottom of the hole.

Frequency of measurement varies. Holes are measured when they near the contact of the conglomerate and monzonite, so this point can be closely determined. After that, measurements are made at about 200 ft. intervals, and again when the hole is completed.

SAMPLING

Equipment

Sampling equipment consists of a dump box, a 6-ft. launder, a sample splitter, and 5-gal. milk cans to catch the samples.

The dump boxes are built of 2 in. by 12 in. plank, lined with 1/4 in. steel plate welded at the joints. The boxes are 12 in. wide, 36 in. long, and 24 in. high--inside dimensions. The launder is constructed of 1/8 in. steel plate, and designed to fit securely, with a 3-in. overlap between the steel lining and wood exterior, at the open end of the dump box. The discharge end of the launder lies on the upper edge of the splitter unit. The slope of the launder is adjusted to insure an even flow of sludge to the splitter, which is a modification of the Ray Consolidated Company splitter[5]. It is, essentially, 3 or more Jones splitters arranged

in series to facilitate gravity flow of the sludge through the successive individual units. The upper two tiers of slots of a four cut splitter are 1-1/4 in. wide, and the lower two tiers are 3/4 in. wide; the slots being staggered by 1/2. The sample portion of the sludge passes straight down and is discharged at the front. The rejects gather underneath and are discharged at the side.

Personnel

It is desirable to have as samplers young engineers or geologists. This type of help has not been available, so it has been necessary to employ less skilled men, which in turn has meant special training and supervision for the sampling crew. Every detail of catching, preparing, washing character samples, and panning for concentrates had to be taught in most cases. Sampling is supervised by shift bosses and engineers who visit the rigs periodically during sampling operations.

Technique

The churn drill holes are sampled at 5-ft. intervals, from the contact of the conglomerate and the monzonite. to the bottom of the hole. Character samples are taken at 20-ft. intervals in the conglomerate.

The sludge is dumped from the bailer into the dump box, and flows through the splitter. The driller is cautioned to ease the bailer into the box to avoid surge and splash. The sample portion is caught in a 5-gal. milk can and a bucket of rejects is caught; a portion of which is washed free of slimes and saved as a character sample. A uniform volume of rejects from each 5-ft. interval is panned to obtain a concentrate which is dried and sacked. An engineer examines both the character sample and concentrate to de-

termine, megascopically, the rock type, alteration, kind, and relative amounts of mineralization. The character samples are filed for future reference, and the concentrate is sent to G. M. Schwartz of the United States Geological Survey for selected samples to be polished and examined by microscopic methods.

The sampler fills out a "Churn Drill Report", noting the depths at which the samples are caught, color of sludge, condition of hole, depth at which water was encountered, fault zones as evidenced by gouge, and any other pertinent data relative to the hole. The spaces for rock type and mineralization are filled in by the engineer who examines the character and concentrate samples.

The size to which the sample is cut depends largely on the size of hole being drilled, and if drilling is under water. A 12-1/2 in. hole may require a cut to 1/256 of the total volume of sludge bailed, and a 4 in. hole is cut to 1/8 or 1/4 of the original volume of sludge. The sample is cut so there is a minimum of about 15 lbs. of dry solids.

Drillers are required to bail holes clean. This is relatively simple above the water table, and takes about 3 or 4 trips into the hole. However, after the holes are under water, it is never possible to get a really clean hole, as there is always a certain amount of slimes in suspension. In this case, a few tests are made to determine how many bailings are required to pick up the largest part of the sample, and leave the hole relatively clean for the next run. On numerous occasions it has been possible to obtain samples of the slimes in suspension by carefully bailing a hole after it had been shut down for periods of 24 to 48 hrs. Assays of this material have agreed very closely with the assay of the last run made, so that no serious errors are introduced by leaving this material in the hole.

No attempt is made to shut the water off by casing.

Drying & Preparing Pulps

The samples caught in milk cans are gathered up, usually at the end of the shift, and hauled to a centrally located drying room. They are dried on a series of 9 heavy steel drying tables, each about 3 ft. square. Steam, which is generated to 20 lbs. gauge pressure in a manually-operated vertical fire tube boiler, using diesel oil for fuel, supplies the heat for the drying tables.

Other drying room equipment consists of a Jones dry splitter, 2 Braun type UA pulverizers, small platform scales, and a table for rolling pulps.

The operating personnel of the drying room consists of a foreman on day shift, with three dryers and grinders; and lead men on afternoon and graveyard shifts, with 2 or 3 helpers. The drying room operates 24 hrs. per day for 7 days a week.

As samples are brought in from the rigs, they are lined up in sequence and dried on the steam plates. Since there is such a large amount of sludge to be dried, the time of drying is more than an hour per sample, and may extend to 3 hrs. for exceptionally heavy, slimy samples. After drying, the sample is scraped off the table into a large pan and quarted by means of a Jones splitter. Three of these cuts are put in paper sacks; the fourth being set aside for pulverizing. Each of these cuts will weigh from 2-1/2 to 3 lbs. Paper tags are used, designating the sample number by hole as well as by footage. The cut which is pulverized and then thoroughly rolled, is divided between five different pulp sacks holding from 150 to 200 gms. each. One of these pulps is sent to the assay office of the St. Anthony Mining and Development Company for field control.

The other four are sent to the Magma Copper Company in Superior where one pulp is assayed directly for total and oxide copper; two are used in making up composites for total copper, oxide copper, gold, silver and molybdenum determinations. The fourth pulp is held in reserve.

Two of the three sacks of rejects are stored in tunnels at San Manuel, and one is stored at Superior. These are held in reserve for possible future examinations, or for further assay.

Records of the holes, listing each individual sample number, depth of footage, date of sample, weight of cut, number of splits made to obtain this cut, and the name of the sampler are kept in the bucking room. These records are all filed.

ASSAYING

The pulp sent to the St. Anthony Mining and Development Company is assayed for total and oxide copper. The method of assay used is a modified procedure of the short iodide method, wherein the end point of the starch-iodine blue was made much sharper, and gives results close enough for field control.

At the Magma Copper Company the total copper content is determined by standard electrolytic methods; oxide copper is determined by the long iodide method. Results of these assays check very closely with those of Ledoux & Company and Union Assay Office.

Gold, silver and molybdenum are determined from composites. These composites are made up of pulps representing portions of the hole of the same general mineralization or assay character, but do not usually exceed 100 ft. of hole.

If the average of individual assays of total copper checks within 0.03 percent with the total copper determination of the com-

posite by Magma Copper Company and an umpire assayer, those values are accepted as being correct.

In the office at Superior, records of each hole are compiled showing all determinations made on individual and composite samples, the accepted assay, rock type, and mineralogy.

TONNAGE & GRADE

Churn drilling to May 1st has blocked out approximately 119,170,000 tons of ore, which will average 0.80 percent copper. This total contains 14,600,000 tons of low grade oxidized material which averages 0.65 percent total copper, 0.36 percent oxide; 29,640,000 tons of mixed oxidized and sulphide ore which averages 0.82 percent total copper, and 0.58 percent oxide copper; and 74,930,000 tons of sulphide ore which averages 0.82 percent total copper, and 0.05 percent oxide copper.

These tonnage and grade figures are of a preliminary nature, and are kept up to date with the drilling as closely as the required checking of assays allows.

The following procedure is used to compute the tonnage and grade:

The drilling area is divided into rectangles and triangles as the spacing of drill holes demand, with a drill hole at each corner of the rectangles; and at the apices of each triangle. Care is taken to arrange the pattern of geometric figures so that one drill hole is not a common corner for more figures than necessary. The area, calculated from the horizontal distance between holes, is multiplied by the average of the thickness of the columns of ore encountered in the holes making up the corners of the figures.

Twelve and one-half cu. ft. per ton is the factor used to convert the volume to tonnage. The grade for each block is the weighted average of the ore columns of the holes used.

The bottom sample, or in some case, more than one sample is thrown out as a precaution against salting of marginal material by higher grade samples above. The cut back at the bottom of an ore column depends on the condition of the hole, grade of ore, and whether there is a gradational change in copper content at the bottom of the column, or a sharp break. The tonnage is calculated from drill hole data with no allowances for an extension of the ore body beyond the boundaries of completed holes. This eliminates the necessity for constant revision of such an estimated extension of ore, due to the continually changing drilling pattern.

At intervals tonnage and grade calculations are computed by two other methods as checks. In the second method, the average area of adjacent cross sections is multiplied by the distance between the sections to compute the volume. The grade is computed from the weighted average of the ore columns of holes in the two sections. The ore columns are adjusted as in the first method.

The third method involves plotting the area of influence of each hole, and using the adjusted ore column for thickness, to compute a tonnage and grade figure for each area.

ACKNOWLEDGEMENT

Acknowledgement is due the officers of San Manuel Copper Corporation for permission to use the information presented.

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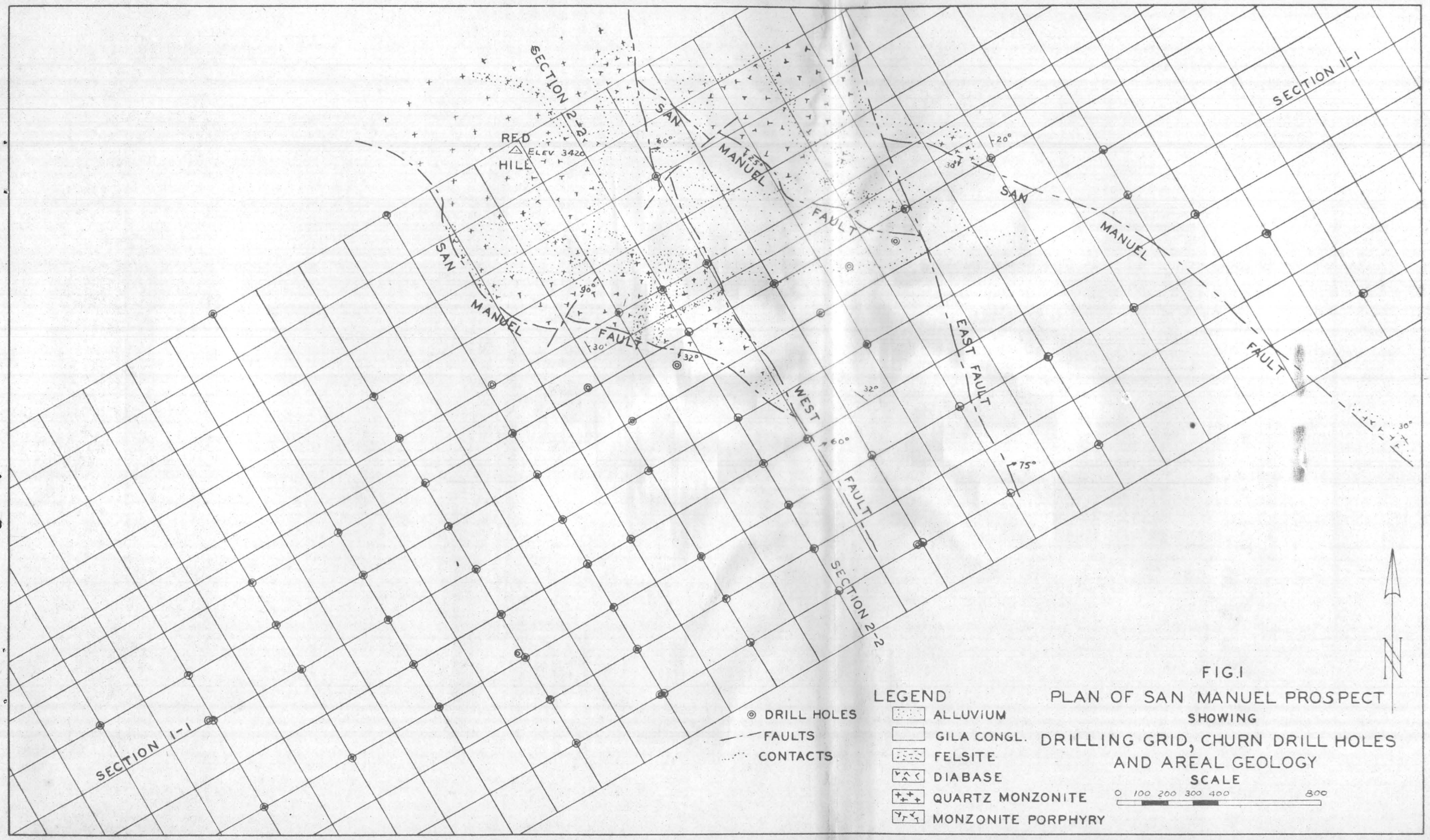

FIG. I

PLAN OF SAN MANUEL PROSPECT

SHOWING

DRILLING GRID, CHURN DRILL HOLES AND AREAL GEOLOGY

SCALE

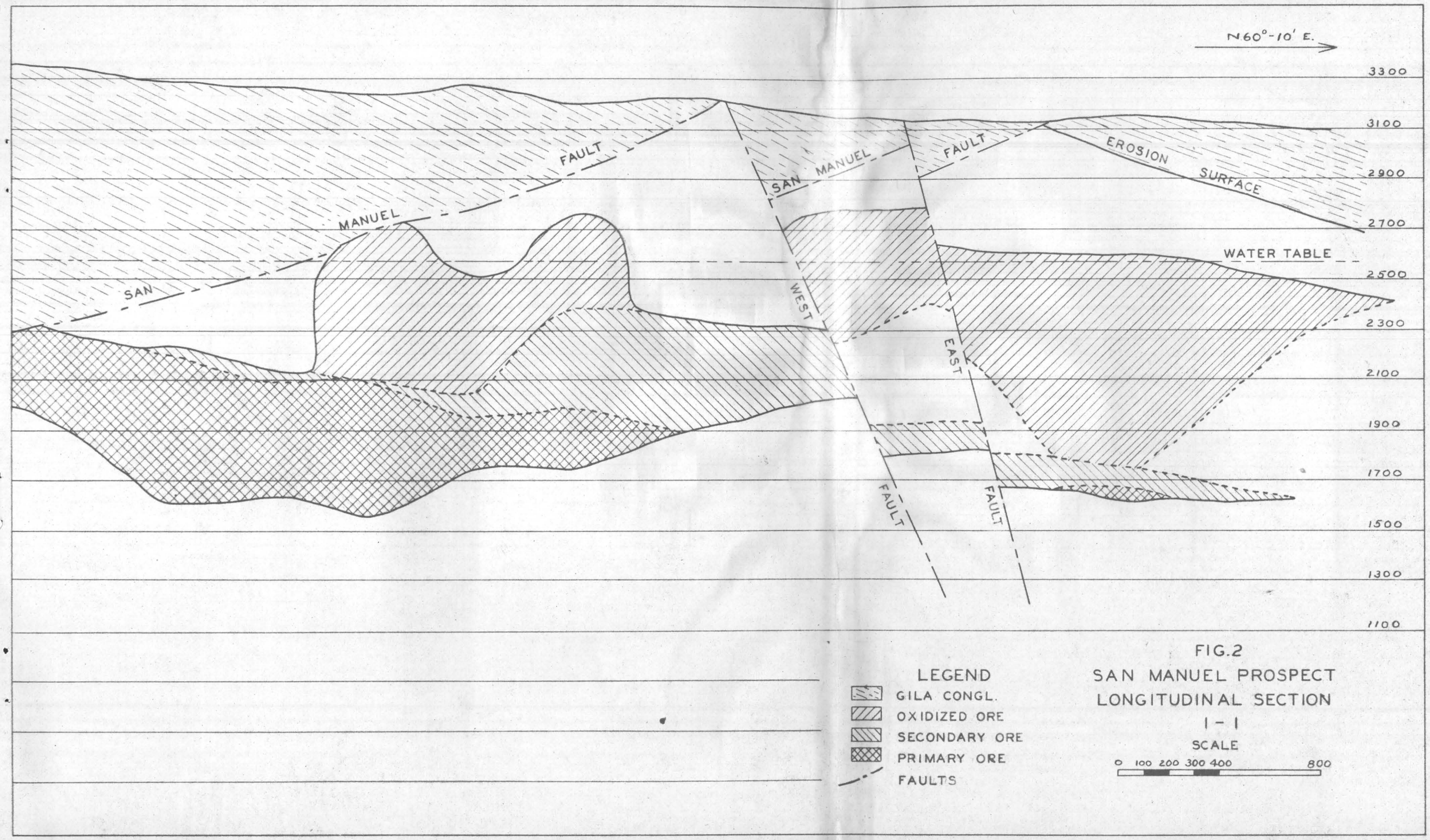

FIG. 2
SAN MANUEL PROSPECT
LONGITUDINAL SECTION
1 - 1

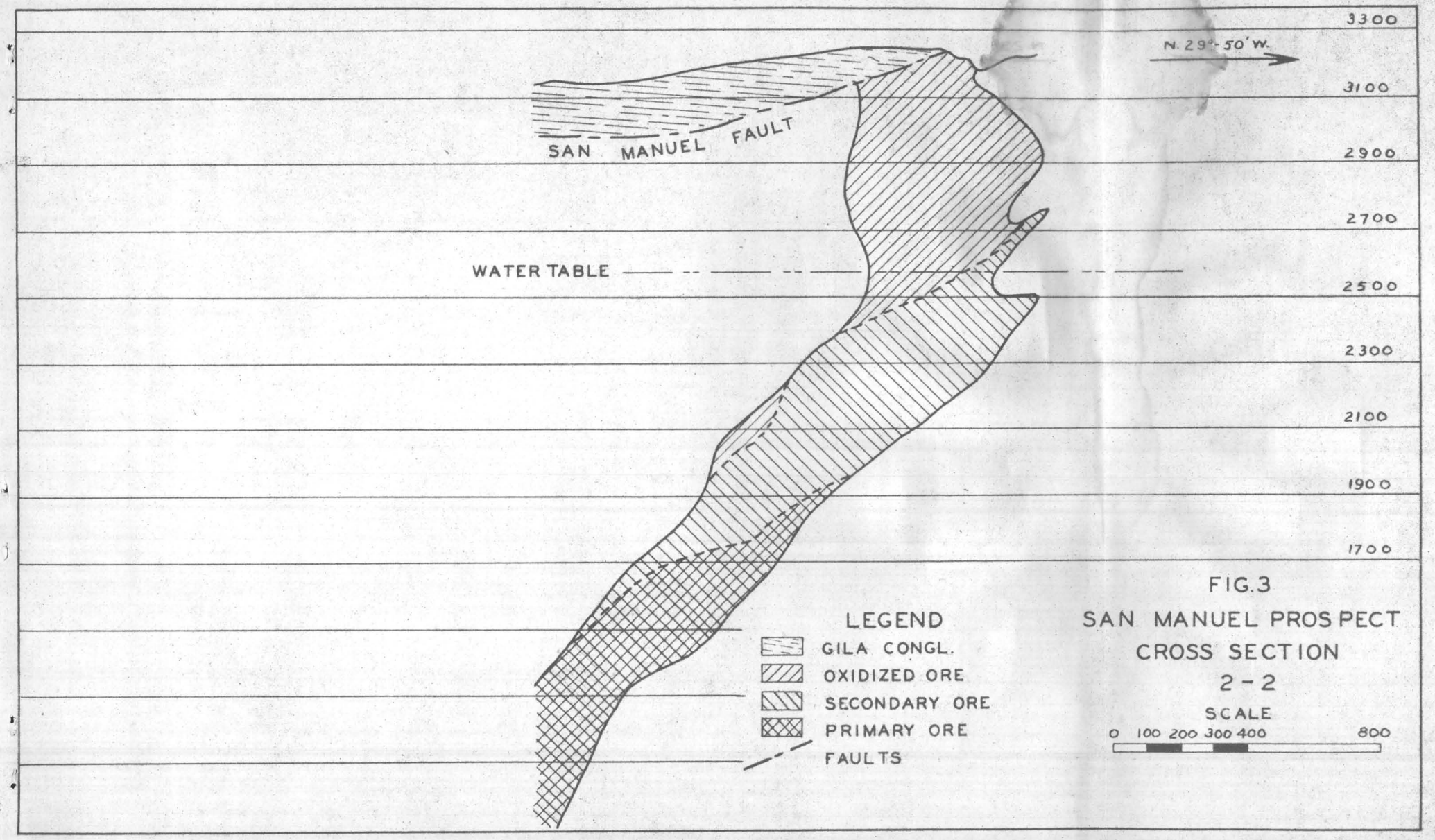

FIG. 3
SAN MANUEL PROSPECT
CROSS SECTION
2 - 2

R. B. MULCHAY
FEB 1954

SHAFT WORK AT SAN MANUEL

American Institute of Mining and Metallurgical Engineers

Arizona Section

December 7, 1953 Tucson, Arizona

C. L. Pillar, Development Superintendent

SAN MANUEL COPPER CORPORATION

SHAFT WORK AT SAN MANUEL

Since the beginning of the actual underground mine development work at San Manuel started in the spring of 1948, we have sunk two shafts and are now in the process of starting sinking operations in three new additional shafts.

The sinking of our No. 1 and No. 2 Shafts has been very well covered by two previous papers given by Mr. Buchanan: one given to this section of the AIME in 1948 and one at the American Mining Congress in 1951. In view of this, this paper will handle the three new shafts, outlining the design features of the shafts and the methods that are and will be used to sink them, based on the experience obtained during the sinking of No. 1 and No. 2 Shafts.

A review of our past sinking operations is briefly submitted.

No. 1 Shaft having four compartments--two 6'-0" x 6'-6" and two 6'-0" x 5'-0"--with the ground supported by steel sets lined with reinforced concrete was sunk through relatively loose blocky quartz monzonite to a depth of 1643 feet at an average rate of 88.1 feet a month, 2.93 feet a day, or .186 feet per manshift. The first 1100 feet of the shaft was sunk dry. Water was encountered at 1100 feet depth and by the time the shaft bottomed at 1643 feet depth we were obtaining an inflow of 1130 GPM. Our best speed was made from 500 to 800 feet depth; in two months we sank and concreted 310 feet or 5.25 feet per day or .292 feet per manshift.

No. 2 Shaft is a three compartment timbered shaft, each compartment being 5'-6" x 5"-7½" was sunk to a depth of 2064 feet. The

first 710 feet through firm Gila Conglomerate, the balance through broken altered monzonite prophyry, the mineralized rock of the area. This shaft was sunk at the rate of 110.5 feet per month, 3.68 feet per day, or .307 feet per man shift. Our best month was 138 feet. Water was first encountered at 317-foot depth, and by the time the shaft was bottomed at a depth of 2064 feet, we had a total shaft inflow of approximately 1300 GPM.

No. 3A and 3B Shafts

In order to hoist the 35,000 tons per day mine production at San Manuel, two identical four-compartment shafts are being sunk. These shafts are 195 feet apart and will be known as 3A and 3B Shafts. In addition to handling the ore hoisting, they will exhaust an estimated 400,000 CFM of mine air.

Each shaft will have four identical compartments, 6'-6" x 7'-0" in the clear, requiring a rock excavation of 31'-6" x 9'-2" or approximately 288 square feet in area, and will be sunk through Gila Conglomerate to an approximate depth of 1750 feet. Both shafts will have two large permanent stations, one located on the 1475 level and one in the vicinity of the 1400-foot level. The upper stations will receive the majority of the exhaust ventilation which will by-pass the 1475 level pocket area. The 1475 stations will serve as access into the shafts for men, supplies, and ventilation. A 275-foot sump below the 1475-foot level will provide the necessary room for dumping pocket, skip loading station, spill pocket, sump pumps, and provision for future sinking operations. The 3B Shaft will contain three temporary booster pump stations at depths of approximately 375, 750, and 1125

feet. The number and location of temporary booster stations in 3A Shaft will depend on the ability of 3B Shaft to handle the water in the No. 3 area.

Both shafts have been collared and are now at a depth of approximately sixty feet. Each shaft has a 6'-6" x 5'-0" concrete service tunnel entering the manway compartment ten feet below the surface. The tunnels extend in a giant "Y" from each shaft through the center of the permanent hoist house to the electric sub-station. It will house all pipe lines, power lines, signal and control circuits that will be carried down the shafts from the surface installation.

Both shaft collars were installed by excavating 25 feet of unsupported raw shaft using a clam shell on a crane for the mucking operation. Five 12" WF 64-pound bearer beams, 36 feet long, were placed across the top of the shaft at the collar elevation. They were included in the collar fabrication and supported the immediate sets hung below. After placing the collar concrete the extended ends of the bearer beams were cut off flush with the outside of the concrete shaft walls.

It was found advantageous to place the Riddell Mucker in the shafts as soon as possible. In both cases the mucker was installed when the shafts were approximately 25 feet in depth, placing the track and running gear on the second set and using the crane to handle a two-ton muck bucket. The remainder of the collar excavation and concreting procedure was typical of our proposed sinking operation which is described in full.

CREW. The sinking of each shaft will be done by a crew of six men per shift in the bottom who will perform all operations of drilling, mucking, placing steel, concreting and installing manway and service

lines on contract. A top lander, truck driver, and hoistman will provide the necessary surface service for each shift. In order to better coordinate the surface service with the crew on the bottom, a two-way blast-proof loudspeaker system is to be installed. This will allow the bottom crew to talk with the surface top lander and hoistman, and also allow the top lander to talk with the crew in the bottom.

HOISTS. Sinking hoists for both shafts are double-drum clutched hoists driven by a 250-HP motor, have a 15000-pound rope pull and hoisting speed of approximately 850 feet per minute. Each drum will wind 3000 feet of 1 1/8" rope, carry a 1500-pound crosshead locked to the rope, and a 38-cubic foot sinking bucket hooked six feet under the crosshead. The sinking hoists will operate in the No. 1 and No. 2 compartments of each shaft.

The fourth compartment of 3B Shaft will be serviced by a 100-HP single drum hoist with a rope speed of 350 feet per minute. Its main function will be to handle the sinking-pump gear in the bottom.

DRILLING AND BLASTING. All drilling is with hand-held jack-hammers of the 55-pound class. A common air and water manifold, equipped with line oilers, serves as a rack to carry the drills with their individual hoses in and out of the bottom as a unit. Alloy drill steel and "one pass" friction-type bits complete the drilling equipment.

The conglomerate is relatively easy to drill, but at times rather tough to pull. To date we are using about fifty-six 6½-foot holes. Fourteen rows of four holes with a relieved "V" cut across the center of the second or third compartment. Holes are drilled approximately 6½ feet deep to pull a six-foot round. To date very little overbreak

has been experienced on the sides. Holes are loaded with 125 pounds of straight 40% gelatin and detonated with regular delay electric blasting caps. Loading to date has been one stick of powder per foot of hole with a 2½- to 3-foot burden.

MUCKING. The Riddell Shaft Mucker, with counter-weighted 3/8-yard clam shell, handles the mucking operation in both shafts. To date the mucking track has been set on the steel sets, covering the first three compartments. The dividers are left out of the steel sets hung under the mucker and are placed after the mucker is moved down. To date the ground has been good and can stand a 20- to 30-foot unsupported sump. In the event the ground does require close support to the bottom, the full steel set will be placed minus the No. 2 divider. This allows the two middle sets open to work in. Good operators can reach the muck in the end compartments by the pendulum swing of the clam. The bucket in the No. 1 compartment can be swung into the No. 2 compartment for loading at the bottom. However, the open sump at the bottom allows freer, more efficient use of the Riddell Mucker. The entire crew is used in the mucking cycle; one on the mucker, one on the hoist signal bells, one guiding the clam in the bottom, the balance used picking down the walls and squaring out the corners. The last few tons of loose muck are either hand loaded or scraped to the center and left.

The muck is hoisted to the surface bin in 38-cubic foot buckets. The bucket dumps consist of an underslung car moved in and out of the shaft by an air cylinder in which is incorporated a conventional key-hole slot and slide which overturn the bucket by securing the tail chain. This car completely covers one compartment of the shaft while the bucket is dumping. The complete surface operation, including setting

the crosshead on the dumping chairs, is controlled by the top lander from the top of the muck bin. It is the same type of dump used on our No. 1 and No. 2 Shafts with good success from an operational, maintenance, and safety standpoint.

GROUND SUPPORT. The ground support is composed of steel sets on 6-foot centers set in a minimum of twelve inches of heavily reinforced concrete with the inside of the shaft steel positioned six inches inside the concrete. Reinforced concrete curtain walls, twelve inches thick, support the shaft at the dividers. A 4'-0" x 3'-0" arched window is formed in each curtain wall of each set. This is to provide for attachment of the guide brackets on 3'-0" centers and acessibility from one compartment to another. These shafts are different from those we have previously sunk in that their entire structural steel support is incased in reinforced concrete with the exception of the areas where the guide brackets are attached. This has been done to insure permanency and strength, to prevent excessive corrosion, and to streamline the shaft to handle the large volume of exhaust air at maximum efficiency.

In placing the ground support the steel sets are hung first, then lined and blocked with temporary two-inch lagging between the steel and the ground where required. Past practice has been to support 48 to 60 feet of shaft in this manner before concreting. The sets themselves consist of 6" H 25-pound end and wall plates 11½"-wide box beam dividers fabricated with two vertical six-inch channels and horizontal 3/8" x 4" plate strapping. Each set is hung from ten 3½" x 3½" x 5/16" angle studdles.

At the beginning of the concrete cycle the set, 48 to 60 feet below the last concrete and approximately 18 to 30 feet above the bottom, is

sealed off, and the concrete placed back up the shaft to the previous pour, one set at a time. The reinforcing steel is composed of horizontal 3/4" diameter rings on six-inch centers with vertical 3/4" rods on twelve-inch centers. It is made in baskets or panel sections welded together on the surface. This practice has proved a time saver on the bottom and insures the correct placing and spacing of the bars in the shaft.

The concrete forms are comparable to steel lagging fourteen inches wide and weighing ninety pounds. They are supported and held in place vertically by walers of light 6" WF supported from the dividers and end plates. Each compartment is formed on all four sides with special steel arch forms bolted between the curtain wall panels and on the outside of the end wall panels to form the necessary windows and recesses for the guide brackets. The form panels are bolted together with speed bolts and blocked in place against the inside flange of the walers. The concrete side of the forms are brushed with form oil prior to pouring. The forms are stripped in a minimum of twenty-four hours and used on the sets above with the exception of the window and end wall recess forms which are left a minimum of seven days.

The concrete is a 1:2:4 mix of cement, screened river sand, and crushed rock with the addition of posolith to give high early strength. The aggregate is fed to a one-half yard electric-powered mixer at the shaft collar from a 40-ton batch plant. From the mixer the concrete is dropped through an eight-inch pipe into a small bottom gate hopper hung just above the pouring floor. Small rubber tired buggies are loaded from the hopper, trammed and dumped into the forms. This method of placing the

concrete cuts down on aggregate segregation and brings the pour up evenly in the forms and hence avoids pushing the sets out of alignment. With the standard seven-man shaft crew the concrete can be placed in the forms as rapidly as it can be properly mixed on top. The concrete is vibrated in the forms during the pouring by compressed air vibrators.

When water is encountered in these shafts the placing of good concrete will becaus considerably more difficult. Our best results under heavy water conditions in No. 1 Shaft were obtained by making our forms as water tight as possible and allowing the collected water to overflow the forms. This meant placing the concrete under water and allowing the excessive water to decant off. This procedure caries some cement and fines with it, but considerably less than when the water is washing through the forms. Roofing over the forms, back forming around concentrated flows, and piping through the forms have been tried with only moderate success, mainly because, regardless of the time and effort spent in such precautions, there is still ample water trapped in the forms which will badly wash sections of the concrete unless the forms are water tight. In pouring under wet conditions we have increased the cement in our mix by 50% to offset the loss in the forms. Test cylinders taken in the forms have averaged approximately 3000 RSI dry and the same under wet conditions when the proper precautions are made. One and one-half-inch diameter "weep holes" will be drilled in both concrete side walls of each compartment on 12-foot vertical spacing to relieve the hydrostatic pressure in water-bearing ground.

WATER-PUMPING. The amount of water that will be encountered in these shafts is purely problematical. Our geologists have estimated that we will first encounter water in the Gila Conglomerate at a depth of approximately 300 to 350 feet. Their statement is based on water table data from churn drill holes in the surrounding area. At our No. 2 Shaft water was encountered in the Gila Conglomerate at 300-foot depth and at the bottom of the Gila, 710-foot depth, the inflow was 300 GPM. If this is any indication we can expect from 1000 to 1500 GPM inflow into the shaft by the time it is bottomed at 1750 feet depth.

It is hoped we can keep 3B Shaft in the neighborhood of 200 feet ahead of 3A Shaft in order that the heavy pumping in the area can be accomplished through one shaft which will be equipped with an additional auxilliary pump hoist.

The first water encountered will be handled in the bottom by an eight-stage deep-well type pump with 440-volt, 1750 RPM, 75 HP driver, having a rated capacity of 500 GPM at 420 feet of head. This unit is close coupled, 14 feet long and weighs 4000 pounds. Similar units handled the water in the bottom of No. 1 and No. 2 Shafts very successfully during their sinking operations. The capacity will be larger than required at first, but the ease with which this pump is handled with an auxilliary hoist presents no more problem than a smaller unit. The pump will be connected to an 8-inch discharge line by a 50-foot length of $4\frac{1}{2}$" high-pressure hose. Power will be transmitted from the surface by 600-volt No. 2 portable cable and controlled by a 110-volt manual switch which actuates the starter on the surface. The set-up is flexible, allowing the pump to be lifted and lowered during blasting

or mucking without delay for breaking or making connections. The pumps require no priming or by-pass lines; as long as the lower bowl is submerged it will handle any amount of water up to its capacity.

A small booster station will be cut in the neighborhood of 375-foot depth to provide for pumps to relay the sinker discharge to the surface. This station will be the full width of the shaft--12 feet high and 25 feet deep. It will accommodate four 100 HP, 440-volt, 3600 RPM, single stage horizontal centrifugal pumps rated at 600 GPM at 425 feet head, and four 500-gallon surge tanks that will feed the booster pump suctions. Pumps are operated automatically by float switches in the surge tanks.

Additional booster stations will be placed at approximately 375-foot intervals; as the sinking progresses this will place them at 750-, 1125-, and 1475-foot depths, the 1475 being the main haulage level station.

This sinker-pump discharge will be put through 10-inch cyclone desanding cones in parallel before emptying into the 500-gallon surge tanks. The desanding cyclone will be a new addition to past shaft pumping procedures and somewhat of a "guinea pig." Its purpose will be to take most of the coarse sand from the sinker discharge before it is run through the series of booster pumps, thus eliminating in part the excessive maintenance on this pumping equipment.

In order to power the 1125- and 1475-foot stations, it will be necessary to bring 2300-volt feeder lines to the 1125 and set up the required transformer capacity to handle our 440-volt power there and below. We plan to use the relatively new air-cooled transformer packaged units for this temporary power installation.

To prevent large quantities of water falling freely in the shaft

over vertical distances in excess of 50 feet, water rings will be formed by leaving the concrete lining out for a vertical distance of two feet above each concrete "seal off" set which is spaced approximately 48 feet apart. A headboard around the bottom of this recess will form a ditch 18" x 12" around the perimeter of the shaft. This ditch will trap most of the water falling down the shaft walls from the weep holes and leakage through the concrete above. A 300 GPM motor pump, set below the ring and fed by gravity from it, discharges the water into the sinker columns.

As the upper rings dry up with the shaft advance the pumps are moved to a lower ring. It will also be possible to connect two or more rings together in order to concentrate water in the lower rings for pumping.

The water rings serve two purposes: the first improving the working conditions at the bottom of the shaft and, secondly, they keep the load on the sinker pumps down. Our past experience has been that a sinker in good condition will lift 500 GPM of clear water 400 feet, but drops off to 350 GPM when worn and handling dirty water during the mucking operation. We can handle two sinkers on the bottom where necessary but prefer to trap as much of the water as possible before it gets to the bottom.

The wear and maintenance on the sinker pumps will be high. Past experience has shown the life of a sinker pump will be about 48 feet of shaft excavation and concrete work. During that time it will have handled as high as 150 tons of sands and slimes in the water it has lifted, most of which will be taken out by the desanding cyclone before entering the booster circuit.

VENTILATION. The sinking operations will be ventilated by 7000 CFM of fresh air carried in 24-inch light-walled vent pipe, the discharge of which is usually kept at the bottom of our shaft concrete. One $7\frac{1}{2}$ HP, 440-volt, axial vane fan, rated at 7000 CFM under four inches of water gauge and located in the surface service tunnel far enough back from the saft collar to prevent picking up the exhaust air, supplies the ventilation for each shaft. Past experience has shown that as soon as the shafts become wet the fans will be used only immediately after blasting.

The shafts will be sunk using the permanent headframes which are conventional "A" type frames 181 feet in height. Sheaves for sinking will be set in the frame with their centerline 90 feet above the collars. Temporary muck bins of 180-ton capacity will be fabricated in the headframe structure. This will leave one side of the shaft free for sinking operations. At the end of the sinking operation, a 5000-ton ore storage bin will be erected at each shaft. The temporary sinking bins will be removed and the permanent sheaves installed on the top of the headframe to complete the surface installation.

The finished shafts will provide for two 18-ton capacity "Jeto" type bottom dump skips in balance running on steel guides at a speed of 3000 feet per minute in the No. 2 and No. 4 compartments. The No. 3 compartment located between the two skipways will serve as the manway which will carry ladders, piping, and power cables. The number one compartment will carry an auxilliary service cage to handle men, maintenance supplies to the loading stations, and shaft spillage. In short it will be the service elevator. Each shaft will be a separate identical operating unit capable of handling one-half of the mine production.

The permanent ore hoists, which will be installed and placed in service at the end of the sinking period, will be two of the highest powered hoists in the world. Each identical unit is capable of delivering 860 tons per hour from a maximum depth of 3000 feet or 18,000 tons in a 21-hour operation. The hoists will be double drum with geared dual D.C. motor drive. The drums will be cylindrical, 15 feet in diameter with 116-inch face and wind 3000 feet of 2¼" rope in two layers. Each hoist will be powered with two 3000 HP DC 600-volt motors. A fly wheel motor generator set for 80% power peak equalization powered with a 4000 HP, 2300-volt induction motor driving two 2500 KW generators on each side of the fly wheel will furnish the direct current power for the dual hoist motors.

These hoists will be equipped for semi-automatic operation; that is, the skip tender in the underground loading station will press a button as soon as the skip is loaded. This will start the hoist and take it through its entire cycle automatically until the skip comes to rest in its dumping position in the headframe. At that time the other skip is spotted in the loading position at the underground station and the cycle is repeated.

No. 4 Shaft

The No. 4 Shaft is being sunk 600 feet to the east of our present No. 1 Shaft. It will be a part of the main mine service and supply yard. Its primary function will be to handle caged supplies from the surface to the underground mining operations and provide for the intake of approximately 300,000 CFM of the mine ventilation.

The shaft is roughly square with curved ends. It has two 8'-0" x 14'-0" in the clear cage compartments and two semi-circular end compartments five feet wide at the apex of the curve, one of which provides manway, pipeway, and serviceway for power lines. The other side provides additional cross-section area for the large flow of ventilation air and can be converted to additional pipe and cable ways when such is required. The shaft will require a rock excavation of 16'-0" x 27'-6" or approximately 400 square feet in area and to my knowledge will be the largest shaft to date sunk in this state. The shaft will be sunk to a depth of 1550 feet through relatively dry altered blocky quartz monzonite.

It will have three large stations. The shaft entrance to them will be placed during the sinking operation by the sinking crews. A ventilation station at 1350-foot depth will receive the major portion of the ventilation air, keeping it off the lower supply stations. Supply stations will be located at both the 1415 and 1475 levels. The shaft will bottom at 1550-foot depth allowing 75 feet of sump below the 1475 level for sump pumps and provision for future sinking operations.

No. 4 Shaft has been collared and is now 96 feet deep with the finishing touches being put on the headframe and sinking hoist installations. It is planned to start full sinking operations with a contract crew some time next week.

A 6'-6" x 5'-0" concrete service tunnel connects the manway end of No. 4 Shaft with the sinking hoist house, permanent hoist house, and electric substation. Similar to the No. 3 installation, it will house

all pipe and electric lines that will be carried down the shafts from the surface installations.

In general the methods and equipment previously described in No. 3A and 3B Shafts are being used to sink No. 4 Shaft.

No. 4 Shaft differs in its drilling operation because of the larger excavation area, and requires 72 drill holes, 12 rows of six holes using a conventional "V" cut across the center of one of the cage compartments. Approximately 125 pounds of powder are used on a 6-foot round. Loading averages roughly 3/4 of a stick per foot of hole. The altered quartz monzonite breaks and pulls with greater ease than the tougher conglomerate at No. 3, but also presents more of a problem in overbreak.

Due to the size of No. 4 Shaft, two identical Riddell Shaft Mucking units are used to good advantage. Each unit runs the width of the shaft in a cage compartment. Most of the muck at the ends of the shaft is easily handled by the clam shell when it is swung sideways.

Each mucker loads the bucket working in its compartment. Hoisting during the mucking cycle is in balance with the exception of dumping the bucket in the headframe and moving the lower bucket from its hang-up point to the bottom of the shaft which is done singly out of balance.

The ground in No. 4 Shaft requires immediate support within eight to ten feet of the bottom. This is accomplished by hanging the regular steel sets and lagging the ground behind them with regular two-inch lagging. This support close to the bottom does not interfere with the Riddell Mucking operation.

The ground support, like the No. 3 Shafts, is composed of steel sets on 6-foot centers set in 12 inches of heavily reinforced concrete. Here reinforced concrete is used only around the perimeter of the shaft;

no curtain walls are poured. The sets have wall and end plates of 6" H 25-pound beams with 8 WF 31-pound dividers across the width of the shaft at the ends and 8" x 10" WF 39-pound beams through the center. Dividers are supported on each end by posts of heavy channel fabricated back to back. The center dividers use an additional 8-inch 15-pound B section post through the center to give them additional strength. This heavy posting of the dividers is done to stiffen and strengthen them across the 14-foot span in order that the alignment of the four guides in each cage compartment will not be subject to distortion after the shaft is placed in operation. The circular ends are also designed to be self-supporting and prevent transmission of ground weight on the sides of the end dividers which may result in distortion of guide alignment. Each set is hung from fourteen $3\frac{1}{2}$" x $3\frac{1}{2}$" x 5/16" angle studdles.

The concrete line is placed six inches inside the steel and allows a minimum of 12 inches of concrete. The forming and placing procedure is similar to that used at No. 3 Shafts. It is simpler and faster than the No. 3 process due to the absence of curtain walls.

Due to the fact that No. 4 Shaft is being sunk a distance of 600 feet from No. 1 Shaft, which is 1643 feet deep and has been draining this area for the past two years, we do not expect any appreciable water in the sinking of No. 4. We will also be under No. 4 Shaft with drifts on the 1415 and 1475 levels shortly, which should complete the drainage of this area in time for No. 4 to be sunk dry. In the event some water may be encountered, it cannot be far above the 1415 level and can be drained through it by the use of diamond drill holes.

The ventilation during the sinking of No. 4 Shaft is similar to that described at the No. 3 Shafts.

This shaft is being sunk using its permanent headframe which is conventional "A" type frame 109 feet in height. Small 6-foot diameter sinking sheaves are set on the permanent sheave bearing plates for the sinking operation. A temporary muck bin of 180-ton capacity is placed on the open side of the headframe. At the end of the sinking operation this bin will be removed and the permanent sheaves installed to complete the surface installation.

The finished shaft will provide for two $6\frac{1}{2}$' x $13\frac{1}{2}$' double deck supply cages in balance each traveling on four 4" x 6" mahogany guides at a speed of 1500 feet per minute. Each cage will handle 100 men, an eight-ton development locomotive, a loaded development car of concrete aggregate or track ballast, or two fully loaded 11-foot timber trucks. It will be possible to cage all portable underground equipment through this shaft with the exception of the main haulage locomotives and ore cars.

The permanent service hoist which will be installed and placed in service at the end of the sinking period will be a double drum, geared dual DC motor drive. The drums will be cylindrical, 15 feet in diameter with 90-inch face and wind 3000 feet of $2\frac{1}{4}$" rope. The hoist will be powered with two 700 HP, 600-volt motors. A motor generator set powered with a 1750 HP, 2300-volt induction motor driving two 600 KW generators, one on each side, will furnish the direct current power for the dual hoist motors.

Acknowledgment is made to Mr. J. F. Buchanan, Mr. L. I. VanDalsem, and members of the San Manuel Copper Corporation staff for the help received in the preparation of this paper.

GEOLOGY OF THE SAN MANUEL MINE

ROCKS

The San Manuel orebody is a disseminated copper deposit in quartz monzonite, monzonite porphyry, and diabase.

Quartz monzonite, which is Pre-Cambrian[1] in age, is exposed in an area including No. 1 and No. 4 shafts and extending about 2,400 feet southeast from No. 1 shaft. It is also exposed in the bottom of No. 2 shaft and in a zone roughly 500 to 1,200 feet southeast of No. 2 shaft having its long dimension along a northeast-southwest axis. Churn drilling has shown all these exposures to be parts of a contiguous mass of quartz monzonite lying within and beneath the ore zone.

Late Cretaceous[2] monzonite porphyry comprises the bulk of the remaining rock encountered below the conglomerate. In general, the porphyry occurs in two large masses that are separated by a projection of quartz monzonite south of No. 2 shaft. The porphyry masses converge to the east. The north area of porphyry is a sheet-like[3] mass lying above quartz monzonite. The south area has not been bottomed by any drilling south of the orebody and may be a stock.[4] Monzonite porphyry also occurs as dikes cutting quartz monzonite. These dikes become more numerous with proximity to the larger masses of porphyry and generally trend northeast-southwest.

Several diabase dikes, intrusive into quartz monzonite and monzonite porphyry, have been exposed in the mine openings. All the diabase dikes are mineralized similarly to the rocks they intrude.

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[1] References and notes appear at the end of the paper.

Andesite porphyry dikes have been noted in a few instances. Some of these dikes are unmineralized while others are lightly mineralized with pyrite. The mineralization and hydrothermal alteration in all cases are of a different character than that of the intruded rock. Andesite porphyry is probably, as Schwartz[5] suggests, the intrusive equivalent of the lava flows of the Cloudburst formation, the mineralization being of a later age than the ore mineralization.

The Cloudburst formation was cut by No. 3A and No. 3B shafts and by drifting on the westernmost part of the 1475 Level. This formation contains two members. The upper member is a conglomerate made up mainly of quartz monzonite fragments but containing some fragments of all the older rocks in the area. The matrix tends to be arkosic, but granitic sand and gravel are not uncommon. The conglomerate is poorly cemented but relatively well sorted compared with the overlying Gila(?) conglomerate, having beds of sand and gravel alternating with beds of larger boulders. Slickensides are frequently seen on bedding planes. Tuff beds are found in the conglomerate and in the underlying member which is composed of interbedded flows, flow breccias, and conglomerate. Where not distorted by drag adjacent to the San Manuel fault, the average strike of the conglomerate is N 5° W and the average dip is 30° to the east.

The contact of the Cloudburst formation with the overlying Gila(?) conglomerate in No. 3A shaft was an unconformity. In No. 3B shaft the contact was marked by an irregular tuff bed but here, and in the exposures of this contact on the 1475 Level, the bedding of the two formations is parallel or very nearly parallel and, in the 1475 Level exposures, it was not possible to pin-point the exact contact due to the similarity of the two formations.

A small tuff bed near the bottom of 3A shaft was mineralized with pyrite and a trace of chalcopyrite. This mineralization is later in age than the ore mineralization since the Cloudburst formation is not mineralized where it is seen in depositional contact with mineralized porphyry on the surface.

Rhyolite dikes were intersected in several places on all the levels of the mine. Rhyolite is not mineralized by hypogene copper minerals but is occasionally stained by migratory chrysocholla where the dike is in, or close to, the zone of oxidation. Chlorite was noted coating joint faces of several rhyolite dikes. Rhyolite dikes are known from surface exposures to intrude the Cloudburst formation but do not intrude Gila(?) conglomerate.

Gila(?) conglomerate was cut in the upper parts of shafts Nos. 2, 3A, and 3B, and on the 1475 Level southwest of No. 2 shaft. Fragments composing the upper portion are derived mainly from basic flow rocks while the lower beds are derived from granitic rocks. The Gila(?) conglomerate contains no hypogene mineralization or alteration. The average strike is N 50° W and the average dip is 35° to the northeast.

STRUCTURE

The oldest of the important post-mineral faults is the San Manuel fault. In its underground exposures, this fault has an average strike of N 66° W and an average dip of 26° to the southwest.

Typically, there are from one to three feet of red gouge and slickensides on the fault surface. On the footwall side, immediately beneath the red slickensides, are from five to fifteen feet of gray gouge

with irregular slick faces aligned more or less parallel to the fault and sometimes containing harder fragments with slightly rounded edges. Beneath the gouge, the underlying rock is badly fractured and grades into less fractured, normal rock about 50 to 100 feet below the fault surface.

The conglomerate found in the hanging wall in exposures east of the Hangover fault is not badly disturbed but contains a few small slips close to the fault surface. West of the Hangover fault, besides showing small slips adjacent to the San Manuel fault, the Gila(?) and Cloudburst conglomerates show considerable drag for a distance of 300 feet above the plane of the San Manuel fault.

There are a number of faults which strike N 20° W to N 25° W and dip steeply to the east. These faults exhibit varying amounts of normal fault movement and are younger in age than the San Manuel fault. The most important of these are known as the East fault, the West fault, and the Hangover fault, the latter being in reality a zone of faulting.

A large reverse fault lying between the north and south limbs of ore was intersected by drifting on the 1475 and 1415 Levels. This fault, which is known as the Vent Raise fault, strikes N 59° E and has a dip which varies from vertical to about 70° to the southeast. On the 1475 Level, the Vent Raise fault cut the East fault or a branch of the East fault and is therefor considered to be younger than the northwesterly striking, easterly dipping system of faults.

MINERALIZATION AND HYDROTHERMAL ALTERATION

The ore lies within a mineralized zone having a width on the

order of 8,000 to 9,000 feet and a known length in excess of 9,300 feet. To define ore, an arbitrary assay cut-off of 0.50% copper is used.

The upper and western portions of the orebody are separated into two branches by an area of lean, weakly altered hanging wall rock. These two branches converge at depth so that, in cross section, they may be likened to a "U" or "V" leaning to the northwest. The northwesterly limb of ore is referred to as the North orebody and the southeasterly limb as the South orebody.

The North orebody has a strike of about N 60° E and dips approximately 50° to the southeast. The western portion of the South orebody has a strike very nearly parallel with that of the North orebody, or N 60° E, which gradually swings about to the east where the strike is approximately N 45° E. The dip of the South orebody is variable, the upper portions dipping southeast, and the lower portions arching under to the northwest and joining the North orebody. To the east, the central area of lean material gradually diminishes to a core and then disappears as the two limbs join to form a single mass of ore.

The main primary minerals are chalcopyrite, pyrite, and quartz, with minor amounts of molybdenite. These minerals are distributed quantitatively into three zones which are known as the ore zone, the hanging wall zone, which lies between the North and South orebodies, and the footwall zone, which surrounds the ore and hanging wall zones.

The ore zone, which averages approximately 0.80% copper, contains most of the chalcopyrite, quartz veinlets, and molybdenite. Pyrite is present in about equal proportions to chalcopyrite. The yellow sulfides are finely disseminated through the rock but also appear as veinlets or, more often, scattered on fractures. Molybdenite usually forms a

slick coating on fractures or accompanies quartz veinlets and is present in sufficient quantity to be recovered metallurgically. The ore zone is altered by a "sericite-pyrite-chalcopyrite"[6] type alteration which is recognized in underground exposures by the moderate gray color of the rock, the presence of altered but easily recognizable biotite, and the presence of pyrite and chalcopyrite in moderate quantities.

The hanging wall zone is simply a lean version of the ore zone with lesser amounts of all the primary minerals. This zone is gradational with the ore zone and the amount of mineralization is dependent on the distance from the ore zone. Alteration here is classed as "marginal biotite"[6] type and is recognized by the dark gray appearance of the rock, fresh-looking biotite, and weak pyrite-chalcopyrite mineralization.

The footwall zone is radically different from the ore zone except for the area where it grades into ore. Pyrite is the chief mineral here and occurs in much greater quantities than in the ore zone. The strongest pyrite is found south of the ore zone where it composes about 10% to 20% of the rock by weight. North of the ore zone the pyrite content is somewhat less and gradually diminishes to a very small amount about 1,500 feet north of the ore zone. Chalcopyrite is rarely seen in the footwall zone, and then usually as a tarnish on pyrite. Quartz veinlets are very rare and molybdenite does not occur in visible quantities. The footwall zone is altered by "hydromica-pyrite"[6] type alteration which is characterized by the light gray appearance of the rock, absence of recognizable biotite, and the abundance of pyrite.

A distinctive chlorite[7] alteration occupies an irregular zone within the South orebody, having its long dimension in a northeast-southwest direction, parallel to the long dimension of the South orebody.

The chlorite is found chiefly as a coating on fracture surfaces which is locally very strong, and, in a few places, permeates the rock.

It was originally thought that a close relationship existed between chlorite alteration and the better grade of copper mineralization. A comparison of this zone of chlorite alteration with the better than average grade ore shows many broad similarities, though not always in detail. Due to the fact that chlorite also alters post-mineral rhyolite, it is believed that any similarities that exist are due to a mutual structural control rather than a mutual source of mineralization.

Ore occurs similarly in quartz monzonite, monzonite porphyry, and diabase, in a zone which is roughly coincident with the contact of the large sheet-like mass of monzonite porphyry with the intruded quartz monzonite. This relationship does not hold true at the eastern end of the orebody but here the ore is diminishing in cross-section. It is difficult to relate the ore directly with the intrusion of porphyry since it shows no preference for either rock and does not occur similarly where the large mass of porphyry to the south intrudes quartz monzonite. It may be theorized that the similarity in shape of the sheet-like mass of porphyry and the ore zone, as well as the later chlorite alteration, may be due to a mutual, though unknown, structural control.

OXIDATION AND SECONDARY ENRICHMENT

The upper portions of the orebody, with the exception of the western end, and much of the central lean hanging wall zone have been oxidized or partially oxidized.

The chief minerals resulting from oxidation and enrichment are

chrysocolla, chalcocite, and various iron oxides. Cuprite, native copper, and black copper oxides are frequently seen where the oxidized zone grades into the chalcocite zone. Copper carbonates are very rare, only one small specimen of malachite having been recognized in all the drifting that has been done.

In order to better understand the overall oxidation pattern of the area, as mine openings and diamond drilling make more data available considerable thought has been directed toward separating and classifying stages and types of oxidation.

It is possible to separate oxidized rock into two general types depending upon the color that the iron oxides impart to the rock.

The first type, identified by a deep red or reddish-brown color, is seen at the surface, in the upper parts of shafts No. 1 and No. 4, and in diamond drill holes extending above the top of the sulfide ore at the east end of the South orebody.

This type of oxidation which comprises much of the oxidized ore zone, with the chief ore mineral being chrysocolla, is characterized by complete or nearly complete oxidation of the copper minerals and a relatively sharp line of demarcation between the oxide ore and the underlying sulfide. The absence of leaching and migration of copper from the oxidized zone to form an appreciable enriched chalcocite zone is quite marked in the San Manuel orebody. Thus, the chalcocite zone with variable amounts of oxidation and local enrichment is not important tonnage-wise so that, metallurgically speaking, the ore can be mined as clean sulfide with a minimum of mixed sulfide-oxide ore.

Oxidation of this type occurred in at least three stages: first, during the erosional cycle preceding deposition of the Cloudburst

formation; second, during the erosional cycle preceding deposition of the Gila(?) conglomerate; and, third, during the present erosional cycle. Much of the oxidized ore explored by churn drilling in the central and eastern portions of the mine is probably "red" type oxidation.

The second type of oxidation is characterized by the yellow to tan color of the rock which is probably due to a lack of hematite and the presence of limonite and goethite[8] and is typical of all the oxidized ground encountered in the deeper mine openings. Oxidation of the sulfides is not always complete and many rather large areas of primary sulfide rock or partially oxidized rock are surrounded or nearly surrounded by "yellow" type oxidation. The outer margins of this oxidation generally contain chalcocite as well as primary sulfides and much iron oxide stain, but very few oxide copper minerals. Enrichment is very slight, if any. The copper content is usually not increased to any marked degree and occasionally is less than that of the primary ore. Chalcocite, and sometimes native copper and cuprite, occur along the top and sides of deep tongues of this "yellow" oxidation in about the same quantities in which they are found beneath them so that there is often an inversion of the normal sequence of oxidation with primary ore lying above secondary ore which in turn overlies oxidized ore.

Deep tongues of "yellow" oxidation are known to follow rhyolite dikes, the San Manuel fault, and other northwesterly-trending post-mineral faults. In some cases no structural control is evident but it may be surmised that the path of oxidation was controlled by local differences in permeability of the rock.

This type of oxidation invades the sulfide ore locally along certain faults and in some fringe areas, but has not been found to be of

particular economic importance. The intensity of oxidation of ore minerals and the volume of such material relative to the volume of sulfide ore is so limited that it does not affect stope layout or draw. The distinctive coloration and mineralogical characteristics do, in fact, provide a marker and guide for the stope engineers in evaluating their draw data in some instances.

"Yellow" oxidation extended much deeper between the two limbs of ore than in the ore itself and extended deeper than previous oxidations so that, in cross-section, the oxidized zone has the shape of a trough with its sides impinging on the two limbs of ore.

ACKNOWLEDGMENT

Acknowledgment is due the officers of San Manuel Copper Corp. for permission to use the information presented.

J. D. Palletier
November, 1956

1 Peterson, N. P., Geology and Ore Deposits of the Mammoth Mining Camp Area, Pinal County, Arizona, Arizona Bureau of Mines, Geological Series No. 11, Bulletin No. 144.
2 Creasey, S. C., Personal communication.
3 Schwartz, G. M., Geology of the San Manuel Copper Deposit, Arizona, Geological Survey Professional Paper 256, p.9.
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5 Schwartz, G. M., 1953, op. cit., p.12.
6 Schwartz, G. M., 1953, op. cit., pp. 19-32.
7 G. M. Schwartz says of a specimen of this material collected on the 1285 Level: The index of refraction 1.594 and very low birefringence indicate chlorite, but the X-ray pattern is very nearly that of antigorite. (Winchell considers antigorite a member of the chlorite group but most investigators include it in the serpentine group.) I think for the present that we may call it chlorite."
8 Schwartz, G. M., 1953, op. cit., p.37, p.57.
Schwartz, G. M., Oxidation and Enrichment in the San Manuel Copper Deposit, Arizona, Economic Geology, Vol. 44, No. 4, 1949, pp. 273, 274.

Deep Hole Prospect Drilling At Miami, Tiger, and San Manuel, Arizona

by E. F. Reed

SAN MANUEL
1962

K 8104 ? IN PART ?

TABLE 9. - Man-shifts per foot of drift and drift footages, 1415 ventilation drift, arch steel-supported drift

Period	3/22/54 to 7/55	7/54 to 1/55	1/55 to 5/3/55
	Man-shifts per foot of drift		
For period....	1.22	1.16	1.08
To date.....	1.22	1.18	1.11
	Drift footage		
For period....	598	1,603	1,193
To date.....	598	2,201	3,394

Block Development

To assure a steady production from the block-caving operations, the ore bodies (north and south) were divided into self-contained units (so-called full-protected block caving) that could be developed, caved, and mined gradually, independently, and according to schedule. The bodies were divided by levels in the vertical plane into 600-foot lifts that, in turn, were divided into panels and blocks in the horizontal plane. Since each block had a limited daily output, 8 to 13 blocks had to be drawn for the total daily production; thus, several blocks had to be developed ahead of production. In general, production at San Manuel comes from one level while the next level is being developed.

The main, or south, ore body was laid out in panels numbered from northeast to southwest, and panels were subdivided into blocks numbered from northwest to southeast. Block 6-1 was the most northwesterly block in the sixth panel from the northeast boundary of the ore body. Panels were normally 210 or 140 feet wide; the length of the blocks ranged from 150 to 270 feet, depending on the number and spacing of the grizzly drifts in the block.

In general, the all-gravity caving method was used. On the first lift the slusher-gravity method was used for boundary blocks where the ore column was low. Development costs for the slusher-gravity method were less than for the all-gravity method.

Annual data pertaining to footage of mine workings driven for block development is listed in table 10.

Gravity Blocks

Raise Stations and Transfer Raises

Some time after haulage-panel, grizzly-fringe, and fringe-panel drifts were completed, development of the block began with excavating transfer raise

stations in panel drifts on the haulage level. Drift lagging was removed, and the ground was drilled and blasted to make room for the station and to hang temporary (joker) chutes. Quite often it was necessary to grout the arch of the drift before drilling and blasting was done. Details of this procedure are discussed later in this paper.

TABLE 10. - Summary of block development footage per year

Year	Raise stations	Transfer raises, feet	Grizzly drifts, feet	Draw raises, feet	Undercut drifts, feet	Undercut pillars, square feet
1954.....	1	161	323	--	--	--
1955.....	127	14,286	6,154	11,235	6,238	20,535
1956.....	106	17,457	10,223	15,183	14,999	171,914
1957.....	154	15,180	9,488	16,660	11,381	145,270
1958.....	154	22,165	11,653	24,541	20,350	321,897
1959, (first 6 months).....	58	10,936	5,890	12,540	12,460	222,459

The stations are 6-1/2 feet high, 11 feet long, and as wide as the haulage drift. Raises were driven to 2 feet above sill sets on the grizzly level, and backover raises were driven as shown on figure 21. The raises measured 4 by 4 feet inside timber and were lined with 6- by 8-inch cribbing. Where hard, abrasive rock was encountered, or anticipated, 3- by 4- by 3/8-inch angle iron for armoring was installed on the crib rings. The last round was drilled in the transfer and backover raises to form an arch for natural support until the grizzly drifts were driven.

Permanent steel chutes, which were practically free of ground support and may be described as floating, were installed on trunnions supported by the two caps of adjacent drift sets (fig. 22). The undercut guillotine door was operated up and down by an 8-inch-diameter air cylinder with a 36-inch stroke. The door passed through a slot in the bottom of the chute and was held in guides at each side (fig. 23). The door had 3/4- by 2- by 36-inch extensions to prevent the guide from filling with muck. Cylinder and door guides were supported by H-beam and angle iron which was bolted to the bottom of the chute slide. This type of chute, with its 36- by 39-inch opening, gave ample room for barring down or blowing out the chute throat, and the positive undercut action of the door made it virtually impossible for a boulder to block the chute open.

Three men per shift comprised a raise-station crew. Each crew was equipped with an 8-ton battery locomotive, six 5-ton cars, a rocker shovel loader, a stoper, and an air-leg drill.

Each of six two-man transfer-raise crews drove two raises, single shift. Two stopers and two single-drum, air-driven hoists were used by each crew to drive a total of 1,350 feet of transfer raises per month, for an average advance of 4.55 feet per man-shift.

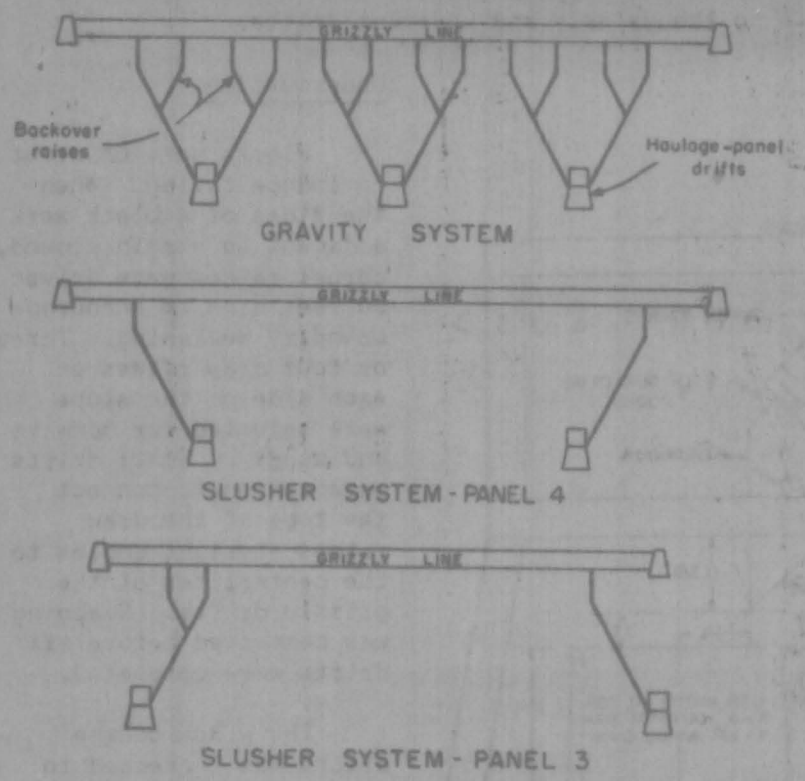


FIGURE 21. - Schematic Sections Contrasting Development of Gravity and Slusher Blocks.

Grizzly Drifts

The grizzly level was 60 feet above the haulage level. Grizzly drifts were driven on 30- to 45-foot centers along the length of the block after the transfer and backover raises were completed. At first, the drifts were supported with 10- by 10-inch and 12- by 12-inch timber sets. Because of high timber repair costs, circular rigid steel sets, circular yieldable steel sets, and monolithic concrete were tested. Subsequently, ground support by concrete lining was adopted as the method of support. Details as to design, results, and costs are discussed in a subsequent section on concreting. Figures 24, 25, and 26 show the various methods used.

The grizzly drifts were driven 8 feet wide by 8-1/2 feet high in rock section (fig. 27). The ground was supported by 6-foot rock bolts and chain-link fencing. Badly fractured ground was supported by light timber sets. Concrete, 1 foot thick, was poured around the top of transfer raises to a depth of 5 feet below sill level, and grizzlies of 45-pound rail with 12-inch maximum spacing were installed. Next, forms were placed in the drifts, and concrete was poured at least 1 foot thick on ribs and 1-1/4 feet thick in the arch. Completed drifts measured 4 feet wide by 6-1/2 feet high (fig. 28). Five two-man crews per shift drove the required 987 feet of grizzly drifts per month, with an average advance of 1.38 feet per man-shift.

Draw Raises

The first undercut levels were placed 25 feet above the grizzly level. This interval was reduced to 20 feet when the drifts were supported with concrete lining. After these grizzly drifts were completed, finger or draw raises were driven to 2 feet above the undercut level. Normal spacing of the raises was 15 by 17.5 feet. They were 5 feet in diameter at the grizzly level, with the last round belled-out to 8 feet in diameter at the undercut level. The raises were lightly timbered. Four two-man crews averaged 1,920 feet per month,

with an average progress of 10.85 feet per man-shift. Figure 5 shows the relationship of the draw raise to the grizzly and undercut drifts.

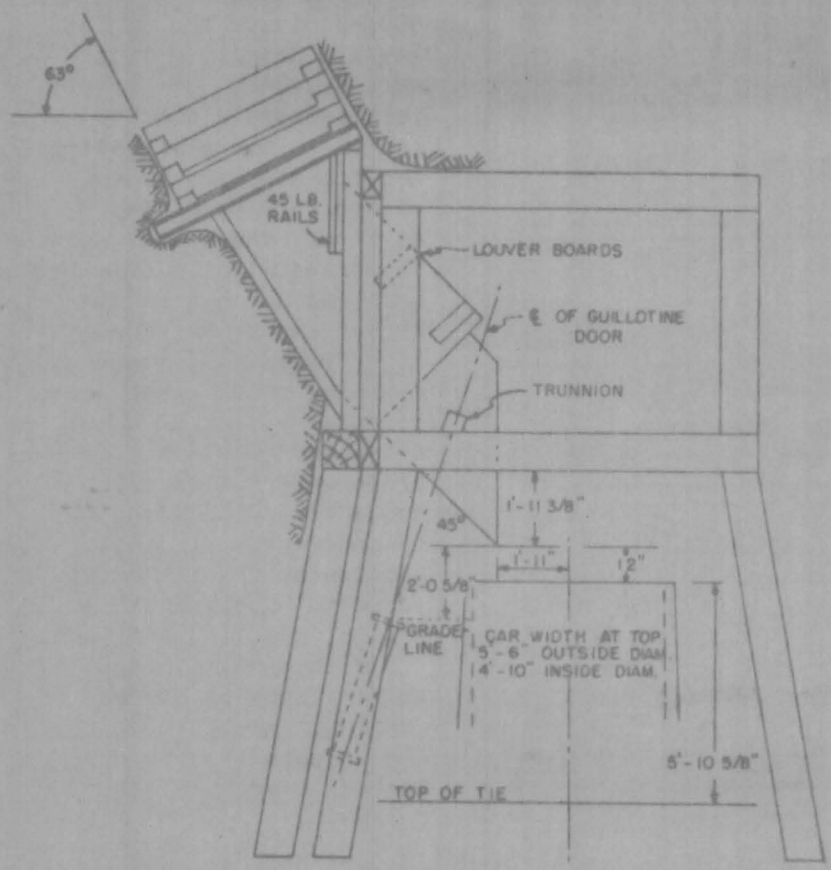


FIGURE 22. - Loading Chute for Transfer Raises.

Undercutting

Blocks were undercut to induce caving. When the sides of a block were adjacent to virgin ground, corner raises were driven 30 feet high to encourage boundary weakening. Three or four draw raises on each side of the stope were selected for manways and supply. Small drifts were driven to connect the tops of the draw raises at right angles to the centerlines of the grizzly drifts. Slabbing was commenced before all drifts were completed.

The width of the drifts was increased to 15 feet, and the backs were slabbed to a height of 15 feet above the undercut floor. This height was considered necessary to take care of the swelling of broken rock and caving action. The slabbing operation was started at one corner of the block and progressed toward the opposite corner. The undercutting was completed by drilling and blasting from one side of the pillars remaining between the drifts. Drawing commenced immediately to give room for good caving action.

Timber was used on the undercut level to assure safe working conditions. Peeled pine poles, 6 inches in diameter and 6 feet 6 inches long, 6- by 8-inch timber, and 2- by 12-inch lagging were used. None of this timber was treated (fig. 29).

Timber was augered, loaded, and shot with the pillars. Delay electric caps, used in pillar blasting, were wired in parallel series, with not more than 25 primers in any one series. Each series within a circuit was tested with a galvanometer, and then the circuit was wired to a No. 12-2 conductor cable from the power source. All blasting lines passed through an interrupter switch and from this switch to the main pillar blasting switch installed on an independent electrical circuit.

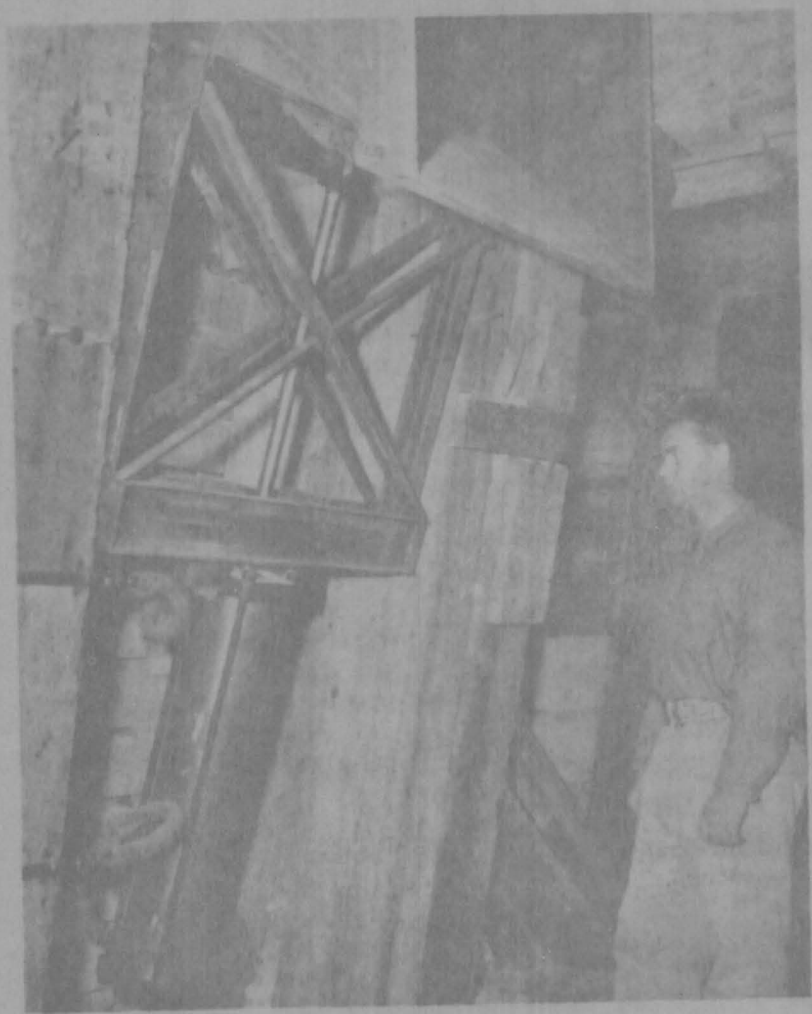


FIGURE 23. - Air Cylinder and Guides for Operating Chute Gates.

After each pillar blast, the broken material was drawn to give room for the next pillar blast and to determine by visual inspection that no part of the blasted pillar was left standing. If any of the pillar remained standing, it was removed before the succeeding pillar was blasted.

Twelve two-man crews, eight men per shift, undercut 30,000 square feet per month.

Slusher-Gravity Blocks

A slusher-gravity system was worked out at San Manuel for use on the first lift where the column of ore is low. Slushers were used in panels 3 and 4, and others are planned. This method requires considerably less development than the all-gravity system, but extraction costs are higher. Figure 7 is a schematic illustration of drifts and raises used in the all-gravity system. Figure 19 shows the differences in transfer raises of the two methods.

Blocks worked with slushers have two haulage drifts; the all-gravity system has three. Less than one-half the transfer-raise footage is required for the slusher-gravity system. Draw raises to the undercut level are spaced 17.5 feet apart, and grizzlies are constructed of 90-pound rail with 13-1/4-inch openings. A single transfer raise, with 4- by 6-foot inside measurements, serves one-half of the draw raises on a slusher-grizzly line. The undercutting procedure is identical to that discussed under the all-gravity blocks.

Double-drum, 2,600-pound slusher hoists powered by 30-hp. electric motors operating on 440-volt, 60-cycle alternating current are used. The drums are 14 inches in diameter. The pull and tail cables are respectively 130 feet of

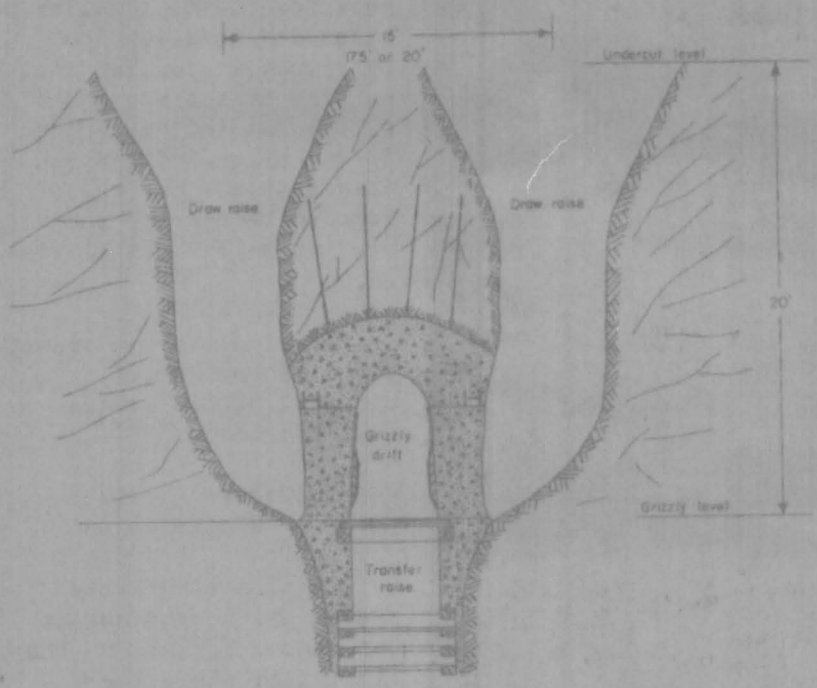


FIGURE 24. - Concrete-Supported Grizzly Drift and Draw-Raise Entrances.

5/8-inch and 250 feet of 1/2-inch wire ropes. The 42-inch scraper is a 1/4 box-type with a capacity of 13 cubic feet (fig. 30).

Ground Control

Heavy ground has been a major problem at San Manuel. Several types of timber sets were tried, and rigid and yieldable circular steel sets were tested. All materials tested were unsatisfactory because of high repair costs. Concrete support tests were started in August 1956. Results were satisfactory, and the decision was made to use concrete support in all future grizzly and panel drifts.

The following information is abstracted from previous papers.[11 12]

Although the initial cost of concreting is higher than when other materials are used, appreciable savings result because costly drift maintenance is minimized. The following additional advantages are apparent:

1. A safer and cleaner working place is available for the chute tapper, thus increasing the tons per man-shift.
2. The fire hazard within a block is virtually eliminated.
3. Less skilled workmen are required for initial development, and fewer skilled timbermen are needed for repair work.
4. More efficient ventilation is possible since drifts are smooth lined. Most concreted grizzly drifts remain open after a block is completely drawn and can be used selectively for air courses.

The first concrete placing was done with two stationary pneumatic placers located on the 1415 grizzly level. A 1/2-cubic-yard mixer was used for surface

[11] Ward, M. H., Underground Concreting at San Manuel Mine, San Manuel, Arizona: Proc. Arizona Section Meeting, AIME, Tucson, Ariz., December 1958, 11 pp.

[12] Pillar, C. L., The Placement and Use of Concrete Underground at the San Manuel Mine, San Manuel, Ariz.: Proc. Am. Min. Cong., Denver, Colo., September 1959, 15 pp.

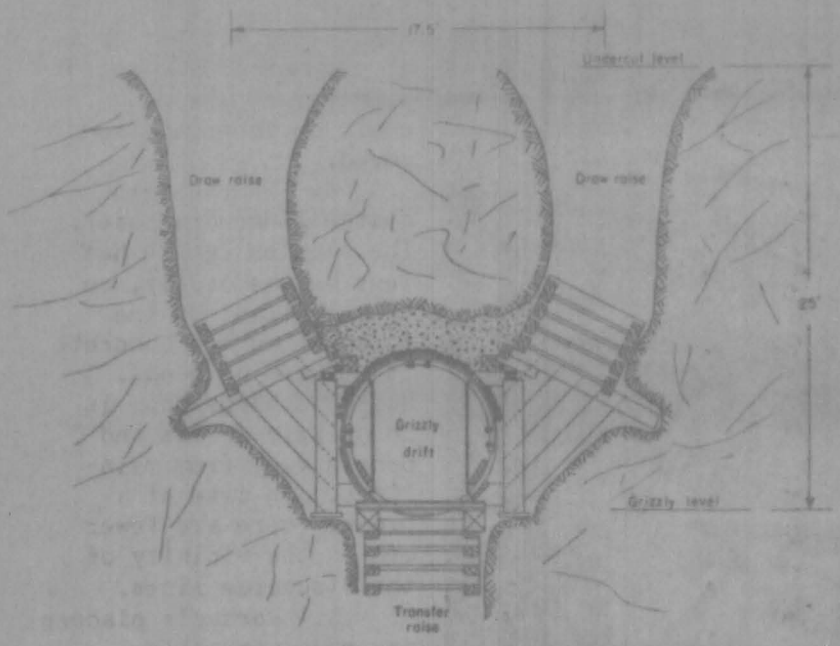


FIGURE 25. - Steel-Supported Grizzly-Drift and Timber-Supported Draw-Raise Entrances.

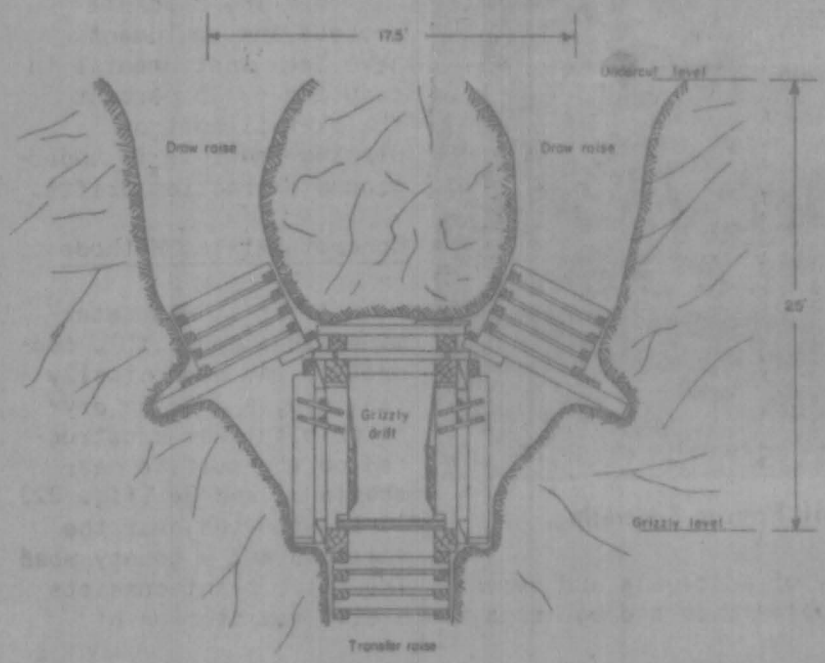


FIGURE 26. - Timber-Supported Grizzly-Drift and Draw-Raise Entrances.

mixing. Sacked cement was hand-opened and dumped into the mixer hopper. Concrete was poured down a vertical exploration churn-drill hole lined with 8-inch casing into a receiving hopper, from which it was chuted one of the two placers. Permanent, 6-inch, Victaulic pipelines were taken off the main feeder line into the panel drifts. Concrete was blown for as much as 2,000 feet.

In August 1957 the stationary placers were replaced by portable placers using the Flocrete process (fig. 31).

Several advantages of portable placers are evident:

1. Less manpower is required. With portable placers, 5.9 cubic yards per man-shift is poured, compared with 2.7 cubic yards per man-shift poured with conventional pneumatic placing.

2. A better concrete is obtained. Disseminated air causes the concrete to flow rather than blast from the discharge pipe. This prevents segregation and honeycombing.

3. The number of plugs in the discharge line is decreased. The maximum pipe length is reduced from 2,000 to 400 feet. If a plug occurs, it is easier to find and break loose. With stationary placers, concrete

FIGURE 27. - Grizzly Drift Prior to Concreting.

has filled and set up in as much as 1,000 feet of pipe before the plug could be found and removed.

4. Shorter discharge lines are safer. The shorter length has less volume of air, so the resultant of the force of moving concrete is reduced at bends. Most discharge pipe is in grizzly drifts and panels away from mainlines. In case of a break, there are fewer men in the vicinity of the discharge lines.

5. Portable placers are more versatile. Small pours of foundations, floors, stations, and so forth that would not merit stringing pipe from a stationary placer can be poured efficiently.

6. The Flocrete process and equipment have been instrumental in reducing by 25 percent the overall cost of placing concrete in underground extraction drifts.

Concrete Mixing Methods

After it was determined that monolithic concreting was economically feasible, a 300-ton drybatch plant was constructed on the surface near shafts 3A and 3B (fig. 32). It is situated near the railroad and a county road for convenience of delivery of aggregate and cement. The batch plant consists of elevators to lift aggregates into storage bins and a silo for storage of bulk cement.

FIGURE 28. - Grizzly Drift After Concreting.

Concrete for the first level (1415) is delivered dry batched to a concrete mixer by three dump trucks, each with four 1-1/4-cubic-yard compartments (fig. 33). The mixer is a 1-1/4-cubic-yard double-drum paver mounted on tracks. At a signal from the underground remix station, concrete is mixed and poured into a hopper mounted over a vertical churn-drill hole with 8-inch casing. The hopper is covered with a 3/4-inch reinforcing rod grid of 3-inch squares, which screens out large rock that may cause plugging. Concrete for the second level is delivered dry batched to the No. 1 shaft, where a track-mounted 1-1/4-cubic-yard mixer located at the shaft collar mixes and pours wet concrete down either of two 8-inch standard pipelines mounted in the shaft's service compartment.

Concrete Mix

A mix that yields 1-1/4 cubic yards of concrete consists of 705 pounds of type 11 portland cement; 1,300 pounds of clean washed sand having 5 percent minus 100-mesh; 2,200 pounds of string cubical minus 1-inch screened coarse aggregate sized within limits of ASTM specifications; 48 gallons of clean, pure water; 14 ounces of air-entraining agent for added workability; and 21 ounces of a retarder and strengthener. The mix yields concrete that permits fast and efficient placing and has an average 28-day strength of 3,500 p.s.i. This mix gives a slump of from 5 to 6 inches at the mixer and 3 to 4 inches in the forms. It is necessary to cool

FIGURE 29. - View in Undercut Drift.

the mixing water during the hot, dry summer. The portable, electric-powered, water-chilling plants are capable of pro-during 20 g.p.m. of water at 40° F. The chilled water reduces the temperature of the aggregate so that concrete temperatures at the mixer are kept at approximately 70° F. The concrete loses considerable water during its journey between the surface mixer and the underground forms (fig. 34). A water-to-cement ratio of 0.58 to 1 is maintained; this keeps the concrete workable but does not allow the accumulation of free water in the forms. The set of the concrete is retarded by the use of additives so that concrete can stay a maximum of 3 hours in the portable placers and be discharged satisfactorily.

Underground Remixing Station

At the bottom of the churn-drill hole most of the wear and shock caused by the falling concrete is taken by a 16-inch-diameter pipe header (fig. 35), which receives and chutes the concrete into a remixer. A replaceable, belt-on bottom for the header has been developed.

Between the surface and the 1415 level the 8-inch casing has several slight bends where above-average abrasion occurs and an occasional hole is worn through the casing. To change the position of wear points, the casing is rotated 90° after each 1,000 cubic yards of concrete is poured. At the same time, an additional 30 feet is welded to the casing at the surface, the casing is lowered, and a corresponding amount is cut off at the remixer station. A method was developed to detect a hole in the casing before escaping concrete can accumulate around the outside of the casing and cement it in place. The 8-inch casing is suspended inside a 12-inch drill casing in which 10 g.p.m. of water is circulated from the surface into a funnel at the remixer station. If a hole is worn in the casing, milky water and particles of rock and sand are observed immediately in the funnel as the concrete descends.

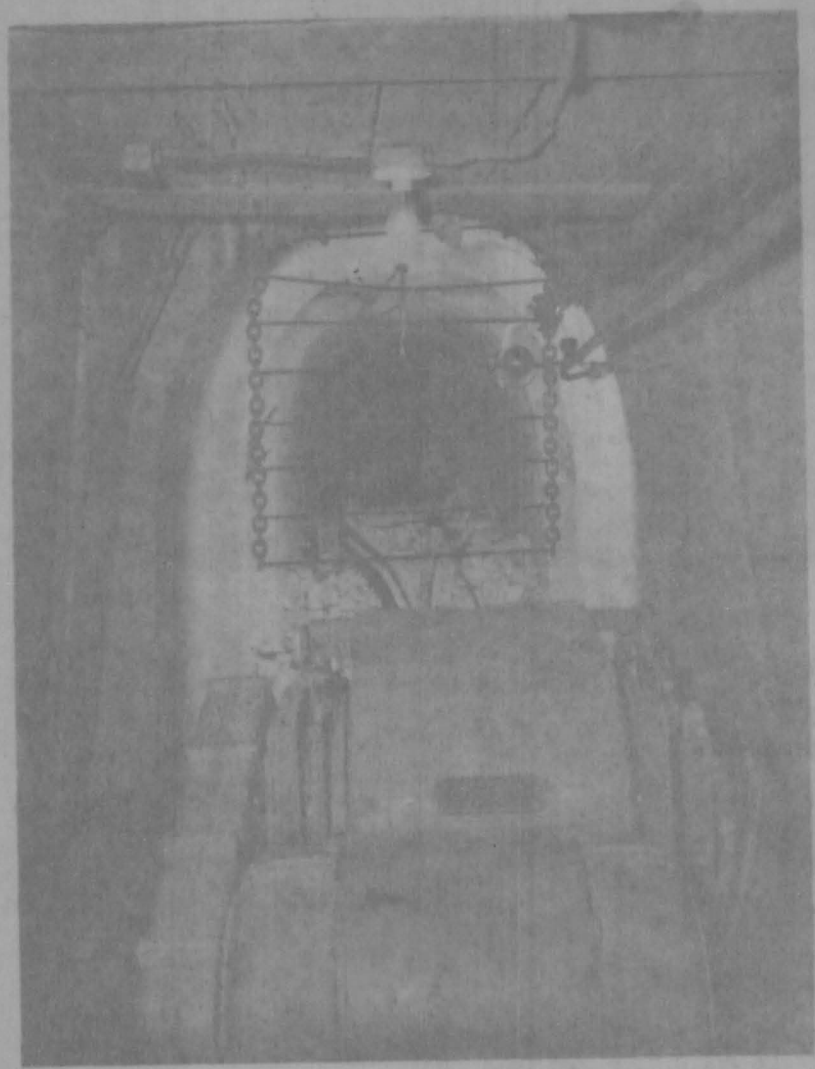


FIGURE 30. - Double-Drum Hoist and Scraper in Concrete-Lined Slusher Grizzly Drift.

From the pipe leader the concrete flows into a 6-cubic-yard remixer, where it is agitated until loaded into placers (figs. 36 and 37). The remixer is a 5-1/2-foot-diameter tank, 8 feet long with a side section cut out. A spiral agitator built around a 4-inch tubular shaft extends the length of the tank. Each end of the shaft is supported on water-lubricated bearings to prevent cement freezing. The agitator is driven by a chain-connected 10-hp. air motor.

Communications

A separate telephone circuit and light-bell system are used for communicating between remixer and surface mixer. A standardized system of bells is posted for ordering and indicating receipt of concrete. This prevents the pouring of several batches on top of each other should the casing become plugged between surface and remixer station.

Concrete Forms

Wherever possible, drifts that are to be supported with concrete are driven without timber support, using 6-foot rock bolts with wire mesh for pre-concrete ground support (fig. 27). Light timber sets are used where the ground will not stand with rock bolting. The sets have sufficient span for placing of forms between them. The light timber sets are left in place, and concrete is poured around them. When excessive pressures develop before concreting, it is necessary to replace the light timber set with a steel set of 6-inch, wide-flange section. This is done only where absolutely necessary, as structural steel imbedded in the concrete has proven detrimental to concrete support of heavy, moving ground and also where secondary blasting is required. It is the practice at San Manuel to keep as much steel out of the concrete as possible.

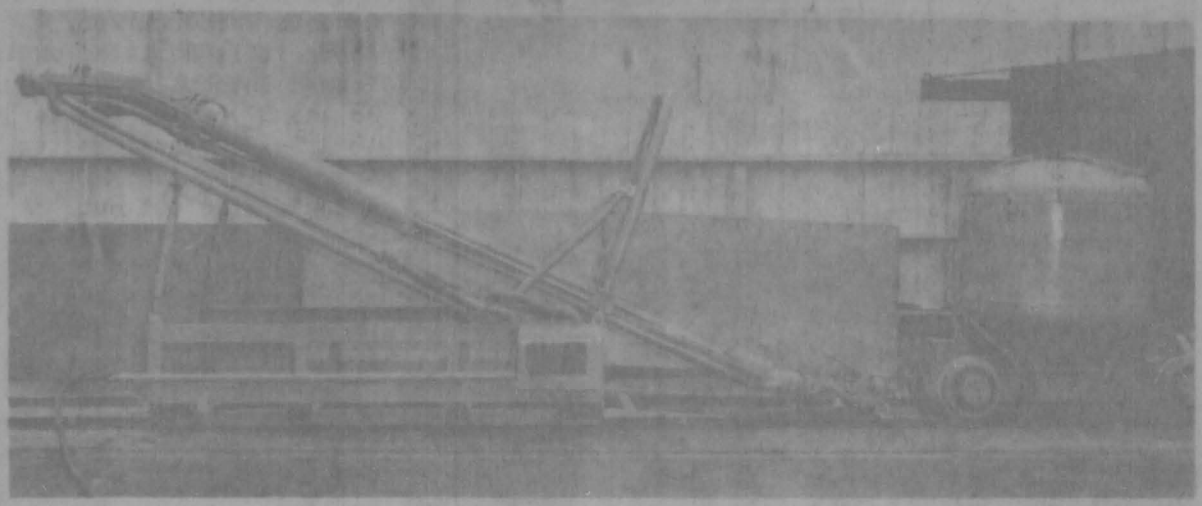


FIGURE 31. - Portable Concrete Placer.

FIGURE 32. - Batching Plant.

Steel imbedded in the concrete is confined to wear beams at the brow of draw raises, eyes, hooks, and brackets for control-board cables, safety chains, pipelines, and slusher sheaves. No reinforcing steel is used in concrete in the mining areas.

In block grizzly and slusher drifts that contain no track and are at right angles to tracked panel drifts, forming is accomplished by using portable, light, strong plywood panels. These forms are made in the mine carpenter shop from 5/16-inch plywood with 2- by 6-inch wooden ribs. The standard panel is 5 feet 10 inches long, which is an even interval for 17-foot 6-inch drawpoint

FIGURE 33. - Concrete Mixer at Churn-Drill Hole.

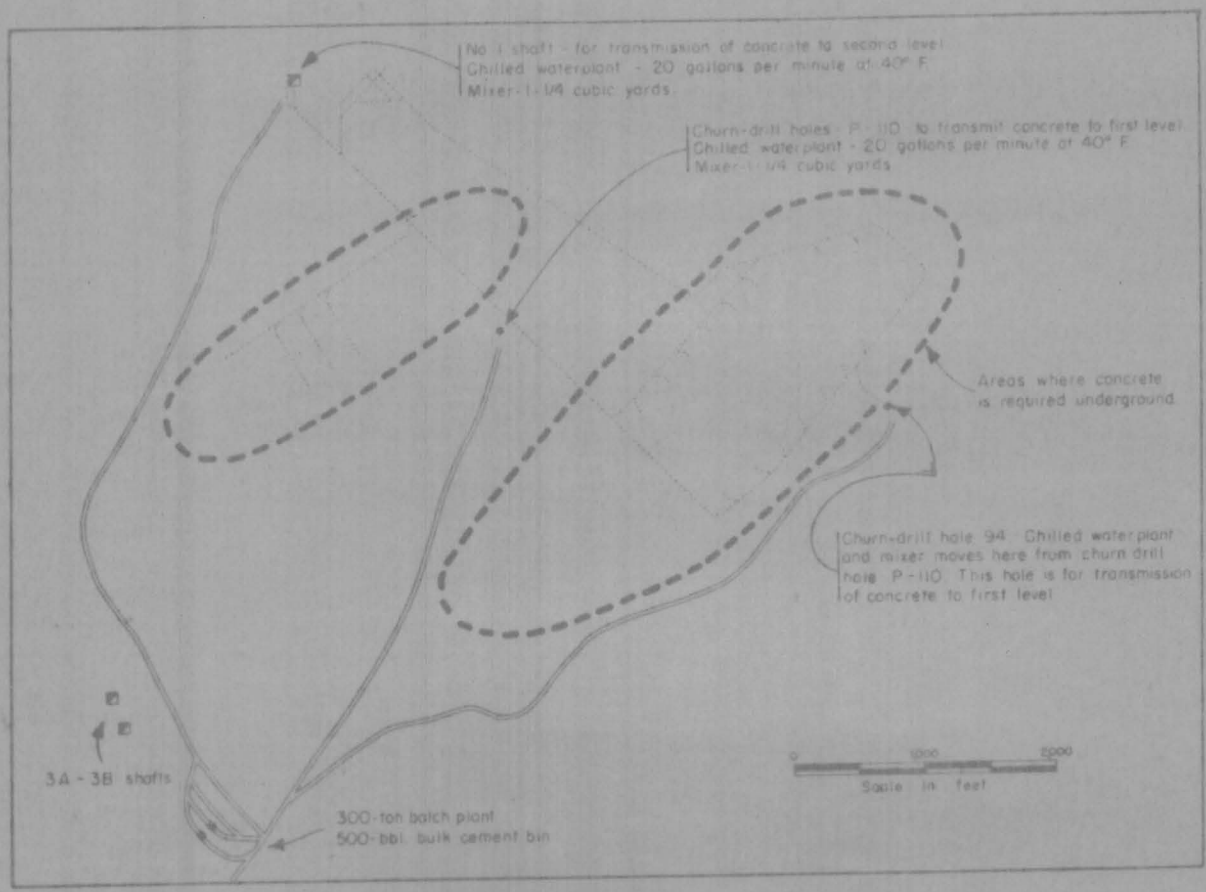


FIGURE 34. - Location of Concrete Batch Plant, Mixing Stations, and Areas Where Concrete is Used Underground.

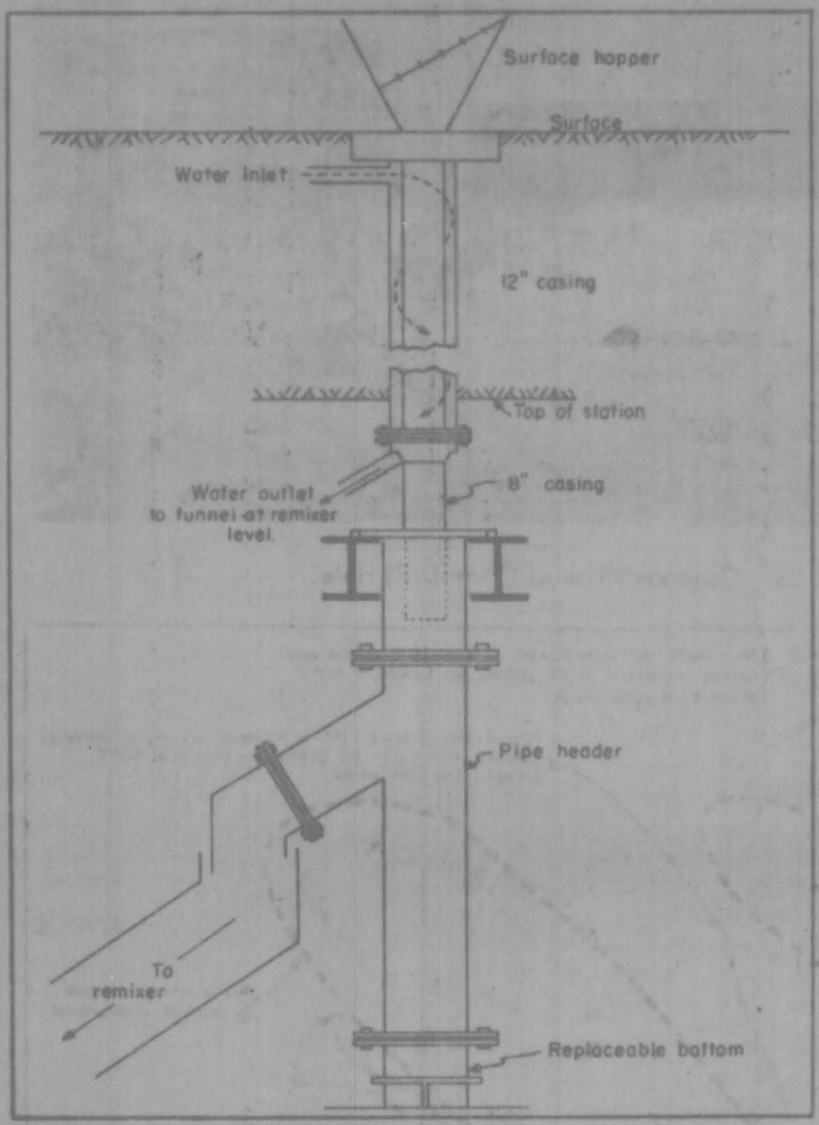


FIGURE 35. - Sketch of Casings and Header.

spacing. Grizzly sections are composed of two straight side panels and two arched back panels bolted together with 5/8- by 4-inch machine bolts. A slusher drift sections uses an additional 12-inch-wide panel bolted between the two arched back panels along the top center of the drift. Properly placed holes in the form panels allow quick placing of eyes, hooks, and brackets that are to be imbedded in concrete. In grizzly drifts, forms are set on leveling sills laid on the raw drift floor.

The drift floor between the side of the forms and the rock walls is dug to solid rock and cleaned of all loose muck. In slusher drifts, a 12-inch concrete floor with imbedded wear rails is poured between the concrete drift walls. A narrow slit in the form allows 1-inch rough lumber to be sectionally placed between the form and the ground to make draw-raise box-outs. These 1-inch lumber box-outs are braced with 3- by 6-inch studding. After forms are set, lined, and leveled, they are stull-blocked to the ground and well braced internally with rough 3- by 6-inch timber. Drifts are driven, and forming is done to allow a minimum thickness of 18 inches of concrete in drift walls and arched roof. Finished grizzly drifts (fig. 28) are 4 feet wide and 6-1/2 feet high, and slusher drifts are 5 feet wide, 6-1/2 feet high (fig. 30).

At first, the panel-frinoe drifts (fig. 38) were formed in a manner similar to the grizzly and slusher drifts. Now, circular, jumbo-mounted steel forms are used. These forms are in 10-foot lengths. They are transported and expanded into position by a hydraulic-powered, track-mounted jumbo. In the stripping process, the jumbo contracts the forms, and they are then in a

FIGURE 36. - Underground Remixing Station.

position to be transported through the existing forms to any desired location for placing. Light steel bubbles bolted to the outside of the forms are used to form grizzly and slusher drift entrances. Early performance with these forms represents a 50-percent reduction in labor costs over previous methods. Panel drifts a are driven and forms are placed to allow a finished circular section of 8 feet 6 inches, with a minimum thickness of 18 inches of concrete.

Concrete Placing

A train of portable Flocrete placers mounted on trucks is pushed from the remixer station to the forms. Each individual placer is connected o to an underpass discharge spout and emptied (fig. 31).

The placer is a 1-3/8-cubic-yard circular tank with a conical bottom. Compressed air at 90 p.s.i. is introduced through a turnip-shaped valve in the bottom of the placer to force out concrete through a 6-inch bottom-discharge pipe. Air is disseminated throughout the batch as it is gravity-fed to the bottom. A spring holds the turnip valve against its seat until discharge starts. A foot-operated air vibrator is mounted on the outside of the placer and is used to loosen stiff concrete.

The opening at the top of the placer is sealed airtight with a rubber-covered hatch, which is mounted on a swivel and swings aside for loading. If the discharge line plugs, the air seal is broken by opening a pressure-release valve at the top of the tank.

A quick-coupling attachment is mounted on the underpass and is connected to the placer. Air is supplied through a 2-inch pipe attached to a manifold

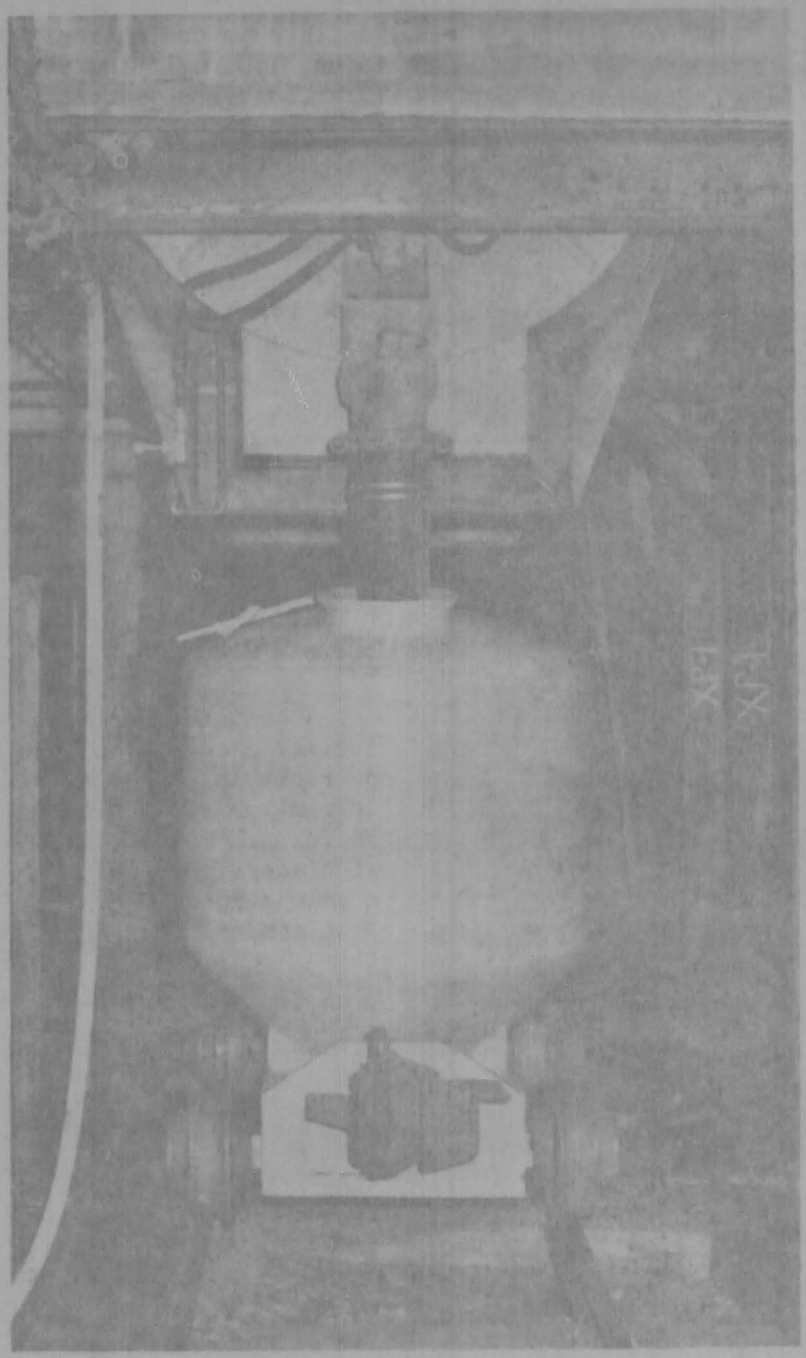


FIGURE 37. - Remixer Discharging Into Placer.

with two 1-1/2-inch protruding tapered fittings, which are sealed against tapered seats of a casting on the placer (fig. 39). Quick-coupling hooks, operated by an eccentric lever, draw the placer into position and lock it for an airtight seal between discharge pipe and air inlets. After each placer is emptied, it is uncoupled from the underpass and pulled back sufficiently to clear the underpass discharge header. The header and pipe are hoisted above placer heights by two double-acting air cylinders (fig. 40). The empty placer is pushed through the underpass and uncoupled from the train. The train is pulled back, the discharge header is lowered, and the next placer is brought into position and emptied (fig. 41).

An 8-ton locomotive pulls a train of four to six placer pots that contain 5-1/2 to 8 yards from the remixer station to the forms. Two trains are used during the placing operation, and two sections of forms are poured simultaneously so that both trains may be run to one form if a plug occurs. The tram distance of placers ranges from 1,000 to 4,000 feet. Telephones and audio phones are located throughout the mine and are used in dispatching concrete to the different drifts.

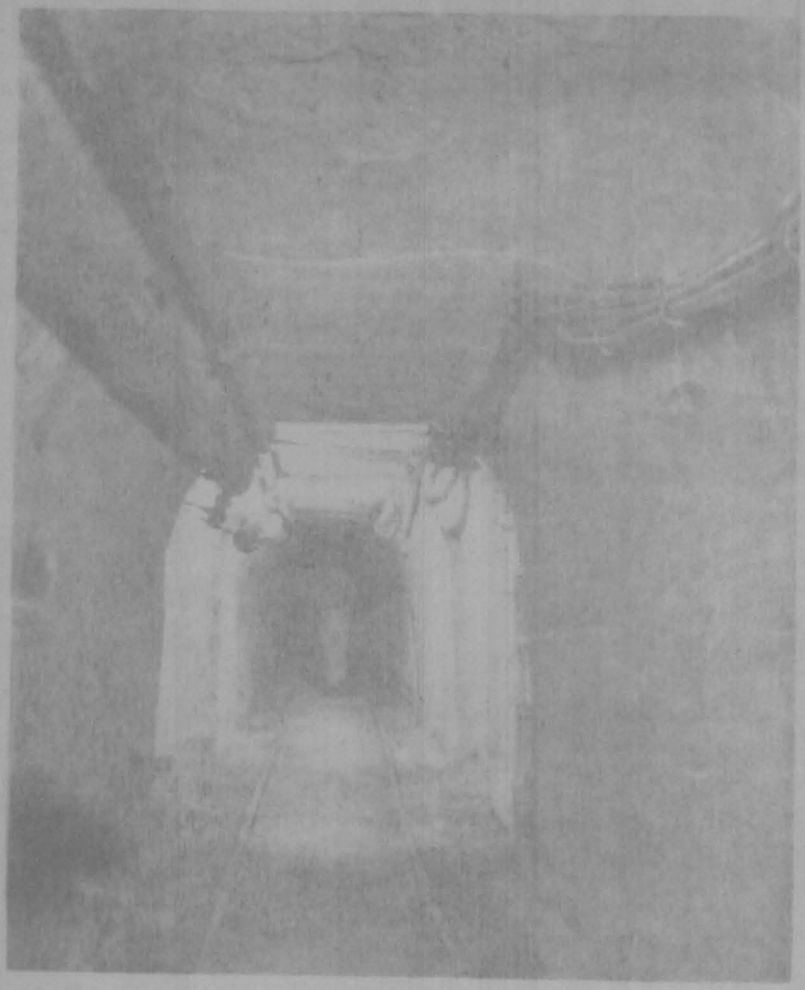


FIGURE 38. - Concrete-Lined Panel-Fringe Drift.

One man works on top of the forms to shift the discharge pipe for even filling. Internal air vibrators are used to consolidate the concrete. Sections of discharge pipe are removed as the form is filled and the pour retreats.

When the pour is completed, a bundle of rags or "rabbit" is forced into a cleanup header. The header is then connected to the underpass; air is introduced through the header, and the rabbit is forced through the discharge line to clean it. Usually two rabbits are run before the line is free of all concrete.

Discharge pipes are made from 6-inch lightweight spiral tubing in 5- and 10-foot lengths. Elbows of 45° and 90° bend and 2-foot radius are fabricated from standard pipe. This smooth bend reduces wear and plugging in the lines. All connections are made with 6-inch Victaulic couplings. The average length of the discharge line is 100 feet around one 90° bend, but pours through 400 feet of pipe and three 90° bends have been made without difficulty.

Concrete Crews

An average of 150 cubic yards of concrete per day is now being placed on the two mining levels. The pouring operation is accomplished on the "A" shift, and the forming operation on both "A" and "B" shifts. The crews for this work are shown in table 11.

This crew has placed a maximum of 230 cubic yards of concrete in one 8-hour shift. The underground placing and forming crew work under a contract bonus system whereby they are paid each week for the work they accomplish.

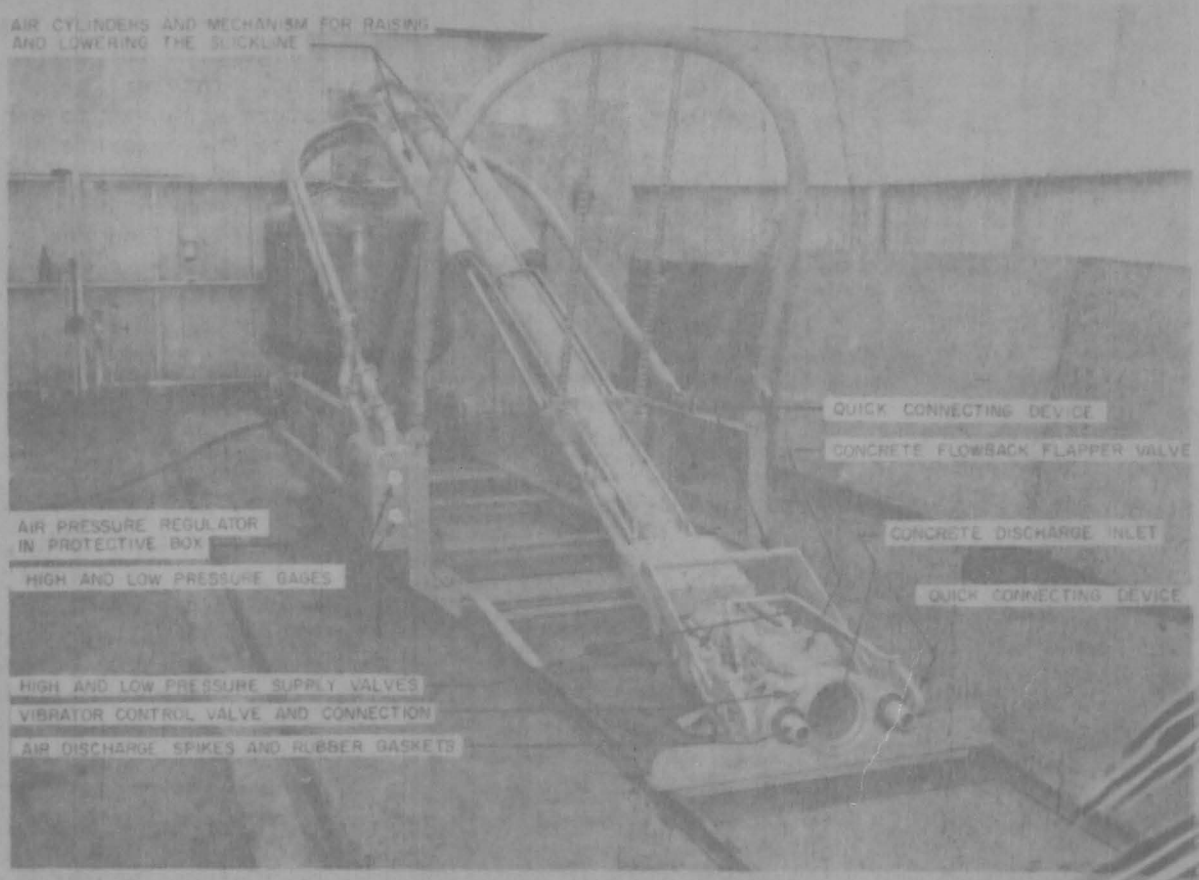


FIGURE 39. - Front View of the Underpass.

FIGURE 40. - Slickline in Raised Position; Placer Pot Passing Through Underpass.

FIGURE 41. - Coupling Placer Pots to Slickline.

Costs

Costs, as shown in figures 42 through 46, cover all direct supervision, labor, and material, and some equipment maintenance. For a fair and conservative evaluation, these costs were reduced to a 1956 base. This was done by reducing the costs for work accomplished after July 1956 by the same percentage that mine labor has been increased, roughly 3.5 percent a year. It will be difficult to compare these costs with costs from other operations, owing to the different manner in which each individual company keeps its own cost accounts.

TABLE 11. - Crew required for concreting operations

	"A" Shift	"B" Shift
Surface crew:		
Batch-plant operator	1	0
Truckdrivers	3	0
Mixer operators	2	0
Total	6	0
Underground placing crew:		
Leadman	2	0
Remixer operators	2	0
Motormen	3	0
Placer operators	3	0
Puddlers in form	3	0
Pipe riggers	0	2
Total	12	2
Underground form crew:		
Leadmen	2	2
Form men	5	5
Helpers	5	5
Total	12	12
Grand total	31	14

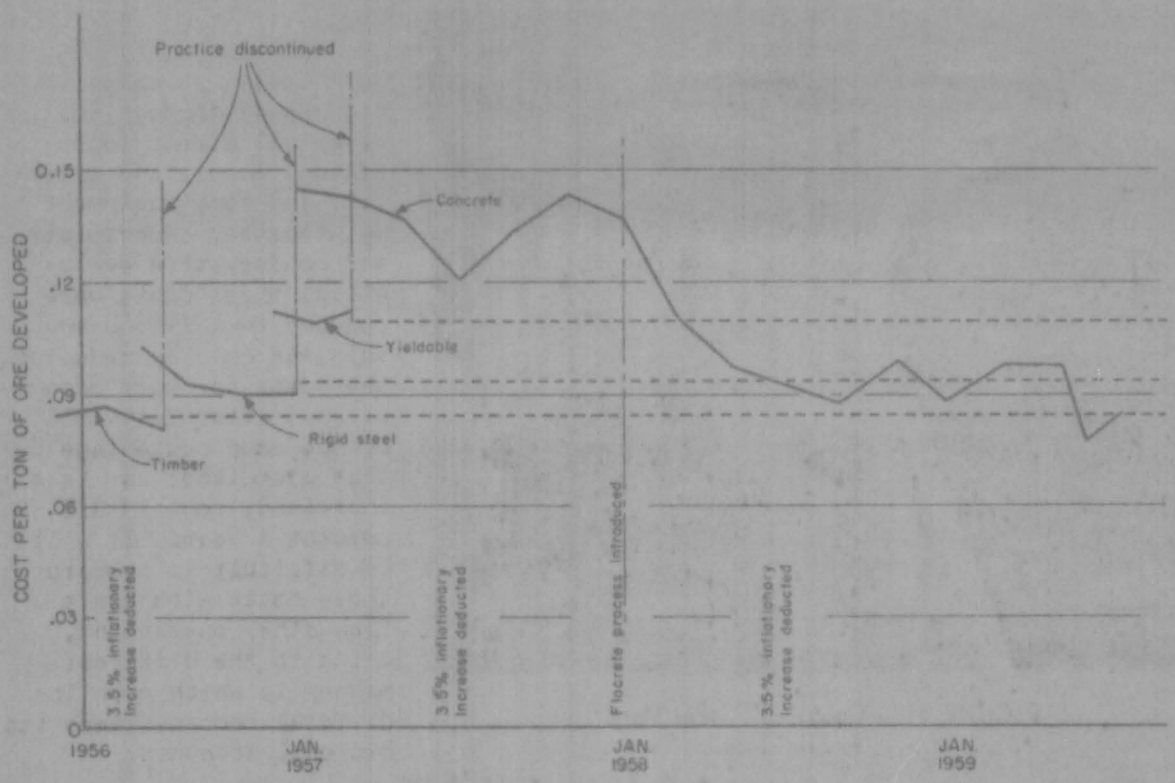


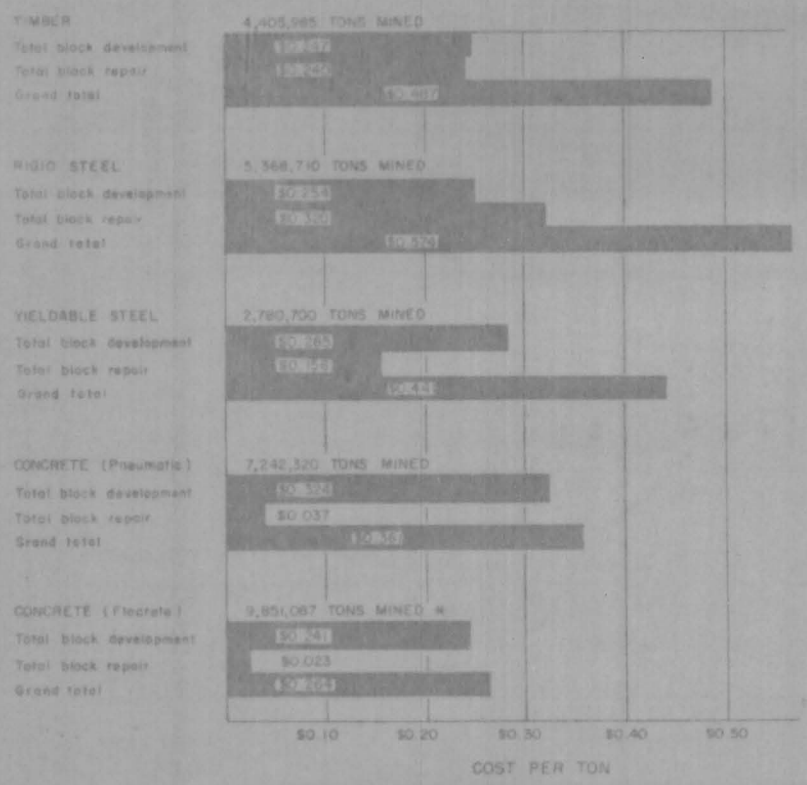
FIGURE 42. - Block Grizzly-Drift and Draw-Raise Costs Per Ton of Ore Developed.

Grouting

Grouting at San Manuel has been very effective on the haulage level in retarding rock movement, preventing timber breakage, eliminating runs of loose material during timber repair, and reducing overbreak during the excavation of raise stations. Grouting was tried first on turnouts where rock was very loose. Good results were obtained, and the practice was extended to timber repair and raise-station excavations.

Grouting is placed where the rock becomes loose or where heavy pressures develop above the caps of timber sets in haulage drifts. It is accomplished by drilling 1-1/2-inch holes in rings on 5-foot centers through timber to depths of 10 and 20 feet in the rock. A 1/2-inch-diameter pipe is inserted in a hole, and a grout pump is attached.

Grout consists of water, cement, and a little bentonite. Bentonite promotes flowability and plasticity while the grout is being placed. Usually grout will plug the cracks in the rock, but occasionally cracks and holes must be plugged with wood or paper cement sacks. The grout is pumped at 300 p.s.i. At first, four to five holes were drilled in a ring, but experiments proved that two holes to a ring were sufficient. Sometimes the 10-foot holes are grouted first at pressures considerably less than 300 p.s.i. The 20-foot holes are grouted last, and always to a pressure of 300 p.s.i.



* Does not include blocks that have not been under full draw for a minimum of 60 days.

FIGURE 43. - Block Development and Repair Costs.

Loose, unconsolidated fault zones in haulage drifts have been grouted before timber repair. The present practice is to pump the grout into the drift backs in such areas and allow it to set. The broken timber is then removed, and the back is excavated for clearance. Thus, no spiling is required. With this repair practice, main haulage drifts are closed for very short periods.

Raises located in loose, unconsolidated material are driven by first grouting the area. After the grout has set, the raise is drilled and blasted to the desired size. Timbering and overbreak are reduced.

Mining

Caving Characteristics

The ore at San Manuel has good caving qualities and generally flows readily from draw raises. After final blasting of undercut pillars, a block is drawn rapidly (as much as 3 feet a day). The general procedure is to draw uniformly from all fingers. After initial caving, the goal is an average daily draw of 18 inches, with an equal draw from all points; however, a draw of 2 feet a day is not considered excessive.

In most cases caving progresses rapidly until it reaches the overlying Gila conglomerate. Visual observations on the 1285 level disclosed that no void existed in the porphyry between virgin ground and the caved material.

Ore fragmentation varies with the hardness of the rock. Ore from the eastern part of the mine is harder and coarser than that from the western part. This difference is apparent in the tonnage broken on the grizzlies. On the east end 7.5 percent was broken, compared to 1.8 percent on the west end. For a short time after undercutting, the fragmentation is coarser than it is after the draw is well established.

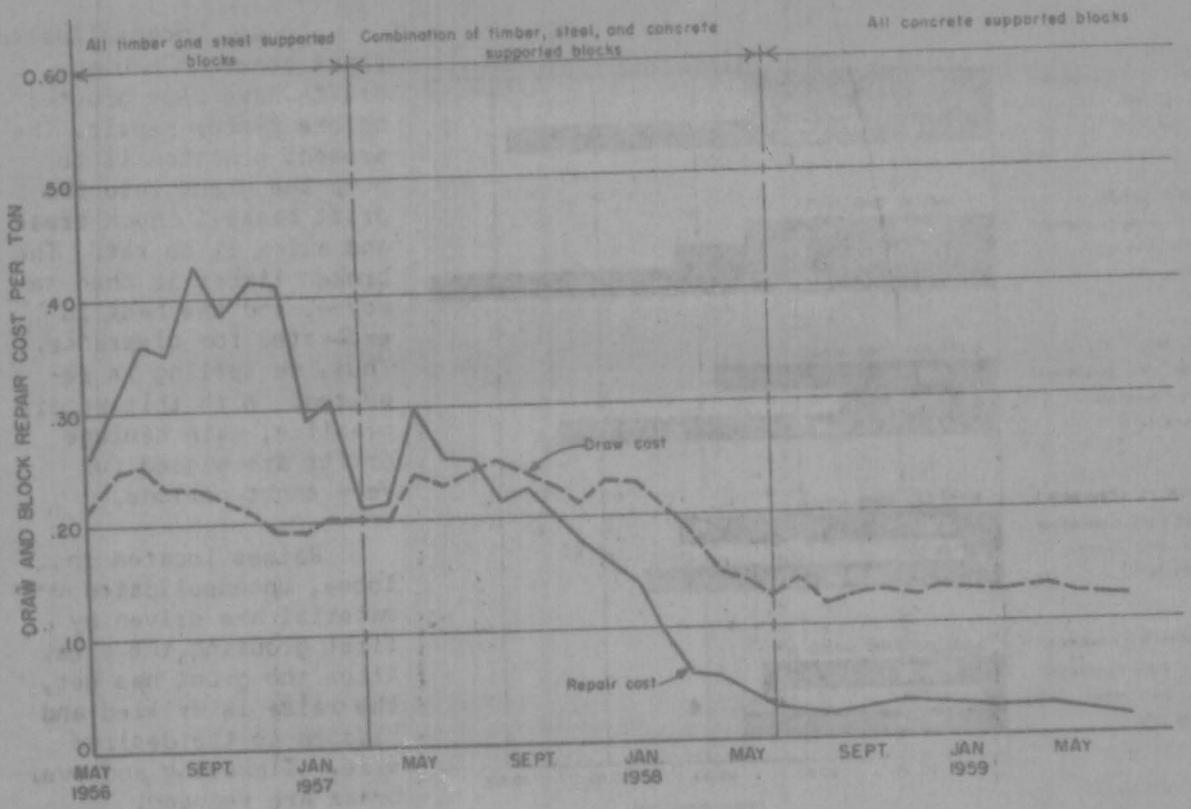


FIGURE 44. - Draw and Block Repair Costs.

Production Planning

Detailed planning is conducted 1-1/2 years in advance by the engineering department. These plans give monthly scheduling for shaft sinking, level development, block development, block sequence, and production.

Ore reserve estimates and block expectancies are compiled primarily from drill logs. Development headings on the grizzly level are sampled every 10 feet; haulage workings are sampled every 20 feet. The reserve estimates are adjusted periodically to the development sample results. The present ore cutoff is 0.55 percent recovered copper, but this figure varies with production conditions and requirements. Tables are used to convert total copper to recovered copper, depending on the oxide content.

The progress of development headings is controlled by a weekly development efficiency report which compares actual progress with quotas based on the development schedule. Block production quotas are set by the mine superintendent, and these are compared with actual production on the daily mine production report.

Both development and production progress are reviewed monthly.

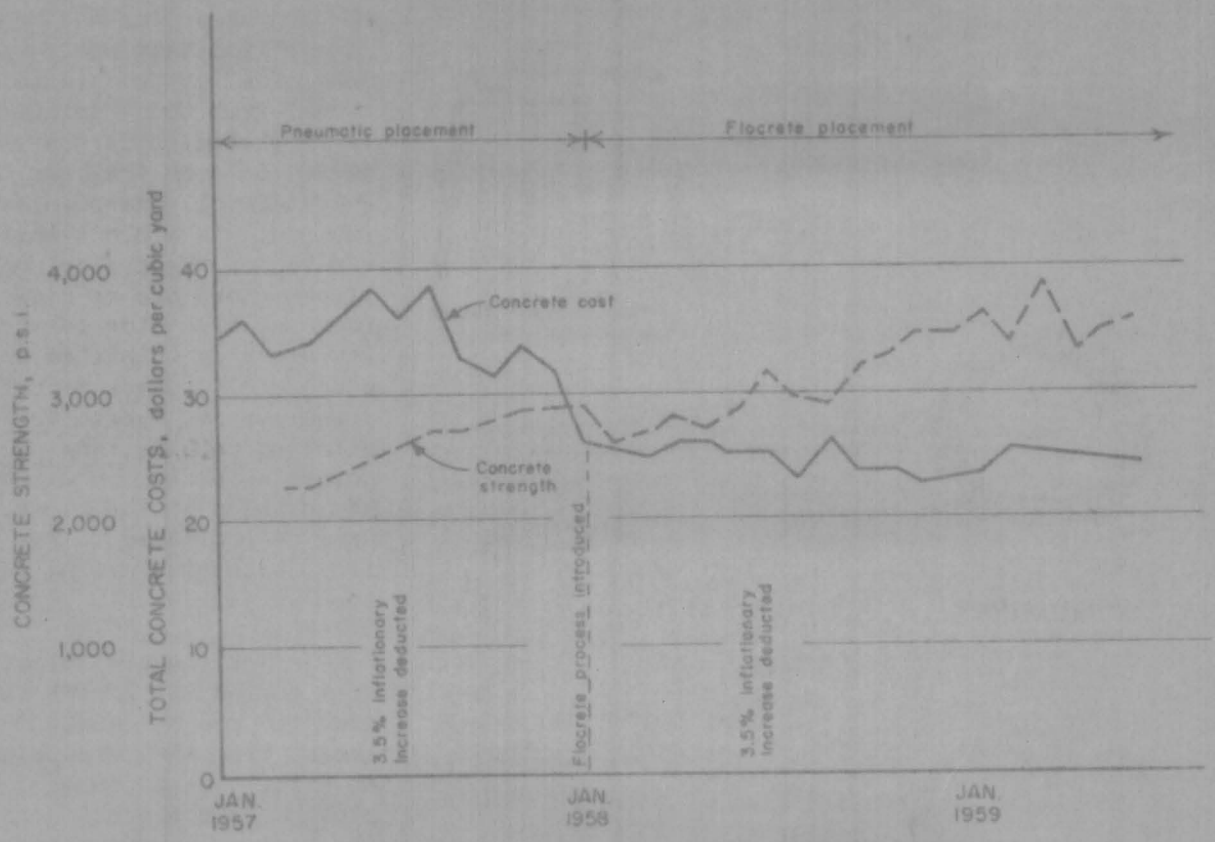


FIGURE 45. - Total Cost of Underground Concrete Forming and Placing, and Compressive Strength of Concrete.

Stope Records

Diamond-drill and churn-drill sample information is plotted on cross-section sheets. From these sheets, longitudinal sections parallel to the long axis of the ore body are prepared on 30-foot spacing.

Stope books are prepared that contain the tonnage and grade for each block and a sheet for each draw-raise line within the block. These sheets show three basic types of information: (1) The profile of the upper limit of ore against which the actual draw may be plotted; (2) the geology of each line, including details of the ore zone and the overlying capping; (3) and a graph where the grade of ore drawn may be plotted against expectancy. Each stope book contains a composite plan showing block layout and its relationship with the haulage level, and a summary sheet of block statistics.

Draw Practice

Draw from a block is controlled as soon as the first undercut pillar is blasted. As pillars above each drawpoint are blasted, ore is drawn rapidly until about 500 tons has been drawn. This permits inspection to ascertain that

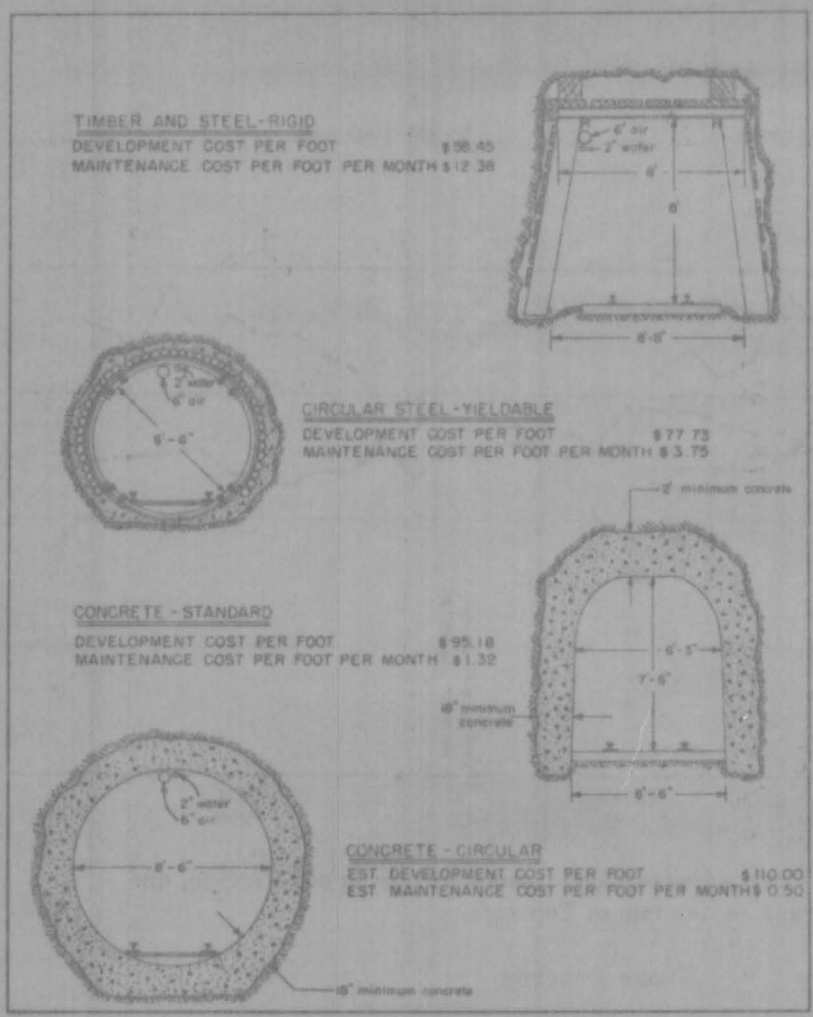


FIGURE 46. - Types and Costs of Stope Panel-Drift Support in Heavy Ground Areas.

all of the pillar has been removed and that space has been made for the ore to cave. When the undercut is completed, orders are given to "even the back" and start all drawpoints running. In a block where the top of the ore column slopes, more ore is drawn from the high side so that the block is completed as a unit. If a back is not drawn evenly, excessive dilution results, and points of excessive weight may develop in the block. Draw is increased to relieve excessive weight in some areas.

After the preliminary draw period, an effort is made to draw an equal amount from each drawpoint. An erratic draw causes piping and ore packing in the various drawpoints. If a drawpoint is not pulled regularly, the ore packs in it.

Rate of Draw

Experience has shown that an average rate of draw of 18 inches of ore in place, or about 35 tons per draw raise per day, is the most effective rate. In order to produce 33,000 tons of ore per day, it is necessary to have 8 to 10 blocks in full production, and to have this output supplemented by tonnage from blocks closing out or coming into production.

Chute Tappers and Blasters

Chute tappers and blasters fill the transfer raises according to their draw orders. Broken ore usually is measured by filling the raises before trains are loaded. Tonnage is estimated if trains are loading while ore is running into the transfer raise from draw raises. The capacity of an all-gravity transfer raise is 35 tons in the main branch and 20 tons in the back-over. In slusher blocks the tonnage drawn is estimated by counting scraper loads. When transfer raises above a haulage panel drift contain a trainload

of ore, the dispatcher is notified, and he sends a train into the loading chutes. Tonnage drawn is cross-checked with the car reports of haulage loaders.

Chute blasters do all of the ordinary secondary blasting in the blocks with block holes and bombs of 45-percent gelatin dynamite, and instantaneous electric blasting caps.

High hangups and high packs in draw raises are brought down by specially trained "high-pack" crews. Sometimes a crew can free a hangup by climbing ladders and placing charges of explosives directly against the boulders. At other times it is necessary to erect timber stagings from which the boulders can be drilled and blasted.

Blasting is done only at lunch time and at the end of the shift. The shift foreman is responsible for blasting from a central location after an accounting of all personnel has been made. This blasting procedure does not delay operations and allows time for the smoke and dust to clear before men reenter the blocks.

Modifications in Draw Practice

To better cope with existing conditions, several modifications of the original plans have been made. At first, excessive time was consumed in breaking boulders, clearing flooded grizzlies, and opening small draw raises. Later, the 8-pound hammers used by the chute tappers were replaced by 16-pound hammers. Grizzly rail spacing was increased from 9 inches to 12-1/2 inches; this allowed about 90 percent of the boulders to pass the grizzly. The size of the transfer-raise opening was increased and the grizzly was lengthened by 2 feet. The change in grizzlies virtually eliminated flooding.

In the original timbered blocks, draw raises were cribbed to a height of 25 feet above the grizzly sill, and the unsupported part of the draw raise was driven with a diameter of 3 feet. This cribbing took weight after undercutting closed the raises. Later, the cribbing was eliminated, and the draw-raise diameter was increased to 5 feet. The draw raises were belled out at the undercut level to a diameter of 8 feet. This has greatly reduced the hangups in the raises.

The vertical distance between the floor of the grizzly drift and that of the undercut level was reduced from 25 to 20 feet. This brought the hangups in the draw raise closer to the raise mouth and hence made them more accessible for blasting. The shorter raise reduced the block development cost. No appreciable increase in weight on the grizzly level has been noted.

Based on the caving experience of other mines, it was orginally planned to draw 12 inches of ore in place per day. After modifications were made on the grizzly level, the rate of draw was increased to 18 inches per day. This increased rate of draw reduced the active blocks required to meet the production quota. It also increased fragmentation of the ore and indicated better copper recovery from blocks initially drawn. Pillar blocks are drawn at a slower rate than those with four sides in virgin ground.

Draw Control

Procedure

The chief engineer and planning engineer maintain a 1½-year tonnage and grade production forecast. This is corrected periodically to actual tonnages drawn. A monthly tonnage quota for the various blocks is set by the mine superintendent, who, with the general mine foreman and stope engineers, works out an allotted tonnage for the three shift foremen. A balanced tonnage per shift is maintained.

The stope engineer gives draw orders, compiled from the quotas, to the draw boss. The chute tapper is assigned to a grizzly line and makes a copy of the tapper report. He tries to fill his quota. The tonnage drawn is entered on the stope engineer's draw-order sheet and submitted to the tonnage department. The drawn chute tonnage is adjusted to the tram report once a week. When a block is being undercut or sealed off, the stope engineer may request an adjusted drawsheet at more frequent intervals.

Drawpoints are sealed off temporarily for chute repairs and to break up piping to control dilution. They are sealed permanently when the ore reaches the cutoff grade of 0.55 percent recovered copper. Recovered copper is a computed value equal to the total copper assay less the sum of the oxide copper and tailing loss.

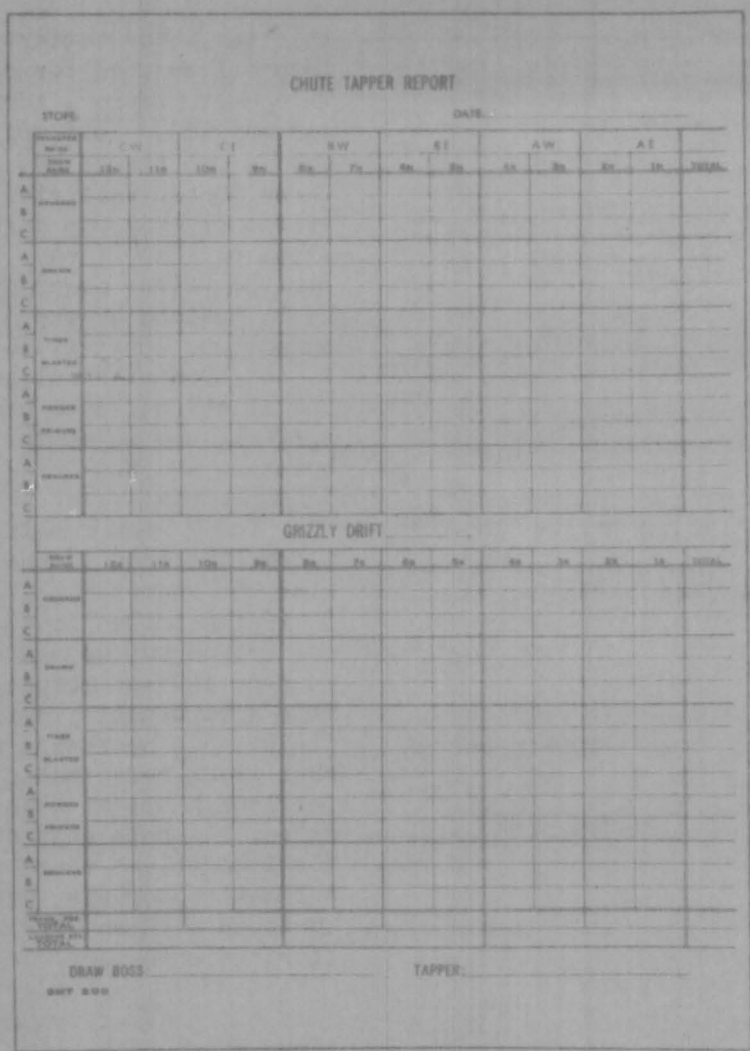
Reports

From various daily mine shift reports (figs. 47 through 53), the production department prepares daily and monthly tonnage reports (figs. 54 through 56), assay compilations (fig. 57), draw-tram comparisons, and special studies of production records requested by the mining department.

Sampling and Grade Control

Ore reserve estimates are compiled from drill logs. To help delineate the ore, primary development headings on grizzly levels that are entering or leaving the ore zone are sampled at intervals of 10 feet. When in ore, development headings on haulage levels are sampled at intervals of 20 feet. The interval is reduced to 10 feet where the headings approach the limits of the ore body. The results of the sampling of development headings are used when the estimate of ore reserves is adjusted periodically.

For grade control of the blocks, a 5-pound sample is taken at each drawpoint from each 100 tons drawn. Sample results are given regularly to the stope engineer. He may request that samples be taken from each 50 tons of ore drawn when the grade at any drawpoint approaches the cutoff point. When the grade reaches the cutoff point, he compares the physical appearance of the rock with the assay and decides if the raise should be sealed or held for subsequent sampling.


CHUTE TAPPER REPORT

STOPE: DATE:

GRIZZLY DRIFT

DRAW BOSS: TAPPER:

FIGURE 47. - Chute-Tapper Report Form.

"Hold" signs are painted on boulders in drawpoints which have been overdrawn, show early dilution, or are very low grade and ready to be sealed permanently. Drawpoints are sealed after the assay returns from three consecutive samples fall at or below the grade cutoff. At the discretion of a stope engineer, temporarily sealed drawpoints are reopened occasionally to test for grade improvement.

In low-grade block-caving operations, large tonnage and close grade control are essential for economical operation. The stope engineer attempts to draw the block evenly without cutting tonnage.

Underground Transportation

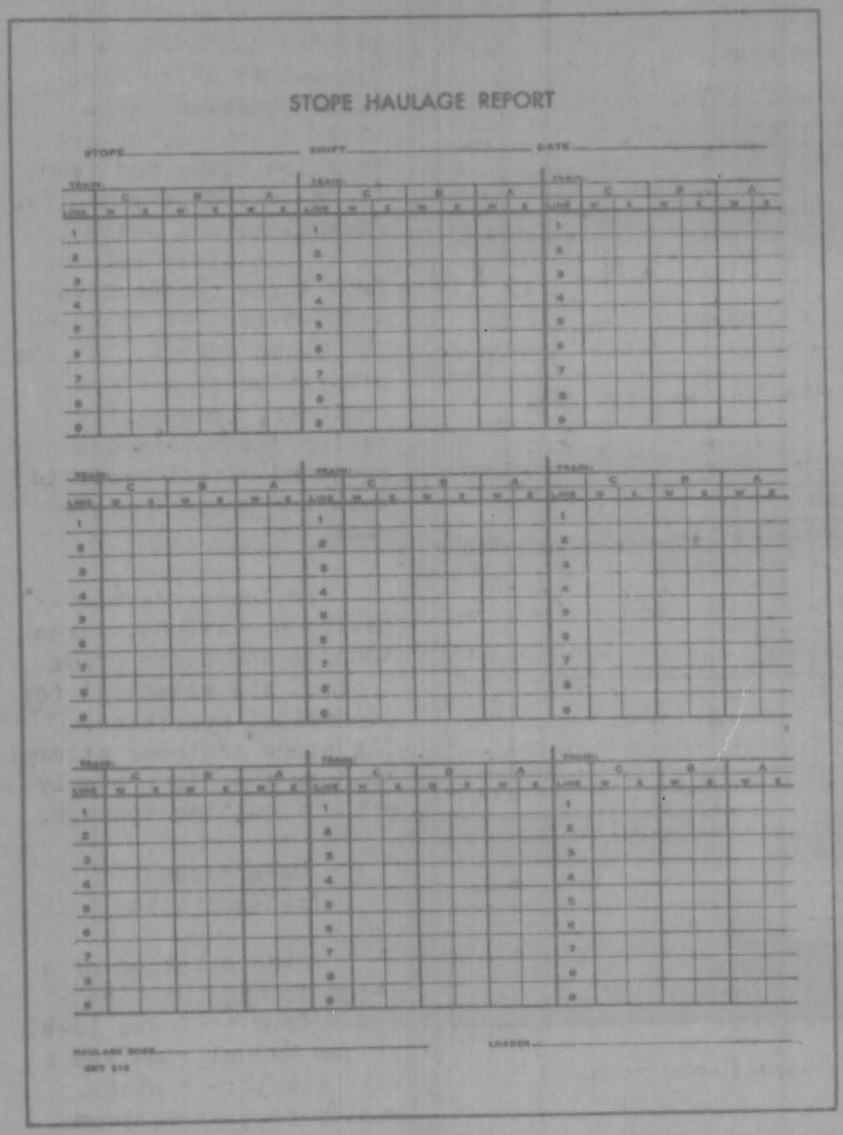
Transportation of a large tonnage at high speed from a single level at San Manuel requires a very flexible haulage system to give maximum access to stoping areas, minimize delays, and provide for safety. With these requirements as a guide, the haulage was laid out as a simple double loop with one-way traffic.

The following is abstracted from a paper[13] presented by C. F. Cigliana, general mine foreman at San Manuel.

Equipment

Haulage equipment consists of fourteen 23-ton locomotives and two 12-ton locomotives that operate from a 275-volt direct-current trolley line and

[13] Cigliana, C. F., Ore Transportation at San Manuel: Min. Eng., vol. 10, No. 5, May 1958, pp. 573-576.



STOPE HAULAGE REPORT

STOPE _____ SHIFT _____ DATE _____

TRAIN	C		B		A		TRAIN	C		B		A		TRAIN	C		B		A	
LOAD	W	E	W	E	W	E	LOAD	W	E	W	E	W	E	LOAD	W	E	W	E	W	E
1							1							1						
2							2							2						
3							3							3						
4							4							4						
5							5							5						
6							6							6						
7							7							7						
8							8							8						
9							9							9						

TRAIN	C		B		A		TRAIN	C		B		A		TRAIN	C		B		A	
LOAD	W	E	W	E	W	E	LOAD	W	E	W	E	W	E	LOAD	W	E	W	E	W	E
1							1							1						
2							2							2						
3							3							3						
4							4							4						
5							5							5						
6							6							6						
7							7							7						
8							8							8						
9							9							9						

TRAIN	C		B		A		TRAIN	C		B		A		TRAIN	C		B		A	
LOAD	W	E	W	E	W	E	LOAD	W	E	W	E	W	E	LOAD	W	E	W	E	W	E
1							1							1						
2							2							2						
3							3							3						
4							4							4						
5							5							5						
6							6							6						
7							7							7						
8							8							8						
9							9							9						

HAULAGE BOSS _____ LOADER _____

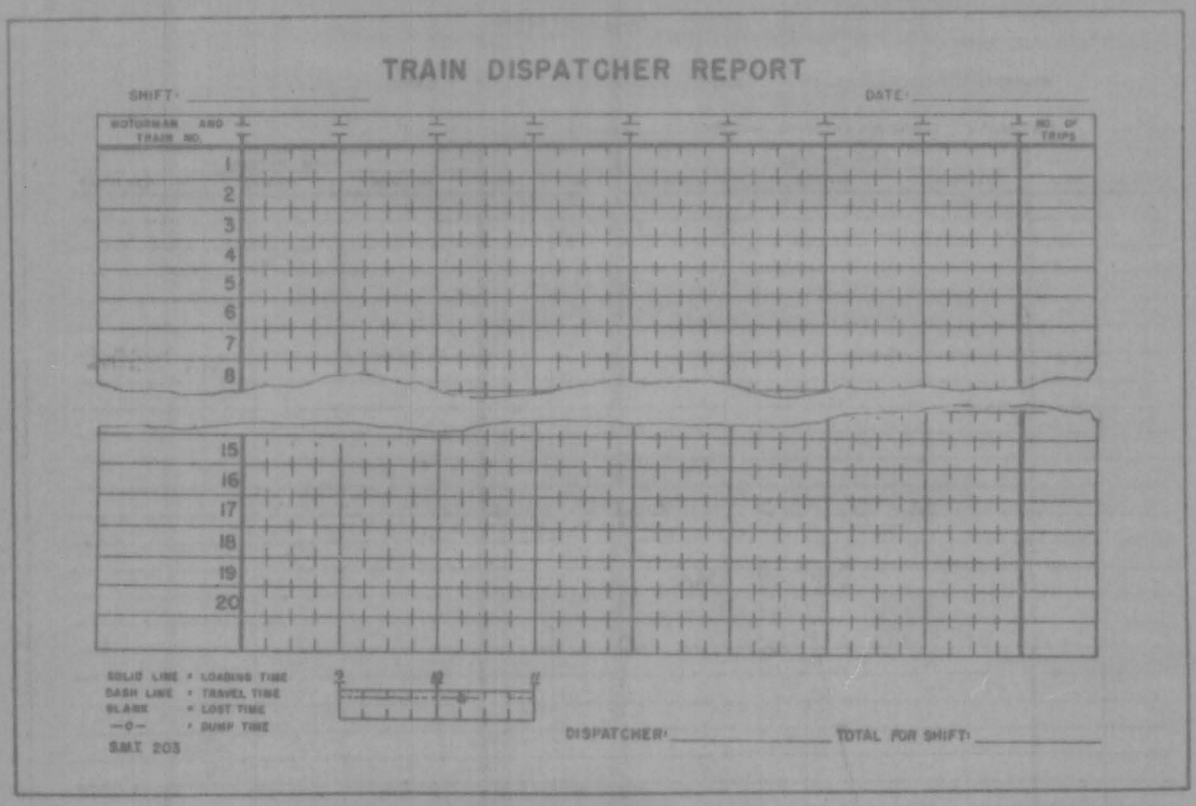
FIGURE 48. - Stope-Haulage Report Form.

thirteen 8-ton battery locomotives used for development, repair, supply, and service. Full voltage is maintained by three 50-kw. rectifier stations; they are parallel so that if any station fails, power can be maintained by the other two. One station is on the north haulage, one is on the south haulage, and one is at the hoisting shaft.

The 23-ton trolley locomotives (64 inches wide, 56 inches high, and 21 feet 6 inches long) are driven by two centrally hung 120-hp. motors. This arrangement provides the balance necessary for best track performance for long hauls at high speed. At both ends of the locomotives there are automatic couplers, each equipped with an air-operated uncoupler.

All locomotives have three types of braking on all four wheels--air braking, hand braking, and dynamic braking--and are also equipped with overspeed and deadman controls.

Control is set up for selective series or parallel operation of traction motors in either direction. Acceleration is obtained by moving the main drum clockwise, and dynamic braking is provided by moving the main drum counterclockwise from the off position. The controller is equipped with a deadman tip-up handle, which must be held down to apply power to the locomotive; if it is released, the circuits are opened, and power is withheld from the traction motors. At the same time emergency airbrakes operate. If the locomotive exceeds 15 m.p.h., the overspeed contact will automatically open, removing power from the traction motors, and brakes will be applied through an emergency air valve in the run or brake position. When the motor speed has been sufficiently decreased, power again can be applied.



TRAIN DISPATCHER REPORT

SHIFT: _____ DATE: _____

MOTORMAN AND TRAIN NO.									NO. OF TRIPS
1									
2									
3									
4									
5									
6									
7									
8									
15									
16									
17									
18									
19									
20									

SOLID LINE = LOADING TIME
DASH LINE = TRAVEL TIME
BLANK = LOST TIME
—O— = DUMP TIME

SMT 203

DISPATCHER: _____ TOTAL FOR SHIFT: _____

FIGURE 49. - Train-Dispatcher Report Form.

The handbrake is applied by a 12-inch wheel operating a square-threaded brake screw. Airbrakes exert pressure on the brake linkage in the same way as the handbrake. A pressure cylinder is connected to the standard hand-operated brake linkage, and a governor cuts the compressor into the circuit when the pressure is too low to be effective.

Eleven trains of fifteen 12-ton cars each are needed to haul the ore. Each car is equipped with one stationary coupling and one rotary coupling and a local innovation--a safety coupling used to prevent cars from uncoupling in transit. The box-type steel cars have rubber-cushioned draft gear and are mounted on two four-wheel roller-bearing trucks with 14-inch-diameter gray chilled cast-iron wheels.

The entire haulage operation is controlled by a dispatcher from whom clearance must be obtained before trains are moved into the haulage loops.

Train Loading

There are three phases in ore transportation--loading, traveling, and dumping. In the first phase, trains are loaded against the track grade to take

POCKET DUMP REPORT

SHAFT: ____________ SHIFT: ____________ DATE: ____________

ESTIMATED TONS BEGINNING SHIFT: ____________

TRAIN	TRIP	CARS DUMPED			TRAIN	TRIP	CARS DUMPED		
		ALLISON	MASON	DEVELOPMENT			ALLISON	MASON	DEVELOPMENT
TOTAL					TOTAL				

SKIPS HOISTED: ____________

SKIP TENDER: ____________ DUMPMAN: ____________

BMT 504

FIGURE 50. - Pocket-Dump Report Form.

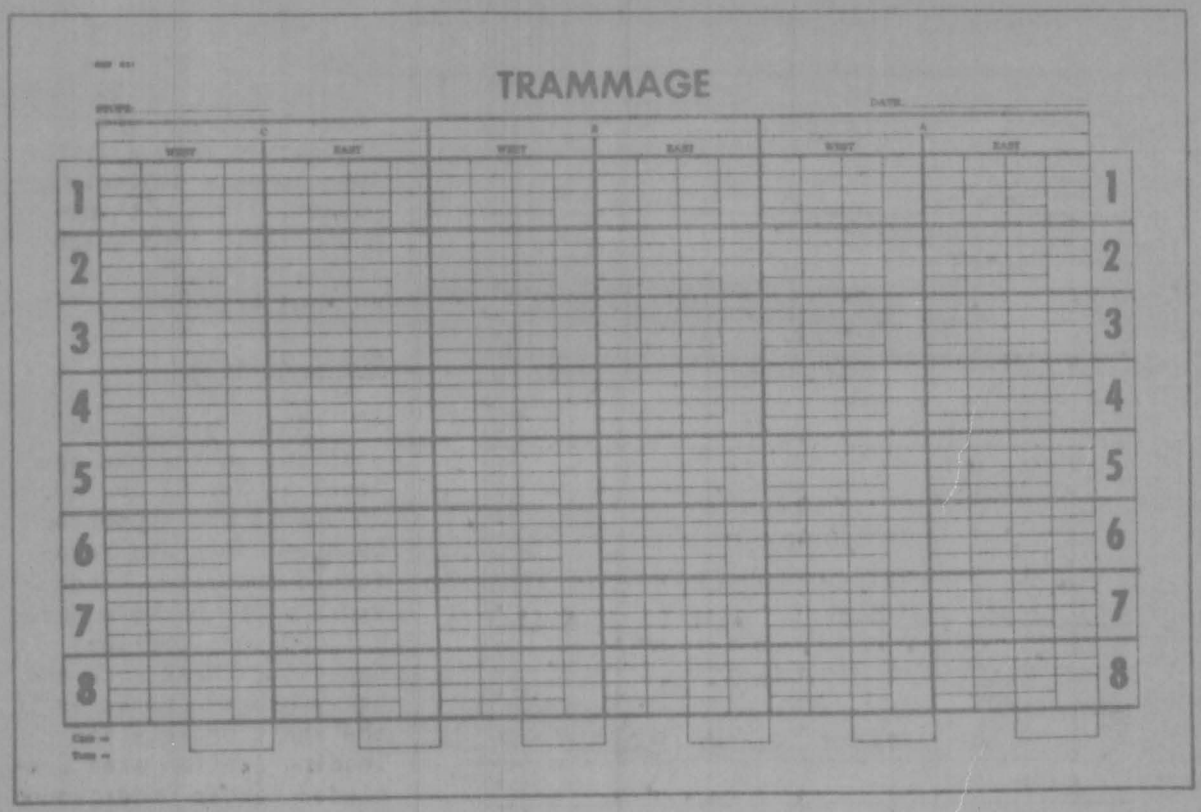


FIGURE 51. - Trammage Form.

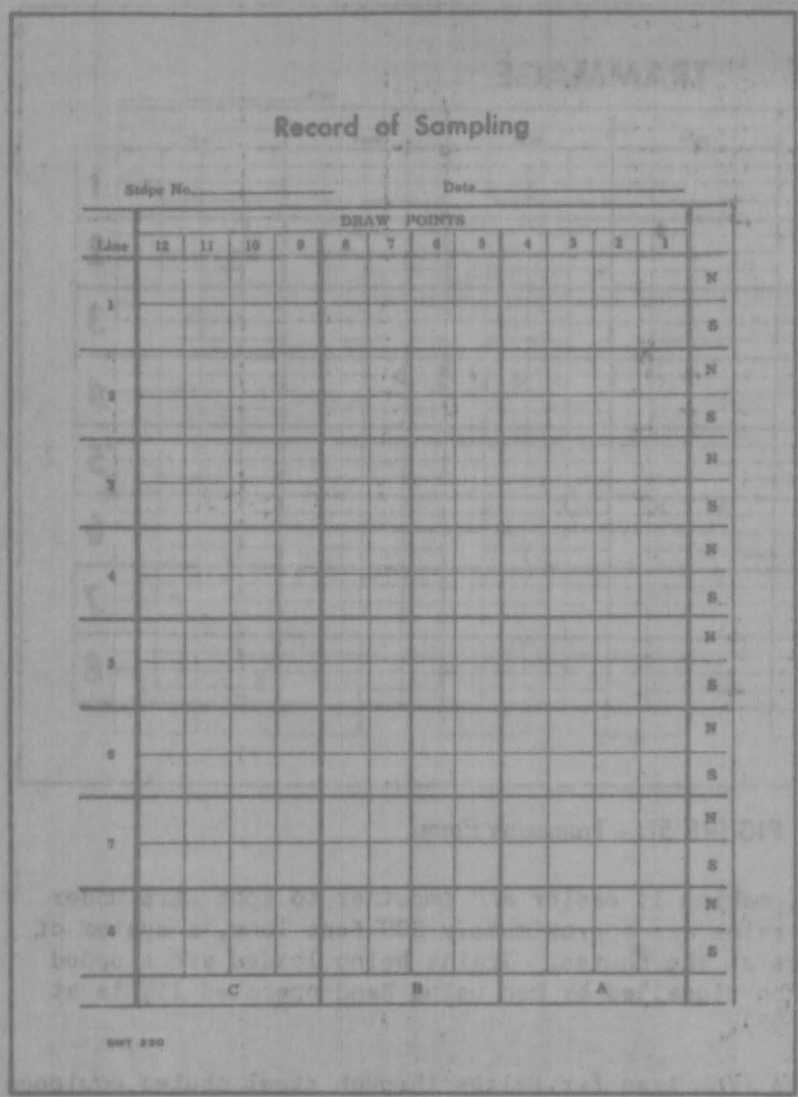
all slack out of couplings, making it easier and smoother to spot cars under the loading chute. Since trains are approximately 300 feet long, a system of signals is used to spot cars at the chutes. Trains being loaded are stopped or moved by the operator when signalled by men using hand-operated lights at the loading stations (fig. 58).

Ore flows into the cars from transfer raises through steel chutes equipped with undercut guillotine gates (fig. 23).

The mine operates on a three-shift-a-day basis, but blocks are drawn on a staggered basis of two shifts, leaving the third shift for maintenance and repair work. Cleaning the drifts of spillage from chutes during loading is one of the jobs done on the third shift. The spillage is loaded with a rocker shovel and a specially constructed track cleaner that draws power from the trolley line (fig. 59).

Hauling and Dumping

When a train is loaded, the operator obtains clearance and hauling instructions from the dispatcher and proceeds into the north haulage, clearing the panel drift and closing the haulage block signals. He then observes posted speed limits and block signal procedure while traveling the haulage loop to the rotary dump.



Record of Sampling

Stope No._____ Date_____

Line	DRAW POINTS 12	11	10	9	8	7	6	5	4	3	2	1	
1													N
													S
2													N
													S
3													N
													S
4													N
													S
5													N
													S
6													N
													S
7													N
													S
8													N
													S
			C				B				A		

SMT 220

FIGURE 52. - Record of Sampling Form.

Cars are dumped three at a time in a tipple that rotates 180° (fig. 60). The ore is discharged into a 1,000-ton-capacity concrete pocket, lined with 60-pound rail, from which it is hoisted to the surface by skips.

Shaft Pockets

Shaft pockets and stations on the haulage level in No. 3A and No. 3B shafts are supported by steel sets and reinforced concrete. The storage ore pockets have a 1,000-ton live-load capacity. Weak rock and heavy ground pressures in the shaft pockets and loading station area precluded larger underground storage areas.

The skip loading arrangement is laid out to measure the required volume of ore from a storage pocket through an air-powered undercut-guillotine-type gate into a measuring basket or skip loading pocket. The pocket chute door is operated by a 16-inch air cylinder with a 56-inch stroke. The skip loading pocket (fig. 61) discharges its load directly into the skip through an air-powered overcut arc-type gate. Provision was made by planning engineers for the installation of skip loading chairs, but these were unnecessary.

Hoisting is semiautomatic in that the automatic controls that move the skip from the loading position to the dumping position are actuated by the skip tender at the loading pocket.

Skip Loading

Two men load the skip to avoid the hazard that may occur when especially wet ore is handled. Electrical signal lights are used to coordinate the

loading operation. One man draws the ore from the storage pocket into the measuring pocket. The other man loads the skip from the measuring pocket and then sets into motion the semiautomatic hoisting cycle. Automatic controls take the skip from the loading position to the dumping position. Ore skips are run in balance.

Personnel and Performance

A typical haulage shift is organized as follows:

Haulage-shift boss	1
Jigger boss	1
Dispatcher	1
Rotary-dump operator	4
Skip tenders	4
Operators, 23-ton locomotives	11
Operators, 12-ton locomotives	2
Chute blasters	11
Chute tappers	11
Cleanup man	1
Total	47

This organization is capable of hauling a maximum of 13,124 tons in a single shift. The maximum for three shifts is 35,321 tons. The average ore tonnage per man-shift on haulage only is 245.

A breakdown of the average time consumed in the four phases of the haulage operation shows the loading time to be 24.43 minutes, hauling time to the rotary dumps, 12.94 minutes; dumping time, 4.64 minutes; and return trip, 12.12 minutes. This totals 54.13 minutes per train, and each train averages 6.4 trips per shift.

Shaft Stations

No. 4 shaft stations on the haulage and grizzly levels are 22 feet wide by 11 feet high, rising to 17 feet at the shaft. All stations are supported with steel sets and reinforced concrete. The stations can handle the largest items of equipment and supplies used in the mine. Stations in other shafts are large enough to handle the equipment needed for development headings driven from the shafts and to provide room for pumping equipment needed in sinking.

Underground Equipment Maintenance

A regular schedule is followed for all maintenance and lubrication of underground operating equipment. Electric slushers are inspected and lubricated each day. Spare slushers are kept in stock to replace worn machines. When necessary, steel chutes are repaired on each shift. Haulage cars are inspected daily, and 15-car trains are sent to the maintenance shop for lubrication and preventive maintenance. A cycle of these cars is completed each month. The cycle for the smaller 5- and 7-ton cars is 2 months. About 1,800 pounds of lubricants is required for a cycle of all mine cars. Rotary dumps, loading equipment, and hoists are inspected and lubricated daily.

DAILY TONNAGE
COMPUTATION SHEET

DATE_____

STOPE ORE

STOPE	A SHIFT		B SHIFT		C SHIFT		TODAY		CARS THIS MONTH
	CARS	TONS	CARS	TONS	CARS	TONS	CARS	TONS	
TODAY									
MONTH							CAR FACTOR		TONS

DEVELOPMENT TALLY

A SHIFT		B SHIFT		C SHIFT	
PLACE	CARS	PLACE	CARS	PLACE	CARS

	TONS	BCTRAM
3A POCKET		
3B POCKET		
NO. 1 SHAFT		
TODAY		
MONTH		

CARS TO SMELTER	TODAY	MONTH

DUMP TALLY

POCKET	A SHIFT		B SHIFT		C SHIFT		TODAY		MONTH TOTAL	
	BC	SC	BC	SC	BC	SC	BC	SC	BC	SC
3A										
3B										
TODAY										
MONTH										

FIGURE 53. - Daily Tonnage Computation Sheet.

Three hours each Monday and Friday are set aside for minor preventive maintenance. Major maintenance is done during holiday periods, and emergency repairs are undertaken as needed. A large locomotive shop is situated on the haulage level in the No. 1 shaft area. Three large haulage locomotives are rotated through the shop at a time; one locomotive goes through for major overhaul, and the other two are exchanged periodically for preventive maintenance. The cycle requires about 1 week.

Communications

Good, dependable communication is one of the key factors in a safe, high-speed, efficient underground mining operation. To coordinate a working force of approximately 900 men and its equipment, three methods of communication are used: A standard telephone system, a two-way audio system, and a radiotelephone system.

The following information about the communications system is abstracted from a paper[14] by C. L. Pillar, mine superintendent.

Telephone System

The standard telephone system is mentioned briefly because it is highly conventional and has been in standard use in industrial operations in this country for the past 50 years. The main function of this system is to connect underground operations with surface operations and service facilities.

[14] Pillar, C. L., Mine Communication System at San Manuel: Proc. Ann. Metting AIME, San Francisco, Calif., February 1959, 9 pp.

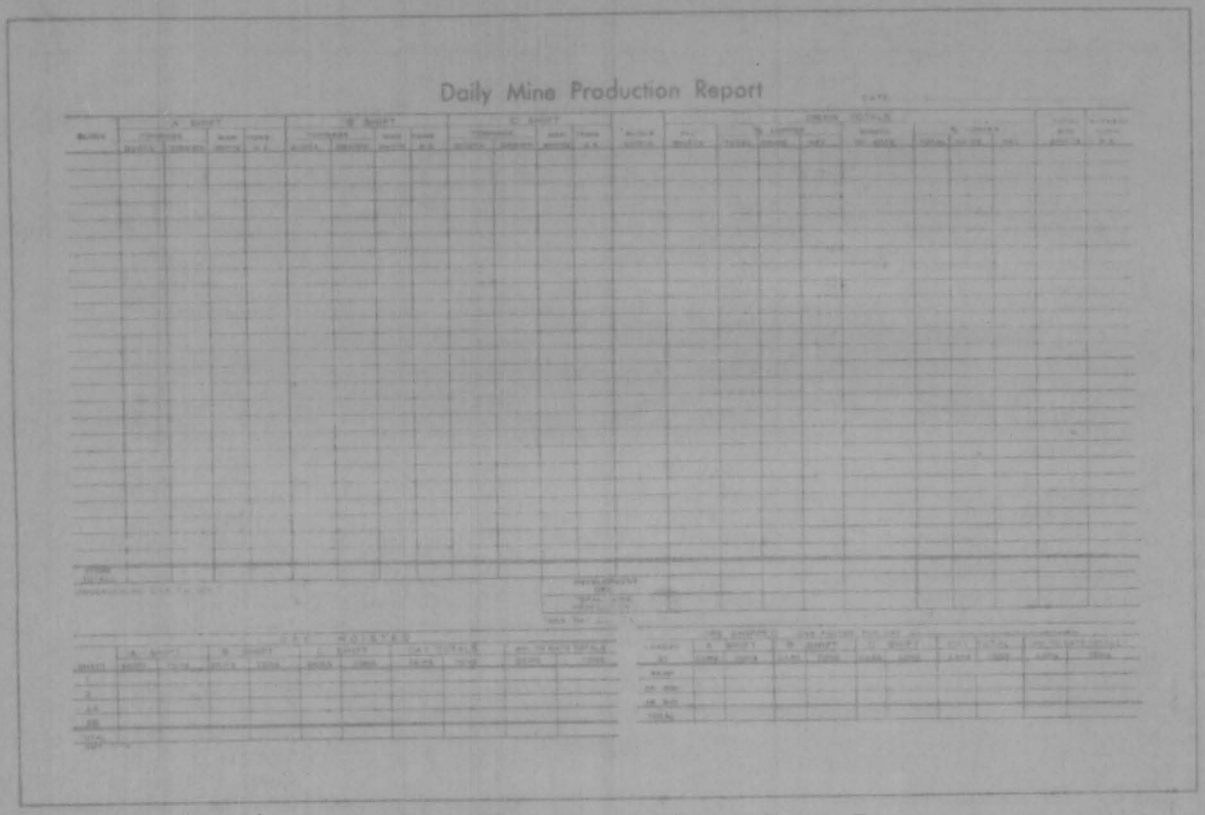


FIGURE 54. - Daily Mine Production Report Form.

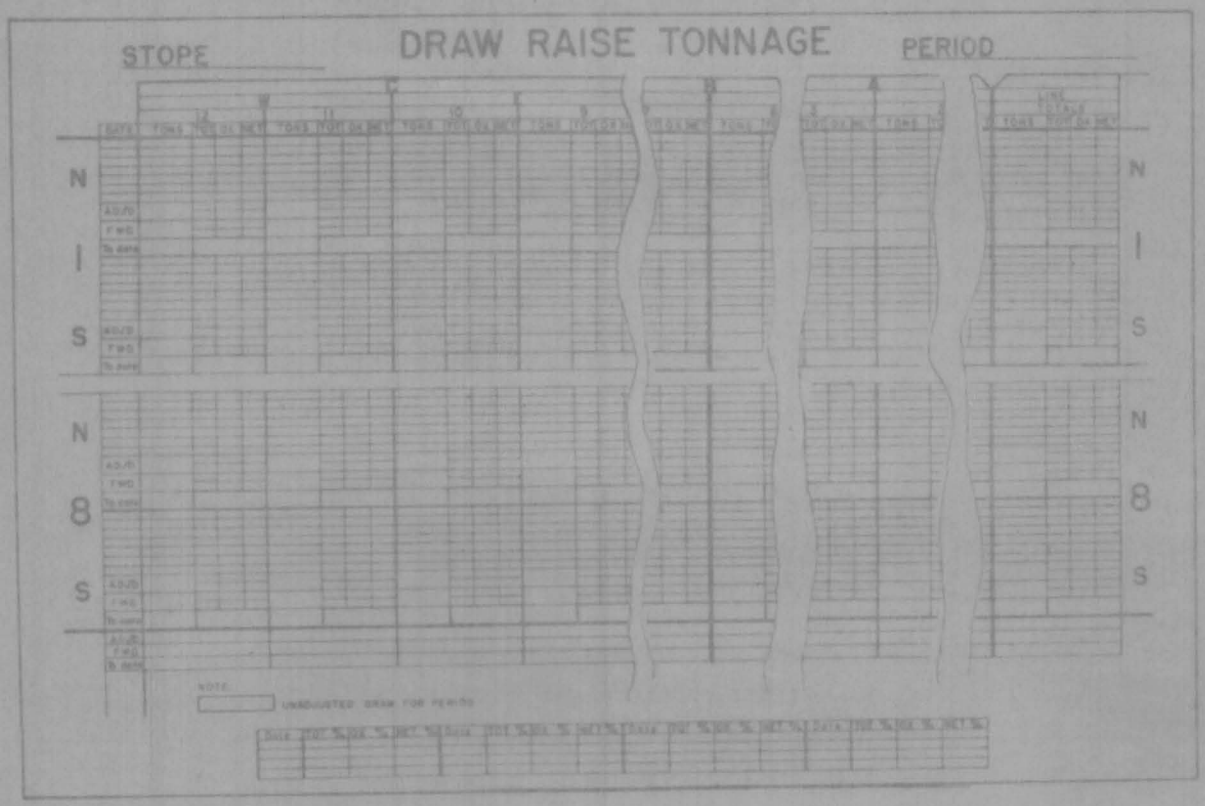


FIGURE 55. - Draw-Raise Tonnage Form.

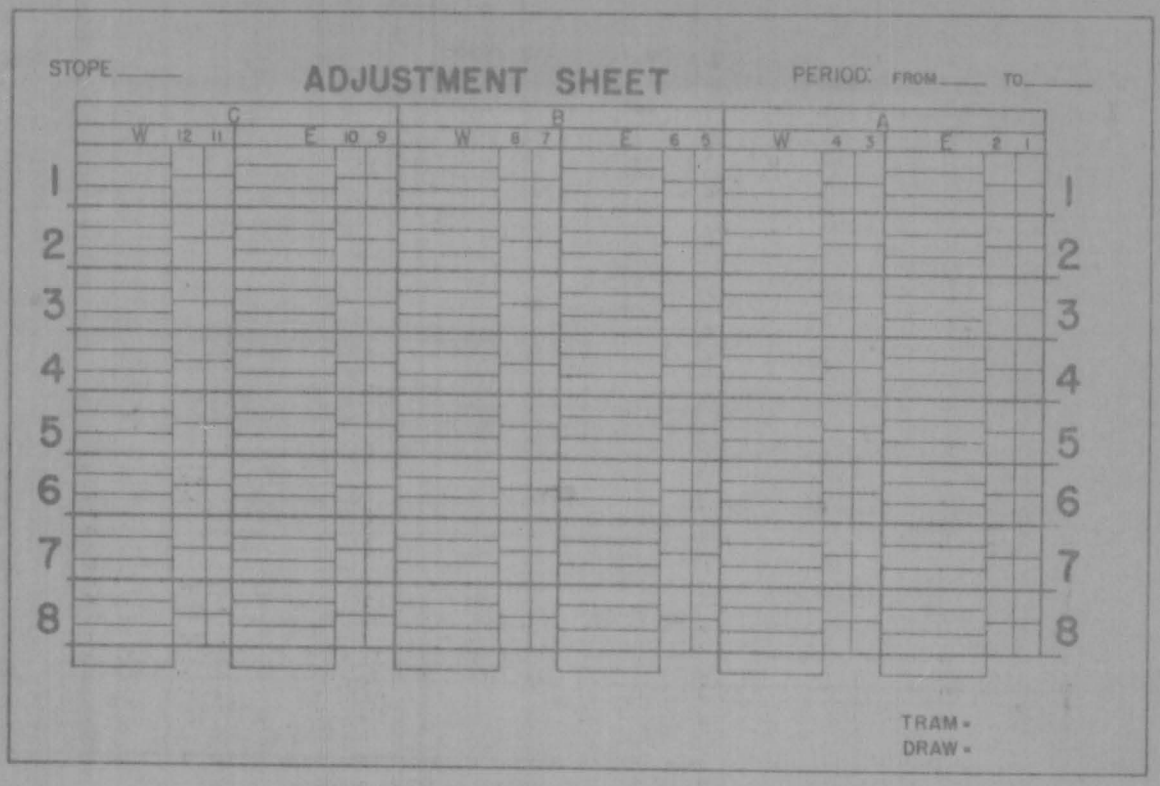


FIGURE 56. - Adjustment Sheet Form.

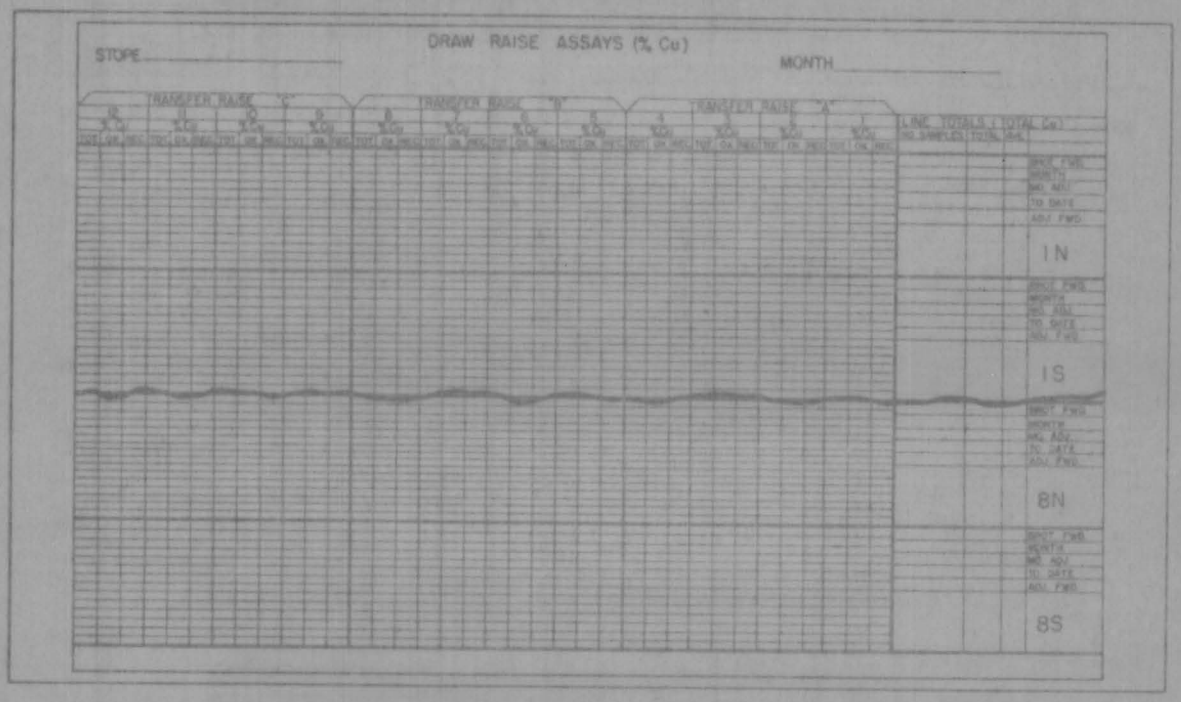


FIGURE 57. - Draw-Raise Assays Form.

GENL. INF BOOKLET 4/1968

MAGMA COPPER CO.
SAN MANUEL DIV.

CONCRETING AT SAN MANUEL
HW SEANEY – R.L. TOBIE
MIN. ENG. 11/65

DRAW CONTROLS AT SAN MANUEL
RL TOBIE – E.K. STALEY
MIN. ENG. 6/67

CHANGES IN MINING SEQUENCE
TO COPE WITH GROUND WEIGHT
AT SAN MANUEL
J.D. PELLETIER 11/68
COPY OF COMPANY REPORT

MINING, MILLING AND SMELTING
METHODS SAN MANUEL.
IC 8104

CAN INST PUB 1957, PAGES 107 – 138
X DEVELOPMENT OF LONG
BLAST HOLES AT NORANDA

MAGMA COPPER COMPANY
SAN MANUEL DIVISION
SAN MANUEL, ARIZONA 85631

May 16, 1969

Mr. Roland Mulchay
2732 Wren Road
Salt Lake City, Utah

Dear Roland:

The accompanying collection of articles on methods and practice at San Manuel should provide the information you requested. All that I can add to this information is that changes and improvements have effectively eliminated many of the heavy ground problems encountered in the early days. We see no serious problems in raising daily mine production to 60,000-70,000 tons by mid-1971.

I hope you have a good trip to Bolivia and that we will have an opportunity to see you when you return. We hope to be moved into the new house by then so you should plan to stay with us and take a look at the mine first hand.

Keep your head down.

Sincerely,

Wayne

W. H. Burt
Asst. General Manager

WHB/dck
Encls.

Automation of batching and mixing, plus close control of the operation, have brought about more efficient, economic . . .

The automated batch plant at Magma Copper Co.'s San Manuel mine was installed in 1964. The plant can also be operated manually.

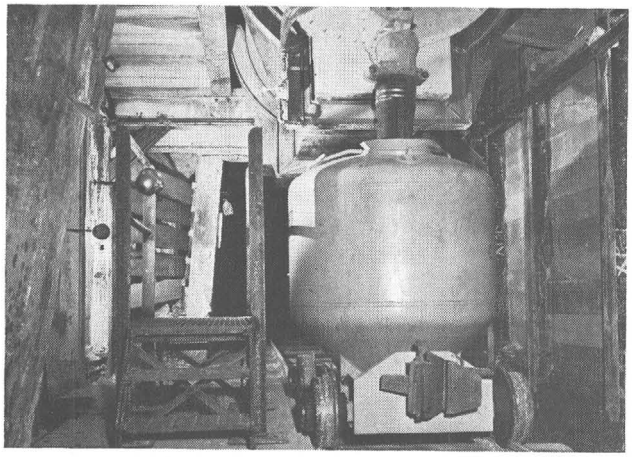
Concreting at the San Manuel Mine

H. W. SEANEY and R. L. TOBIE

SOCIETY OF

[redacted]

[redacted]



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[redacted]

[redacted]

[redacted]

Draw Control At San Manuel

R. L. TOBIE and E. K. STALEY

The president of the Mined-Land Conservation Conference foresees good growth opportunities for both . . .

Coal and the Electrical Utilities in the West

Based on a talk given by
ARNOLD E. LAMM

SOCIETY OF

Copper Precipitation Methods At Weed Heights

HOWARD W. JACKY

SOCIETY OF

Part III

MINING BY BLOCK CAVING

Fundamental Considerations in Block Caving
Phillip Bucky ..... page 129

Concrete for Ground Support in the Kelley Mine
Martin Hannifan ..... page 147

Discussion
..... page 157

FUNDAMENTAL CONSIDERATIONS IN BLOCK CAVING

By Philip B. Bucky

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BACKGROUND

Mining by block caving is described as consisting of those procedures which utilize the earth's barodynamic or gravitational forces to stress portions thereof, when and as desired, to produce fragments of a size suitable for handling. In addition, it provides a means for the further fragmentation of large ore particles and a transportation or transfer system that economically transports the ore to the shaft bottom or mill.

At the 1898 meeting of the Lake Superior Mining Institute, Mr. E. F. Brown reported block caving as having started at the Pewabic Mine on the Menominee range in Michigan in 1895. This was described according to Figure 1. A block of ore which ranged up to 200 feet in width and of a height of 100 feet was prepared as follows:

1. Two parallel open stopes, 8 feet wide and 200-250 feet apart, were driven between the hanging and the foot walls to within 20 feet of the level above or the capping.

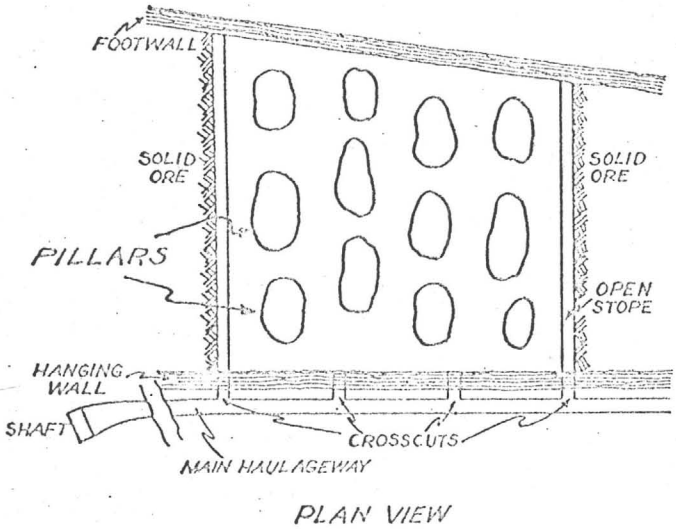
2. The bottom of the block was then undercut to a height of 7 feet leaving pillars for support.

3. The pillars were then drilled and blasted. This resulted in a block of ore that was free on the bottom and ends. The forces acting on the block were those due to its own weight, the capping above it, and the side thrust between the hanging and the footwalls. It took several weeks after the pillars were blasted from the bottom, for the block to fail and fill the undercut with caved material. The block, however, was left alone for a period of six to eight months, which coincided with the period that the lake was frozen and no ore could be transported. During this period, it was reasoned, the forces acting on the material crushed it. When drawing commenced, 80 percent of the material passed through a three-inch ring.

After it was believed the ore was sufficiently crushed, timbered crosscuts *M* (Figure 2) were driven by spiling from the haulage drift in the footwall to the hanging. From these, additional spiled drifts *D* were driven on 25-foot centers to the ends of the caved ore. Plats were placed at the ends of these drifts and the ore was allowed to run in. The ore was then shoveled into cars until waste or capping appeared. When waste appeared, a set or two of timbers were blasted and drawing resumed.

Figure 3 illustrates the first changes that took place after the introduction of the Pewabic method. It may be noted that:

(1) The development work on the undercut level consists of a systematic series of drifts and crosscuts connected with finger raises. The pillars may be shot on the retreat as in room and pillar mining methods or en masse under a complete block.



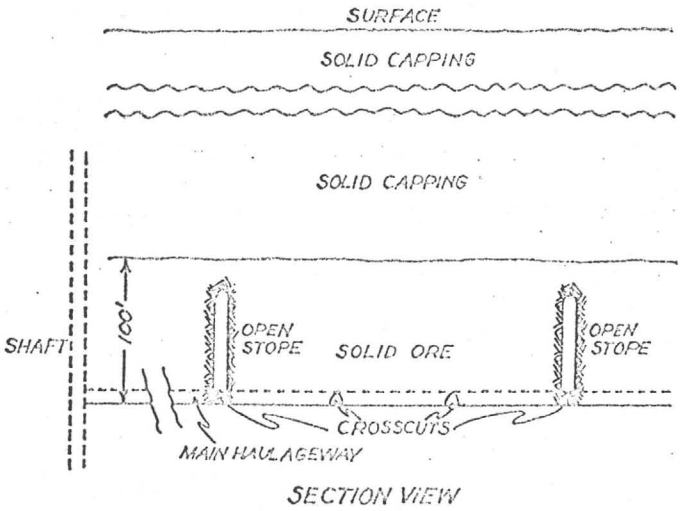
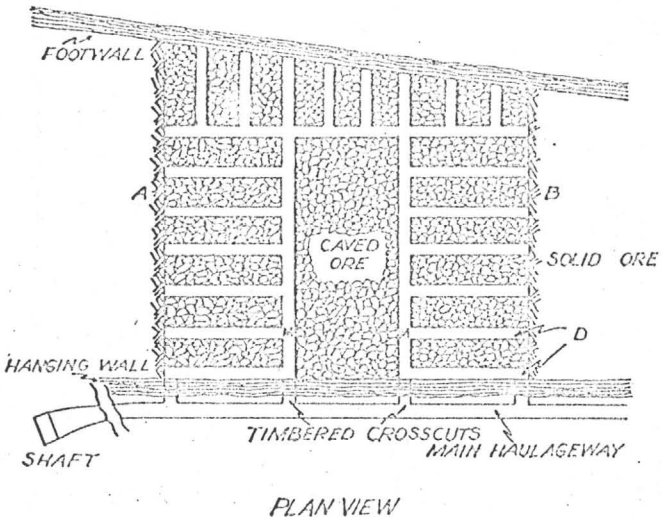


FIGURE 1. Pewabic block prepared for caving.



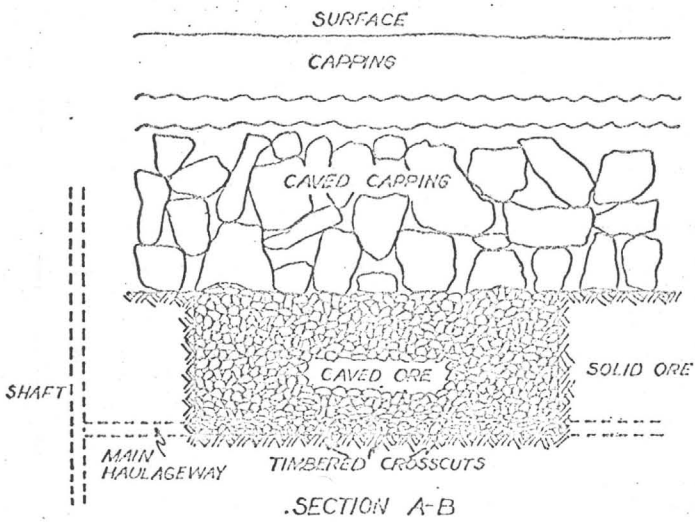


FIGURE 2. Caved Pewabic block.

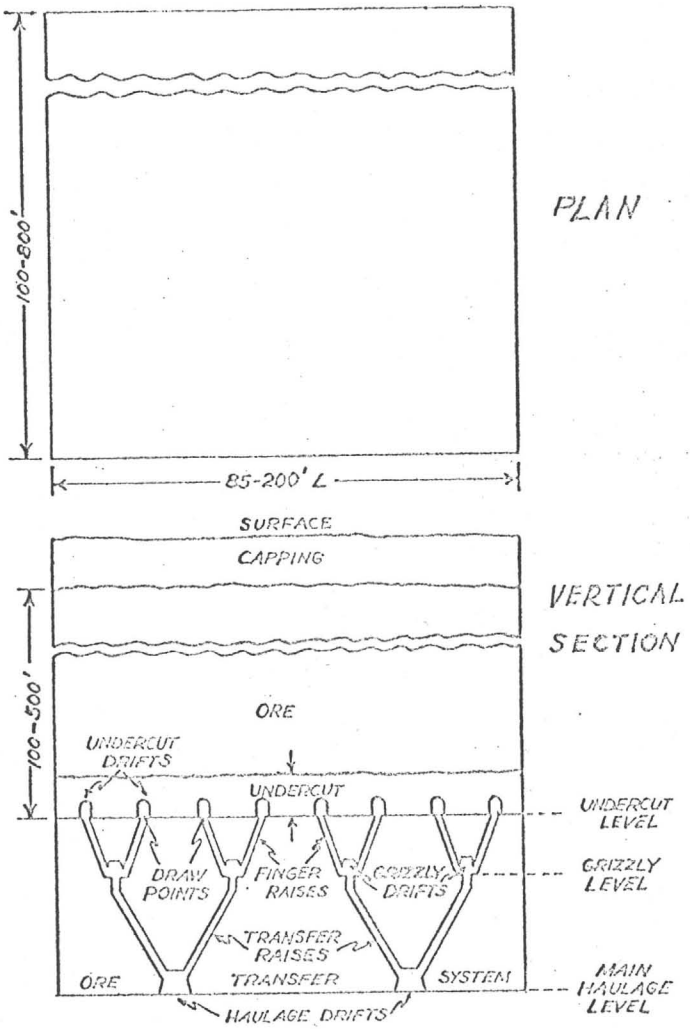


FIGURE 3. Generalized present day block-caving stope.

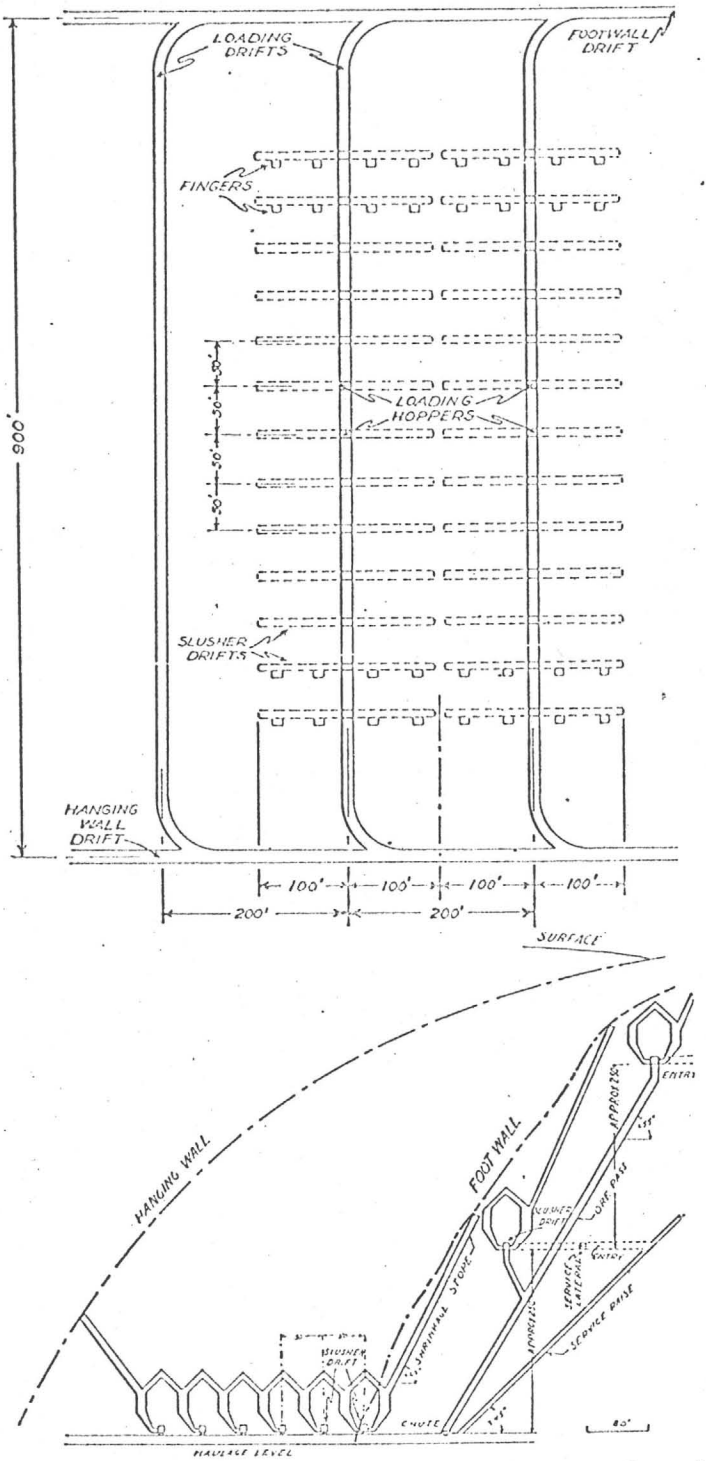


FIGURE 4. Present mining method (top). Present slusher system (bottom).

(2) The ore transfer system consists of draw points, finger raises, grizzlies and transfer raises, which take advantage of gravity for ore transportation.
(3) Boulder blasting is accomplished on a grizzly with particle size control so as to cause as little chute or transfer raise clogging as possible.
(4) Block widths range from 85 to 200 feet +, block heights from 100 to 500 feet +, and block lengths to 800 feet or more.

The above developments eliminated most shoveling, provided a convenient grizzly for handling large boulders and a convenient path for the travel of ore to the loading point where a gate controlled car loading.

There is evidently a realization that the block height or length does not affect its caving quality, and that the block width may be varied considerably.

Important advances (Figure 4) developed at Climax, deleted the transfer raises, introduced the idea of blasting boulders on a solid (slusher) drift bottom, and used the drift for the operation of a slusher which is able to transport and load large particles into cars on a lowered haulageway. They, in addition, sponsored the idea of concrete supported drifts and fingers, and increased the cross sectional dimensions of both.

Block cave mining presents two diametrically opposed problems:

I. That of designing a procedure whereby the earth forces shall fragment or cause an orebody to fail, and

II. That of designing a series of openings in the same orebody which shall convey the ore and allow men and *large equipment* to work efficiently, comfortably and safely.

FRAGMENTATION

It is important to realize that geologic materials, when confined, have the property of exerting considerable force on confining objects, and developing tensile or compressive stresses within themselves when unconfined and subjected to compressive or tensile forces. This is illustrated in Figure 5. If μ, Poisson's ratio in this case, is equal to $1/3$ and the applied compressive load is 2000 psi, the tensile stress developed is 667 psi if the member is unconfined laterally. If the ore strength is less than 667 psi in tension, the pillar will spall or fail along the ribs as shown. It is a view common in pillars in many flat-bedded, weak-in-tension deposits.

If one now assumes an undercut block (Figure 5, lower left) in the field with no end stopes, i.e. confined on all sides, then two sets of forces at right angles to each other act horizontally on the block. The block's lower portion will have a tensile stress equal to 2 x 667 psi, or 1334 psi. If the ore strength is less than 1334 psi in tension, the bottom will fail and keep on failing until the opening gets full when it will stop until some of the ore is drawn and space is available for more to fail to. In this manner,

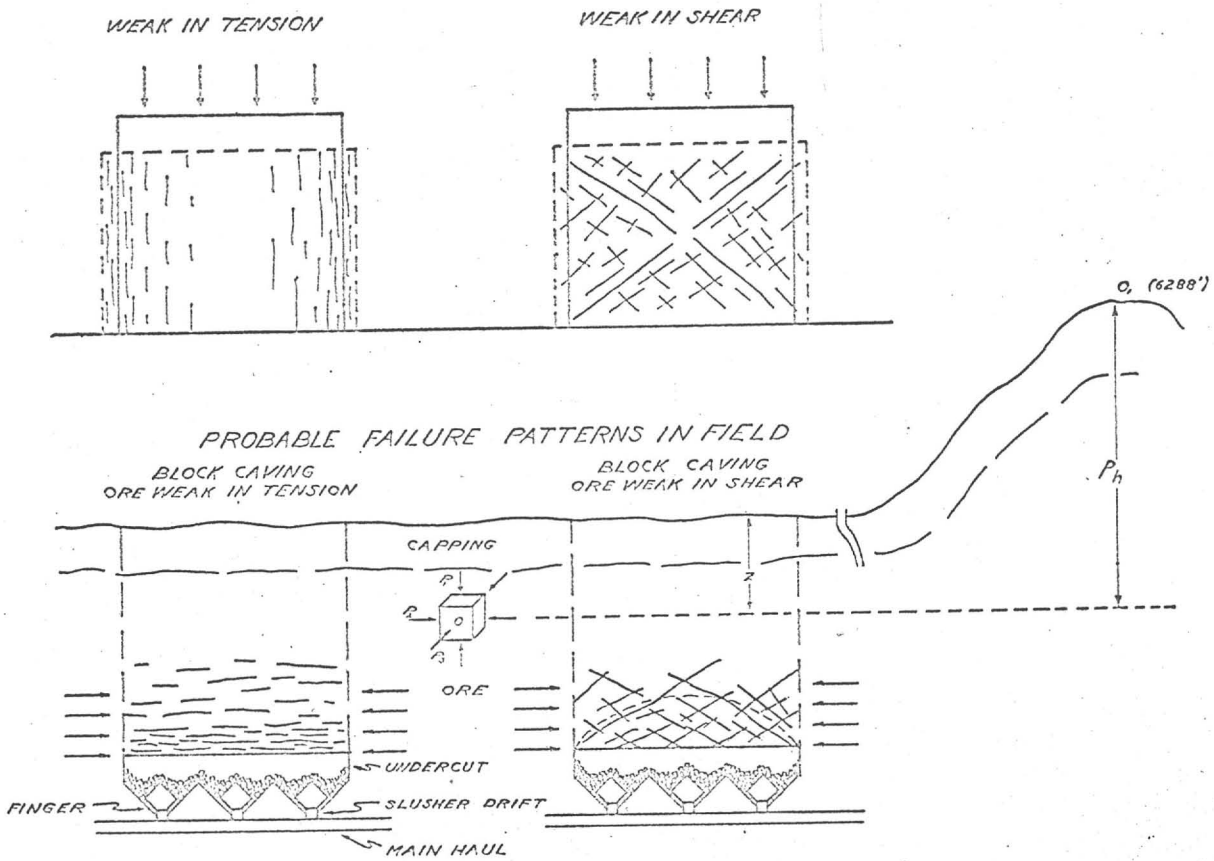


FIGURE 5. Testing machine failure of geologic material.

all the ore and capping in a block may be caved and the stope kept full or drawn as desired.

If the orebody is weakest in shear, then an open stope had best be made at one of the block ends. The maximum shear force available would be equal to one half the unit lateral pressures acting in the block and, in this case, would equal $\frac{2000}{2} = 1000$ psi. If the ore strength was less than 1000 psi in shear on a 45° angle planes to the load direction, then a series of arch failures would result. In a room and pillar mine shear failure is indicated when failure planes are inclined to the direction of the lines of forces.

The presence of weak planes in an orebody is an indication of the kind of failure to be expected.

The value of the shear stress induced by lateral earth forces is obtained from the following formulae

$$S_S = \frac{P_2 - P_3}{2} \sin 2\phi$$

Where S_S = shear stress on plane angle ϕ

P_2 and P_3 = unit lateral forces at right angles to each other.

ϕ = angle which the normal-to-the-shear plane makes with directions of the applied lateral force.

In a uniform material block free at one end

$$P_3 = O \text{ and } S_S = \text{a max. when } \phi = 45°$$

It may be reasoned, therefore, that the ability of a block to cave or fragment is a function of its strength in tension or shear and the value of the forces that act thereon which are mainly compressive.

We now reach the point where we try to determine the forces acting on a block and the initial state of stress at a point in a geologic structure.

The procedure for estimating these values is based on the following conditions and assumptions:

(1) The geologic structure and the maximum elevation inducing lateral forces are known (Figure 5).

(2) The geologic material is elastic and isotropic and all physical characteristics are known.

(3) A state of equilibrium exists, i.e. no lateral or vertical movement is taking place at the area in question.

Under the above conditions, Figure 5, the vertical pressure acting on point O is P_1, i.e. the weight of the material to the surface, the lateral pressures are P_2 and P_3 which must equal each other since the material is isotropic and the body is in equilibrium. If the weight of the geologic material is 1 psi per foot, and the Poisson's ratio $\mu = 1/3$, then P_1 in Figure 5 is 500 psi since point O is 500 feet under P_1. The lateral pressures P_2 and P_3, however, are not in general determined by the depth below the

surface at point O, but generally by the elevation of some other point as O_1, whose height above O is P_h. In this case, the lateral pressures may be shown to be equal to

$$\frac{P_h \mu}{1 - \mu} = \frac{P_h}{2} \text{ if } \mu = 1/3 \text{ then}$$

$P_2 = P_3 = \frac{6288}{2} = 3144$ psi. Point O is in equilibrium with vertical forces of 500 psi and lateral forces of 3144 psi.

The theory as here presented was applied to determine whether the initial state of stress in the Jeffrey orebody of the Johns Manville Company at Asbestos, Quebec, was sufficient to block cave it. Cores of the orebody were obtained and tested. The geologic section to the highest point in the region, Mt. Washington, Maine, (6,288 feet) elevation was studied. The orebody was strong, showing over 800 psi shear. The available forces were sufficient to accomplish the objective and the orebody is now being successfully mined by block caving. The salient points are summarized in the following table.

Jeffrey Orebody
Stress-Estimates for Block Caving
Source for Lateral Pressures.

Mt. Washington, Maine, Elevation 6288 ft.
Distance Asbestos, Quebec to Mt. Washington, Maine = 100 +. Miles

Location	Elev.	Shear Stress Available Psi	Shear Required Psi	Comment
Top of Pit	715	836	836	?
Pit Bottom	500	868.2	836	Probably cave
Haulage Level	0	943.2	836	Will cave

The report stated, "This orebody will probably cave at all elevations of 700 feet or less. Drifts, and haulageways at block bottom will require additional support."

It is reasoned that the same mountain provides the stresses present in the granite quarries in Massachusetts. Here, measurements of freed quarry blocks show length increases, which indicates that the quarry rock is under a stress equivalent in amount to that induced by the height of Mt. Washington. Plans have been made for measuring rock stress and the various moduli in place when the snow clears — probably this summer.

The previously presented material provides evidence

(1) That lateral earth pressures are transmitted by geologic material for great distances and in all directions, under the proper conditions of confinement,

The pertinent data in the block caving procedure at the Jeffrey Mine is as follows:

Elevation Mt. Washington, Me.	6288 ft.
Elevation top of pit, Asbestos	600 - 715 ft.
Elevation pit bottom	400 ft
Elevation haulage level	0
Block dimensions	200 ft. x 200 ft. x 400+ ft. high
Slusher dimensions	9 ft. high, 8 ft. wide
Fingers staggered on 35 ft. ctrs.	44 per block
Slusher drifts 50 ft. ctrs.	8 per block

Slusher drift has over it 18 inches of concrete and 14½ ft. ore.
Slusher drift has under it 18 inches concrete and the orebody.

Original Mining, 1st Block

Blasted all undercut pillars at once (8 ft.) pull immediately—2,000 to 2,700 tons per day. Drew uniformly and checked carefully, the block caved to the surface in 3 months.

The concrete slusher drift top fails in shear, but slowly.

The bottom also fails and cracks, but wears and takes load.

Present production—6 blocks—15,500+ tons per day.

(2) That mining at depth requires that careful attention be given to the general structural geology of the region.

(3) That a high elevation in any region is a basis for estimating lateral pressures at depth.

(4) That mines in rugged country may experience a rapid increase in lateral underground pressures as they proceed in depth.

(5) That an orebody amenable to block caving methods is one containing planes of relative weakness not too far apart, and so situated that lateral barodynamic pressures are available in sufficient intensity to cause failure along these planes.

DIMENSIONAL FEATURES

BLOCK SPAN

This refers to the minimum horizontal block dimension. This span (L) must be less than (L_S) that required for vertical shear failure along the block edges. It may theoretically be as small as desired (Figure 3).

For example, if the vertical shear strength is 1000 psi and the ore weighs 1 psi per foot, then the block span L must be less than 2000 feet.

$$L_S = \frac{2S_S}{W} = \frac{2000}{1} = 2000$$

Where L_S = span for shear failure in feet.
S_S = shear strength of ore in psi.
W = weight per square inch of material 1 foot long.

To insure vertical movement of fragmented ore a minimum L is yet to be established. It may at present be assumed as 40 times the size of the largest particle expected, as providing for free particle movement.

For example, if the cores indicate that many 5 foot particles may be expected, then span L should not be less than 40 times 5, or 200 feet. This meets the requirement that the block shall not fail in shear along the vertical edges.

BLOCK HEIGHT

There is no theoretical limitation on the block height provided the block is in a zone where the lateral pressures P_2 and/or P_3 are sufficient to induce fragmentation.

HAULAGEWAYS AND SLUSHER DRIFTS

These openings may be expected to fail as the blocks do. Available evidence bears this out. With present mining depths and lateral pressures, concrete has been successfully used to increase opening strengths. It is felt, however, that studies and tests will provide support designs that will allow larger spans to be used than at present. There is a considerable challenge here, for an increased cross section will allow the use of large capacity mechanical loading and haulage equipment, and a minimum amount of secondary breaking.

RAISES AND FINGERS

The minimum cross sectional dimensions should be four or five times the maximum size particles they are expected to pass. They should, preferably, be as large as possible. The tendency to fail is here present as in haulageways and slusher drifts and similar suggestions are intended. The minimum cross sectional dimension of these fingers is an important control point on production.

HEIGHT OF UNDERCUT

This dimension should exceed one half the distance between finger edges to insure a draw which will not leave material supporting the overlying block.

THE SUPPORT PROBLEM

Since a block of ore when being mined is stressed beyond the elastic limits and into the rupture range, openings and pillars under or in such a block, i.e. haulageways, slusher drifts and raises, must also be so stressed. To keep them from failing, 6 to 18 inches of concrete is added to their surfaces to provide the additional strength required. Ingenious aluminum forms and methods of mixing, transporting and placing concrete have been developed. The emphases from now on will probably be on stronger concrete mixtures, modified drift shapes, and possibly the pre-stressing of concrete supports.

One may reasonably conclude that at the present time the depths at which block caving operation approaches a limit is the elevation where the

lateral pressures developed due to a maximum regional elevation, approach the crushing strength of the available concrete.

At present, one may raise questions as to the shape and thickness of the concrete support shell required, for slusher and main haulage drifts. Shall they be rectangular, arched, circular, or elliptical, and, if so, in which direction shall the long axis be placed? Vertical or horizontal?

Progress of outstanding significance in the support of workings stressed beyond the elastic limits has been accomplished. Can this work be carried on further? Can we work with support stressed beyond the elastic limits and a safety factor based on time? The presently available field evidence answers in the affirmative.

LAWS GOVERNING FLOW — LATERAL PRESSURES

In an endeavor to find out what takes place in a caved stope, a series of experiments was performed. A partial list is tentatively presented. They are not to be considered as complete or final.

Figure 6A shows the angles of repose assumed by unconfined sand. The sand is free to move on all sides.

Figure 6B shows the angle of repose as having been increased due to its being confined on two sides.

These experiments indicate that a lateral pressure is exerted by the sand when confined.

Figure 6C consists of a box of sand, 1 foot square and 6 feet high. The lateral pressures exerted on the box side were measured. At the top, the lateral pressure was 0. A short distance below (1½ inches) it increased to a maximum and was constant from there to the bottom. A hole was then made in the bottom of the box and the rate of flow of sand against head was carefully measured. The interesting point here found was that the rate of flow (weight flowing per unit time) was a constant irrespective of the head.

The material used in all the experiments was 20 mesh dry sand.

It is interesting to compare the behavior of sand with that of water, i.e. fragments with liquid. The unit lateral pressures of liquids increase with head. The unit lateral pressures of the sand fragments tested remain constant. The rate of flow of liquids through an orifice decreases with decreased head. The rate of flow of dry fragments through an orifice is independent of head and is constant.

For solids the following liquid relationship is indicated by the results at Asbestos:

Confined solid geologic materials transmit lateral pressure, induced in them by a vertical load, in all directions (as water does). The value of this lateral pressure being $P_1\mu/1-\mu$ where P_1=Vertical unit load and μ= Poisson's ratio for the material. A material whose Poisson's ratio μ is ½

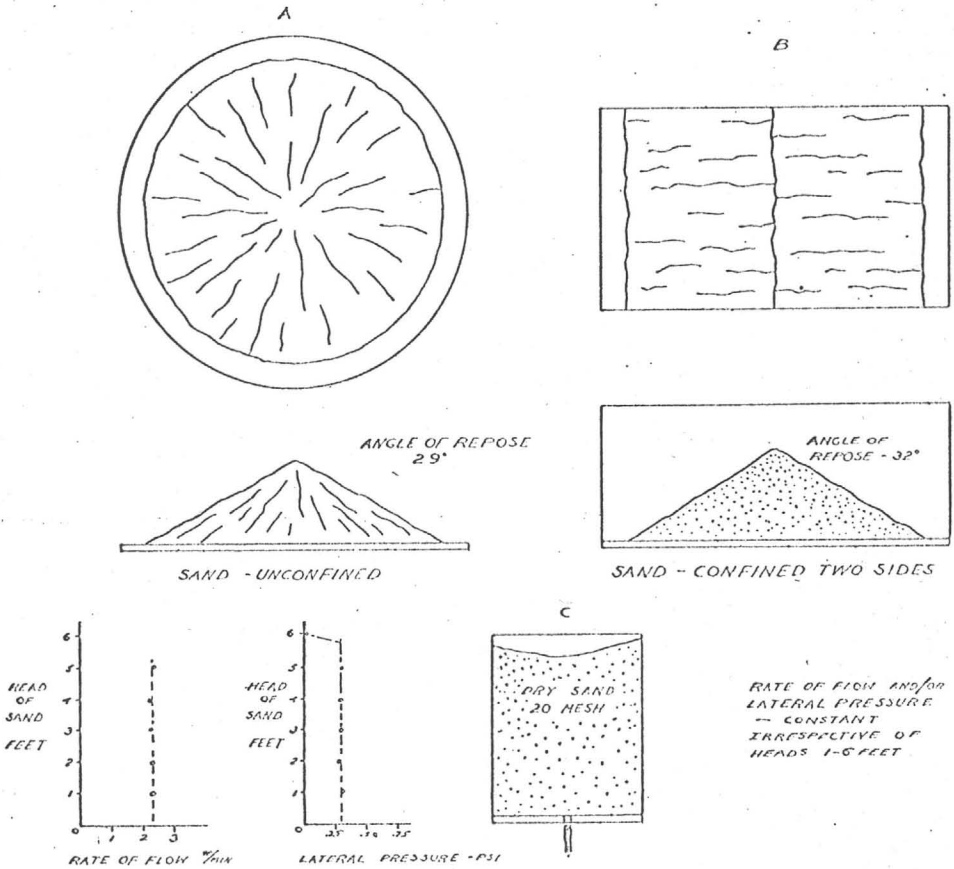
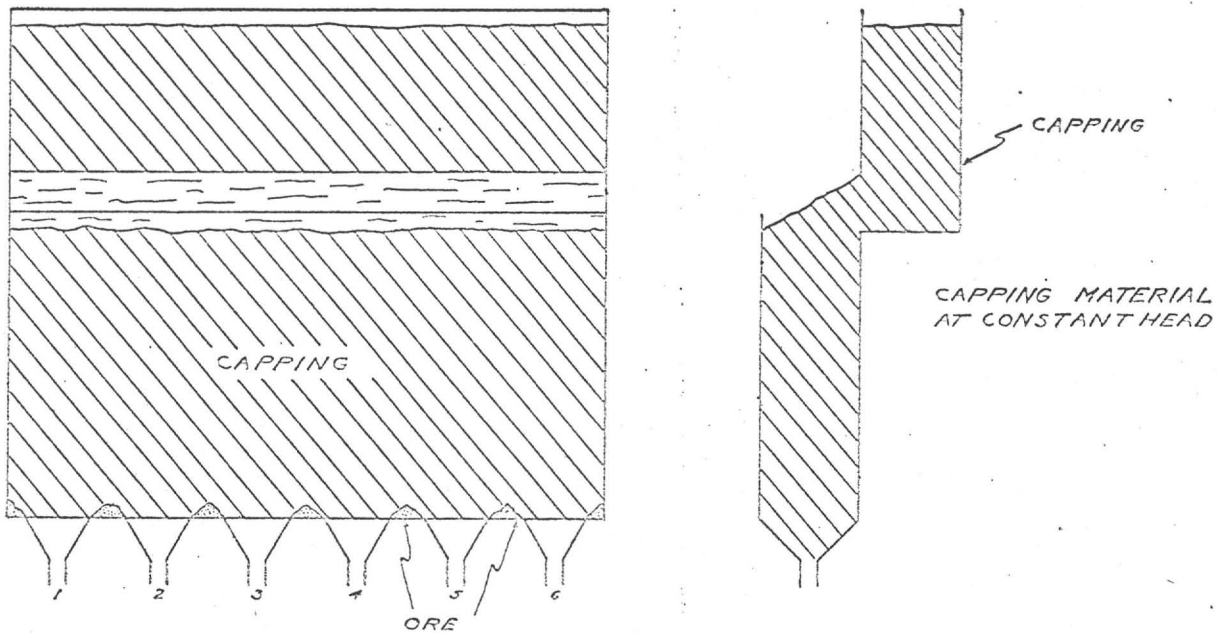


FIGURE 6. Fragmentaulic experiments.

would transmit laterally 100 percent of the applied unit load, the same as water. Confined liquids transmit lateral pressures induced in them by vertical loads (P_1) in all directions. The value of this lateral pressure is P_1.

The laws governing the behavior of fragmented geologic materials (fragmentaulics) are probably better known to mining men than to most others, for they continually deal with them, and yet, it is a virgin field and in its infancy. Study and research along these lines will provide interesting information concerning the supporting power of fragments, the fragment stress distribution around support shells, and the laws governing the flow and mixing action in stopes and bins.



FINAL CONDITION — ALL DRAWING SEQUENCES

[NEGLIGIBLE PERCENTAGE VARIATION IN QUALITY AND RECOVERY OF ORE]

1 - UNIFORM DRAW ALL AT ONE TIME
2 - CONSECUTIVE DRAW - EQUAL AMOUNTS
3 - DRAW TO COMPLETION (50% CAPPING) EACH AT A TIME
4 - DRAW TO COMPLETION AT RANDOM - ANY SEQUENCE

FIGURE 7. Drawing. Ore followed by capping material.

ORE DRAWING

The first ore drawing period is the one concerned with the barodynamic fragmentation of the ore, and the creation of a free surface, and a void space between the uncaved portion of the block and the fragmented or caved material. When the ore separates from the block face, it increases in volume and fills the void, thus causing further caving to cease until some caved ore is withdrawn. For this condition, it appears that a uniform draw from a block is desirable and should be scrupulously maintained until evidence is available that the capping at the surface is moving downward.

The second ore drawing period starts when the ore and capping are all caved and both move toward the draw points. Being interested in determining what could be accomplished with the drawing rate, a model was built as in Figure 7. Asbestos ore and tails were used with an arrangement for feeding tails to the stope as the ore was drawn. In each case, ore was taken from the draw points until the tails appeared to be 50 percent of the amount drawn. Several ore-drawing sequences were used and in each case the amount of good ore left in the stope determined. The interesting result was that within experimental error limits of 4 percent, the amount of ore left in the stope was the same, and that the rate, sequence, or amount of ore drawn from any draw point had practically no effect on the drawing efficiency of this stope. In other words, if the conditions of the experiment were met in the field, then one could draw as one pleased from any draw point, for any amount and for any length of time (i.e. until tails showed 50 percent) without affecting the stope efficiency.

There appears to be tremendous opportunities for increasing the efficiency of block caving operations. High on the list one might consider research to determine the orebody state of stress and the physical properties and behavior of the ore in question, especially when stressed beyond the elastic limits. A relationship considering deformation as a function of stress beyond the elastic limit, confinement, and time should be very illuminating.

Design efforts based on the above information to permit drawpoints and slusher drifts of considerably larger cross section, might well more than double the present production rate per man shift, by permitting the introduction of mechanical loading equipment, or increasing the slusher sizes now in use. The near future holds promises of developments in this field which shall pleasantly affect all interested in the earth's crust and its behavior.

CONCRETE FOR GROUND SUPPORT IN THE KELLEY MINE

By Martin Hannifan

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The Anaconda Company's Kelley Mine in Butte, Montana, is an application of the block caving method to ore zones previously mined of their higher grade ore by square set and fill methods.

The general character of the ore zones differ markedly from the blanket type orebody generally associated with the caving system.

First: They are vein type ore bodies — long, narrow, and deep, measuring 100 to 400 feet in width, up to 1500 feet in length and from 200 to several thousand feet in depth. (See Figure 1.)

Second: About 12 percent of the total ore estimate is made up of backfilled stopes of various ages and sizes, containing timber and rock fill along with recently introduced wet tailing. Many mining blocks consist of over 50 percent this type of material.

Third: Sealed off fires existed in part of the ore zones to be mined. Also, a potential fire hazard exists where remining would admit fresh air to any of the old stope areas. It was necessary therefore to precede block cave mining by an extensive program of fire elimination and prevention. Mill tailings (about 50 percent solids) is delivered by pipeline from surface plants through concrete bulkheads underground where the material is forced into the voids by hydrostatic pressure. This work and the draining of excess water progresses upwards working from several shafts and over 7 million tons will be placed before completion of the project. This type of filling has eliminated the dormant fires and most of the fire hazard in the areas to be mined by block caving. It has also effected considerable improvement in working conditions by removing heat units, decreasing airborne dust by increasing the moisture content of the ore to 6 percent, and by stabilizing loose fill and voids through which new openings are driven to develop cave blocks.

Fourth: The ore zones are capped by waste which varies in depth from 50 to 500 feet. In this we have one of the few similarities with other caving method ore bodies.

BACKGROUND

Since the beginning of production in May 1952, the Kelley Mine has produced 12,000,000 tons of copper ore while successfully overcoming many problems related to the unusual character of the ore material, and the generally disturbed ground in which blocks must be developed and maintained.

Kelley Mine ore may be classified as free caving, but presents many ore handling problems because of its damp, sticky character and the presence of much old timber. Ground strength within the confines of the ore zones is low, and all of these openings require support. There is no problem to

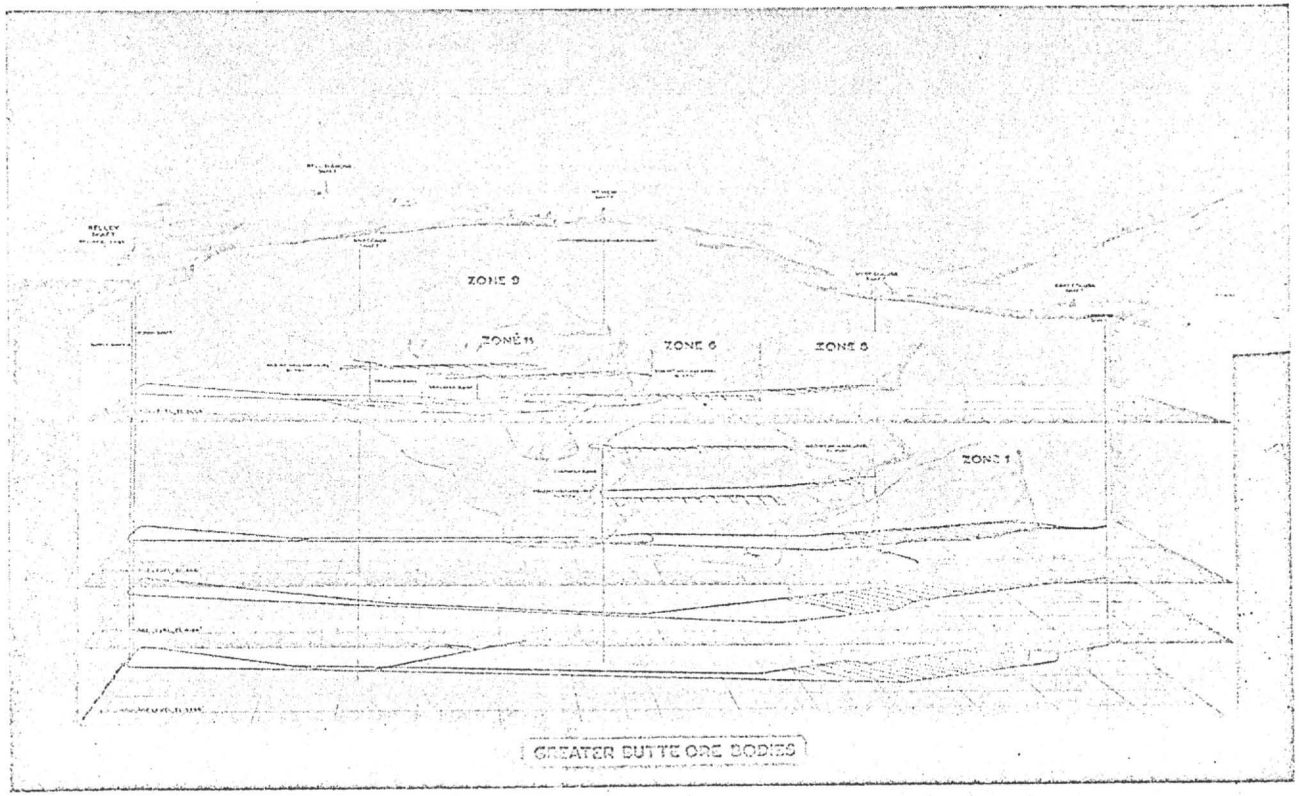

FIGURE 1. Greater Butte ore bodies.

start caving action in the average mining block. The fracturing and some of the expansion in rock volume from solid to broken state has already taken place in the ore zones due to the settlement and compression of rock-filled stopes. As some of the older stope mine operators have told me — "our problem was always to keep it from caving and some of the plums you will get are stopes that got away from us or were left because of local fires."

Boundary weakening of blocks is a minor development job in general, as the character of the ore zones is such that arching and slow caving is improbable. Some weakening is done in blocks under dipping veins, where part of the firm footwall must be caved to assure the drawing of ore vertically above the boundary drawpoints. Close spacing (225 sq. ft. drawpoints) and careful draw distribution minimizes the dilution problem resulting from the channeling or piping characteristic of ore drawing in this type of ore body.

Block design and the support of mine openings have been modified since the start of operations as experience indicated their need.

The adoption of concrete support for the horizontal extraction openings in mining blocks has proved to be the most effective of many changes made in the original mining plans.

Excessive maintenance of timbered grizzly levels was experienced in the first production blocks. Impaired use of these openings also had a serious effect on working conditions, ore drawing distribution, and the general efficiency of the production crews. Several changes in the design and type of grizzly level support were made, as well as a change in the sequence of mining blocks to combat this problem. Early in 1953, concrete support was tried in a particularly troublesome area. Results were so manifest that this method of support was adopted throughout the mine as rapidly as possible.

We had considerable experience in Butte with the use of concrete underground in lining the Kelley shaft and installing innumerable doors, dams, tubes and bulkheads for fire, water, and filling control. We had already in service a ready mix plant of about 200-cubic-yard capacity per day, and several transit mix trucks and two placers. Needless to say, these things made the application of concrete support to cave block openings considerably less of a problem.

To date, about 15,000 feet of grizzly levels (4½′ x 7′) have been concreted, resulting in a remarkable decrease in maintenance costs and vastly improved working conditions.

The original design of a Kelley Mine block is shown in Figure 2. This was a grizzly-control gravity layout supported by timber throughout.

Encouraged by the stability of concrete support on grizzly levels and to combat the problems resulting from damp, sticky ore and the disposal of old stope timber, a change has been made in block design. (See Figure 3.)

GRIZZLY-CONTROL SLUSHER BLOCK:

In this type block, both the slusher level and the grizzly level are supported by concrete. To date, 4,500 feet of slusher levels (6′ x 8′) have been lined with concrete and no repairs or production delay has been necessary because of concrete failure. Some of the other advantages experienced in these blocks are:

(a) Closer spacing of draw raises to minimize the dilution problem resulting from channeling was desirable, and the drawpoint area has now been reduced from 300 sq. ft. to 225 sq. ft. Very minor repairs have been required in these blocks despite the fact that the additional openings required to accomplish this closer spacing

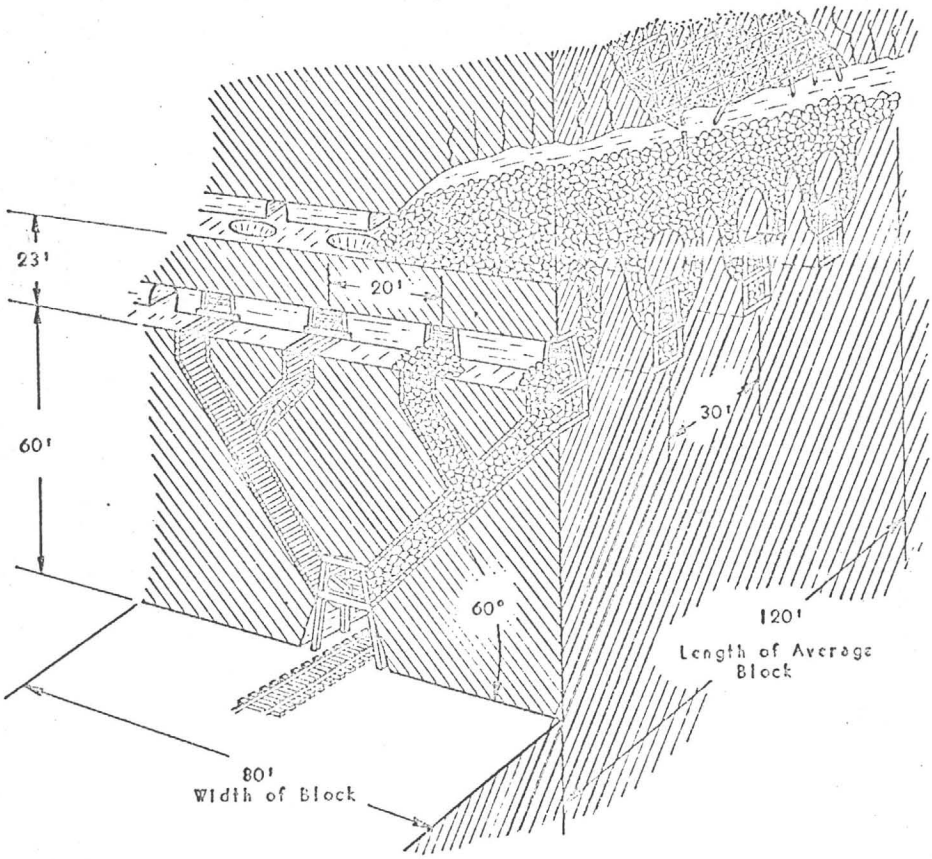

FIGURE 2. Original design of the Kelley Mine block.

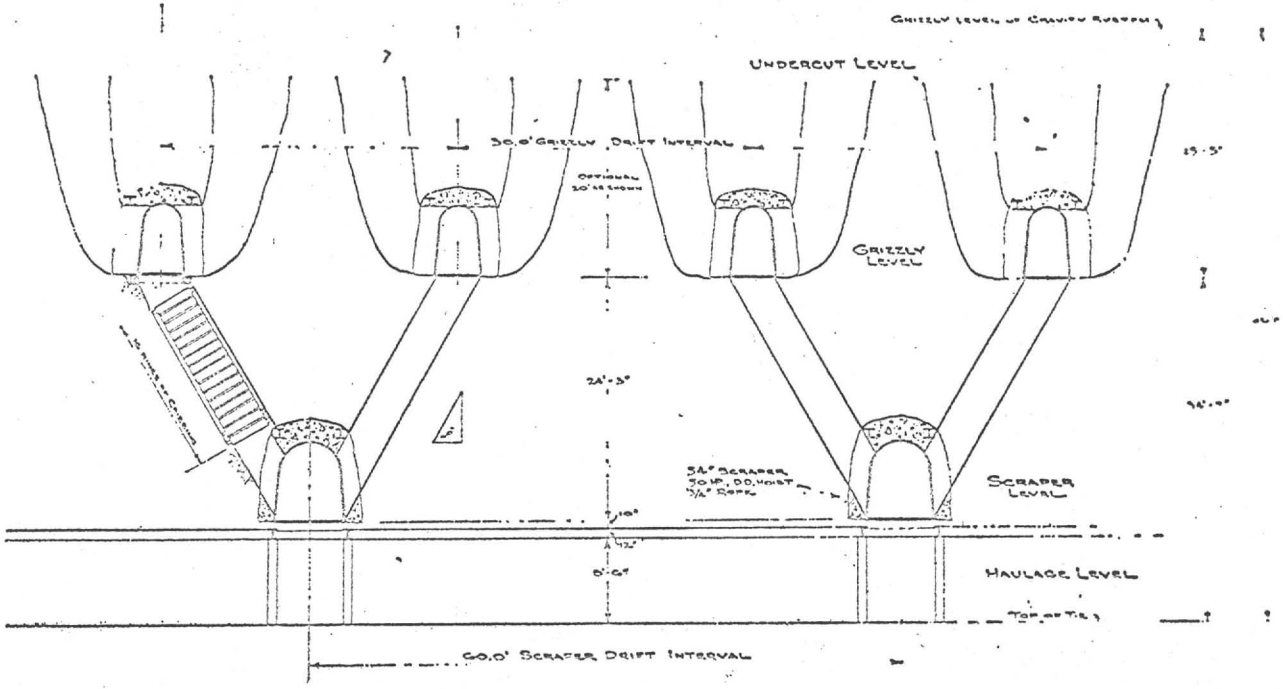

FIGURE 3a. Drawing of new type block design.

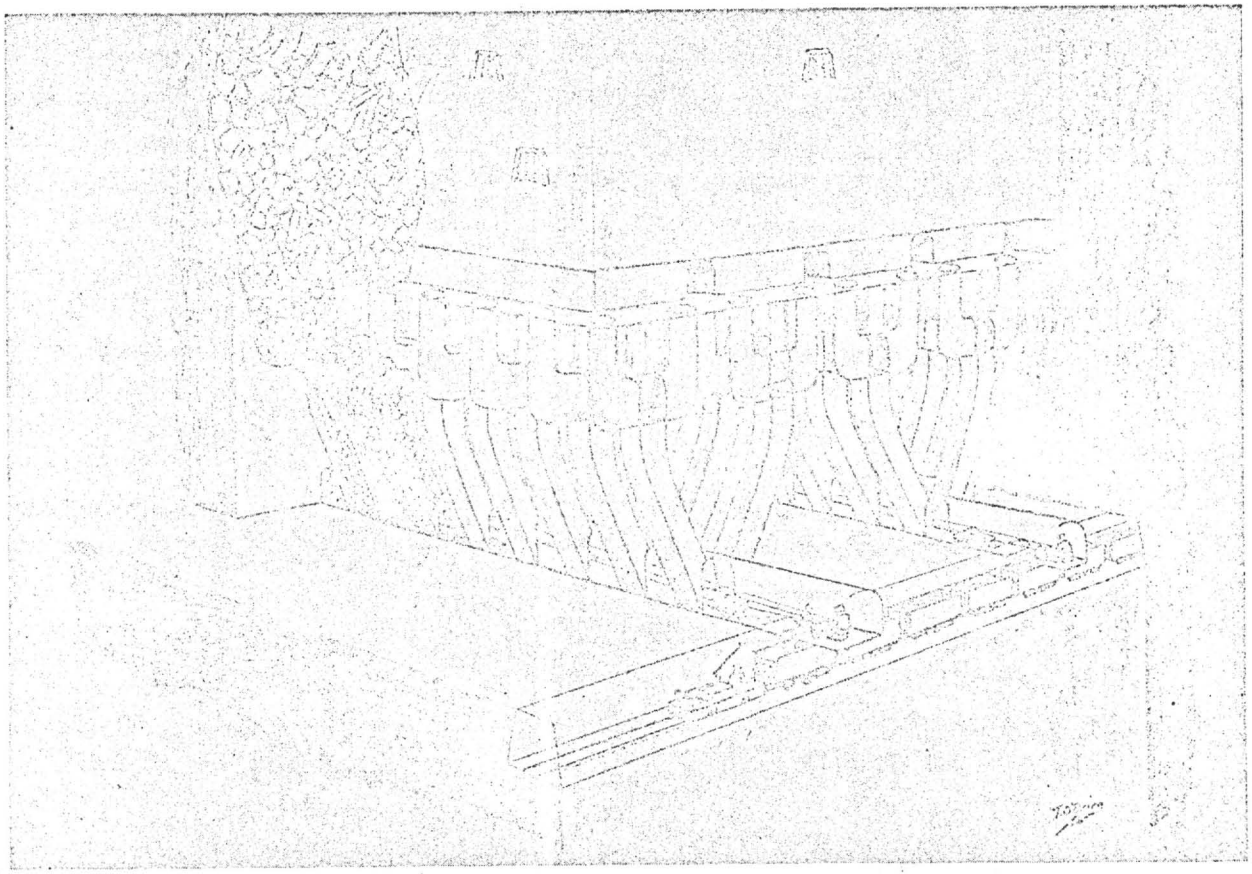

FIGURE 3b. Artist's conception of new block design.

necessarily reduced the size of the ground pillar supports within the block.

(b) Ore drawing and loading efficiency have been greatly improved.

(c) Faster mining is being accomplished with good draw distribution so that fewer units are necessary to meet the production requirements.

(d) Working conditions are safer. Ventilation is better, and the general morale of production crews is improved.

SUMMARY

Following is a summary of some facts of interest resulting from four years' experience in the Kelley Mine during which concrete support replaced timber support in the horizontal openings of mining blocks:

ADVANTAGES

1. Cost of block repairs after caving (labor and supplies) has been reduced 76%.
2. Cost of ore drawing (chute tapping costs per cave ton) has been reduced 33%, because the efficiency of this job (tons per chute tapper shift) improved 68%.
3. Cost of ore tramming (motormen and car loading costs per cave ton) has been reduced 17%, because the efficiency of this job (tons per ore tramming shift) improved 30%.
4. Unimpaired working levels provide cleaner, safer, better ventilated working conditions for production crews.
5. Stability of concreted openings made feasible the adoption of a grizzly-control slusher block designed to combat other problems experienced in the grizzly-control gravity type block.
   (a) The drawpoint area was reduced from 300 sq. ft. to 225 sq. ft. (Closer spacing to minimize dilution as a result of the channeling and piping characteristic of ore drawing in this type of ore.)
   (b) This block design solves the problems of plugged chutes and the disposal of old timber experienced in the original blocks. It also permits handling the maximum rock size the hoisting facilities can accommodate, hence increases ore drawing and loading efficiency.
6. Stability of concreted extraction openings permits maximum daily production per block unit. Good ore drawing control can be maintained, together with rapid extraction, as drawpoints are not stopped for repairs.

7. Concreted openings are fireproof. This feature has special importance considering the presence of old timbered stopes in Kelley ore zones.
8. Concreted openings do not deteriorate with age or ventilation conditions; thus the openings used for extraction remain open for ventilation and other useful purposes.
9. In the deep ore zones, where more than one horizon will be mined, the debris from caved concreted openings will be less troublesome than timber or steel.
10. Handling of timber has been reduced along with its attendant costs and safety hazards.
11. Price of cement has increased less over the last several years than any other construction material used underground.

	1940	1956	% Increase
Timber (ACM pine)	$28.82 m.b.f.	$ 57.99	101.2%
Steel (8" H beam)	61.00 per ton	135.20	121.6%
Cement (Bulk)	2.67 per bbl.	4.76	78.3%

The other ingredients of concrete — rock, sand and water — are locally available, and the cost, in place, is primarily dependent on the efficiency of handling methods; hence, within the control of the mine operator to a great extent.

DISADVANTAGES

1. Comparing the grizzly-control, gravity type, timber-supported block to the grizzly-control, slusher type, concrete-supported block, on a square foot basis, the cost of block development (wages and supplies) has increased 62%.
2. A large proportion of openings must have temporary support in advance of concrete lining. This support may be rock bolts, rock bolts and wire fabric, or arched rail sets and lagging, depending on ground conditions. These openings, before concreting, now cost as much as did the original timber supported openings complete in 1952-1953, due to the increased cost of labor and supplies.
3. Ready mix concrete is difficult, if not impossible, to pass down exposed shaft pipes in below zero weather.
4. There are six separated mining zones on four levels in operation in the Kelley Mine. As a result of this scattered destination for concrete:
   (a) Investment in underground concrete handling equipment is high.
   (b) Application of specialized supervision and crews necessary for best efficiency is difficult.

(c) Unforeseen delays sometimes require the dumping of ready mix, as concrete should be placed within at least two hours of mixing.

5. Underground transportation from the shaft delivery pipe to the form is a problem, as distances are great.

(a) Motor and car tramming must not interfere with ore traffic.

(b) Concrete pumps are bulky, require considerable mechanical maintenance, and are too limited in pumping distances for many locations. Plugged lines cause costly delays.

Pneumatic placers are mechanically simple and will deliver ready mix through pipe over great distances including changes in direction and elevation. However, they present a material segregation and air-consumption problem proportionate to the distance "blown."

CONCLUSION

Use of concrete support has proved very successful in minimizing the serious problems resulting from the failure of timbered extraction openings in the mining blocks of the Kelley Mine.

The reduction in the cost of block repairs alone (76%) more than balances the additional development cost of concrete support for the grizzly level, plus the addition of a concreted slusher level, plus the additional openings required to reduce the drawpoint area 25%.

Credit for the increased efficiency and reduced cost of ore drawing and tramming is shared by the effectiveness of concrete support and the changes in block design to a grizzly-control slusher system.

DISCUSSION

PART III — MINING BY BLOCK CAVING

Round table discussion on block caving; participants:

PHILIP BUCKY, Columbia University, N.Y.
MARTIN HANNIFAN, Anaconda Co., Butte, Mont.
STANLEY WALKER, Climax Molybdenum Co., Colo.
HOWARD FLESHMAN, San Manuel Copper Corp., Ariz.
J. B. FLETCHER, Miami Copper Corp., Ariz.
CLAIRE NISPEL, Kennecott Copper Corp., Ruth, Nev.
M. S. DALE, Kennecott Copper Corp., Ruth, Nev.

MR. ODE[1]

I should like to ask Prof. Bucky to be more explicit about why certain rocks fail in tension and why certain rocks fail in shear. Do you have a mixture of material? Do you have parts of the material becoming plastic and yielding, while other parts are elastic?

PROF. BUCKY

If we have a piece of steel whose compressive and tensile strength are 80,000 psi and shear strength 40,000 psi, there is not much question but that the steel will fail in shear. Now, if we take a rock whose tensile strength is 100 psi and shear strength is 1000 psi and compressive strength is roughly 5000 psi — then, that rock will fail in tension.

MR. ODÉ

I do not see why exactly. How is that tension generated which pulls it apart?

PROF. BUCKY

Will you accept the theory that stress is proportional to deformation? Assume that you do not know anything about this block, but you do know what the stress was in this little square beforehand. Now you go ahead and measure this square in the same direction; you know nothing else about it. Its increase so far as you are concerned is due to 200 lbs. in tension.

MR. ODÉ

Do you assume that every volume element in your pillar is stressed uniformly or not?

PROF. BUCKY

Yes, you assume that here. Assume that I come in before the load is applied to this pillar and I install a strain gage at this point and get a certain reading. Now a load comes on this pillar, and the strain gage registers a deformation which is due to the applied stress of 200 psi tension.

[1]Structural Geologist, Shell Development Company.

MR. ODÉ

If you treat this block completely as a homogeneous elastic piece, then you can assume that the sum of all the stresses must be zero?

PROF. BUCKY

Yes, if the block decreases in height, it is in compression. If the block increases in height, it is in tension.

I have difficulty with some people who say, "Well, here is a material in compression and yet we say it is in tension." Now that is a property of geologic materials to which we have not paid much attention. One of the properties of geologic material is the ability to transmit pressures laterally. We know that water if confined transmits pressure laterally. Now the point we are making here is this, at the Jeffrey mine and at the quarry: that geologic materials when confined to prevent lateral movement will transmit pressure laterally. Rock will transmit a tremendous amount of pressure.

E. P. PFLEIDER[2]

When I was in Sweden this last summer, I found that a Mr. Nels Haft from the Royal School of Mines there had done quite a bit of work in testing strain and stress underground on pillars for the Boledum Company. They were having a great deal of trouble with their pillars failing, so they called in Mr. Haft. Using strain gages in bore holes placed in these pillars, he was able to determine the amount of stress and the direction of stress. From this information, they reoriented their pillars and cut them in size and completely alleviated their problems. Now it seems to me that in line with what Professor Bucky has just said that this type of knowledge from many mines throughout the country would be very useful. It would be a very small expense to start accumulating some of this data, and I think that if the mining companies would initiate it and accumulate it together, that we would be able to march forward much faster than we have. I would like to ask Professor Bucky if he has done any of that work in connection with the stresses around the quarry or underground, say at the Jeffrey mine, to check his observations.

PROF. BUCKY

It is interesting about this quarry. We are in a country that is pretty cold, and they are interested in going underground with quarries. They are down about four or five hundred feet on the surface. The problem of mining dimension block underground is quite a problem. At the quarry we have already done some work in mounting strain gages on the rocks. We are expecting to get the stresses in the quarry, in the face of the quarry, so that we can get a pretty good idea as to how much roof we must leave when we go underground. We expect to have some pretty fair sized spans. We have not made any measurements that would amount to much yet except that we feel very much encouraged. As a matter of fact, I do not

[2]Head, Department of Mining, University of Minnesota.

want to say anything about it because they are too good. It is a rather simple thing to do. A strain gage is mounted on a face of a rock in place, and then readings are taken. Then around the place where this strain gage is attached, a large core is drilled out which will include this strain gage on the surface of the core. After the core is recovered, readings are taken again. The difference, of course, is due to the fact that this piece where the strain gage is mounted has changed in dimensions. If it increased in a certain direction, we call it tension, and if it decreased in a certain direction, we call it compression. That is the method we are following.

PROF. PFLEIDER

Have you tried drilling any holes and putting strain gages in the bore holes, away from the surface to avoid surface effect?

PROF. BUCKY

We have thought about it but have not tried it yet. We still have some problems as to attachment and removal of the gage. We are considering letting some stay in there for a time, to see what happens to the strain gages with time.

DAVID J. VARNES[3]

Professor Bucky, I would like to know how far you want to transmit this lateral stress underground undiminished, that is, due to surcharge? How far would the lateral pressure which is generated here be transmitted out towards the plains? If you drill a hole here down to sea level, let us say, and then drilled one a hundred miles out in the plains, would the pressure be the same at sea level in each of those holes?

PROF. BUCKY

That is a very good question, and one to which we must find an answer. Right now, we seem to have evidence that it is being transmitted for at least 100 miles. We may know at the end of this summer if it is transmitted 700 miles. We are going to try to measure the stresses in the rock at the Jeffrey mine and determine if there is a connection with Mt. Washington some 700 miles away. Right now we are rather skeptical.

MR. VARNES

Have you conducted a theoretical analysis on a semi-infinite medium with a surcharge to see whether the lateral pressure generated by that surcharge at a constant depth decreases or remains constant as the radius or distance from the surcharge increases?

PROF. BUCKY

Theoretically, if you make the assumptions that I made — first, that the material is perfectly elastic; second, that no lateral movement takes place; and third, that no vertical movement takes place — if you make those assumptions, then the situation is exactly like water. There will not be any difference in the lateral pressure. Now, if there is any movement, we

[3]Geologist, USGS, Denver.

know there must be a difference in pressure in the direction in which the movement comes. I believe there is some movement taking place all over, but I believe it is at such a very slow rate, that the change in pressure is comparatively small.

J. H. EAST[4]

I would like to ask Prof. Bucky what happens if you have a fault when this pressure is transmitted for 700 miles. You are going under the assumption that there is not a fault, are you not?

PROF. BUCKY

If the two sides of the fault are in a vertical position, it makes no difference. In other words, if I press on five boards lined up together vertically and in contact, the force will still be transmitted and without loss.

MR. EAST

Suppose they are at a flat angle?

PROF. BUCKY

I am going to get in a jam with you Mr. East, because I am going to say this: if there is no lateral movement taking place and the body is in equilibrium, then the pressure is transmitted. If movement takes place, then there is a difference in pressure. There is bound to be a difference in pressure because of movement, but if no movement takes place and the pressures are applied, then the pressures are transmitted.

PROF. PFLEIDER

I would like to ask if you have a homogeneous rock throughout this entire area, and if so, what type it is.

PROF. BUCKY

I do not know. I am not particularly worried about its chemical composition, but I am concerned about its physical properties and what it will do under varying load conditions. However, I do not really know much about the rocks.

C. P. ERSKINE[5]

Professor Bucky, you say that the material is elastic. How do you get the horizontal forces exerted by a surcharge from Mt. Washington over a local area? I do not quite understand why you would have horizontal forces created by Mt. Washington which are dissipated through the lower lands beyond it.

PROF. BUCKY

That is a question of mechanics and assumptions. If I press on a block, an elastic material, that block wants to increase in dimension. If I press on the block and it is not allowed to increase in dimension, it is going to resist the force so there is a lateral pressure.

[4]Regional Director, U.S. Bureau of Mines, Denver.
[5]Geologist, USGS, Denver.

MR. ERSKINE

But if you press at one point on a block and the rest of the surface of the block is not under pressure and it cannot move laterally, I would expect it to be depressed at the point where you push on the block and then to heave up around it, rather than to have lateral pressures transmitted through the block.

PROF. BUCKY

Here we have a pressure on a block that acts horizontally and tends to push side-ways. It can not move side-ways, but it exerts pressure side-ways. If the material is elastic, then that pressure is induced side-ways.

MR. ERSKINE

Would not the block have to be confined from the top also?

PROF. BUCKY

No, that is where the pressure is applied.

MR. ERSKINE

But the pressure is not being applied over the entire area, however, it is being applied on small areas such as at Mt. Washington.

PROF. BUCKY

If I have a container of water (and this rock behaves much like water) that is a foot in diameter and the top is covered so it can not escape one way or another, (it's totally confined) and I apply a pin-point force to raise the pressure, what will happen?

MR. ERSKINE

But, there you are dealing with a confined liquid.

PROF. BUCKY

And I say I have a confined solid. The interesting part to me is that these confined solids seem to have the same properties as confined liquids, except that the amount of lateral pressure developed depends upon the Poisson ratio of the material, and these are a little bit hard for us to determine at times.

MR. ERSKINE

Along another line, have you considered the possibility of residual stresses in the rock?

PROF. BUCKY

Yes, but I am not certain what a residual stress is. Our geologists who have carried on a lot of this work in the granites talk about the expansion of this rock being due to residual stress. I wish you would define for me the term "residual stress."

MR. ERSKINE

I do not know the conditions of the area, but you are postulating a 6000-foot mountain exerting this pressure. I would propose that there might have been 6000 feet of rock over the area now being mined which has since been eroded. And the stresses still in that rock may act in a horizontal direction. It seems to me that it is easier to explain it that way than to assume the pressure being transmitted over such a great distance.

PROF. BUCKY

If there were 6000 feet of overlying rock and that 6000 feet were eroded, then the pressure would decrease and adjust itself.

MR. ERSKINE

It would adjust itself vertically, but if you are dealing with elastic materials, would it adjust itself horizontally?

PROF. BUCKY

Certainly. If I take a block of elastic material and press down on it, it deforms side-ways; and, if I let up on the pressure, it resumes its original shape.

MR. ERSKINE

But the same thing applies in my original line of reasoning. If horizontal pressure is being exerted, instead of having extreme horizontal pressure relative to the vertical pressure, I think there would be a release in the vertical direction.

PROF. BUCKY

That is quite possible, but consider the Grand Canyon of the Colorado. The erosion at the bottom of the Grand Canyon is considerable, and yet the elevation at the bottom of the Grand Canyon, as I recall, stays the same from year to year. Why?

MR. ERSKINE

I presume it is due to rebound.

PROF. BUCKY

Rebound? Something is moving up at the same rate as material is being worn away, and it must come from someplace within.

MR. ERSKINE

I was interested in obtaining a different viewpoint on the situation. My feeling would be that if material is being removed from above, hydrostatic readjustment would result.

PROF. BUCKY

That may be, but on the other hand, it depends upon the material. What I am considering is just ordinary mechanics of the behavior of geologic material, which I assume is elastic. The existence of lateral pressure originally can be demonstrated by simple mechanics.

JOHN J. REED[6]

I think that there is an important differentiation to be made here, and that is the difference between stress or strain in one direction and stress and strain in three directions. Consider a bar loaded in one direction only; it must have stress in it to be sure. If we put stress in it, it must either stretch or compress. But if we consider three dimensions, as we must when we go underground in examining geologic structures, then it is possible to have stress without strain or deformation. I also believe it is possible to have deformation or strain without stress. Now there are others here who

[6]Head, Mine Research Engineer, St. Joseph Lead Co.

I think can confirm this, and it is a concept that should be considered. I referred to this briefly yesterday when I described the model studies in which I measured modulus or strain gage behavior in a uniaxial compressive test cylinder. Then I used those values in a model loaded in three dimensions, hydrostatic, tri-axially. I had to correct those readings, and as I told you, the corrections turned out to be from 8 to 40 percent. So you see it is of a large order of magnitude.

If we assume an elastic material, in large slabs or a large portion of the earth's crust, and a surcharge is applied as was suggested here, on the basis of the elastic theory we will have deformation close to that surcharge in the underlying regions — that material will be displaced. But the displacements will be limited to a local area directly beneath that surcharge. I mentioned yesterday, and it has been demonstrated mathematically and photoelastically, that the stresses created by an opening in a semi-infinite or infinite medium are localized to that opening. The effects extend about one diameter beyond the opening. The same thing applies to the deformation. Therefore, I cannot conceive of a force due to surcharge extending its effects much beyond the dimension of the surcharge. I think this would be borne out by the theory of elasticity. I suggest that it would be very simple to test it in a concrete slab. I believe you would find stresses developing in the slab fairly close to the surcharge, but not very far away from it.

DR. JOHN W VANDERWILT[7]

I have a question I would like to ask Mr. Fletcher because of his considerable block caving experience. Has there developed through the course of the years a definite relationship between the horizontal cross-section of the block to be caved and its height?

MR. FLETCHER

No, I agree with Professor Bucky. Heretofore, height of the block has depended upon whether the openings below would wear out or cave in before the ore above could be mined out. Concrete has promise because it holds up longer, and it seems to me that a higher height of ore could be mined. Blocks would not have to be taken in successive steps. We mined, for instance, a height of 300 feet of ore on the first block and 300 feet of overburden came down with it. Then we moved down an additional 300 feet and continued on the block underneath. We tried to mine that entire block from 600 feet. In the latter case, the transfer raises, the drifts, everything collapsed before we could mine the ore. Of course, there is the possibility that openings could be repaired to permit mining an additional height of ore. I think San Manuel is trying to determine the maximum height of ore that can be mined in a similar type of rock.

LEONARD OBERT[8]

I would like to ask the mine operators collectively if they would describe or give some estimate of the extent of the undercut area that is

[7]President, Colorado School of Mines.
[8]U.S. Bureau of Mines, College Park, Md.

necessary to initiate caving in the various mines that they represent, both an average value, and also an upper limit. That is, what is the approximate maximum undercut area that they have had to provide before caving starts?

MR. HANNIFAN

Speaking of the Kelley Mine, which as I told you was previously broken material, we can rarely open over 100 square feet before caving is initiated. Only in the hanging wall do we have unbroken material. At Andes Copper, a porphyry-type orebody varying from fairly hard to rather soft material, we actually had an area 4000 square feet undercut and completely emptied. An air-blast condition could have developed but did not materialize because we just quit mining the area and let the opening slowly fill of its own accord. That area actually stood several months. I would say that the minimum and maximum area depend entirely upon the characteristic of the rock. If there are not inherent or existing fractures and the material is rather massive, the area may become quite large. If a lot of fracturing exists in the block, then the area becomes much smaller.

MR. WALKER

At Climax we have different size openings in different parts of the mine. Let us consider the extreme conditions. We had one finger into a 50 by 50 foot area that we were unable to mine. It was in a badly sheared area, and as we came under it with the finger, we were unable to complete the undercutting or to shoot the pillars. We tried to get into it from adjoining fingers and weakening the surrounding ground. We had no trouble at all drawing this particular area — it caved right on through to the top. About all we did then was just to drive a finger to the draw-hole of the stope. We had another area 200 by 300 feet in strong ground that arched. Our most recent trouble has been in much larger area — something like 200 by 500 feet. In this case, we were mining along the top surface of a spherical-shaped orebody. We also had outside pressure exerting on this shell. It would appear to me that the situation resembled trying to crush an egg shell with your hand by applying equal pressure over the entire surface. Although it is a very thin shell, you cannot actually break the egg. I think that in that case, we had that type of situation with Bartlett Mountain exerting a pressure over the curved surface. I think that those are the extremes that we have encountered to date.

MR. FLETCHER

I agree with the gentlemen ahead of me. We have had some very soft ground which would cave over a small area, and we have also had hard ground that required openings over for large areas. It depends entirely on the type of rock.

MR. FLESHMAN

At San Manuel, we have the same type of monzonite porphyry that Miami has. Within this deposit and within the block itself, conditions vary greatly. I would say possibly 10 by 10 feet is the minimum. As far as the maximum is concerned, one side of a block may cave right away, but the

other side may require two undercuts or a shrinkage stope. Our blocks are 210 feet wide and about the same length or slightly longer, depending upon how many grizzly drifts we are using. Within our undercut block itself, it will vary greatly from one side to the other.

MR. NISPEL

Our orebody is similar to San Manuel's. It is generally a soft material, and our minimum is about 10 by 10 feet. Within the panel itself, there may be parts where we open up perhaps five times that area and it supports itself temporarily.

ARTHUR RUFF[9]

When a block is being drawn, what angle of draw is established? Does it go up vertically above the sill or does it go at an angle?

MR. HANNIFAN

I will say vertically above the draw-hole in general.

MR. WALKER

I think vertical draw is a safe assumption, although we have specifically mined a stope in which we planned to get side movement. We have had experience with a very high recovery that could only be explained through side movement. That was obtained by restricting the size of the cave so that we got movement coming in from the side and not directly over the draw-hole itself.

MR. FLETCHER

I would say it is vertical, but we have had side movements that are pretty hard to explain at times.

MR. FLESHMAN

I cannot answer that question, because as yet we have not reached the surface — at least, as of last Saturday morning! We expected the first stope to break through to the surface by the first part of May. The Bureau of Mines has put several monuments on the surface, and they have been tied into the underground coordinates. There are precision level surveys run on these monuments periodically. We believe that the block will cave vertically, and as stated in Peele and other places, eventually the angle of draw will reach 70 degrees. We believe that the original cave is vertical, and afterwards this 70 degrees is attained sloughing along at the sides, with the vertical walls attaining a natural angle of repose.

MR. DALE

Our cave varies somewhere between 70 degrees and vertical. I would say that ordinarily it approaches vertical. It depends partly on the geologic nature of the rock. If there is a hard rock above and weak one to the side, the draw can be controlled so that some of the weak rock to the side can be pulled in. As this vertical wall develops, there may be a tendency for the sides to spall off into the draw area. On the end of block, in almost any case, it seems there would be some force built up that would cave in more from the sides.

[9]Cananea Copper Co., Mexico.

GEORGE CLARK[10]

Mr. Hannifan, you spoke of the voids that were developed in these old stopes. Do you ever have any difficulty in spalling through these in starting new drifts? How do you determine the position of these voids?

MR. HANNIFAN

We do not attempt to locate them, nor do we have any particular difficulty with them. However, one of our failures in concrete was due to a settlement into a void below us which we did not know existed; our foundation fell out from under us. Ordinarily, if we have any knowledge of a void in the area, we would fill it with concrete.

MR. HILL

Mr. Hannifan, in describing the concreting of your slushing drifts, you expressed the thought that there is a possibility that pre-stressed concrete might have an application here. As I understand the purpose of pre-stressed concrete, it is generally a more economical use of concrete by adding steel to take tensile stresses at the full capacity of the steel. This permits the use of smaller beams and smaller slabs than otherwise. As I understand your use of concrete in these mine openings, you do not use any reinforcing at all in your concrete walls and roofs. Have you given any thought to a more economical utilization of materials by putting in some reinforcing to take the tensile stresses?

MR. HANNIFAN

Personally I have not given it much thought because the operators that I have talked to who have been using concrete for some years find that the addition of reinforcement is a detriment rather than a help, the reason being that the percussion causes resonance within the reinforcement which tends to spring it free from the concrete. Failure results sooner than in non-reinforced concrete. As far as pre-stressing is concerned, I imagine the same condition would result. However, if we continue to use the type of block that we have with the slusher level on the blasting horizon, it might have some application, especially at the deeper levels where we know much heavier conditions will exist. This would not be so much an economy but a means of getting higher strengths of support for the slusher opening.

MR. HILL

With regard to the use of steel in concrete as an economy measure, I think that we should bear in mind that the stress concentrations encountered are mainly compressive, with few tensile failures resulting. Steel is not an economical material to use to withstand compression.

I think that rock bolts as used in the walls of metal-mine openings (as opposed their use in coal-mine roofs to build up beams) are most advantageous in giving us the lateral support or lateral restraint that we have already mentioned. Here we are using steel in tension and providing the walls with lateral support. I think that this would be equally effective in the concrete walls of an opening as well as in the natural rock.

[10]Head, Department of Mining Engineering, Missouri School of Mines.

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PREDICTING THE BLOCK CAVING BEHAVIOR OF OREBODIES

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PREDICTING THE BLOCK CAVING BEHAVIOR OF OREBODIES

Block caving is an efficient mass-production method for the underground mining of large orebodies. However, the block-caving method requires a substantial investment in developmental works, based on the assumption that the rock will cave.

Efficient block caving is prevented if the rock forms stable arches or breaks into fragments that are too large to pass easily through the finger raises. Under these conditions, the cost of the additional labor, drilling, and blasting required to assist caving may be sufficient to make marginal orebodies uneconomic or alternative mining methods more attractive.

Reliable methods for the prediction of the block-caving behavior of an orebody are clearly needed at the initial planning stage. These methods should be capable of considering both geological and engineering aspects of a block cave, as experience has shown that the manner in which a block cave is initiated and developed may be almost as important as the geological conditions such as the nature of the rock material, degree of alteration, characteristics of the joints and faults, etc. The initial state of stress in the rock mass should also be considered.

The reliability of various proposed methods for the prediction of block caving behavior was tested in six areas of the Climax and Urad Mines, in Colorado, where block-caving behavior has been

observed over periods ranging from one to six years. Only the methods which proved most successful will be described here.

GEOLOGICAL INDICES

The purely geological aspects of block caving were studied by means of geological indices. These are easily measured and reproducible, numerical properties of the rock mass. Relationships between the various indices and such rock mechanics parameters as compressive or tensile strength and deformation moduli can be established by correlation with test results.[1] The aim of this part of the study was to determine which of the many possible geological indices was most highly correlated with some independent measure of block caving behavior. Clearly, the first step was to set up a measurement of block-caving behavior -- or "cavability."

Measurement of Cavability

At the beginning of the study, there was considerable feeling that a composite index of cavability was required to reflect such properties as the quantity of secondary blasting, size of the rock blocks, sorting characteristics of the muck, ease of fall through ore passes, and ease of loading onto trains. In order to obtain such a composite index, six experienced mine operators were asked individually to rank the areas that they had worked in the Climax and Urad Mines in order from 1 to 10, where 1 indicated excellent caving characteristics and 10 indicated particularly-poor caving characteristics.

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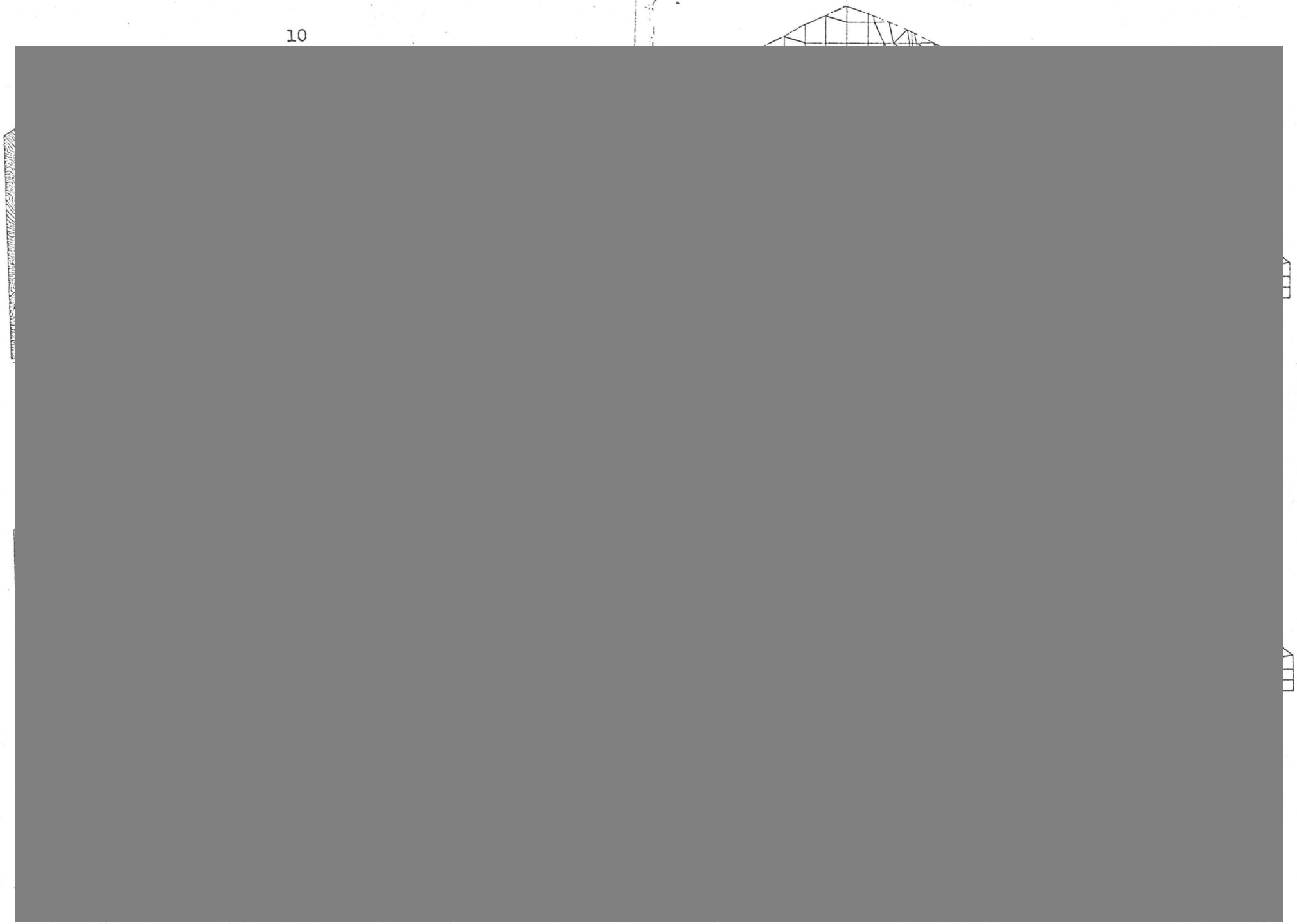
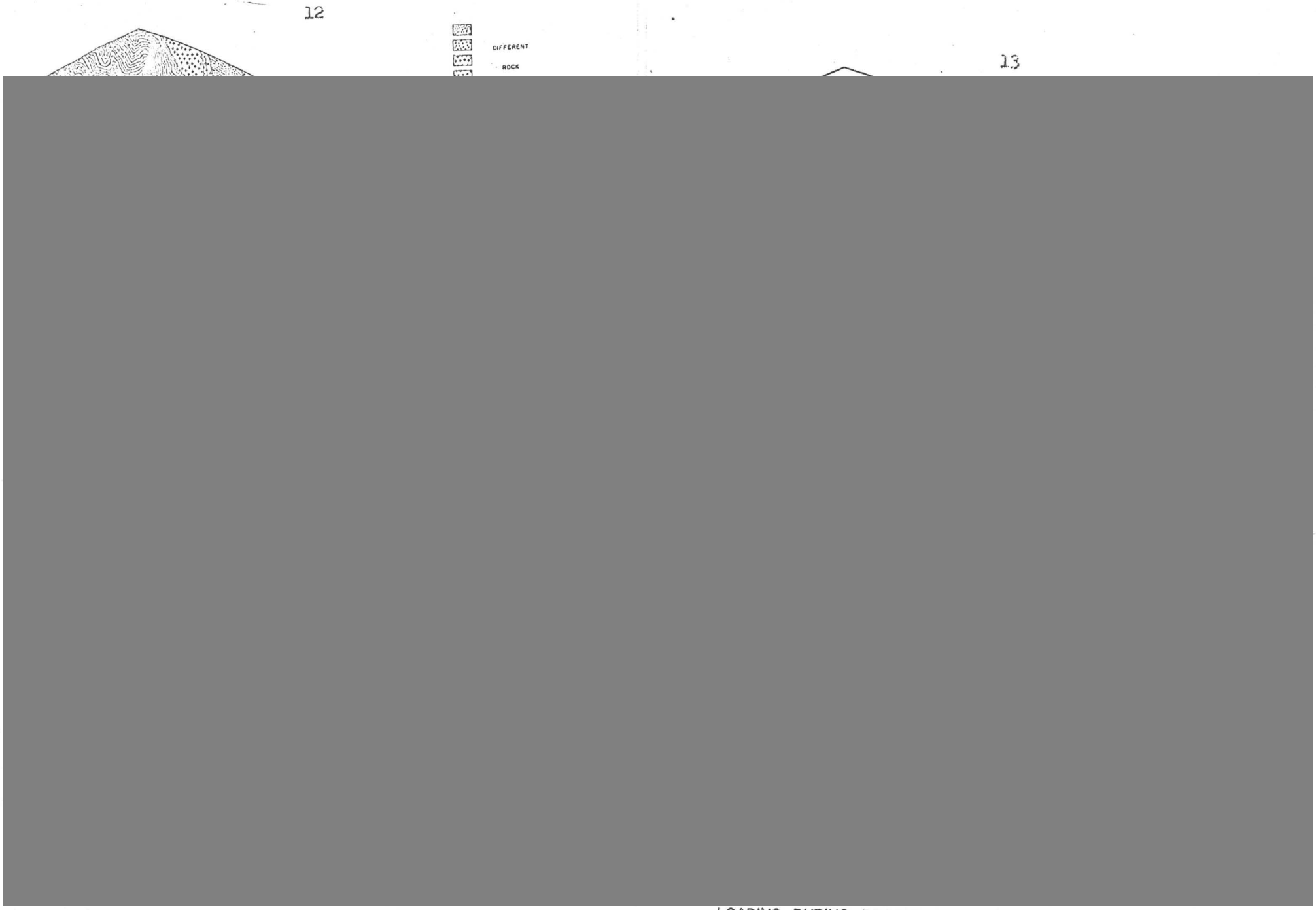
Berkeley. The authors are particularly grateful to Dr. R. E. Goodman





LOADING DURING PROGRESSIVE CAVING.

GENERAL INFORMATION BOOKLET -- APRIL, 1968

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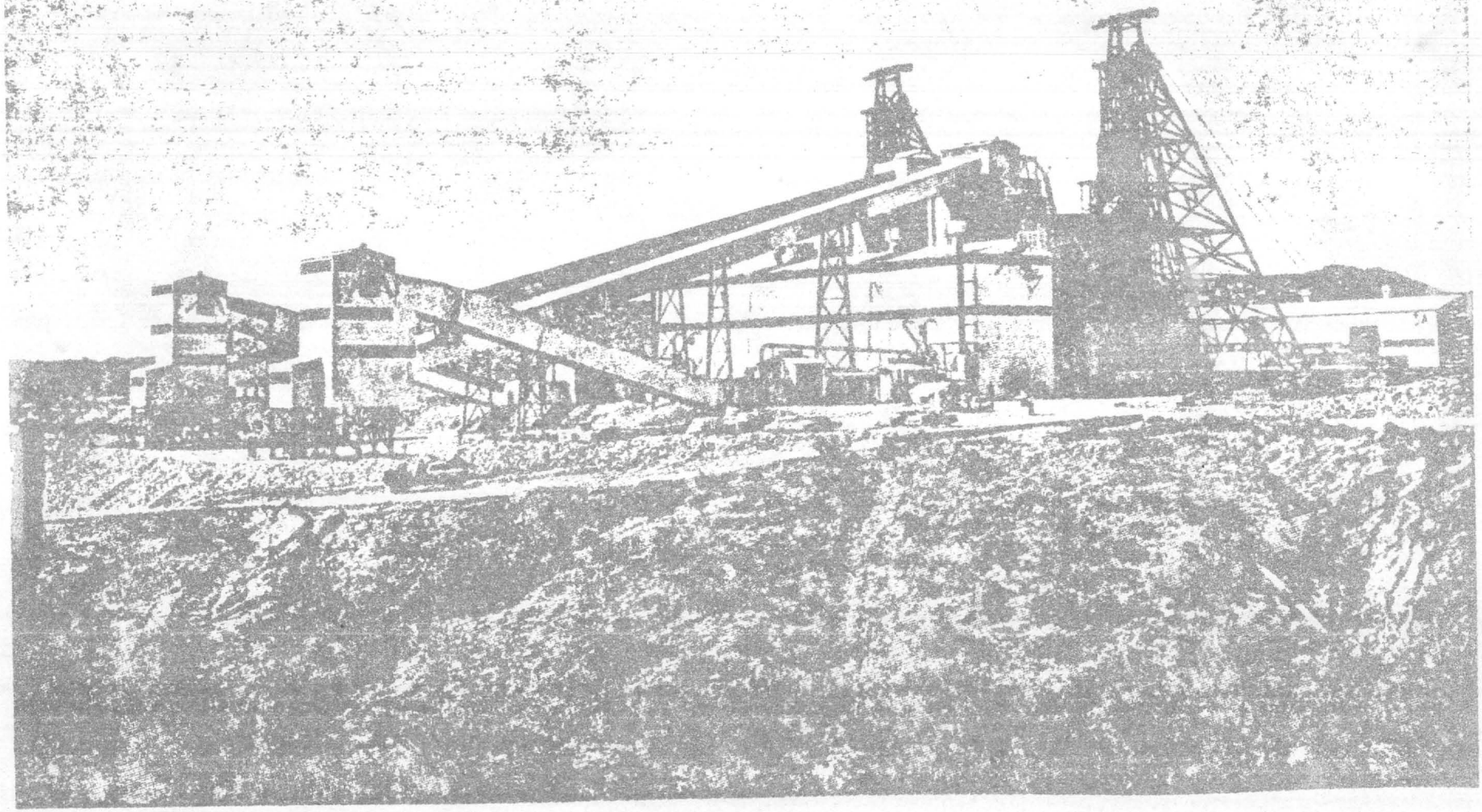


MAGMA COPPER COMPANY
San Manuel Division
San Manuel, Arizona

April, 1968

ORE BODY

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 1900 feet thick. Average over the Southeast (Main) Ore Body is 670 feet thick, and consists mainly of a very competent rock called Gila Conglomerate.

(The last two pages of this pamphlet show sections of the ore body.)

PRIMARY DEVELOPMENT

NO. 1 SHAFT

Depth: 2,262 feet. (For the Second Lift.)
(Sinking to a depth of 3,170 feet is in progress.)

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 hoist, 5'-0" (Below 1475)

Ventilation Downcast.

Structural Steel Sets and Reinforced Concrete.

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

Skips: 4-ton capacity in counter-balance, 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 Size = 1".
Rope Speed = 545 fpm.

No. 1 Shaft -- cont'd

This shaft is used to hoist waste muck 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 transportation of concrete. This is the development shaft, and it will be sunk to provide initial access to future levels.

NO. 3A-3B SHAFTS

Twin Ore Hoisting Shafts, 195 feet apart.

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

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

Ventilation Exhaust Shafts.

Structural Steel Sets poured in concrete for smooth lining. Concrete curtain walls between each compartment, with 3'-7½" x 4' windows in each set.

Steel guides supported every three feet in hoisting compartments.

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

Headframe: 181' to b 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: Two in 3A; one below 1775 SubLevel, Capacity = 1,100 tons.
one below 2075 Level, Capacity = 1,500 tons.
One in 3B below 2075 Level, Capacity = 1,500 tons.

No. 3A-3B Shafts -- cont'd

Skips: Bottom-dump steel skips with alloy steel liners, running on solid rubber tires.
Capacity: 22 tons with + 4.5% 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 face drums with 3,700 feet of $2\frac{1}{4}$" rope capacity in two layers.
Drum Shaft: 27" diameter through drums.
Hoisting Speed: 2,800 fpm.
Two 3,000-hp D.C. motors equipped with M.G. set consisting of one 4,000-hp motor and two 2,500-KW D.C. generators. Steel plate flywheel, 44 tons (approximate) for 80% power peak equalization.

Service hoists are equipped with an 8-passenger cage and 45 cubic feet 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 265' x 50' Crusher Building houses two 42 x 65 gyratory crushers. There are two 96" x 62'-0" pan feeders, with a capacity of 1,500 tons per hour, one fed from the 3A and the other from 3B coarse ore bins. 48" conveyor belts transfer the crushed ore from the crusher building to transfer towers (222'), and back to the fine ore bins (421'). Skip dumping, fine ore bin capacity, and transfer belts are viewed on closed circuit television. The entire automated crusher operation is monitored from control panels at each crusher location.

NO. 4 SHAFT

Depth: 2,158 feet. (For the Second Lift.)

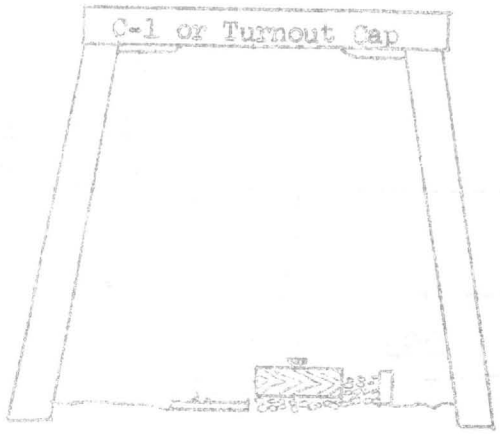
Dimensions: 26'-6" x 14'. Structural steel sets poured in concrete for smooth lining.

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.

No. 4 Shaft -- cont'd

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 feet to b sheaves. Two 14" cast steel sheaves.
Hoist:	Double drum. 15' diameter, 90" face drums. 2¼" hoisting rope. Maximum hoisting speed: 1,500 fpm. Single reduction drive, two 700-hp D.C. motors equipped with M.G. set consisting of one 1750-hp A.C. motor and two 600-KW D.C. generators. Hoist-house equipped with 30-ton crane with 5-ton auxiliary.



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.
39 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 70# Rail is used, 64" is the grade.
45# Rail for Development.
70# Rail Finished Track.

RAIL MEASUREMENTS

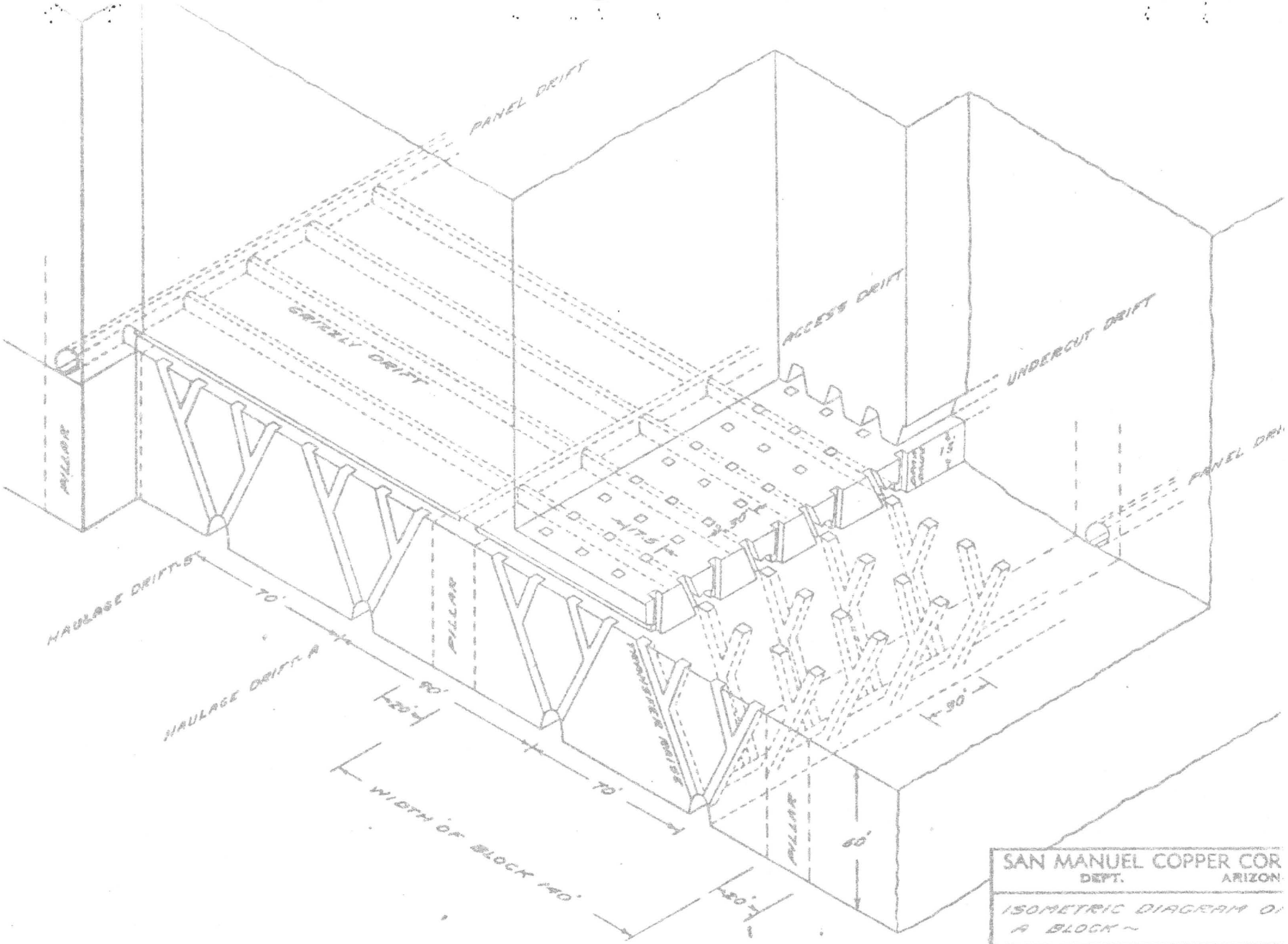
3-3/4" for 45# Rail.
4-5/8" for 70# Rail.
5-5/8" for 90# Rail.

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 70# Rail is used, 64" is the dev'pt grade.
45# Rail for Development.
70# Rail Finished Track.

CONCRETE HAULAGE DRIFT - 45# Rail for Development.
90# Rail Finished Track.

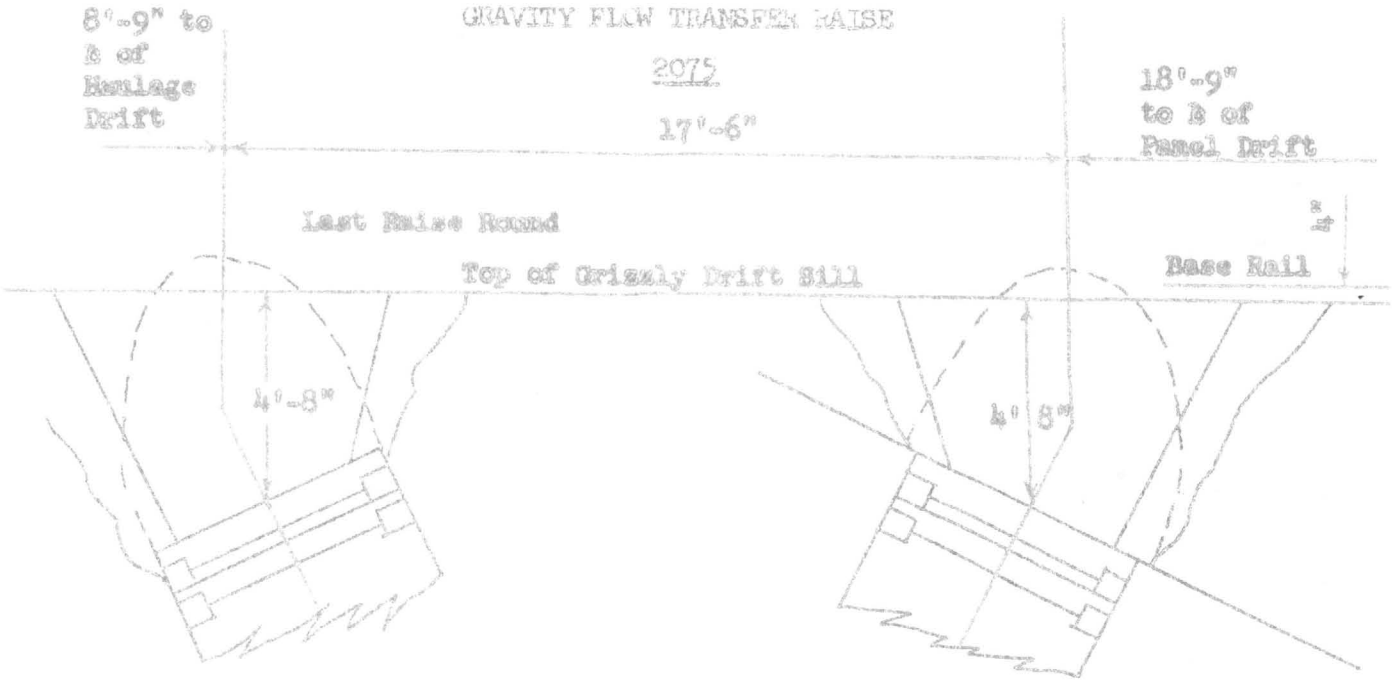


SAN MANUEL COPPER COR
DEPT. ARIZON

ISOMETRIC DIAGRAM O
A BLOCK ~

DRAWN N.H.R. DATE 4/62		DWG.
CHKD.	DATE	A
APPD.	SCALE 1"=40'	ACCT.

BLOCK DEVELOPMENT



4' x 4' inside.

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

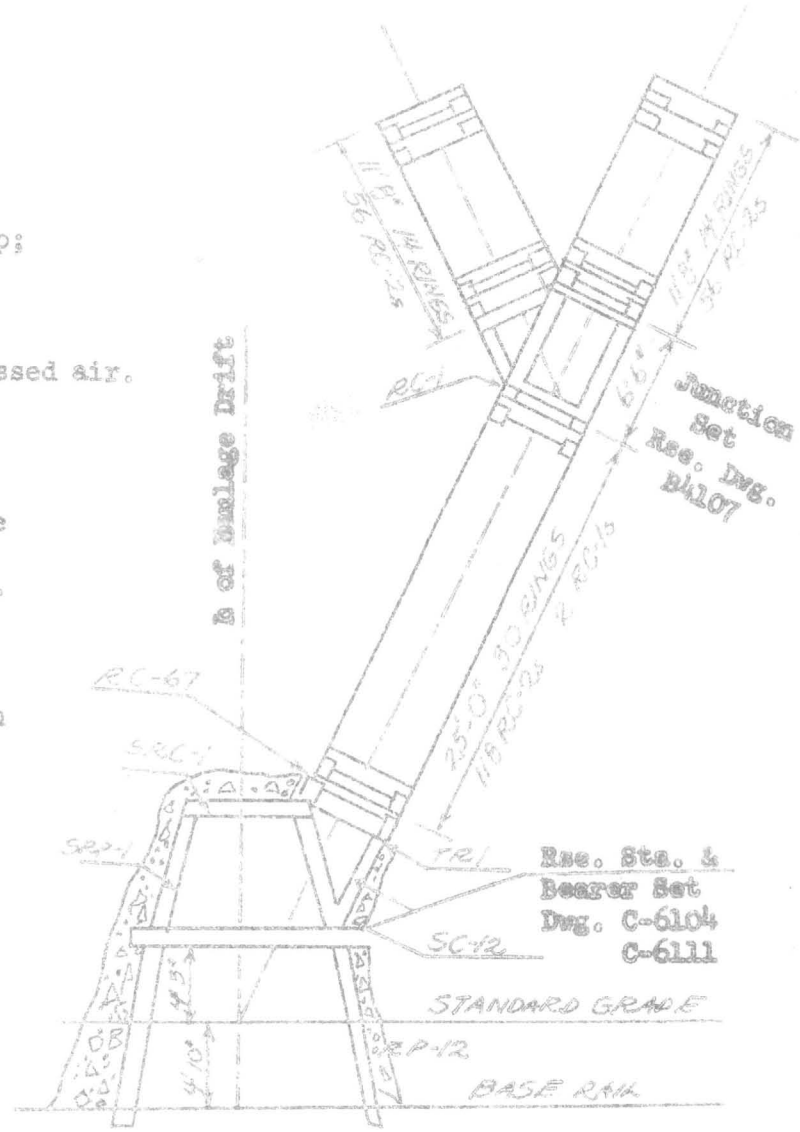
INCLINATION: 63°

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.

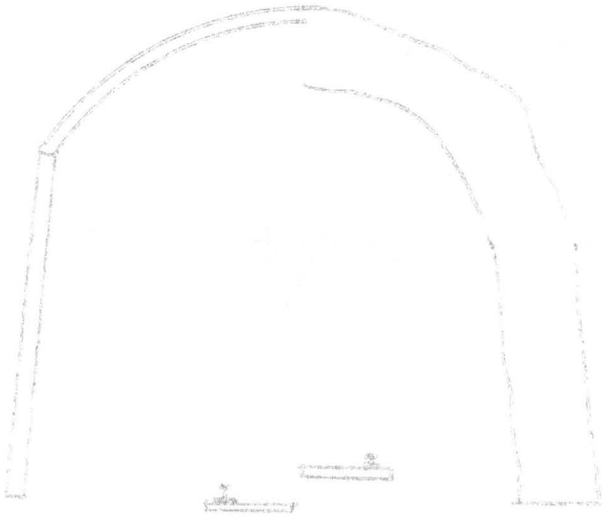
Transfer Raise Stations are cut by the regular drift crew using the same equipment they use to drive the drift. They are cut as the drift heading is advanced.

Raise stations are steel pony sets on top of special drift sets. Lined with non-reinforced concrete.



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.

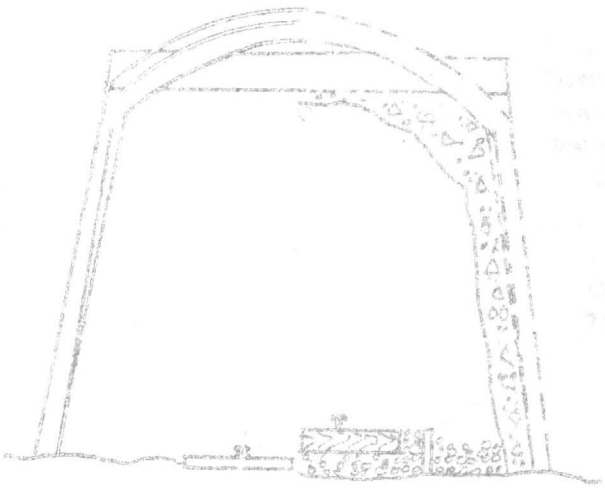
Two-man crews or 3 men in two headings.
29 holes drilled 6" deep.
40 lbs. ammonium nitrate primed
with a stick of 60% Amogel per
hole. Usually fuse blasted.
Ventilation with fans and tubing.



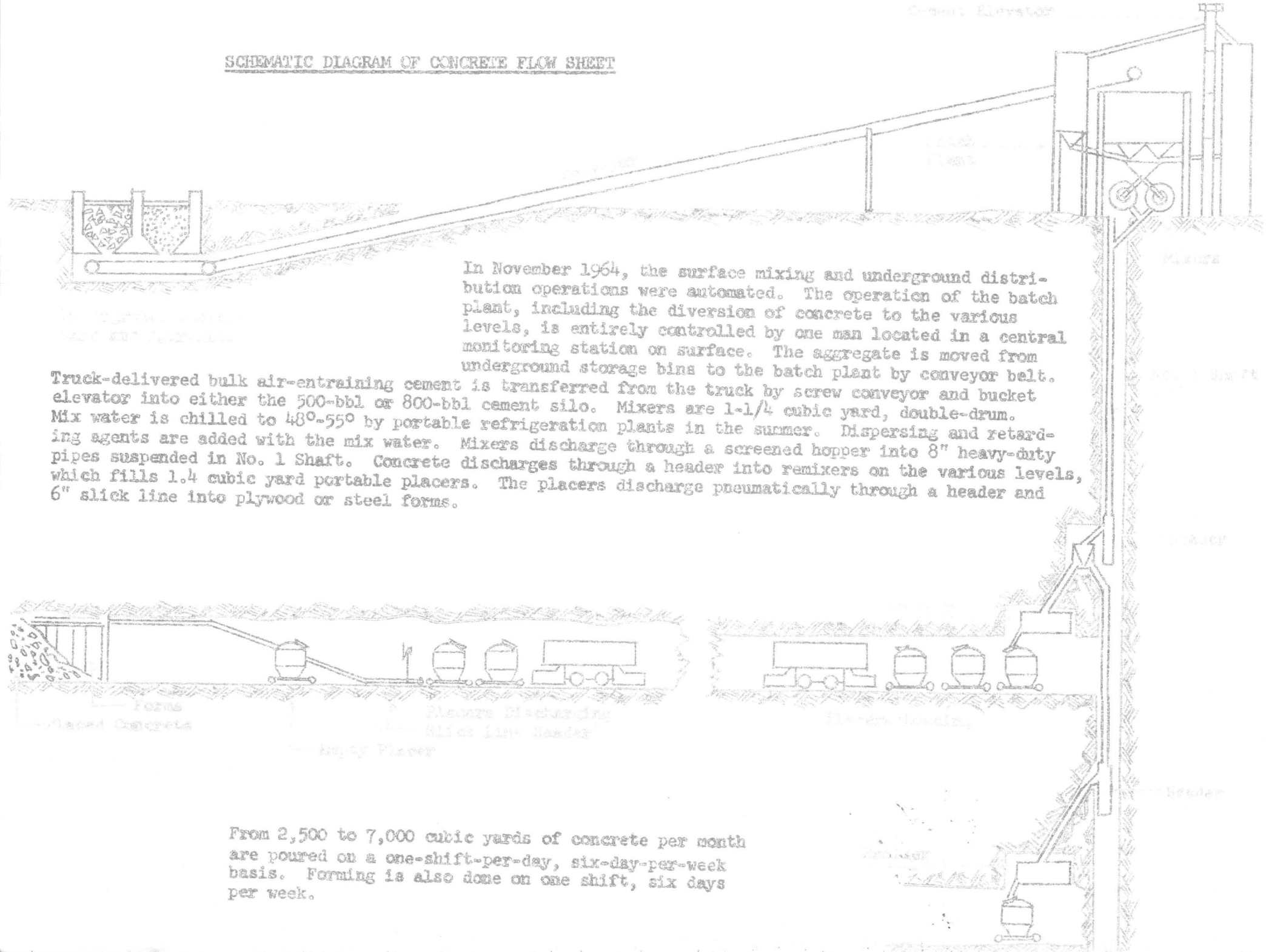
CONCRETE PANEL DRIFT

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

CONCRETE FILLING DRIFT

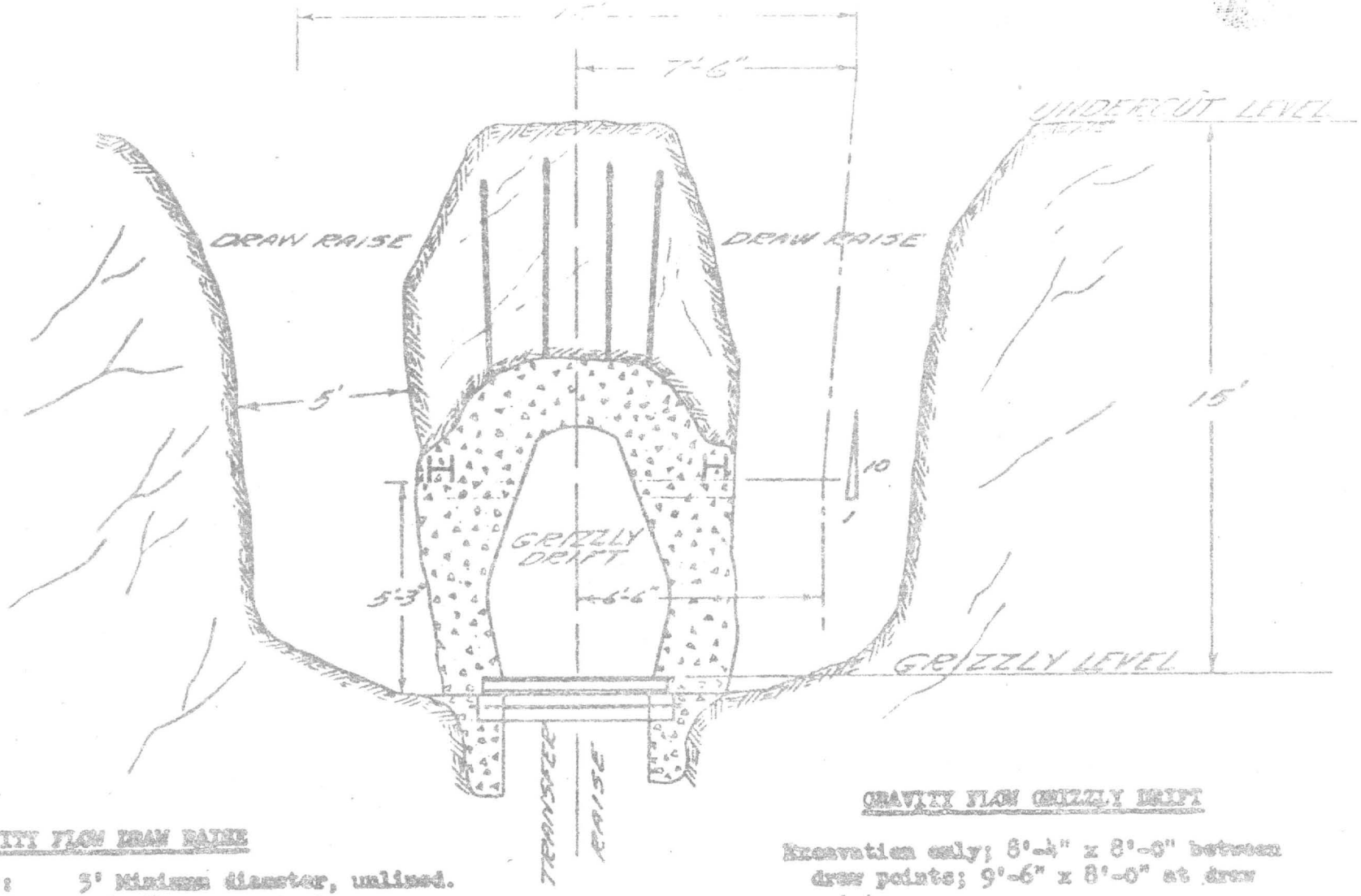


SCHEMATIC DIAGRAM OF CONCRETE FLOW SHEET



In November 1964, the surface mixing and underground distribution operations were automated. The operation of the batch plant, including the diversion of concrete to the various levels, is entirely controlled by one man located in a central monitoring station on 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 screw conveyor and bucket elevator into either the 500-bbl or 800-bbl cement silo. Mixers are 1-1/4 cubic yard, double-drum. Mix water is chilled to 48°-55° by portable refrigeration plants 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 1.4 cubic yard portable placers. The placers discharge pneumatically through a header and 6" slick line into plywood or steel forms.

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



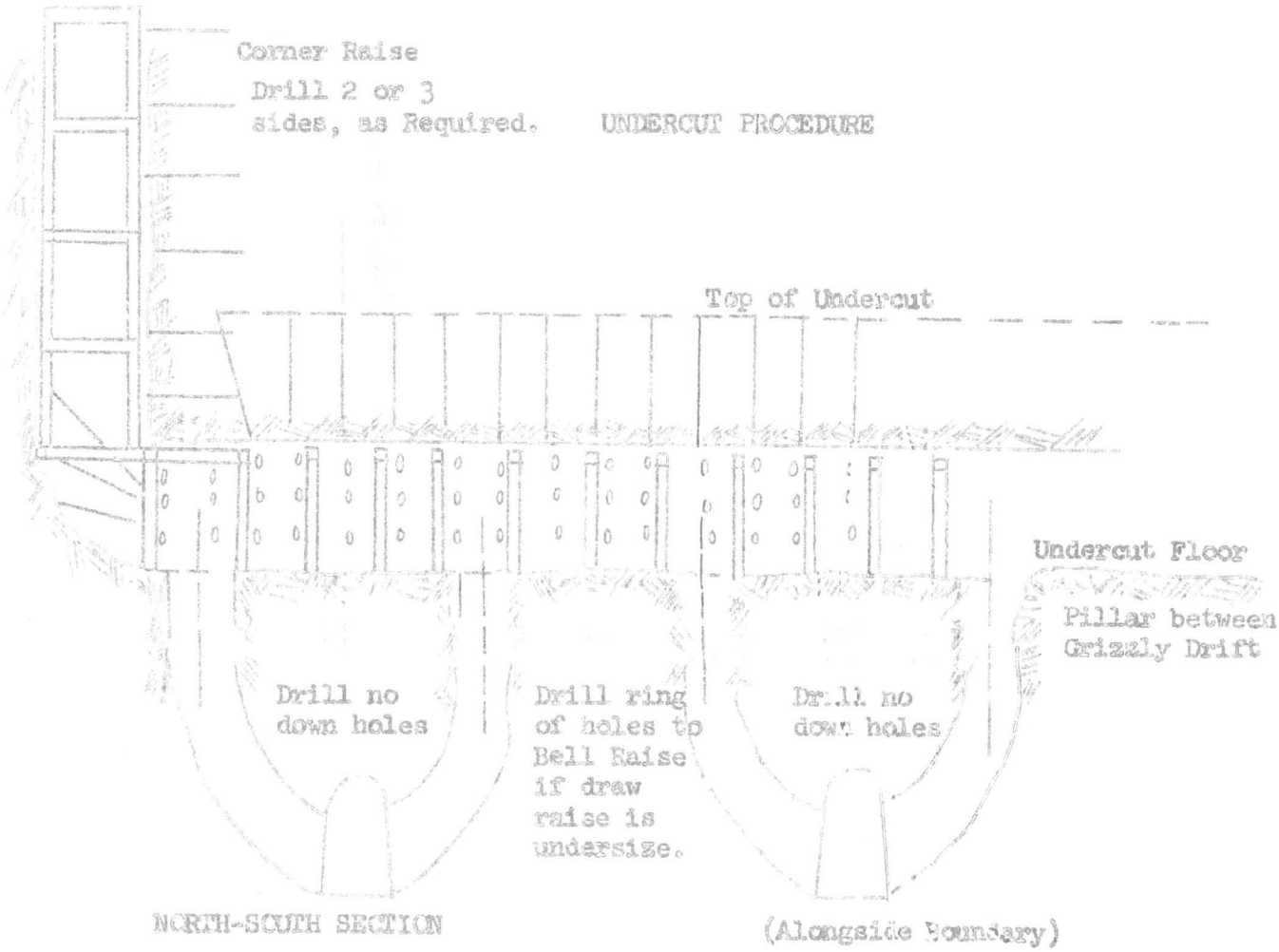
GRAVITY FLOW DRAW RAISE

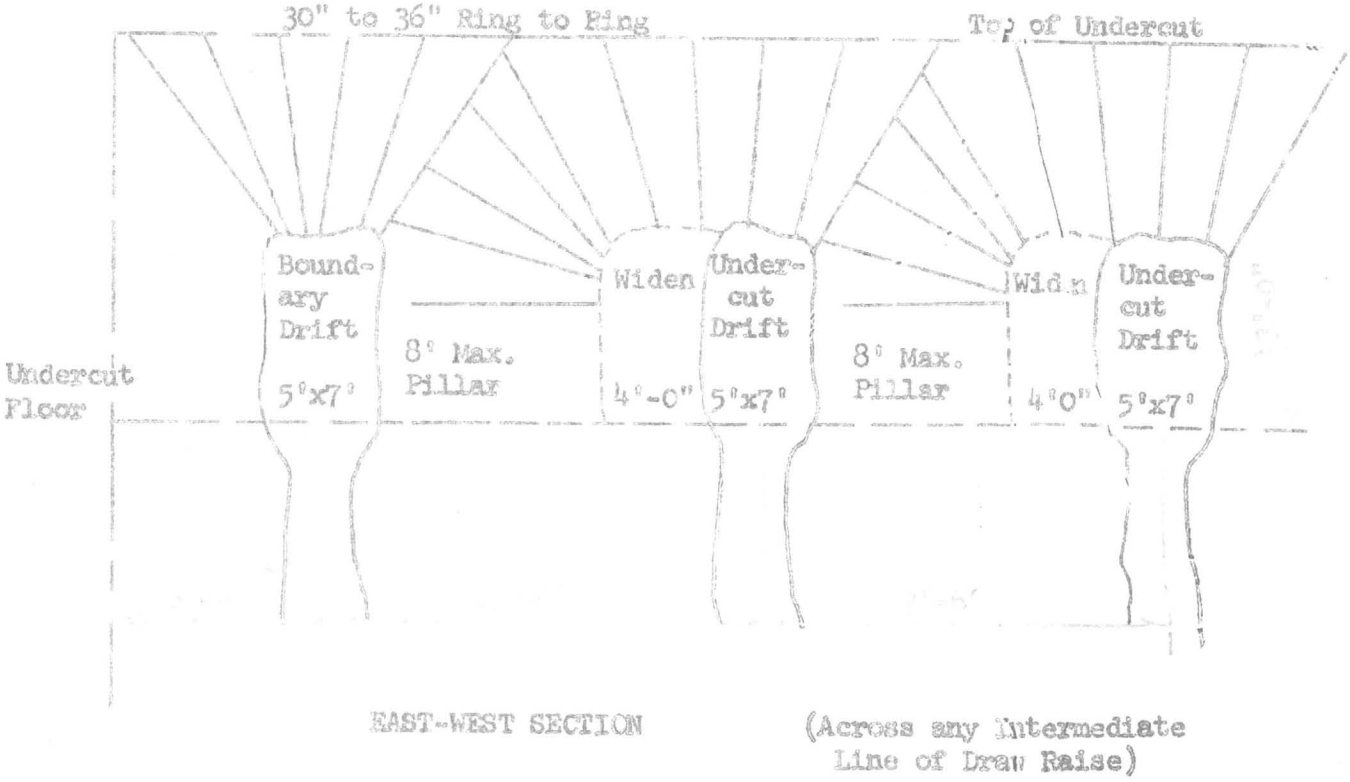
Size: 5' Minimum diameter, unlined.
Length: 15'-0" above grizzly.
Driven after grizzly drift is concreted.

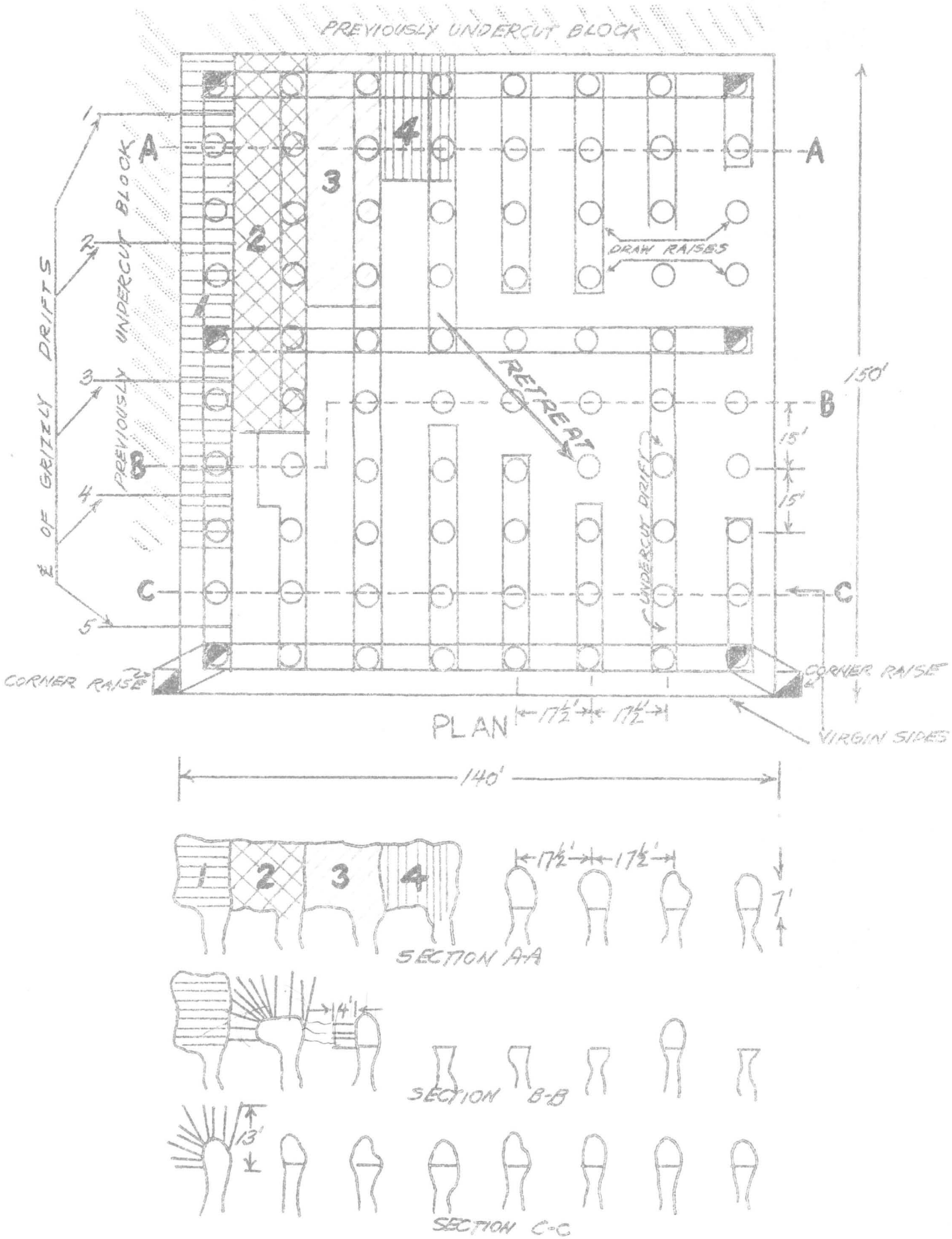
Crews: 2-man crew drives several raises at once. 11 to 24 holes drilled 4' to 6' deep. 27 lbs. ammonium nitrate with each hole primed with a stick of 60% Amogel. Electrically blasted.

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, 8-3/4' 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.


Corner Raise
Drill 2 or 3
sides, as Required.
UNDERCUT PROCEDURE
Top of Undercut
Undercut Floor
Pillar between
Grizzly Drift
Drill no
down holes
Drill ring
of holes to
Bell Raise
if draw
raise is
undersize.
Drill no
down holes
NORTH-SOUTH SECTION
(Alongside Boundary)


30" to 36" Ring to Ring
Top of Undercut
Boundary
Drift
5'x7'
8' Max.
Pillar
Widen
4'-0"
Undercut
Drift
5'x7'
8' Max.
Pillar
Widen
4'0"
Undercut
Drift
5'x7'
Undercut
Floor
EAST-WEST SECTION
(Across any Intermediate
Line of Draw Raise)



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 feet above the grizzly drift floor. They are driven 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 tops of 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 is started 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 feet and timbered if necessary. The remaining pillar, about eight feet thick, is drilled through, and the back of the widened drift is then drilled out to a height of 13 feet 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-foot 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.

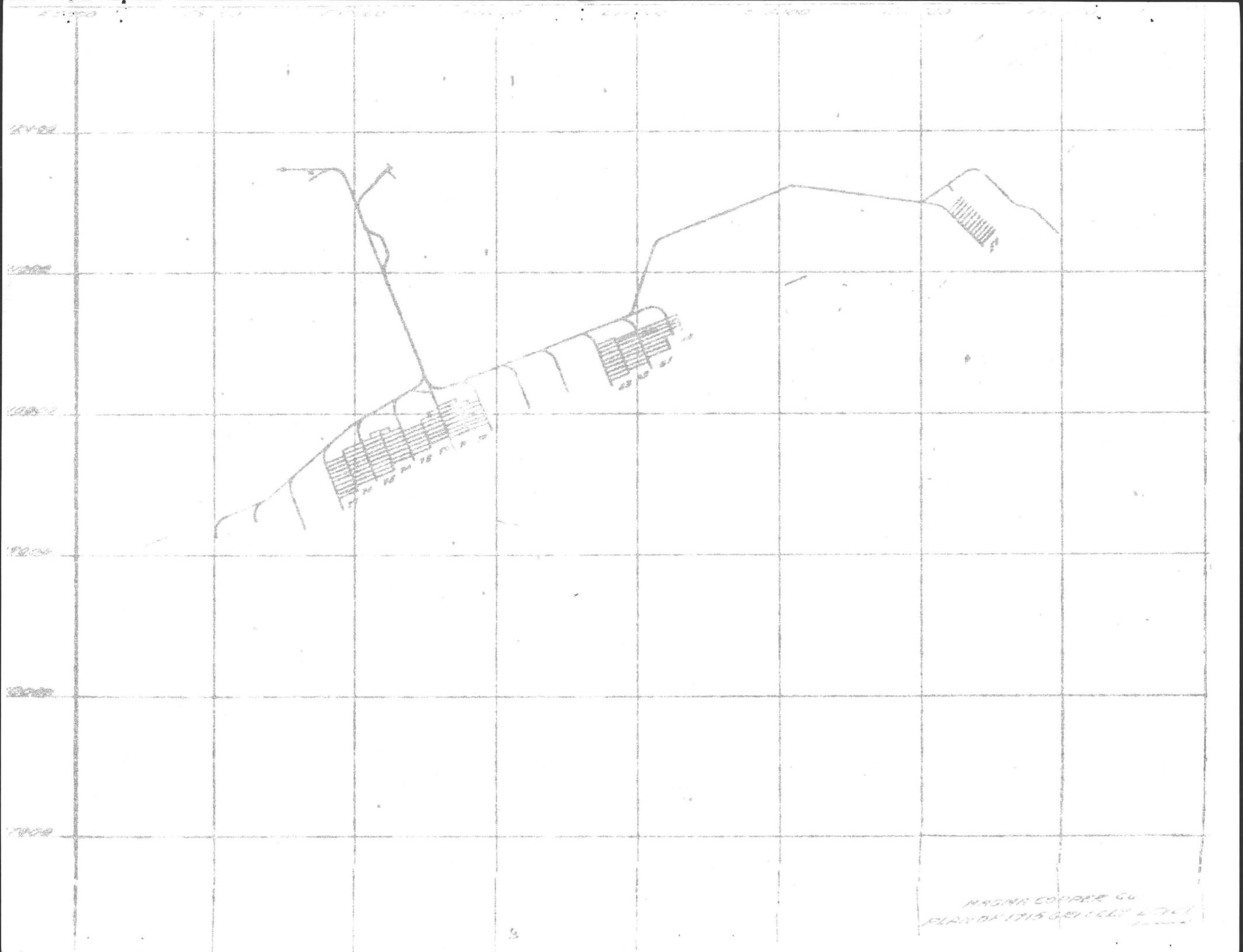
TIMBER:

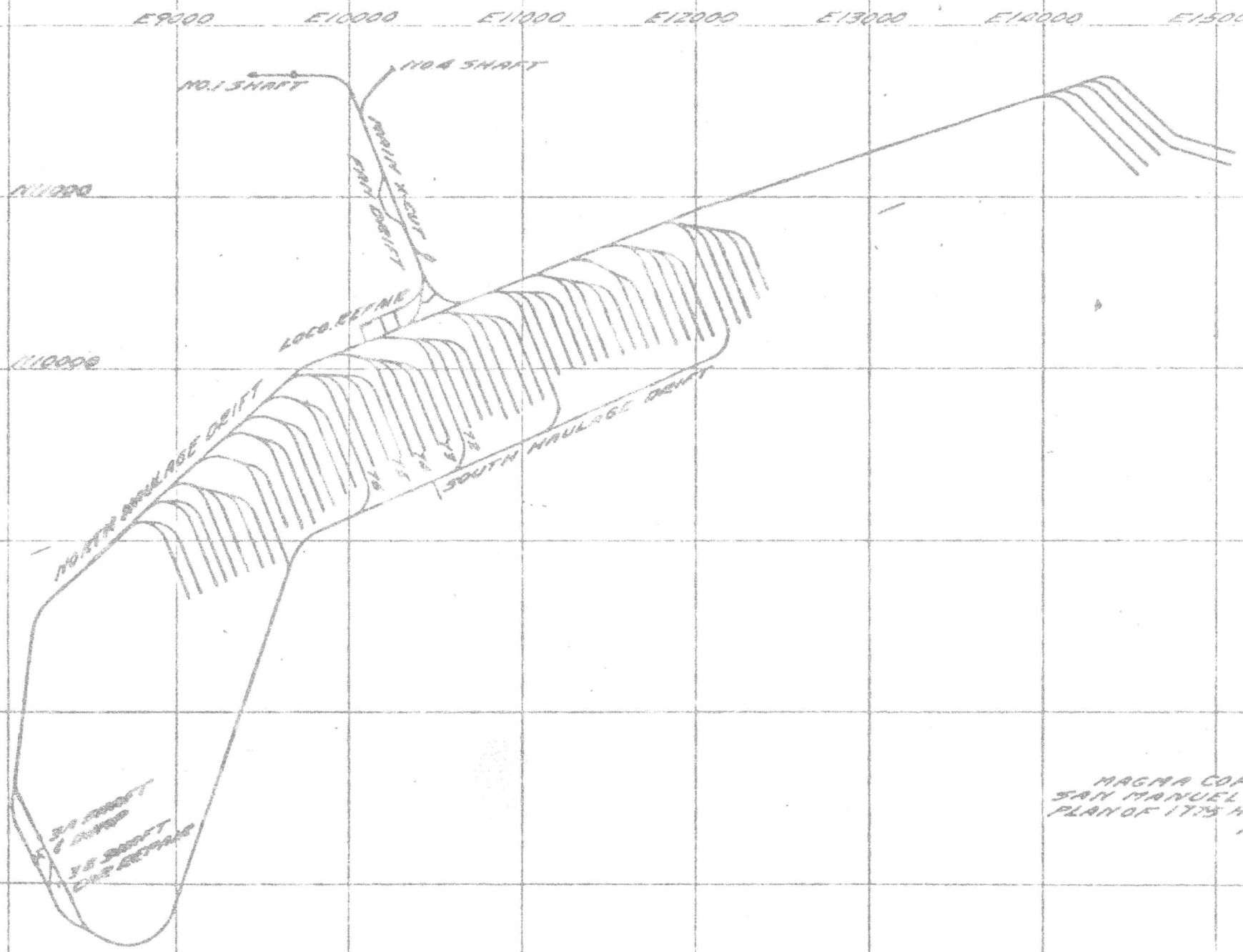
Round posts and 6" x 8" caps with 6" x 8" stringers or sills over raise tops.
2" x 12" side and back lagging.
Raise tops temporarily covered with 2" lagging.

CREWS:

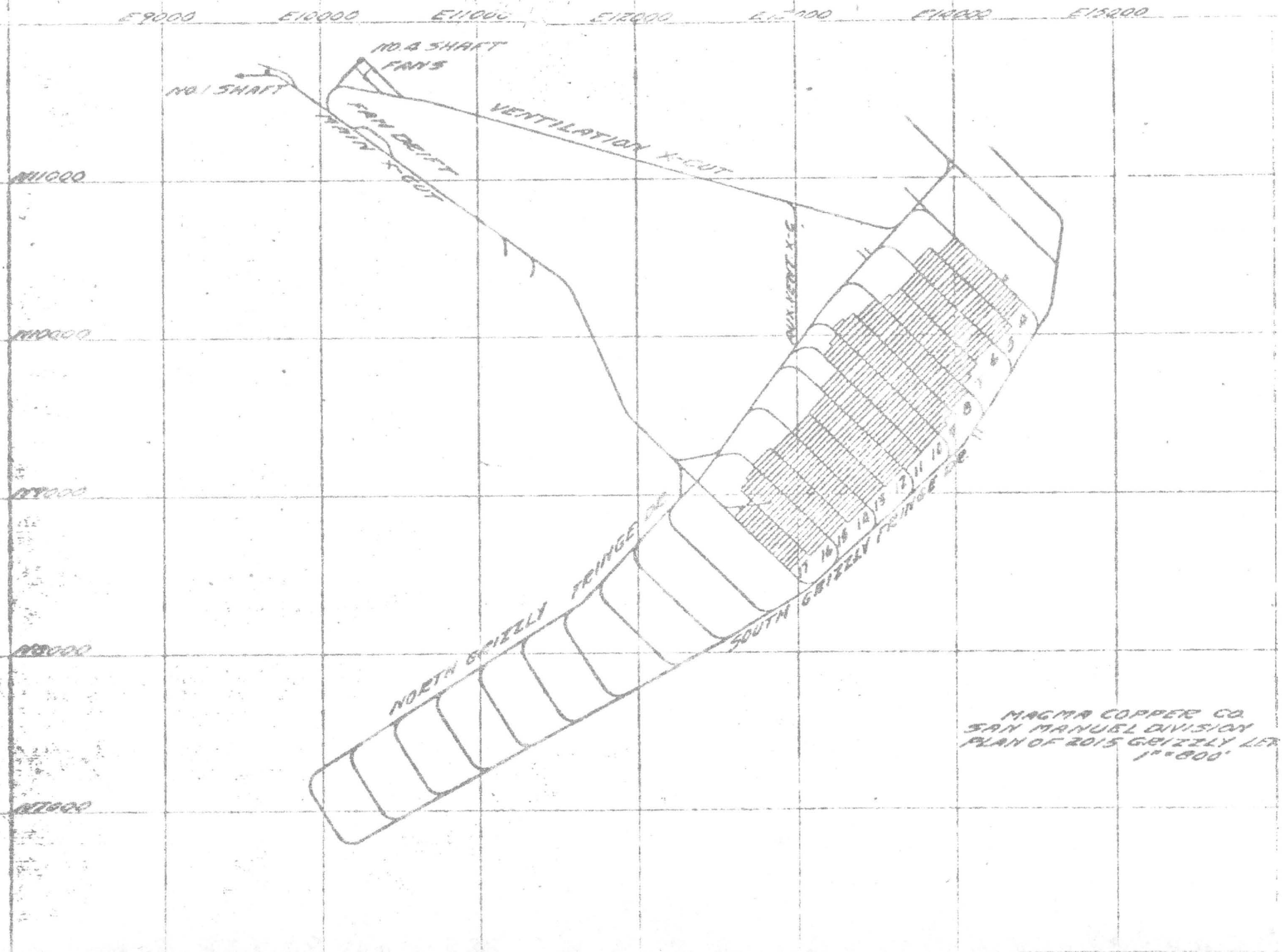
2-man crews drill 10 to 18 jackleg holes and 20 to 30 stoper holes, depending on draw raise spacing and ground condition.

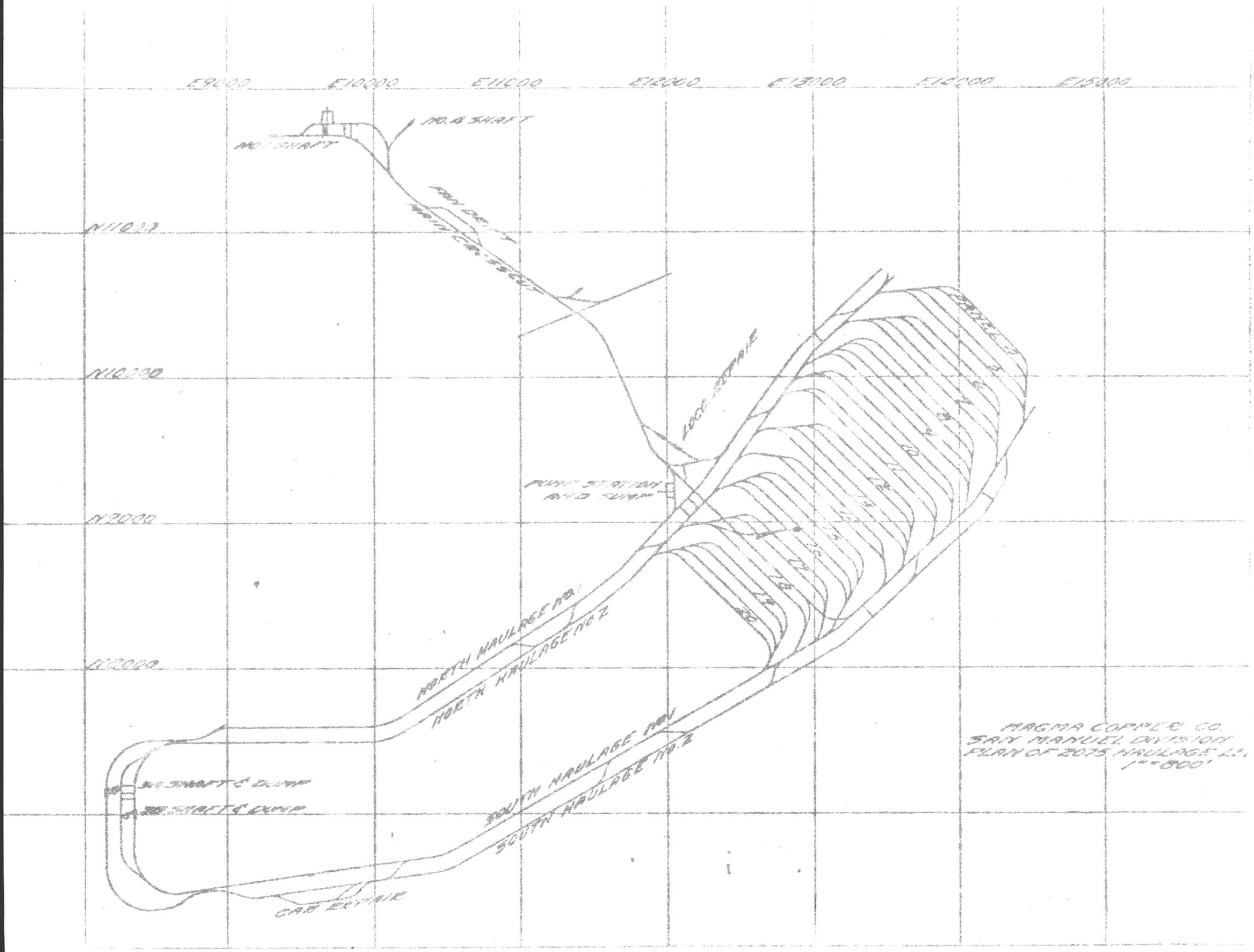
45% N.G. powder or ammonium nitrate primed with a stick of 60% Amogel is used with Millisecond electric blasting.


MAGMA COPPER Co
PLAN OF 1715 GRIZZLY LEVEL



MAGNA COPPER CO.
SAN MANUEL DIVISION
PLAN OF 1775 HAULAGE LEVEL
1" = 500'


E9000
E10000
E11000
E12000
E13000
E14000
E15000
N11000
N10000
N9000
N8000
N7000
NO. 4 SHAFT
FANS
NO. 1 SHAFT
FAN DRIFT
MAIN X-CUT
VENTILATION X-CUT
AUX. VENT. X-C.
NORTH GRIZZLY FRINGE
SOUTH GRIZZLY FRINGE
MAGMA COPPER CO.
SAN MANUEL DIVISION
PLAN OF 2015 GRIZZLY LEV.
1" = 800'


E9000
E10000
E11000
E12000
E13000
E14000
E15000
N11000
N10000
N9000
N8000
NO. 1 SHAFT
NO. 2 SHAFT
LOCO. REPAIR
PUMP STATION AND SUMP
NORTH HAULAGE NO. 1
NORTH HAULAGE NO. 2
SOUTH HAULAGE NO. 1
SOUTH HAULAGE NO. 2
CAB REPAIR
MAGMA COPPER CO.
SAN MANUEL DIVISION
PLAN OF 2075 HAULAGE LEV.
1" = 800'

GENERAL MINE OPERATING DATA

Operating Shifts per Day	3
Operating Days per Week	7
Operating Days per Year (7 Holidays)	358
Production per Day, Tons	40,000
Production per Year, Tons	14,320,000
Active Undercut Area, Sq. Ft.	368,000

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

Block Dimensional Data:

Block Width	140'
Block Length	150' - 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	1,250
High-Pack	50
Held for Repair	80
Held for Grade	20
Total Draw Points	1,400

Rate of Draw, Inches per Day	18

Types of Explosives Used:

Secondary Blasting - 45% (No. 3) Amogel in 1-1/8" x 6" sticks or 60% (No. 1) Amogel in 3/4-lb. and 1-2/3 lb. bags. Initiation is with zero delay electric blasting caps. These are tied into a trunk line to a central underground location where the blasting switches are located.

Primary Blasting - In the larger headings an ammonium nitrate explosive is used except in wet holes. Initiation is by a 1-1/8" x 6" stick of 60% Amogel which, in turn, is ignited by a fuse cap. An 8" 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.

GENERAL MINE OPERATING DATA -- cont'd

Mining Equipment:

Big drifts use a San Manuel-made jumbo with four 8'-booms and four 3½" drifters with 6' screw 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 feed-leg 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 D.C. motors. They haul fifteen 12- to 13-ton capacity cars.

Ore cars are 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 without uncoupling. Car-stops rise and lock the train in position. The Motorman activates the dumper which rotates 180° and returns to the upright position. The car-stops then retract and the Motorman advances three car lengths to the next dump 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

479,000 cfm goes down #4 and #1 Shafts from the surface.
20,900 cfm Miscellaneous intakes, in addition to shafts.

24,000 cfm - 1st Lift (1415-1475 Levels)
160,000 cfm - SubLevel (1715-1775 Levels)
315,000 cfm - 2nd Lift (2015-2075 Levels)

1st Lift - One hi-speed Axivane mine fan, 60-hp, 1750 rpm, 5' diameter on the 1415 Main Crosscut.

Sub-Level Two hi-speed Axivane mine fans: One MXC, 1715, 200-hp, 1160 rpm, 5' diameter; one MXC, 1735 200-hp, 1160 rpm, 5' diameter.

2nd Lift - Three 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, 5' 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 20-hp, high pressure, high speed fans or air movers with tubing.

COMMUNICATIONS

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.

DRAINAGE -- cont'd

No. 1 Shaft is pumping 2,800 gpm, principally from the 2nd Lift;
900 gpm, from 3B Shaft Bottom;
Present pumping is 3,700 gpm Total.

Mine water is pumped approximately 8 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, 1600-hp, diesel-electric.
Trackage:	132-lb. rail, 7-mile haul to receiving bin at reduction plant, level track.

CHANGE ROOM

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, Safety Engineer's Office, Dispensary, Lamp Room, and Dust-Counting Laboratory.

COMPRESSOR HOUSE:

Five 3,500 cfm compressors, delivering at 100-lb. 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.

NO. 1 AND NO. 4 YARDS:

Surface installations in this area include a Machine Shop (including Car Repairs), Electric Shop, Drill Repair Shop, 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 addative storage tanks), Stand-By Power Plant, Compressor House, No. 1 and No. 4 Hoisthouses, Potable Water Treating Plant, and Change Rooms.

NO. 1 AND NO. 4 YARDS -- cont'd

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.

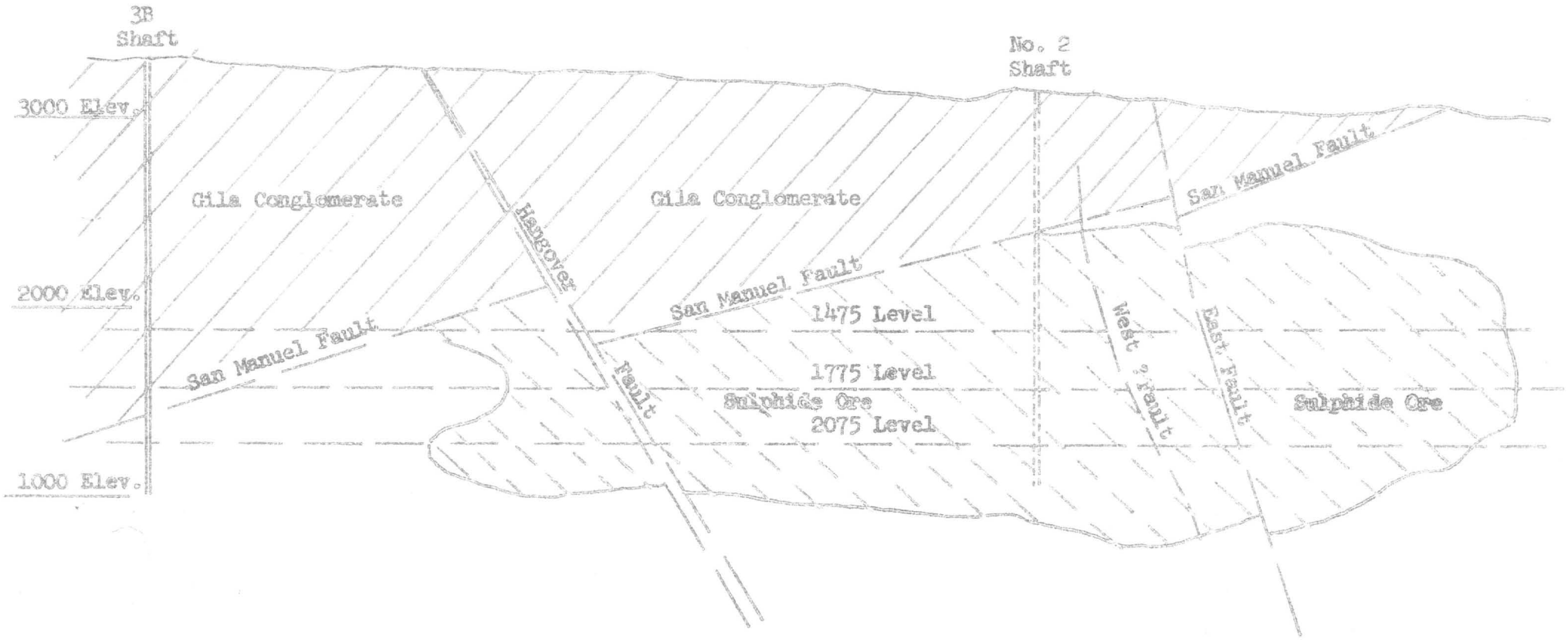
Timber: 12" x 12" drift timber.
2" and 3" lagging, various lengths.
6" and 8" cribbing, pre-framed.
Pole posts - Texas pine for undercut timber.

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.

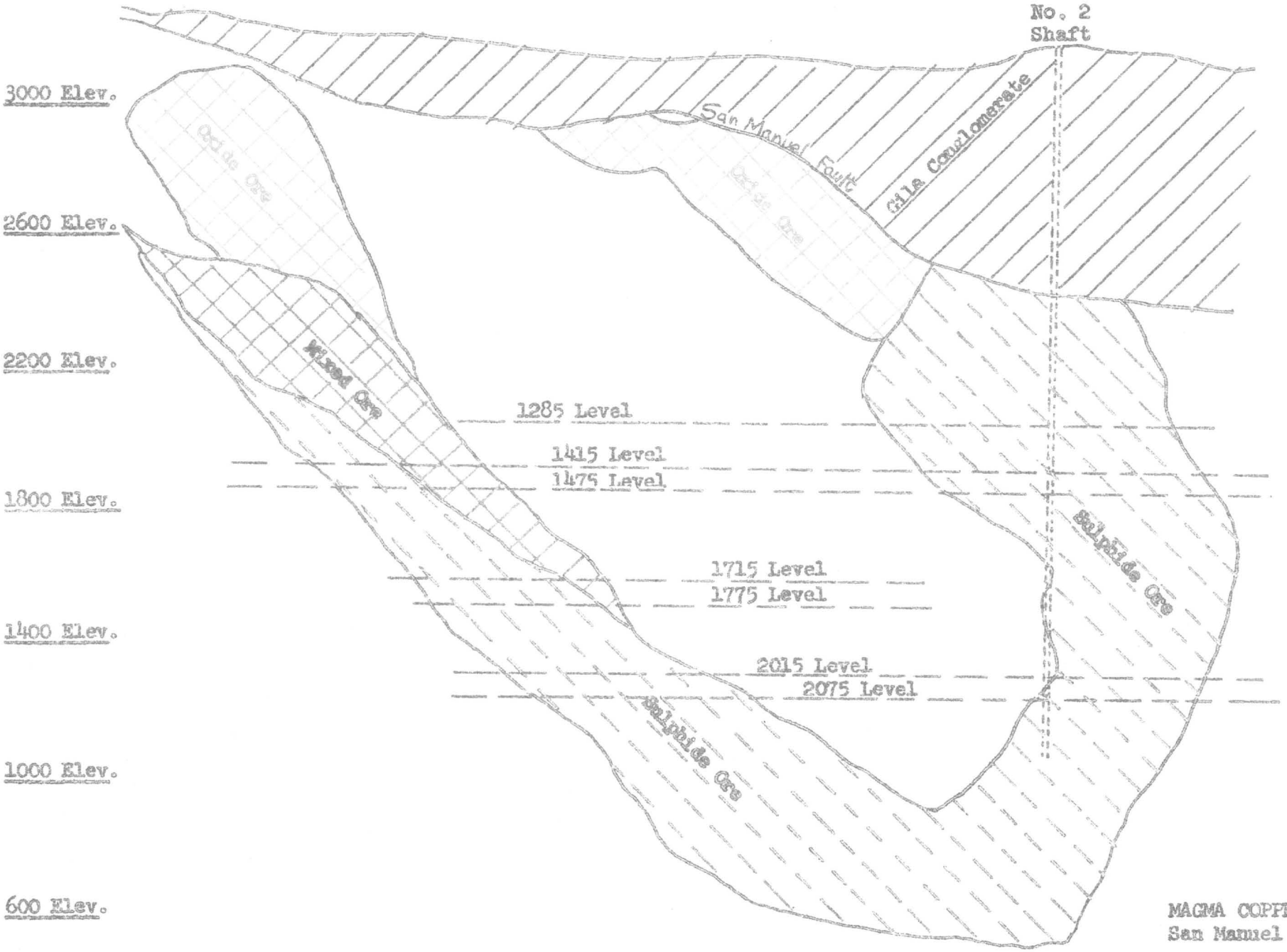
METAL PRODUCTION SAN MANUEL ONLY for 1966

Average Daily Mine Production -- Dry Tons	41,500
Pounds Copper recovered per Ton of Ore Mined	13.9
Copper, pounds	202,779,403
Molybdenum Sulphide, pounds	7,088,366
Gold, ounces	22,396
Silver, ounces	311,699



MAGMA COPPER COMPANY
San Manuel Division

LONGITUDINAL SECTION THROUGH No. 2
AND 3B SHAFTS, LOOKING NORTHWEST.



MAGMA COPPER COMPANY
San Manuel Division

CROSS-SECTION THROUGH
OREBODY N46 W
LOOKING NORTHEAST

CHANGES IN MINING SEQUENCE

TO COPE WITH GROUND WEIGHT

AT SAN MANUEL

by
J. D. Pelletier
Planning Engineer

MAGMA COPPER COMPANY
San Manuel Division
San Manuel, Arizona

November, 1968

CHANGES IN MINING SEQUENCE TO COPE WITH GROUND WEIGHT AT SAN MANUEL

The forces of ground weight resulting from block caving have been a dominant influence in shaping mining practice at San Manuel. These forces play a dual role. On the one hand they may cause extensive damage to mine openings in the area affected by weight. On the other hand they contribute in large measure to the success of the operation by breaking the ore to an easily handled size, by working against the natural tendency to pipe, and by resulting in the extraction of large quantities of ore from pillars and other areas beyond the vertical limits of caving.

Patterns of Ground Weight

With few exceptions, ground weight at San Manuel is associated with the caving action and is not of great consequence in areas more than a short distance from blocks under active draw. Weight usually is noted shortly after undercutting of a block has started. A few blocks in exceptionally weak ground, however, may show signs of weight during development when the draw raises and undercut drifts are driven. (Fig. 1) This development is on close spacing and accounts for about one third of the support area under the block. As undercutting advances across a block, weight from vertical force is noted for distances up to 20 ft in advance of the undercut front. The weight increases with the size of the undercut and is noted on the underlying grizzly level as well as on the undercut

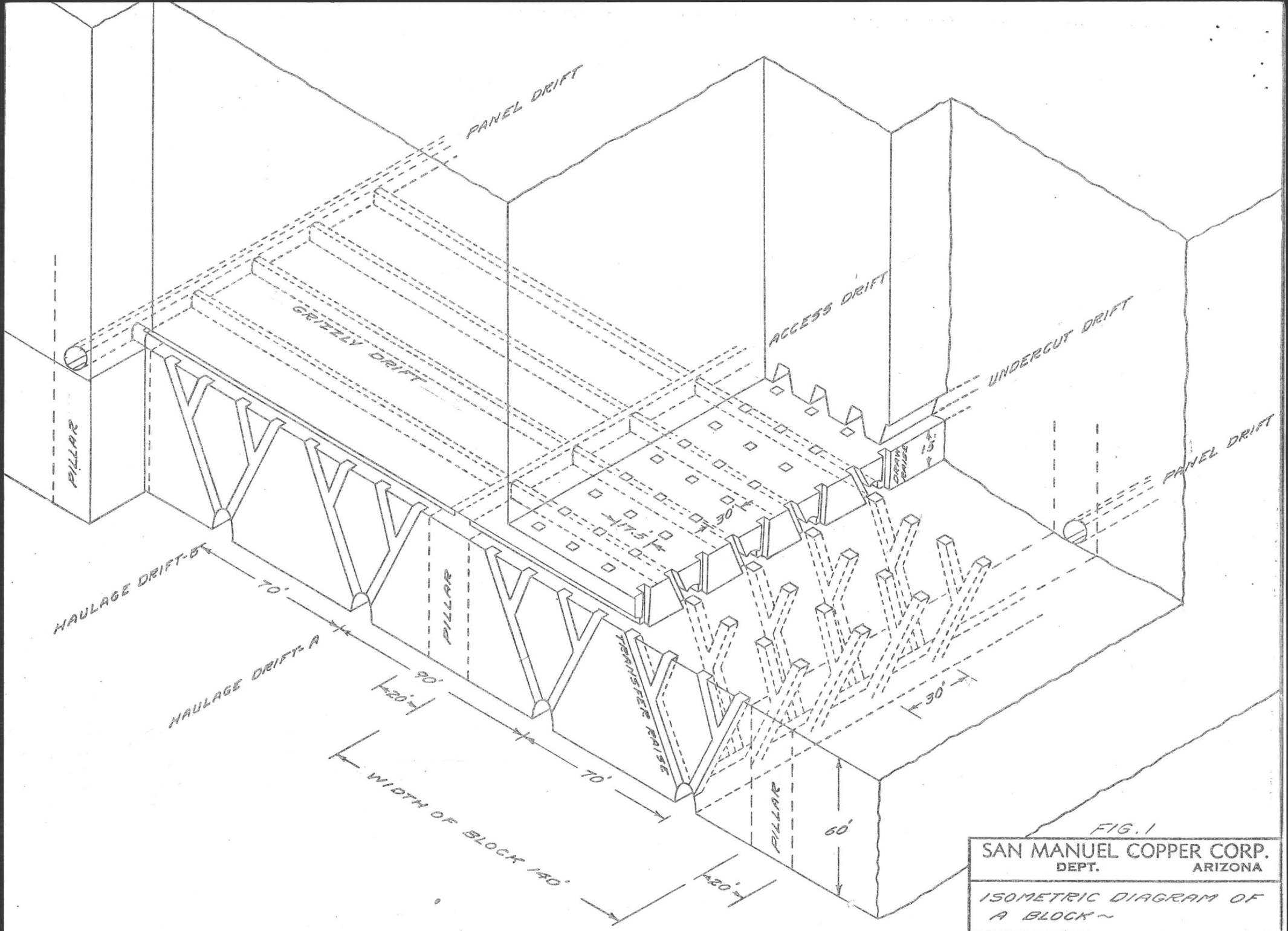


FIG. 1

SAN MANUEL COPPER CORP.
DEPT. ARIZONA

ISOMETRIC DIAGRAM OF A BLOCK ~

DRAWN H.H.R	DATE	DWG. No.
CHKD.	DATE	A

level. The weight is relieved as the undercut front passes over and draw is begun. To prevent extensive damage from the weight associated with undercutting, the final development of a block from driving draw raises through undercutting is accomplished in as short a time as practicable. General practice is to start the undercut in the weakest ground and retreat toward the strongest ground. Where protection of an adjacent panel drift on the grizzly level is important, the undercut is started adjacent to the panel drift and retreats from it if this is possible.

After an undercut has been completed, varying amounts of weight are generally noted around the perimeter of the block. The intensity of this "abutment" weight diminishes with distance from the edge of the block and is not ordinarily significant at distances greater than about 50 ft from the block. Abutment weight appears as a steady pressure on the grizzly level panel drifts adjacent to some of the blocks. (Fig. 2) As evidenced by the failure of the ground support in some of these drifts, the principal stress is directed downward at a moderate angle from the side of the drift away from the block and toward the side of the active block. Where blocks are simultaneously drawn on both sides of the panel drift, the stress is nearly vertical if the blocks have equal draw or, where the draw is not equal, steeply downward toward the block with greater draw. (Fig. 3)

Weight on access drifts adjacent to active blocks is similar to the pattern on panel drifts. The effects of the weight are usually less severe, however, since the access drift is a smaller section and able to withstand greater stresses than the larger panel drift.

Abutment weight often develops on the outermost grizzly drifts of a block as shown by varying amounts of lateral closure of some of these drifts. The principal stress appears to be directed horizontally from

FIG. 2

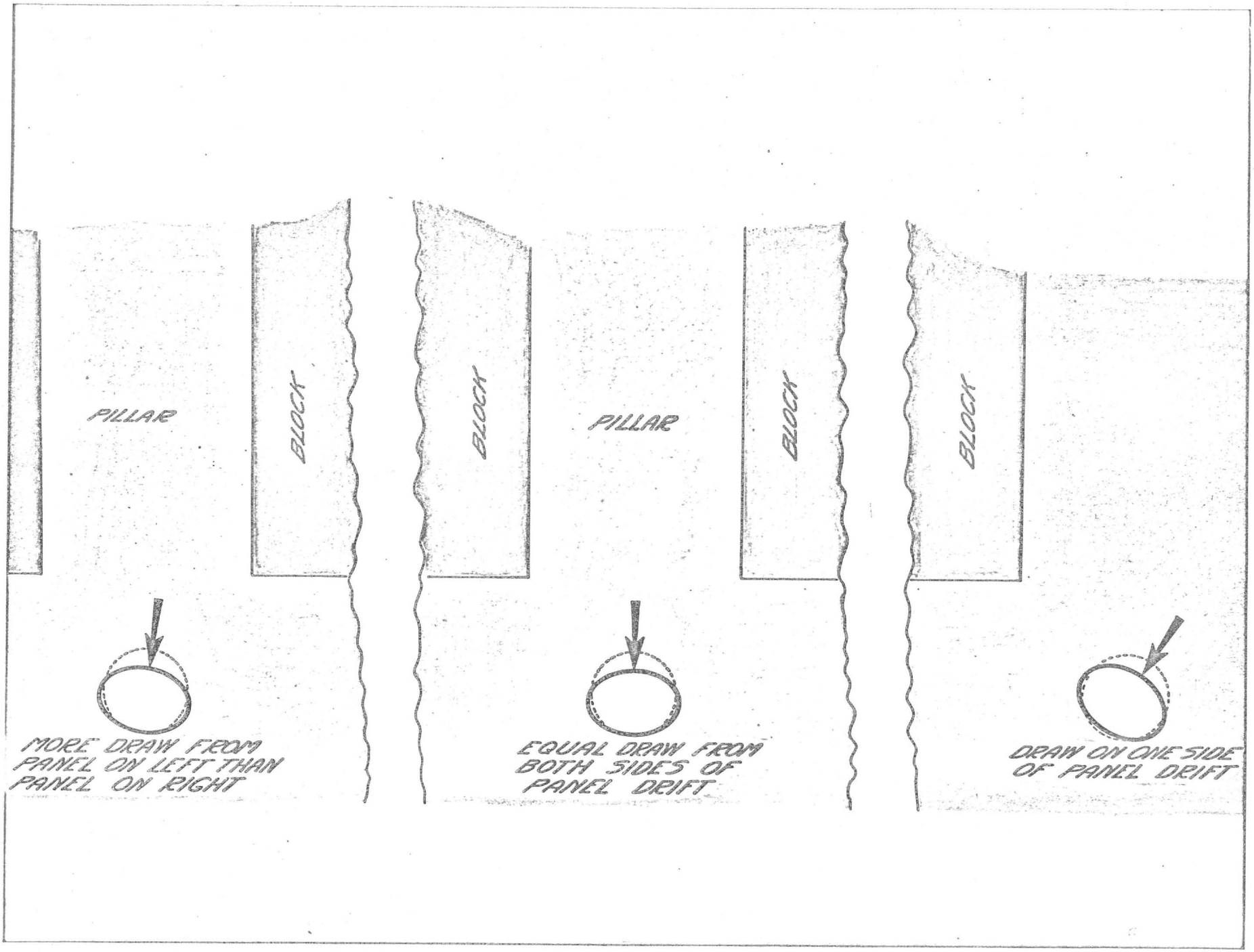

PILLAR
BLOCK
MORE DRAW FROM PANEL ON LEFT THAN PANEL ON RIGHT
BLOCK
PILLAR
BLOCK
EQUAL DRAW FROM BOTH SIDES OF PANEL DRIFT
BLOCK
DRAW ON ONE SIDE OF PANEL DRIFT

FIG. 3

the side of the drift facing the outside edge of the block toward the side of the drift facing into the block. (Fig. 4). Abutment weight on other grizzly drifts within a block is shown by a crushing of the ends of some of these drifts beyond the outermost draw raise positions, or, more rarely, extending to the second raise position or further from the edge of the block.

Draw raise entrances along the grizzly drifts are subject to damage from secondary blasting and transfer raises are subject to wear from abrasion. The weakening of this development by blasting and abrasion is often aggravated by ground weight irrespective of location under a block. This weight can be as intense as the weight near the perimeter of the block. Occasionally, weight on the grizzly drifts can be related to a large mass of ground settling on the undercut or to a relict undercut pillar, although both of these occurences are rare.

Accurate level surveys show a rise of ground occurring beneath a block after it has been undercut and a drop in ground beyond the perimeter of the block. The rise beneath the block amounts to several inches near its center. The drop beyond the perimeter of the block is greatest near the perimeter and diminishes with distance from the perimeter. Decreases in elevation amounting to two inches have been measured at distances of about 150 ft from a block where evidence of drift failure is not apparent. Decreases in elevation as great as six inches have been measured in panel drifts adjacent to active blocks.

Weight on the haulage level is less than on the grizzly level, bearing out the pattern of diminishing weight with distance from the active draw. On the 1475 Haulage Level, where panel haulage drifts were timbered, weight was sometimes noted just beyond the vertical projection of the block limits.

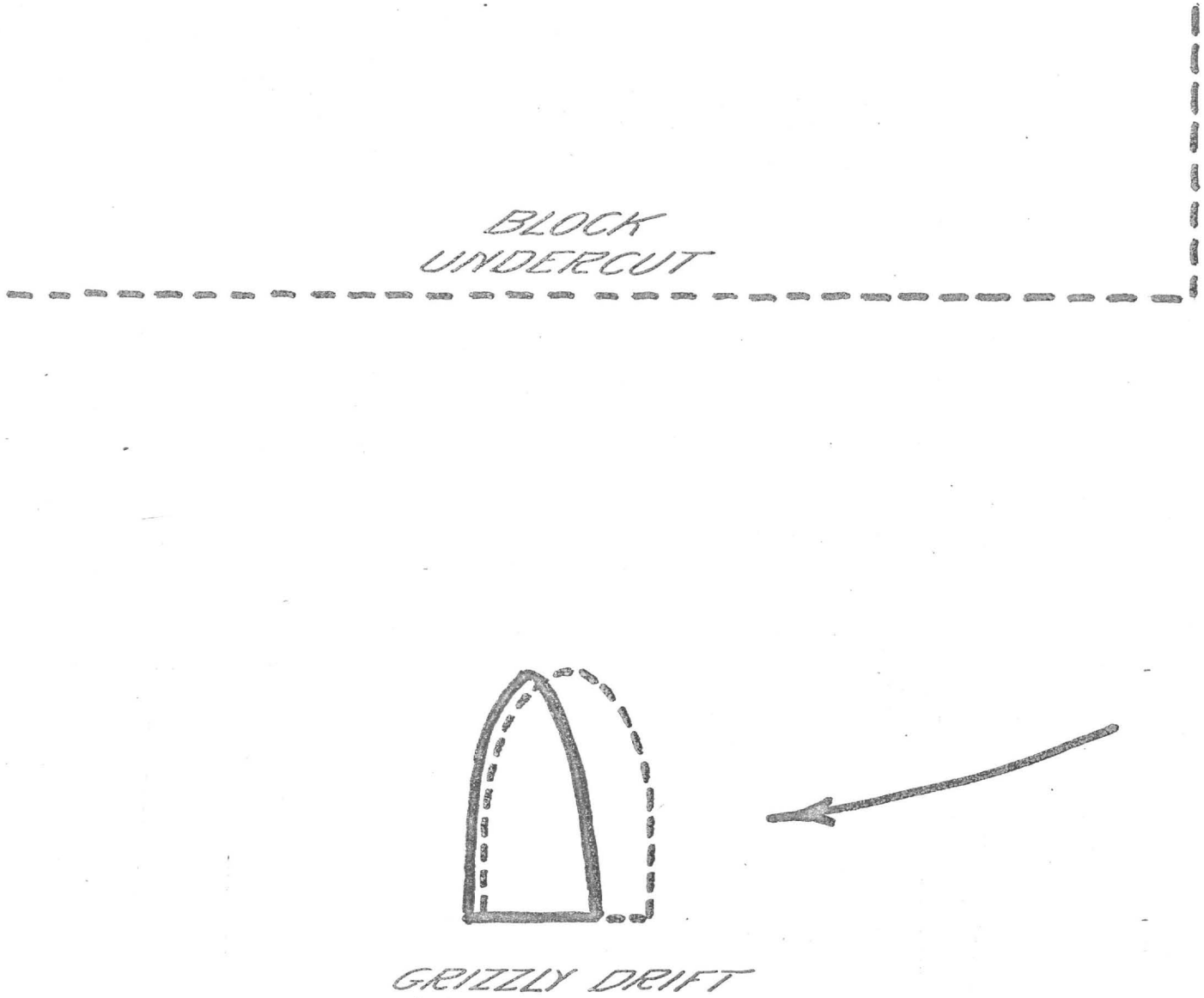


FIG. 4

Although this weight was not generally severe as compared with weight experienced on the grizzly level, in several instances where two active blocks in the same panel were separated by a large pillar such as provided by an unmined block in the checkerboard plan of mining, the entire sections of panel drift through the pillar area were subjected to weight. The effects of this type weight have largely been eliminated through the use of concrete linings in the haulage panels.

A feature of the ground weight is that its intensity is not uniform and that its occurence in sufficient intensity to damage drifts and other mine openings cannot be accurately foretold. A tendency is noted for the severest weight to occur in areas of the strongest ground, although equally severe weight is evident in areas of weak ground in certain instances. Since strong, competent ground tends to cave in coarser fragments than weak, fractured ground, the greater arching capabilities of coarse muck may account for the more frequent occurence of severely intense weight in areas of competent ground.

Another feature of the ground weight is that it does not persist long after draw is discontinued, but usually lets up dramatically within a day or two after completion of draw from the affected area. Under these circumstances it would be expected that a fast rate of draw would result in more intense weight than would a slow rate of draw; however, no consistent relationship can be established between rate of draw and intensity of weight when considered on a block to block basis. Statistically it is difficult to tie the rate of draw to intensity of ground weight with confidence for several reasons. First, while it is usually possible to relieve the weight vertically beneath an area by drawing fast, the effects of the fast draw on adjacent areas, especially the abutments of a block, are not clear. Second,

blocks in coarse, competent ground, which normally exhibit weight, are necessarily drawn more slowly than blocks with easily run muck, which generally exhibit less weight, and the cause of the weight may be related more to the condition of the muck than to the rate of draw. Finally, periods of widespread, intense weight in the mine have coincided with periods when more blocks than actually needed were undercut and under draw, and when development of future blocks adjacent to the active draw was far advanced. This condition necessarily results in a less-than-average draw rate for the mine as a whole while at the same time providing the greatest exposure for damage from the effects of weight.

It is generally believed that the pattern of weight outlined above results when a cavity, or zone of loosened rock, is created by the draw and the unsupported weight of the rock column overlying the cavity, or zone of loosened rock, is borne by the surrounding ground. (Fig. 5). At San Manuel, where the rock column is large as compared with most other block caving mines, it is rare for a block to cave to the surface. Rather, the surface subsides over a broad area and the amount and distribution of subsidence is related to the draw from all of the active blocks considered as a unit even though the blocks may be widely separated. The consistently close relationship of the volume of subsidence to the volume drawn indicates that expansion of the subsiding ground is not great and that voids or large zones of loosely broken rock are rare. On several occasions, when the undercut of a completed or partly drawn block was re-entered by raising and drifting, it was found that re-compaction of the fragmented rock was nearly complete a short time after the draw was stopped. These observations lead to the conclusion that the active draw zone of a block does not extend more than several hundreds of feet above the undercut and that the overlying rock columns causing weight are large.

Figure 5. (THIS WILL BE EXPANDED TO 8" x 10" SIZE FOR FINAL COPIES.)

Weight Problems with Increasing Mining Depth

In April 1962, San Manuel began drawing ore from its second main level, the 2075 Level. This level is 600 ft below the first level and is at a depth of about 1,700 ft below the original ground surface. It was anticipated that the increase in mining depth would result in an increase in ground weight experienced in drifts adjacent to the mining areas. The correctness of this anticipation has been proven by subsequent events.

To compensate for the increase in weight, all second level drifts under the caving areas, both on the grizzly level and on the haulage level, were concrete lined with a minimum thickness of 18 in. on the walls and 24 in. on the back. (Fig. 6 and 7). The practice of concreting the drifts was expanded to include also the drifts through a zone of weaker ground extending several hundred feet north of the initial mining area. The reliance on massive concrete support, backed up with heavy steel repair sets, had worked well in dealing with most weight problems on the first level. On the second level, although the concrete lined drifts have served very well on the haulage level and in the area of weak ground north of the ore zone, this support has not everywhere been sufficient to withstand the ground pressure on the grizzly level near the perimeters of active blocks.

As ground weight developed on the second level, it was realized that the increase in ground support necessary to withstand the heavy pressures encountered near the perimeters of some caving blocks was not practicable, and that the solution must lie in changing the block lay-out or sequence to avoid an excessive concentration of ground pressure. To date there have been three sequence plans used, and a fourth is now contemplated.

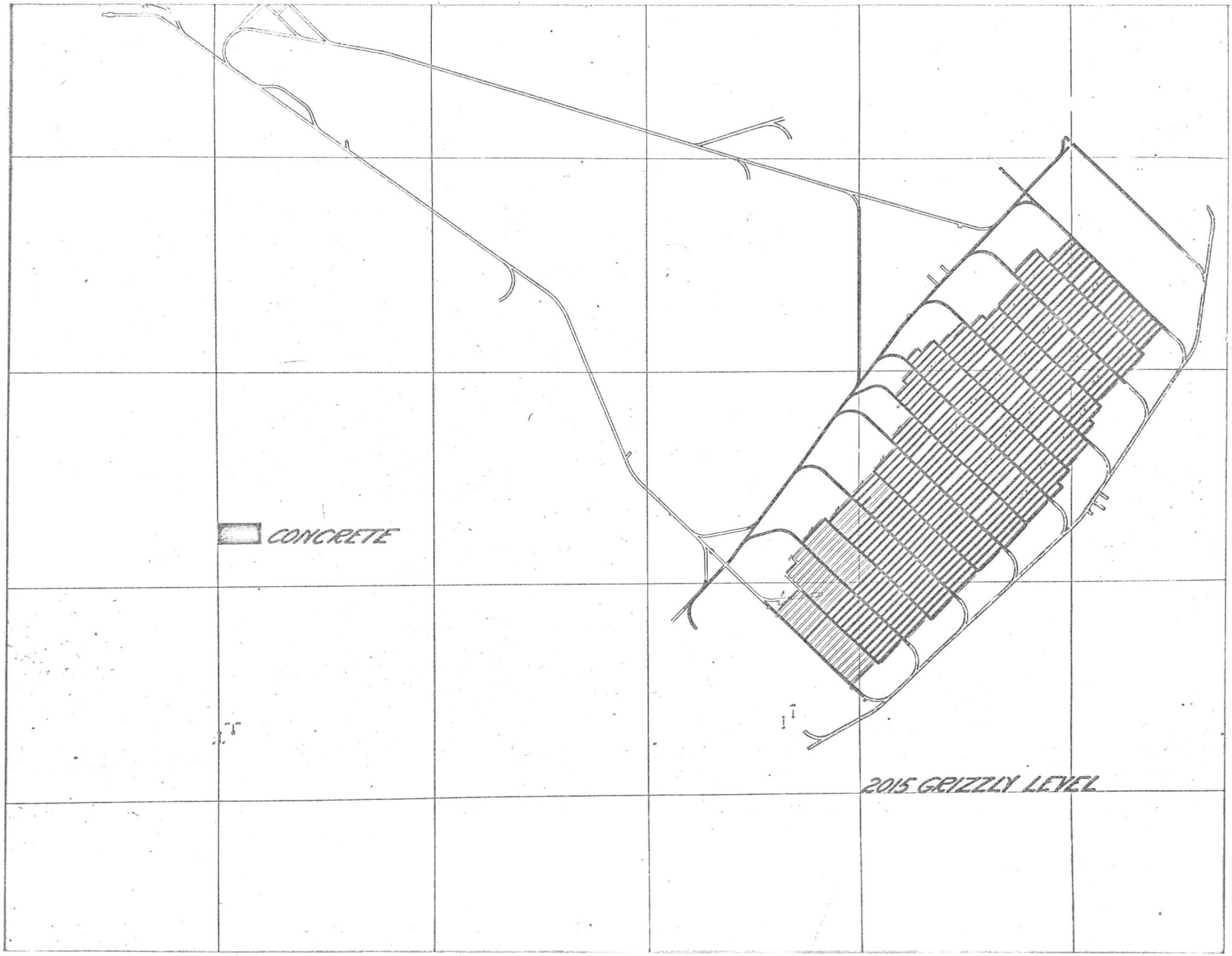


FIG. 6

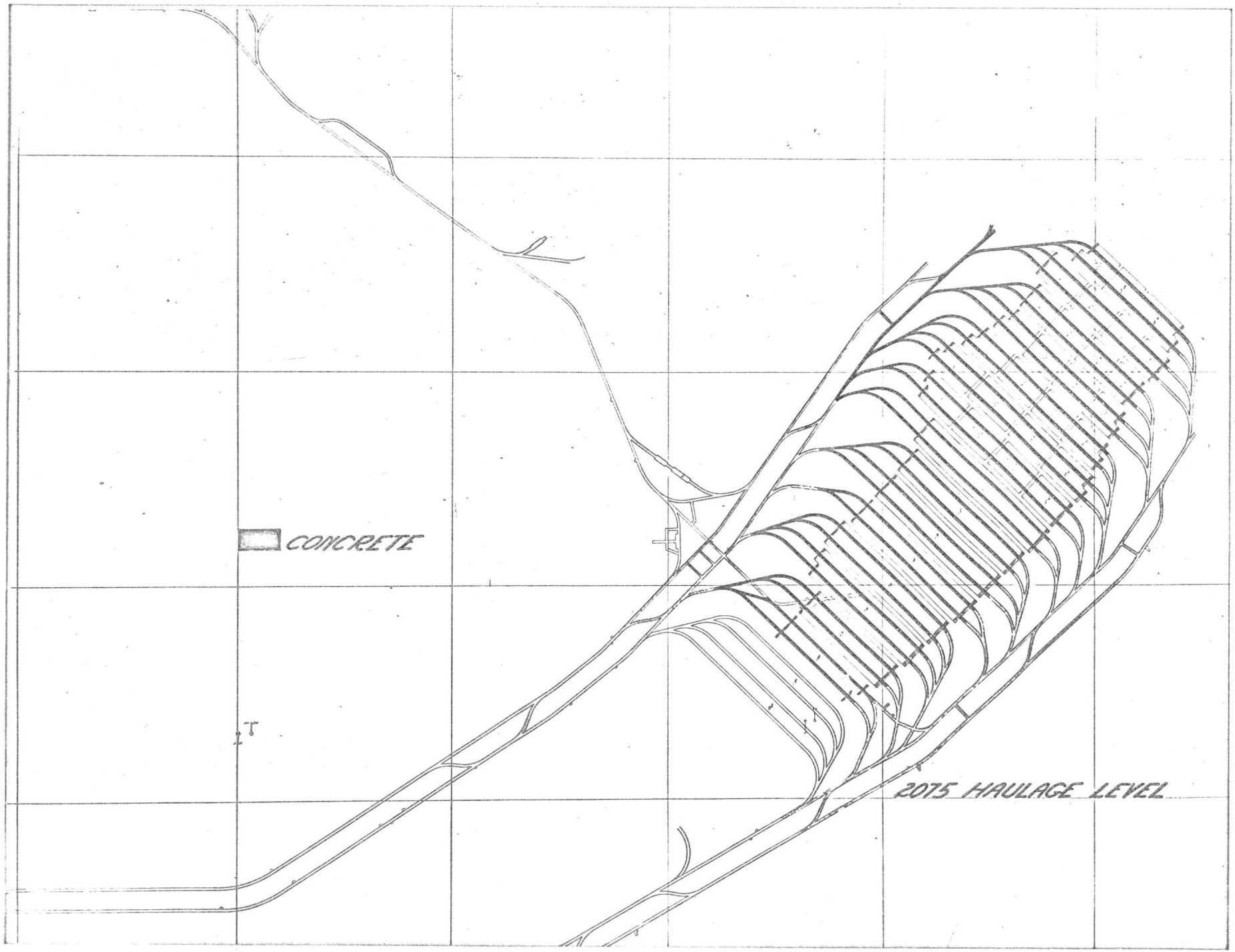


FIG. 7

I - Initial Checkerboard Pattern

The layout and sequence of the initial mining area was a standard "checker-board" plan which had been developed on the first level and had proven very workable. (See Fig. 8). Briefly, the plan calls for dividing the ore zone into panels 140 ft wide separated by pillars 20 ft wide. The panels are normally divided into blocks of five to eight grizzly lines to form convenient working units and fit the width of the orebody. The blocks are separated from each other within the panel by 15-ft wide pillars which serve as a barrier to separate the waste in a completed block from the ore being drawn in an adjacent block. The panels are mined in two "passes" with alternating blocks undercut on the first pass and the remaining alternating blocks undercut on the second pass. Blocks in adjacent panels are also alternated with each other to form the checkerboard pattern. This system has several advantages, among which are: first, an active block is surrounded by inactive blocks which provides a convenient way to exhaust ventilation air from the active block to the haulage level through transfer raises in an inactive adjacent block; second, the caving action is spread out over the maximum area for a given pre-production development; third, more than half the ore is drawn from blocks with all sides against unmined ground; fourth, the loading operation is conveniently spread out in the haulage panel drifts; and fifth, the entire panel can be abandoned in the time it takes to mine two 'passes'.

Six blocks in three panels were undercut in the checkerboard pattern in a five-month period ending September, 1962. The six initial blocks had an undercut area of 165,900 sq. ft., which was sufficient to bring production from the 2075 Level to half the total mine production at a draw rate of about 16 in. per day.

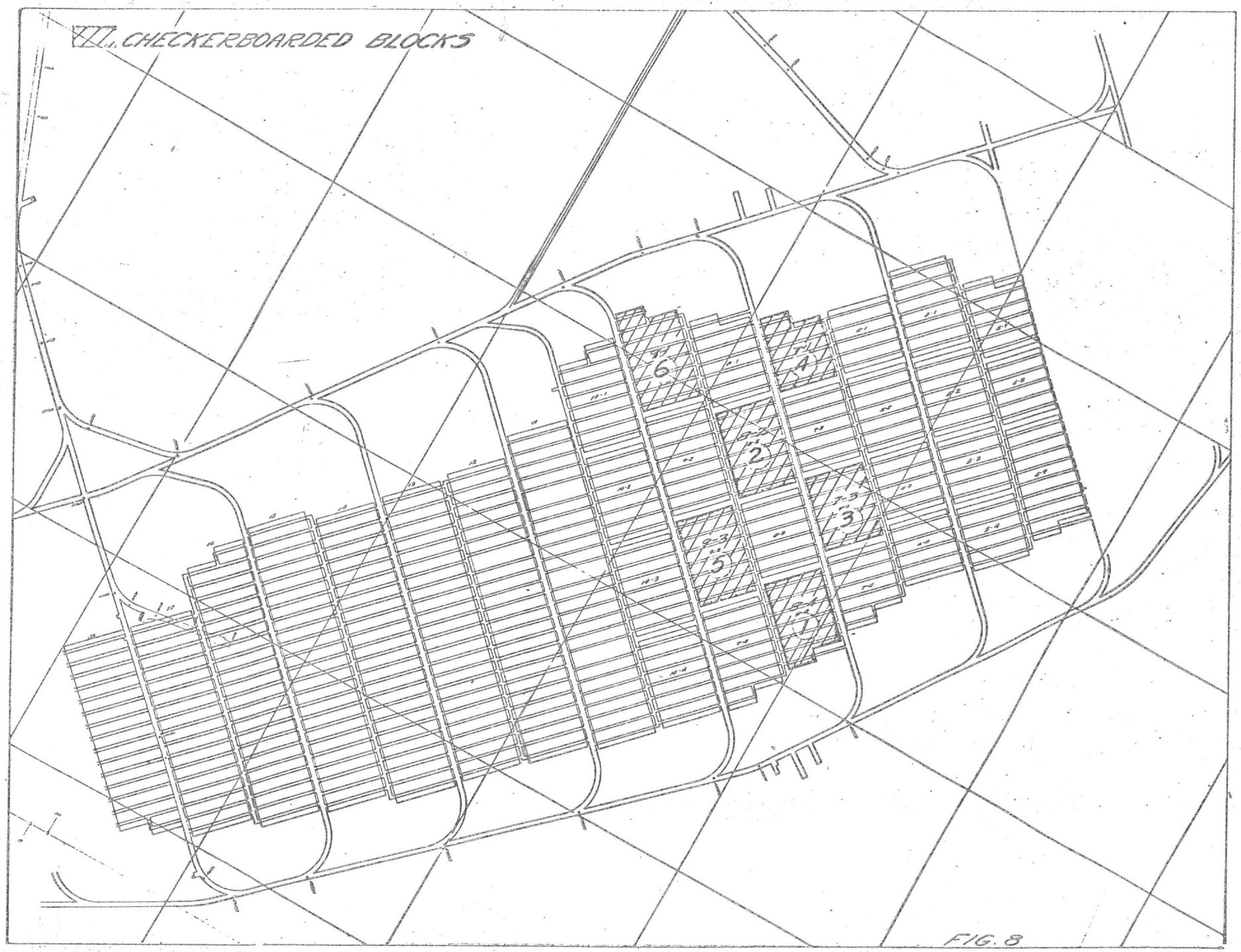


FIG. 8

Some weight was experienced on grizzly drifts as the undercut advanced over them. This weight is normally expected during undercutting and is not usually too serious if the drift withstands the initial shock, as it is generally relieved as draw is begun. Such was the case in the first six blocks on the second level. The rock in the northern two thirds of the ore zone was hard and blocky, and this resulted in considerable transfer raise repair, not primarily because of weight, but because of abrasion. In many instances the blocky condition of the caved ground did not lessen with continued draw, but persisted throughout the life of the blocks.

A very serious type of weight appeared as the second block was being undercut. This took the form of a heavy, persistent squeeze on Panel Drift 8 in the general area between the first two blocks, 8-4 and 8-2. Weight on a panel drift is particularly serious because it threatens to close the access for development, repair, ventilation, and servicing of the draw in blocks adjacent to the weight. Repair was immediately started in the affected area of Panel Drift 8 by installing yieldable steel rings on five-ft centers. The rings were doubled and packed with sacked concrete as the weight continued and the third block, 7-3, which abutted the heavy section of Panel Drift 8, was undercut rapidly in the hope of relieving this weight, or at least of assuring completion of its undercut while this was possible. The weight continued, however, and spread out over the northwest corner of Block 7-3 and over the southeast corner of Block 8-2. In six months time from the initial appearance of weight, sections of Panel Drift 8 were caved completely and re-driving and re-concreting were started. Portions of a 300-ft section of Panel Drift 8 were re-opened several times over a two-year period, until the entire section was concreted grizzly drift size and this smaller, stronger section held. (Fig. 9).

FIG. 9

When the fifth block, 9-3, was undercut, the abutting section of Panel Drift 10 took weight and yieldable steel rings were installed. This weight persisted for some time, but the drift was held open a sufficient size for continued access to Block 9-3.

When undercutting of the sixth block had been completed, no further blocks were undercut on the Second Level for a period of eleven months, during which time the remaining blocks of the First Level were brought into production. Development of future Second Level blocks continued up to the point of driving draw raises and undercutting.

Extensive and continuing repair of transfer raises, grizzly drifts, and draw raise entries was required in Blocks 7-3, 8-2, and 9-1 where the coarse, blocky nature of the caved ground did not decrease with continuing draw. In Blocks 7-1 and 9-3, on the other hand, the draw did not remain blocky for long and repair did not become a serious problem.

In general, the weight pattern developed on the Second Level was similar to that on the First Level except in intensity, which was much greater on the Second Level. The perimeters of active blocks showed evidence of weight. The most intense weight of all was encountered where two blocks were diagonally opposed across a panel drift. However, the intense weight did not appear at every location where two blocks were diagonally opposed, and is rare where two blocks are diagonally opposed across a small access drift. The appearance of weight in Panel Drift 10 alongside Block 9-3 gives evidence of the greater intensity of Second Level weight, since an occurrence of weight around the outer perimeter of a group of blocks was practically unknown on the First Level.

II Change to Doubled Blocks

The great intensity of weight on the panel drifts made it imperative that the mining plan be changed to relieve this weight as far as possible. To this end, it was decided to modify the checkerboard pattern to the extent possible consistent with development work already completed. This was accomplished by mining the ensuing group of blocks in pairs side by side across the panel drift. (Fig. 10). It was reasoned that by drawing on both sides of the panel drift, some relief of weight on the drift could be expected and the potential intense weight where blocks were diagonally opposed could be eliminated with respect to the panel drift if not the access drift. This change retained most of the advantages of checkerboard block caving. It could be fit in with development already completed, but it meant that the mining of some blocks that had been developed through concreting their grizzly drifts would be postponed to a future date.

The doubled blocks, 6-4 and 5-4, and 6-2 and 5-2, were undercut in the last half of 1963 when all development on the First Level had been completed and production from that level could be expected to decrease. Panel Drift 6 passing through the doubled blocks was reinforced with yieldable steel rings prior to undercutting. In the case of Blocks 6-2 and 5-2, raises were inclined over the panel drift from the undercut level and the ground over the panel was shot and drawn to some extent before the advance of the undercut precluded further draw from over the panel. The undercuts retreated from the panel drift toward the access drifts nearly simultaneously in both doubled blocks.

Some weight was taken by Panel Drift 6 in the area between the doubled blocks, but damage to the drift was not extensive and the weight let up shortly after the undercuts were completed. Persistent weight soon became

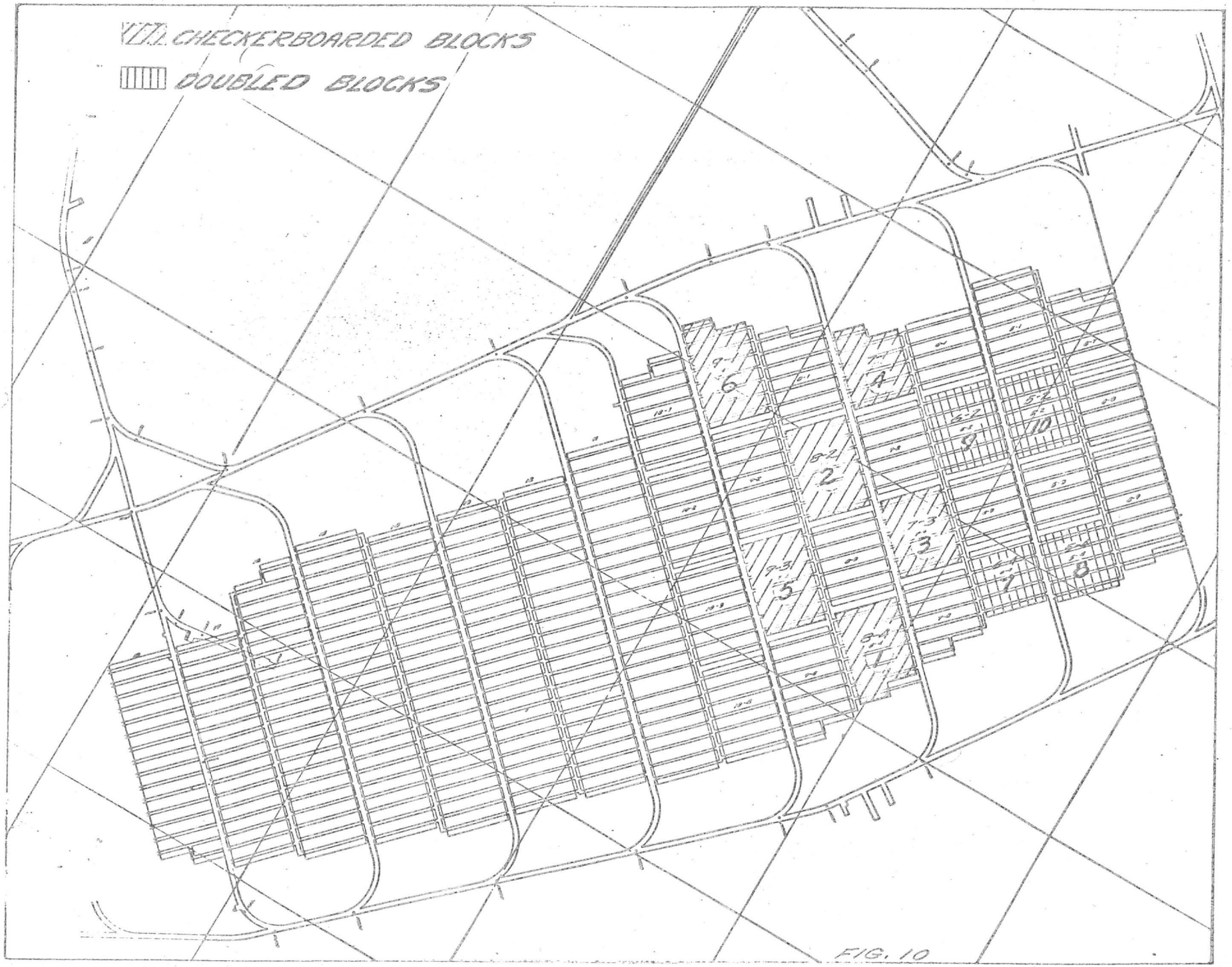


FIG. 10

evident along the south edge of Blocks 6-4 and 5-4 and along the north edge of Blocks 6-2 and 5-2. A 20-ft section of Panel Drift 6 just south of 6-4 and 5-4 became the locus of very intense weight which resulted in closing this section of drift early the following year. It was necessary to re-open this section of drift several times before the weight finally receeded as the draw from the adjacent blocks neared completion. It was thought that a fault structure at this point may have localized the weight. No important faults are known at this location, but many small structures transect the ore zone and this possibility cannot be ruled out.

A section of Panel Drift 6 extending some 30 to 40 ft north from the edge of Blocks 6-2 and 5-2 similarly took heavy and persistent weight. Continuing repair was required to hold the section of drift open. The weight in this area appeared to be spread over a broader area than had been the case south of Blocks 6-4 and 5-4, as evidenced by a more intense squeeze on the grizzly drifts abutting the affected area.

It was theorized that the unusually long unbroken caving front of 300 ft presented by the doubled blocks may in some way bear on the weight that developed. It is noted, however, that there are many other places where long caving fronts were developed without similar concentrations of weight. A case in point, of course, is the north edge of 6-4 and 5-4, and the south edge of 6-2 and 5-2.

All told, the experience with doubled blocks seemed to offer some improvement over the checkerboard pattern with respect to weight, and it was decided to extend the doubled block pattern where possible to the remaining blocks from Panel 5 through Panel 10. At the same time it was resolved to work out a full panel caving system for the blocks west of Panel 10, where development had not progressed to an extent that would preclude this possibility.

The doubled blocks 9-4 and 10-4, 5-1 and 6-1, and 5-3 and 6-3 were all mined without experiencing excessive weight on the adjacent panel drifts, although considerable repair of these drifts was necessary. (Fig. 11). When Block 9-4 was developed for undercutting, the first undercut drifts were driven parallel to the grizzly drifts, rather than at right angles which is the normal practice. This was done on the theory that, by not crossing the grizzly drifts, weight would not be thrown on them. However, the opposite affect resulted, with very heavy weight carried by the pillars between undercut drifts and thus onto the grizzly drifts, and the practice was discontinued when about one third of the block had been undercut in this way.

When the doubled blocks 9-2 and 10-2 were undercut, serious weight developed on the footwall side of a major fault structure in Block 10-2. The undercut had been started over the panel drift between the blocks which lay on the footwall side of the fault. It is believed that a large block of ground broke loose from the footwall of the fault as the undercut advanced toward the fault and the shifting of this ground resulted in the weight which extended from the fault in Block 10-2, across Panel Drift 10, and halfway across Block 9-2. (Figure 12).

A further complication to the weight problem developed in the last half of 1964 and the first half of 1965 as a result of an unexpectedly large overdraw from the blocks in the initial second level mining area and as a result of the changes in sequence that had been made. It is normal practice to schedule block development so that it is completed at approximately the time that the block is needed for production. It is also preferable to have blocks ready for undercutting whenever an uncertainty exists as to when they will be needed, rather than risk not having

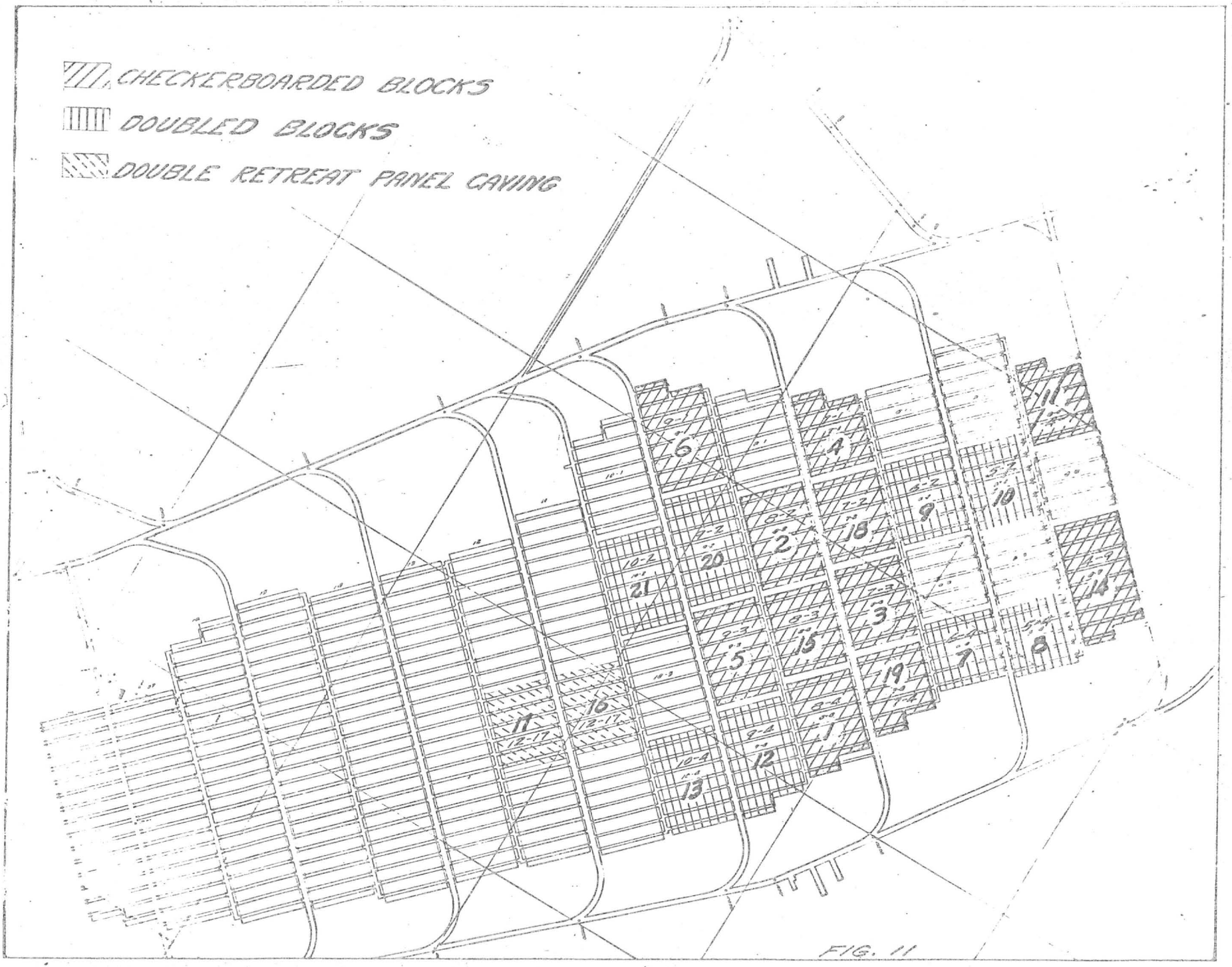

CHECKERBOARDED BLOCKS
DOUBLED BLOCKS
DOUBLE RETREAT PANEL CAVING

FIG. 11

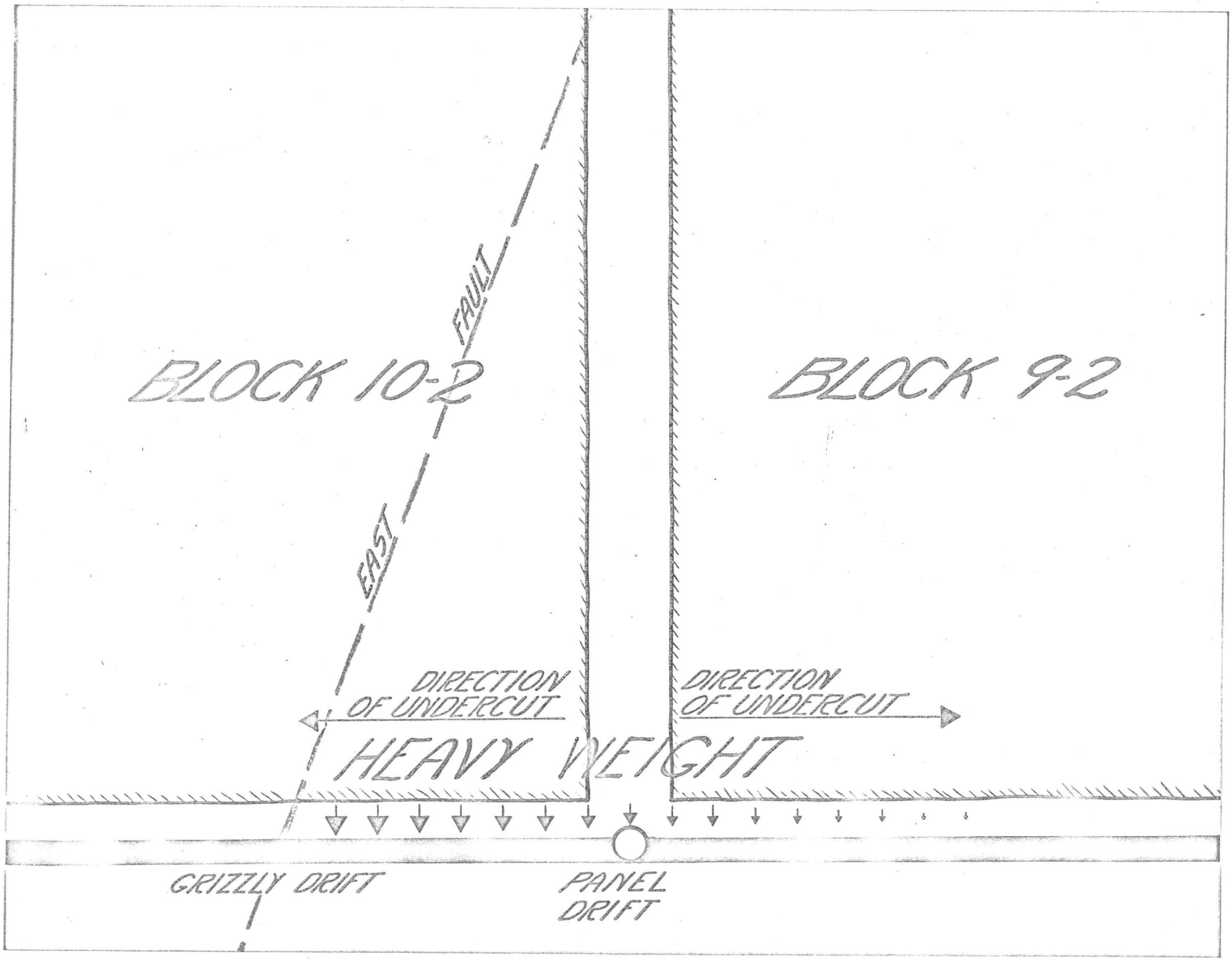

BLOCK 10-2
BLOCK 9-2
EAST FAULT
DIRECTION OF UNDERCUT
DIRECTION OF UNDERCUT
HEAVY WEIGHT
GRIZZLY DRIFT
PANEL DRIFT

FIG. 12

enough development to maintain production. However, the overdraw of the first blocks was so far beyond expectations that a larger than normal number of blocks were pre-developed and awaiting undercutting. Since this development was adjacent to the active blocks, it was subject to weight caused by the caving. In some instances, the damage resulting from this weight was extensive and considerable repair was required.

One way to protect this development from further damage is to undercut the block as soon as repairs are completed. Several blocks were undercut for this purpose, but the resulting slower average draw rate meant that some blocks could not be drawn as well as they should for optimum extraction and weight control. Consequently it is considered self-defeating to go too far with advance undercutting to protect pre-developed blocks. Best results with San Manuel rock have generally been obtained with an average draw rate of about 18 in. per day. Whenever the average draw rate has dropped much below 16 in. per day, repair costs have begun to rise. There are usually some blocks that can be shut down entirely without hurting their extraction or causing weight. When such blocks are available, they have sometimes been shut down and placed in "reserve" to enable other problem blocks to be better serviced.

III - Change to Double Retreat Panel Caving

At the time a decision to work toward a full panel caving system was made, the area west of Panel 10 was not developed to an extent that would preclude such system and this area was designated for panel caving. A significant change in ventilation practice was necessary, since transfer raises would not be available for exhausting air from active grizzly drifts to the haulage level. Special vent raises were driven from the

ends of pony drifts over the haulage panels to the access drift on the grizzly level. This was done with some trepidation since this ventilation development has to be fitted between production development and tends to further weaken the ground beneath the panels. No ground weight has been experienced because of the weakening, however, and this vent system has served very well.

It was also decided to eliminate pillars between undercut units within a panel. Such pillars were already committed in Panels 11 and 12, but were undercut with their adjacent draw units and have not been the site of unusual weight concentrations.

The plan of sequence was to begin with two doubled blocks in Panels 11 and 12. (Fig. 13). The initial blocks were six grizzly lines long, or 180 ft, to assure sufficient width to establish a good cave. The blocks were centrally located within Panels 11 and 12 with retreat planned toward both the north and south limits of the panels by undercutting units of approximately three grizzly lines each (about 90 ft long) until the panel limits were reached. The retreating units were to be undercut when the adjacent central unit was about one half extracted.

The initial cut in the next adjacent pair of panels to the west was also to be made when the initial cut in Panels 11 and 12 was approximately half extracted. The retreat pattern was to be similar to that of the adjacent panels to the east but lagging one undercut unit of about 90 ft from the undercut advance in those panels. This pattern was to be repeated as mining advanced to the west. In this way the over-all shape of the undercut area would develop as a V-shaped wedge with its point advancing toward the west and the central area between the legs of the V would be depleted and abandoned.

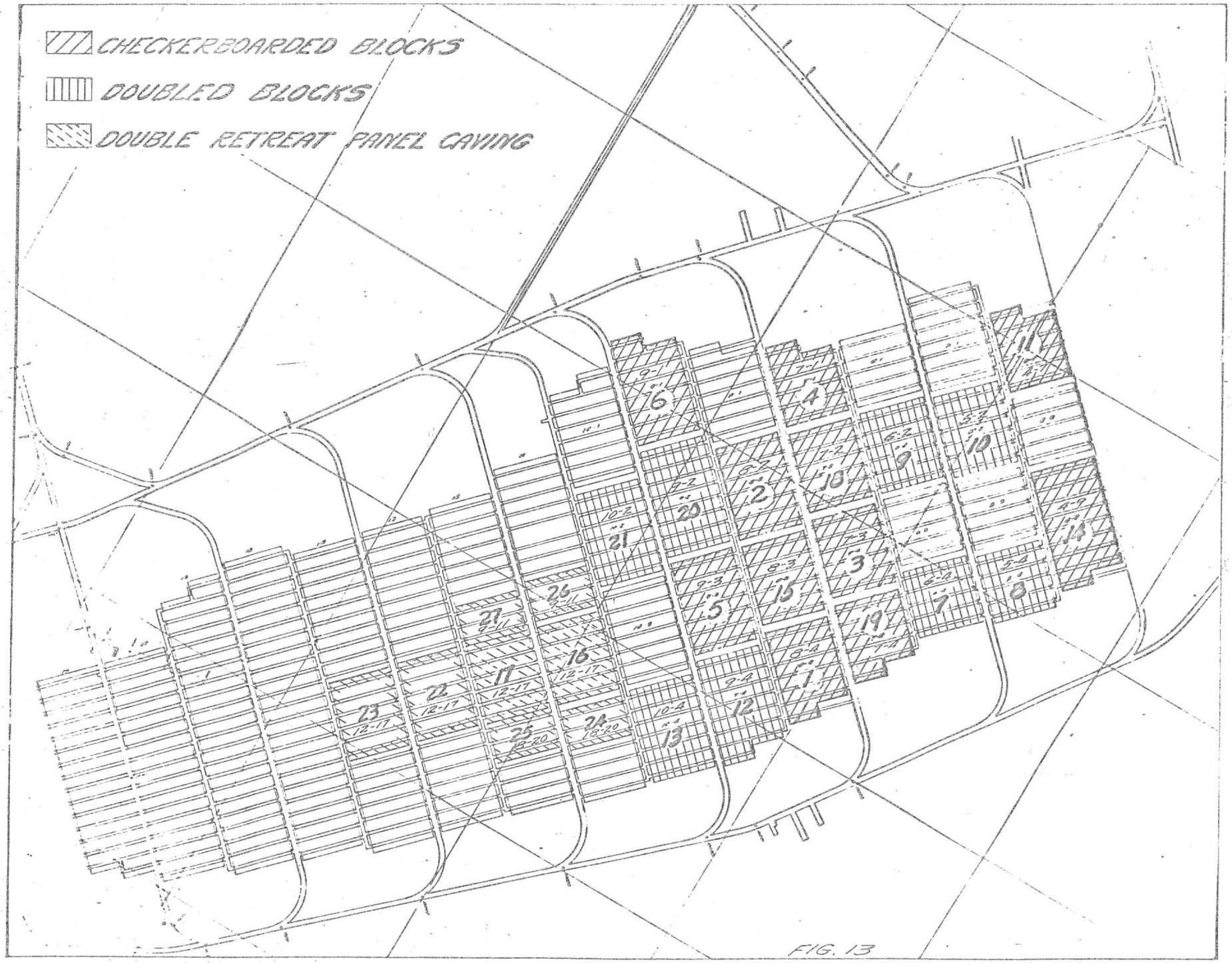

CHECKERBOARDED BLOCKS
DOUBLED BLOCKS
DOUBLE RETREAT PANEL CAVING

FIG. 13

This plan of sequence appeared to offer several advantages:

1. Block-size pillar areas within the panels, which had been the site of intense weight, were eliminated since these areas could be abandoned.
2. The loading advantages of spaced blocks within the panels, such as in the checkerboard sequence, would be largely retained.
3. The improvement in weight experienced on the grizzly panel drifts in the doubled block sequence would be retained.
4. The perimeter weight on pre-developed units could be relieved at will by advancing the scheduled undercut of these units as required.
5. Development access would always be maintained since development was always beyond the outer perimeter of the cave area.

This sequence was extended over an area of five panels. (The initial block of the fifth panel was not doubled with a block in the 6th panel.) (Fig. 14). To date the wedge sequence has been strikingly successful with respect to ground weight when compared with either the checkerboard sequence or the doubled block sequence. Some weight is taken by the panel and grizzly drifts during undercutting, as evidenced by a slight squeeze and moderate cracking of the concrete linings. This weight has not generally continued after completion of the undercut. The most severe pressure on the panel drifts has developed later adjacent to areas that were well along toward depletion and after the total area under draw had expanded greatly in size. Although this pressure has exerted considerable squeeze on the panel drifts in these areas, in no instance has it resulted in complete closure of a panel drift.

Perimeter weight on the outermost grizzly drifts has not been unduly

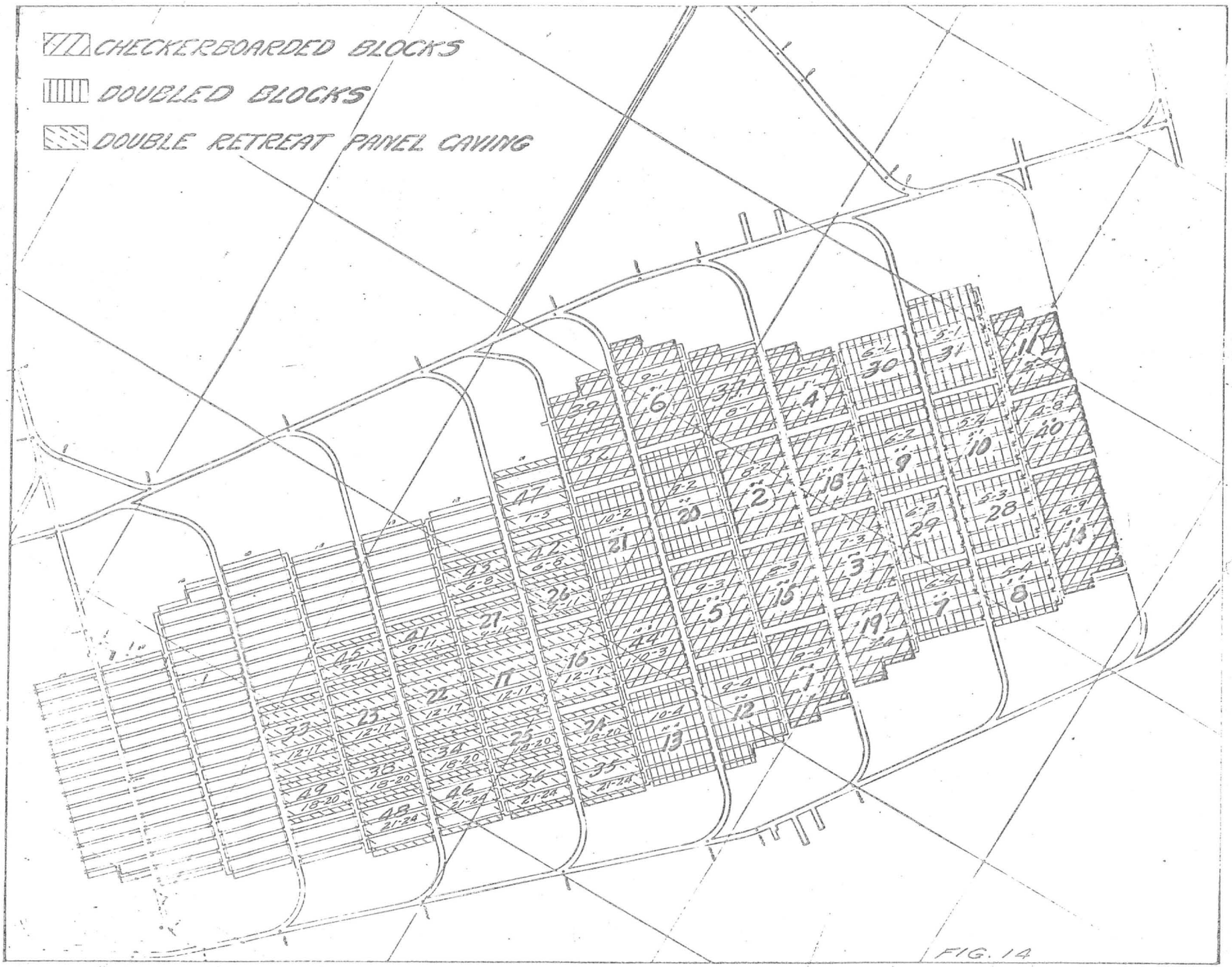

CHECKERBOARDED BLOCKS
DOUBLED BLOCKS
DOUBLE RETREAT PANEL CAVING

FIG. 14

severe. In only one instance the option to advance the scheduled undercut date of a retreat unit was exercised. In this instance the perimeter weight was promptly relieved.

With minor exceptions, the ground mined using the wedge sequence has been of greater than normal coarseness, in part because of the nature of the rock in the area, but also in part because of the lessening of perimeter weight on the advancing edges of the cave front. The less favorable fragmentation, however, can in no way be compared with the great advantage gained through the reduction in ground weight.

In several instances large boulders have set down over a group of draw raises. These boulders were drilled and broken before extensive damage could be done to the underlying grizzly drifts. The effects of secondary blasting in the grizzly drifts, coupled with some weight, probably accounts for most of the damage that is now experienced in these drifts.

It was expected that the change to a panel caving sequence would result in less disparity between the extraction from initial undercuts and the extraction from subsequent undercuts of adjacent units than had been experienced with other sequences. While this has proven true to some extent, the disparity in extractions is still large. Depletion of the initial blocks in the center of the panels has tended to lag nearly to the point of depletion of the next adjacent retreat units. This has resulted in a larger-than-intended number of adjacent grizzly lines under draw at one time (which has proven no serious disadvantage), and in the need to undercut adjacent retreat units at a larger spread in real terms of per cent extraction than is considered desirable for grade control.

IV - Proposed Change to Diagonal Retreat Panel Caving

To improve on the wedge sequence of caving it is proposed to convert to a sequence of diagonal retreat panel caving. The initial undercut of a panel will be at the south limit and retreat units will advance to the north. This will develop a diagonal cave front running from the south edge of the orebody on the west to the north edge on the east.

It is planned to take adjacent retreat units at a spread of one third to one quarter in extraction as compared with the one half spread in extraction required by the wedge sequence. It is also planned to discontinue the practice of doubling the retreat units across the panel drifts in respect to the decrease in perimeter weight now experienced.

It has been noted that the effect of perimeter weight, where present, does not extend more than 30 to 45 ft beyond the cave front. To take advantage of what perimeter weight there is, consideration is being given to decreasing the size of retreat units from three grizzly lines to two or one grizzly line if a workable method of doing that can be developed.

210-7

OUTPUT IN SUB-LEVEL CAVING

by Mr. Arne Gustafsson
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A new method for underground mining was introduced in Sweden at the turn of the century. In a Mining Engineer's Report of 1908 it was mentioned that "caving according to the American method" was now being practised. The method has been practised on an ever increasing scale during the past fifty years and is to-day dominating underground mining in Sweden. The reason why a caving method, which normally yields an unwelcome admixture of waste rock, has reached this prominent position is no doubt to be found in the fact that in Sweden iron ores which can be processed to high grades by relatively simple magnetic and gravity dressing, or such dressing and concentration, are mined on a large scale. The inevitable loss of ore has not made the method unsuitable as long as it is employed for ores having a low value in situ.

Profitability limit

It is of particular interest to learn how serious the disadvantages of admixture of waste rock and losses of ore are and a line of thought will be elucidated below, which will lead to numerical values for arriving at the most favourable conditions for the method.

It is most expedient to begin by determining the present cut off grade for the ore body with the existing plant. In sub-level caving special attention should be paid to the economic limit of the profitability when loading caved rock. This is affected by a number of factors, such as utilization of capacity, market price of the product and consumed materials, freight charges, cost of electric power, wages, etc. In order to avoid making too frequent disturbing readjustments of the operation, an adaption policy should be drawn up, e.g. once a year, based on the figure for the lowest ore content when loading caved rock. In this connection it should, however, be observed that if a modification of the mining method is made, the profitability limit reacts and it will be necessary to adopt a new procedure.

When the existing profitability limit has thus been determined for a fixed period, it is expedient to investigate to what extent the present ore processing plant can handle goods with different ore content. Any possible investments to raise the capacity should, as usual, be calculated against the background of anticipated reduction in costs for the mining operation. By an ore processing plant is here meant the complete production apparatus ranging from the mining area to the consumer.

Limiting factors

The point has now been reached when the variables of the mining method can be discussed and it is desired to know how the ore content of the handled rock will be affected by processing and mining. As loss of ore and admixture of waste rock occur in the mining operation, it is the mining which will be analysed first and the precessing will be adapted afterwards.

Some limiting factors will first be described. The strength of the ore determines the largest economical drift area and the smallest pillar width, etc. which can be driven without reinforcement (development drifting) costing more than what can be gained by increasing the drift area. It is, of course, fully understood that large areas permit higher capacity and open up better rationalisation through mechanisation. Furthermore, the cost of the development drifting will be lower per ton of ore in large drift areas.

Another limiting factor is the blasting property. Rock with poor blasting property results not only in heavier blasting costs, but also in higher drilling costs. The blasting and drilling operations will therefore affect the choice of slice height and drilling pattern. The bore hole diameter will also limit the slice height because the bore hole deviation jeopardizes the rock fragmentation.

Rock fragmentation

After having considered the effect of the above-mentioned factors it is possible to prepare a plan for the mining operation. A guiding principle in this connection must be to give the ore a fragmentation which fits the chuting arrangements and the intake opening of the boulder crushing plant. The handling of ore boulders at the place of loading and in the chute hole is an expensive operation. The diagonal measurement of an ore boulder which comes from the tip of the bore holes (Fig. 1) must therefore not exceed the maximum measurement of the intake opening of the crushing plant.

Consequently, the tip distance in the "fan-cut" if fixed by the burden which, in its turn, should be chosen on basis of the blasting property. Using crushing plant AR-150 and 1 m burden, the tip-distance should be, for example, not more than 1.2 m. The nominal dip distance should, of course, be adjusted downwards by the amount whereby bore hole deviation has occurred as a result of the non-rigidity of the drill steel and incomplete feed fixation. Consistent excess charging of the bore holes results in increased fragmentation, but can hardly prevent the formation of boulders.

The physical possibilities of the broken rock to flow down to the place of loading will now be locked into. The angles of repose for a number of fractions of the existing rock and ore should first be determined. As the moisture content in the small fractions can have a considerable effect on the angles of repose, it is important that the calculations should be made on goods having a moisture content corresponding to that in the mine. The material which exists in the Idkerberget mine proved to have angles of repose of 32°-38°. Furthermore, the angle of repose for, in scale, correctly granulated crushed quarts was fixed at 36°.

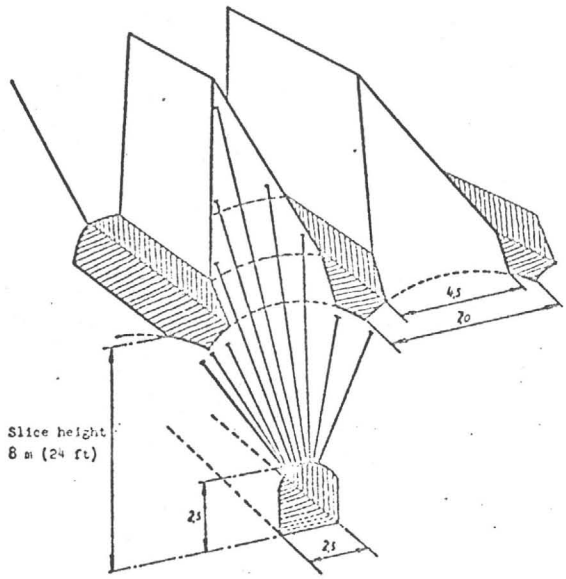

Fig.1 Planned arrangement of sub-level caving seen from the hanging wall caving (not marked).

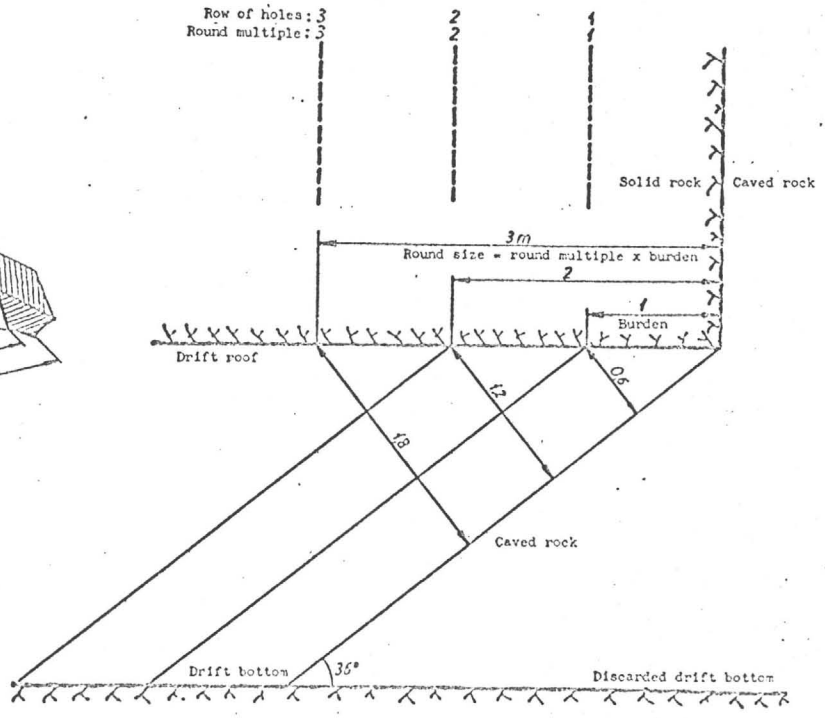

Fig.2 "Narrow passage" at three different round sizes.

When the loading of a sub-level caving round is completed, the fill rock, with possible ore residuals, will rest on the bottom of the drift and form an angle of about 36° to the bottom (Fig. 2). If, for example, in connection with divergent drilling in a vertical plane, the burden is 1 m for the complete round, the narrow passage will not be more than about 60 cm. After normal drilling and blasting, the pieces of ore will then have very small possibilities to penetrate through the narrow passage unless a quantity of waste rock under the angle of respose has been mucked.

As, however, a drilling pattern is drawn up principally with regard to the boulder crusher and the blasting property, a number of rows of holes must be fired simultaneously in order to avoid having "hang" of ore boulders. Short delay blasting is particularly suitable for such operations. A round can comprise one or more burdens: the size of the round is then a multiple of the burden. For example, in Fig. 2 the burden is 1 m and the largest permissible boulder size is 1.5 m. At an inclination of the round of 90°, the largest permissible practical tip distance will be 1.1 m, the required passage 1.2 m, and the round multiple at least 2.

So far, it is only the effect of the ore boulders which has been considered. The waste rock has not been drilled and blasted, but normally consists of a product, crushed during the course of the caving, which has quite a different grain size to that of the ore. The grain size of the caved rock can vary considerably from mine to mine, owing to the abrasive resistance of the barren rock which surrounds the ore body and on the size of the vertical caving. A foot wall with a comparatively flat lateral dip (< 65°) can, however, take up part of the small-grained barren rock, formed by abrasion in the caved material, which would otherwise penetrate through the coarse material down to the place of loading.

Inclination of the round

After having taken into consideration the importance of these factors, it is expedient to study the importance of the inclination of the rounds for the possibilities of the broken ore to reach the loading point and the thereby related admixture of waste rock. For this purpose a number of flow experiments have been carried out in models made of glass. The models were charged with pure magnetite and crushed quartz in correct grain sizes according to scale (Fig. 3). The ore content in the dumping pit fell in step with the loading. The economical limit for loading is drawn in the figure at 20% ore and the ideal would be a curve which ran close to the dotted 100% line. In such a case it would be possible to achieve a complete yield without admixture of waste rock or, in other words, a reversal from pure ore to pure waste rock with lightning rapidity.

As will be seen from Fig. 3, the 107° round inclination is superior and allows loading to about 120% before the lowest profitability limit in the last loaded mine car is reached. This applies to a round multiple of 2 which here corresponds to a round size of 1.8 m. The lost ore residual is in this case about 10% at 120% loading (Fig. 4 top). At a round multiple of 1, a round inclination of 90% is, however, to be preferred (Fig. 4, bottom). The explanation of this is that coarse ores from the upper part of the slice when firing only row of holes obtain more favourable flow conditions in the 90° case (Fig. 5).

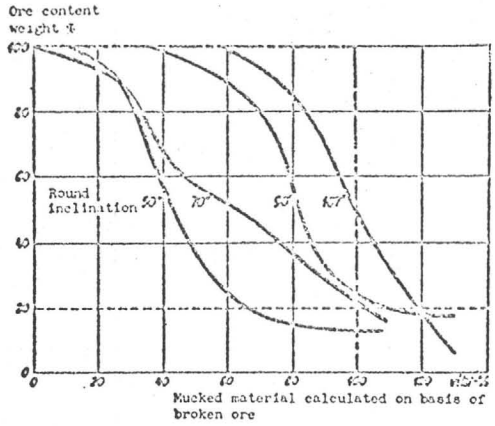

Fig.3 Instantaneous ore content in the mucking heap at 1,8 m round size and different round inclinations; ----- economical limit for ore dilution — · — ideal curve.

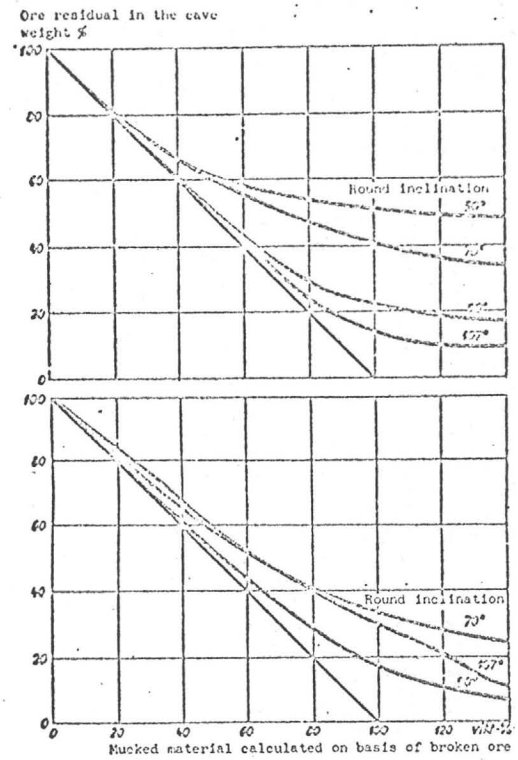

Fig.4 Ore residual in the cave during loading at different round inclinations; top, 1,8 m and, bottom, 0,9 m round size.

The same figure also shows the size of the caved material on the bottom which is exposed to admixture of ore fines in the border layer against the caving. In the 107° case, the ore fines from the blasting should trickle down over the breast in the sub-level drift and to a very small extent be lost in the border layer or fall below the 36° angle of the "old" caved material. The force of gravity should likewise also return the well fragmented ore which, as a result of the surplus blasting force, has penetrated into the coarse caved rock which has rested against the ore slice during the blasting.

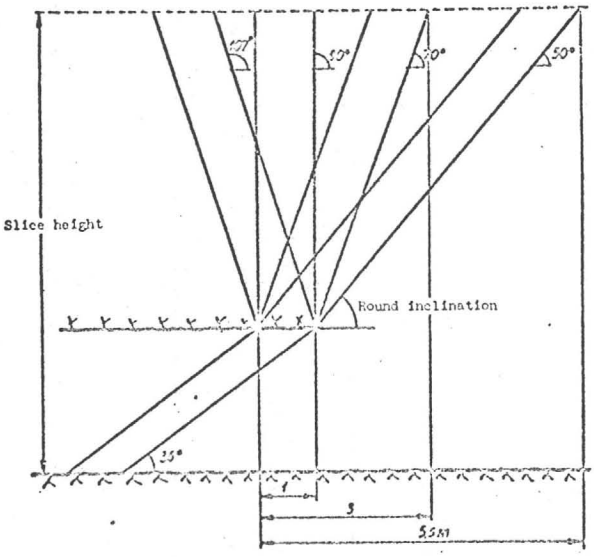

Fig.5 The broken ore exposed for admixture in the border layer against the waste rock at different hole inclinations

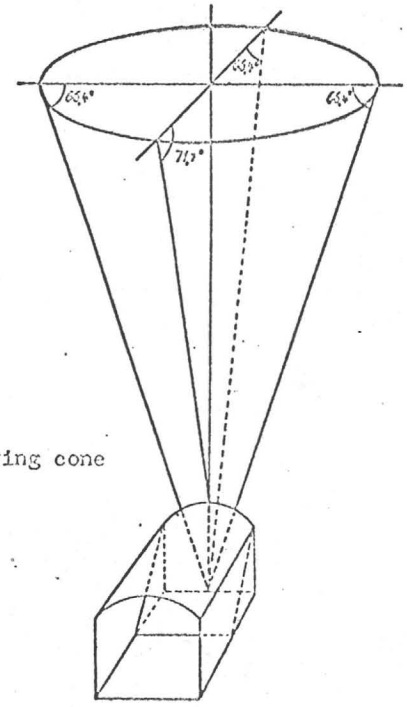

Fig.6 The caving cone

breaking the ore loose
some fragmentation due to excess charge
penetration of small sized ore pieces and fines amongst the relatively coarse waste rock in the caving (Fig. 5)
compression of the caved material which thereby makes it possible for the ore to swell.

The penetration of small sized ore pieces and fines into the coarse waste rock is an undesired effect of the blasting and results in some ore losses, which, however, can be considerably reduced when using round inclinations over 90°.

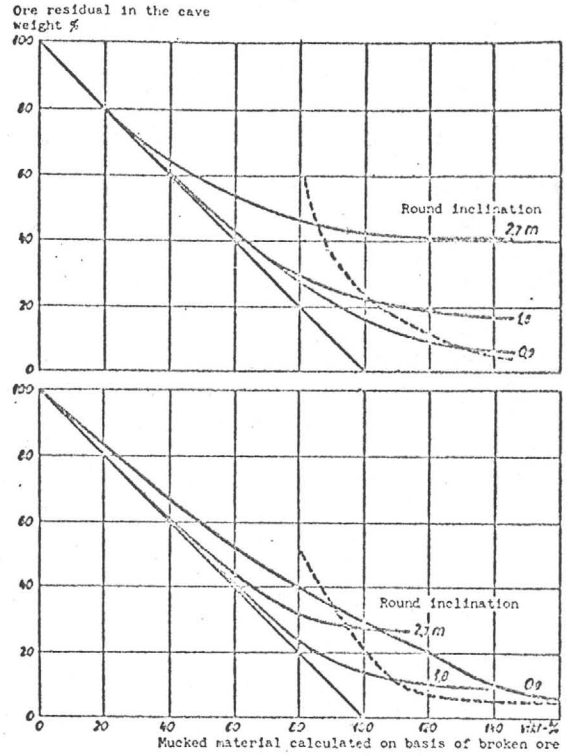

Fig.7 Ore residual at different round sizes; top, 90° and, bottom, 107° round inclinations; ---- economic profitability limit

The caving cone

An investigation was also made at Idkerberget to find out the shape of the body within which the broken ore moves downwards when loading from a draw point. On account of the breast in the drift not being straight, the caving cone (Fig. 6) is not rotary symmetrical, but its basis is generally half circular and semi-ellipse. The obtuse angle between the level of the drift bottom and the generating line which runs backwards in the drift direction is about 107° and for this reason it is not appropriate to work with a larger angle.

As the flow figure has the shape shown in Fig. 6, it would be correct also to carry out the fan cut drilling uniformly with the ascertained semi-ellipse. This should result in all the holes having the same possibility to break when blasting and that the ore obtained from such a blasting would flow down to the draw point. Such a drilling operation (Fig. 1) would also result in the broken ore flowing down to the draw point also from the sides along the generating lines. The drilling must, of course, be carried out with extension steels and the charging made with a flexible rod from a charging machine. Ammonium nitrate explosives should be particularly suitable in this case.

Cost calculation

If the value of the ore in a broken condition in the slice before loading is M kr/t and the cost for processing the waste rock admixture to a finished product is B kr/t loaded rock, and the latter is added to the value of the ore loss, the cost per ton ore of finished product will be

$$K = \frac{100}{100-f}\left(\frac{p}{100}B + \frac{f}{100}M\right) \qquad (1)$$

when p denotes mucked ore and f the loss of ore, both calculated in percentage of broken ore, which is, of course, 100/(100-f) t per ton ore of finished product.

On the assumption, for example, that M is 10 kr/t and B 20 kr/t the following is obtained:

$$K = \frac{100}{100-f}(0.2p + 0.1f) \qquad (2)$$

From diagrams, similar to those in Fig. 7, obtained by means of experiments in a model, it is possible to obtain the values of p and f and, with the aid of the equation (2), to estimate the economical optimum size of round for different round inclinations (Table 1 and Fig. 8).

Table 1. Estimated cost per ton of ore at different round sizes and round inclinations

	Round size m	Mucked %	Ore residual %	Cost Kr
Round inclination 90°	0,3	160	0	32
	0,6	140	3,5	29,50
	0,9	130	7	29
	1,8	110	22	31
	2,7	85	45	39
Round inclination 107°	0,9	165	4	35
	1,8	120	10	28
	2,7	93	28	30

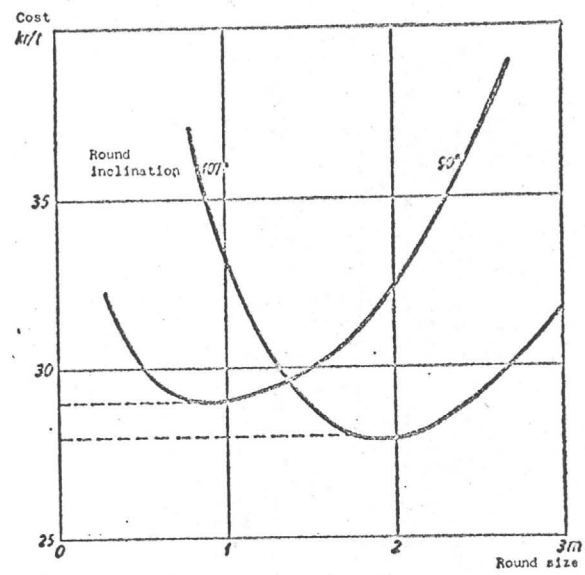

Fig.8 Estimating the optimum round size at two different round inclinations

The curves have distinct minima and if consideration is also taken to the reduction of the setting-up time when loading large-sized rounds, the example indicates that there are obvious reasons for working with 107° round inclinations, 1.8 m sized rounds and loading at 110%. The ore losses would then be about 10% and the ore content of the mucked material from the slices nearly 80%. In carrying out experiments with a model changed with slightly more than 2 t material, this result has been confirmed and, in view of the prospects which this result opens for the most favourable sub-level caving, a reorganization will be made at Idkerberget (Fig. 1).

Concluding remarks

With these reflections I have wanted to describe a method to modify sublevel caving and to deal with the output problem. As a mine deposit must be regarded as a limited utility, it is necessary to evaluate the ore losses which different mining methods occasion. By an adjustment of the processing and mining to the physical flow possibilities which exist in sub-level caving and the economical conditions which apply to a deposit, it is thus possible to obtain the most favourable output.

In Sweden, sub-level caving is, as a rule, driven to about 145% with 70° round inclinations. A reorganization of the method on the basis of what has been proved above, should result not only in about 20% lower production costs, but should also increase the ore production by about 10%, considering the existing underground and haulage capacity.

Latest developments and draw control in sub-level caving

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Synopsis

The increasing use of the sub-level caving mining method in large orebodies is due, mainly, to the ease with which the method can be mechanized, recent developments in mining techniques and a better understanding of the theory of sub-level caving. The present paper is intended to illustrate that one of the key factors in the success of this method, especially in mines with thinly disseminated ores, is the provision of an efficient and effective draw control service.

The sub-level caving theory and the latest layout designs based on it are described briefly to provide the necessary background for the account on draw control practice. Draw control methods in use at present are then described and compared. The subject matter sources are limited to Mufulira Copper Mines, Ltd., where a strict grade control is in operation, and to several Swedish iron mines, where draw is controlled almost wholly by visual means. Then follows an account of a proposed tonnage/grade control which Mufulira hopes to introduce in the near future when the production increases from 80 000 tons per month to 200 000 tons per month and a strict grade control is liable to become difficult to operate effectively.

Sub-level caving in various patterns of blasting and extracting was the earliest caving method used in mines with weak and incompetent ground conditions; in spite of a high dilution and relatively low recovery, it was regarded as the only practical method available. With mechanized development and long-hole drilling equipment, improved blasting techniques and the advent of trackless extraction units, this method is being increasingly considered and applied in base metal and ferrous mines with large, thick and competent orebodies. This is because it can be readily mechanized, it is safe, standardization of equipment presents fewer problems, selectivity in variable-grade orebodies is possible and a large proportion of the development is in ore; furthermore, in countries where labour shortages are an acute problem, the method has obvious advantages. These factors all contribute to keeping costs down and the method is now often preferred to block caving or large sub-level open stoping.

Due to the inherent characteristic of sub-level caving, in that the thin column of ore drawn with each fan blast is bounded by the solid wall of the next fan on one side and the caved waste on all the other sides, the problems of recovery and dilution are as great as in any other method. During the last ten years a considerable amount of research has been carried out in many countries, both in the form of scale-model tests and of studies conducted under natural conditions. This research has shown that by introducing improved layouts based on the draw characteristics and employing modern mining techniques a good recovery in conjunction with a dilution comparable to that obtained with a carefully controlled block cave method is possible. It has been indicated, however, especially in base metal ores, that a strict positive draw control is essential to the success of the method. The term 'positive' in this context means obtaining a comprehensive draw grade and extraction tons record to stop the draw at the desired point and also to gain reliable recovery and dilution figures in each drawpoint. Many iron ore mines in Sweden, due mainly to the simple and cheap mineral dressing process and ease of ore identification, rely on what can be termed a 'negative' control, i.e. the draw is controlled visually by the foreman or loading operator and only overall mine recovery and dilution figures are usually obtainable. It is considered that, if a form of positive control were introduced, most of the mines could achieve a significant saving in mining, hoisting and milling costs per ton of salable product. This is important in view of the iron ore marketing difficulties being experienced at the present time.

The purpose of this paper, after briefly describing the latest development in theory and practice of the sub-level caving mining method, is to explain the present methods being used in controlling the draw and to indicate the intentions of certain mines to modify and improve this control. Some suggestions have been included which are based on the author's experience at Mufulira Copper Mines, Ltd., and observations made during a five-week tour of Swedish mines. It should be borne in mind that most mines considering or using sub-level caving are fairly large, with a crude ore production in excess of 1 000 000 tons per annum. The mines require or have available a good technical service organization, but smaller properties could, with modifications, introduce an effective control on lines similar to those proposed.

Theory of sub-level cave draw

So far as is known, the bulk of research into the sub-level caving mining method has been carried out in Sweden and, in particular, by the Royal Institute of Technology, Stockholm. Janelid and Kvapil's paper[1] on the principles and theory of sub-level caving is probably the most authoritative to date and should be studied by all those intending to introduce this method. Model experiments and underground studies at Mufulira,[2] carried out initially without knowledge of the Swedish work, were found to be in close agreement with many of Janelid and Kvapil's findings. Experimental work is also known to have been carried out in Australia at Mount Isa. It is not possible here to expound all the characteristics involved in a sub-level caving draw, but those that have a major influence on the

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method of draw control, and therefore on the understanding of it, are described.

Shape of draw

Basically, it has been found that the gravity flow of blasted or caved rock under pressure obeys the same laws and is similar to the flow of granular material in a bunker. The ground entering a drawpoint flows from a zone in the shape of an ellipsoid. The actual ground drawn flows from what Janelid calls the 'ellipsoid of motion'; this, in turn, disturbs a larger zone, called the 'limit ellipsoid', which is approximately fifteen times the size of the ellipsoid of motion (see Fig. 1). If the

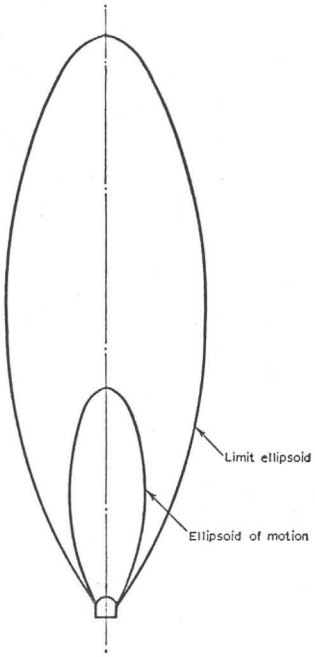

Fig. 1 Ground under pressure is drawn from a zone in the form of an elongated ellipsoid, termed the 'ellipsoid of motion'. This disturbs the ground in a larger zone, termed the 'limit ellipsoid'. From Janelid and Kvapil[1]

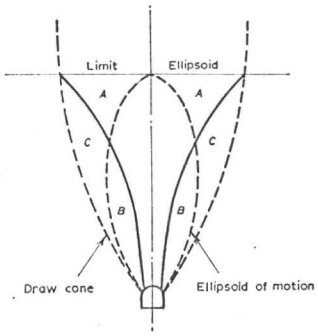

Fig. 2. Relationship between visible draw cone and ellipsoid of motion and limit ellipsoid. Material drawn from *B* is replaced by material gravitating from areas *A* and *C*. From Janelid and Kvapil[1]

material being drawn consisted of laminations of different colours, and could be viewed through a glass plate, the draw would appear to be in the form of a cone with the apex at the drawpoint and the sides slightly concave (see Fig. 2). The eccentricity of the ellipsoid of motion is dependent on the average size of the rock and the speed of draw; for Mufulira ore, and probably most blasted rock, the draw cone has an angle of approximately 40°, or the minimum angle of draw to the horizontal is approximately 70°.

Draw rate

Understanding the relative draw rates within the ellipsoid of motion is also important for the purpose of controlling draw. The velocity of motion is greatest near the drawpoint opening and along the centre-line of the gravity flow. It decreases as the vertical distance from the opening and the horizontal distance away from the centre-line increases. Along the perimeter of the limit ellipsoid the motion is zero. Model experiments and underground studies in producing areas at Mufulira have clearly illustrated the very high draw rate of the column vertically above the drawpoint in comparison to that at the fringes of the draw cone. The caved waste lying vertically above the drawpoint gravitates rapidly down to the drawpoint and cuts off a high proportion of the ore flowing down from the sides of the fan. In the model tests this ore loss comprised approximately 20 per cent of the blasted fan tonnage. With the checkerboard layout of the crosscuts on alternate levels, however, this lost ore, which is often largely undiluted, is drawn from the drawpoints on the level below and overall recoveries in excess of 90 per cent can be expected (see Fig. 5(*a*)).

Dilution

The way in which the waste rock dilutes the ore column in a non-uniform and fluctuating manner is one of the most important factors that must be understood in order to apply an effective draw control. It is known that (*a*) dilution of the ore column commences soon after draw from a newly blasted fan has started, usually before 10 per cent of the ore has been recovered; (*b*) during the life of the fan the dilution rise or grade drop does not occur in a uniform manner: it is quite common to find the drawpoint muckpile containing 40–50 per cent waste soon after commencing draw, several loads later the percentage waste dropping again to less than 10 per cent. (Figs. 3 and 4 illustrate the

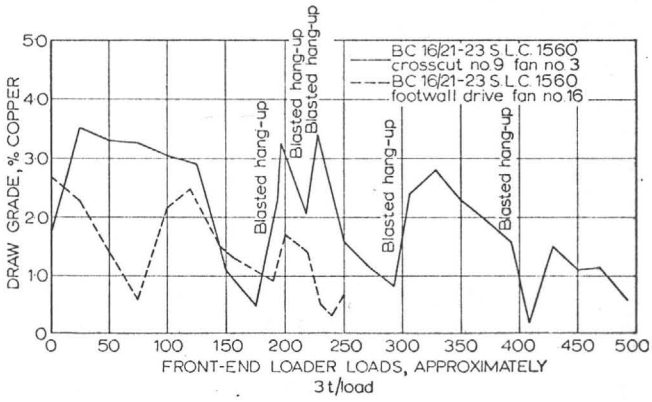

Fig. 3 Typical grade fluctuation during draw from a fan is shown from the draw analysis of two fans picked at random from a recent underground test at Mufulira

fluctuating draw grades obtained during the life of fans in natural conditions underground and in model tests at Mufulira[2]); and (*c*) ultimately, most of the dilution comes from the waste lying vertically above the ore being drawn.

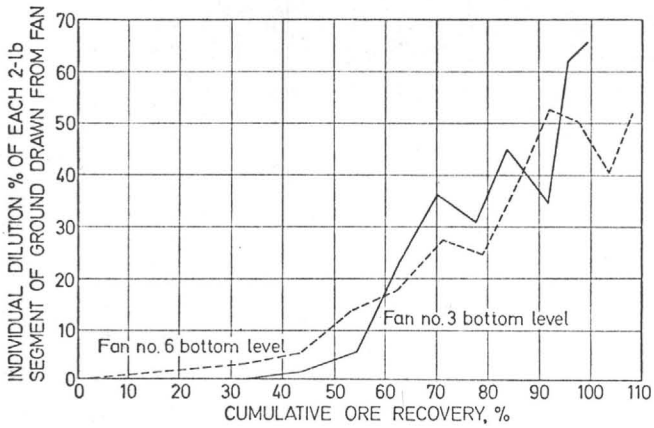

Fig. 4 Graph showing fluctuation of waste during draw from a fan in the model tests

Considering statements (*a*) and (*b*), it has been concluded that, owing to the shape and size of blasted rock, hang-ups continually occur. These hang-ups occurring in the ore allow waste from the previous ring position or from either side to gravitate into the draw column. Sometimes hang-ups can be seen from the drawpoint, but in most cases they are not visible. The size and extent of these hang-ups vary considerably in magnitude and this is the main reason for applying a positive draw control under the supervision of experienced draw control officers. Although a good fragmentation throughout the ore column will reduce the number and size of the hang-ups, they still occur with sufficient frequency to rule out the possibility of obtaining a predicted dilution increase and therefore cutoff point. This was shown in model tests which were carried out under relatively ideal conditions. Also, at Kiruna, Sweden, where the iron ore is fairly soft and friable and good fragmentation is common, graphs similar to those in Fig. 3 were seen. Fans that have been poorly blasted for a variety of reasons will obviously affect the draw and, often, this situation is not immediately realized by operating officials.

Concerning statement (*c*), it has been found, with blast-hole fans angled at 70–90°, that the bulk of the waste drawn during a normal draw comes from the area above the fan position. This is due to the relatively fast vertical draw rate. The waste causing the fluctuation in grade during the life of the fan can come from the previous fan position and from above if a hang-up causes funnelling.

It is considered that improved layouts and blasting techniques can reduce grade fluctuation but can never prevent it.

Other factors influencing the draw

It must be emphasized that there are many other factors which influence draw in sub-level caving. Among these are (1) the shape and size of the rock being drawn: the smaller the average rock size the greater the eccentricity of the ellipsoid; (2) the percentage fines and moisture content: dilution and recovery are affected according to whether the fines are predominantly ore or waste (this factor is only significant when the percentage fines is high, as may be expected with weathered or friable rock); moisture affects recovery adversely and if the content is high where the fines content is also high, it can be a major problem; and (3) the size of extraction opening: where the drawpoint is much wider than the loader shovel, uniform loading across the whole width of the drawpoint is essential otherwise recovery could be adversely affected.

All the above factors were dealt with in more detail by Janelid and Kvapil.[1]

Present and planned sub-level caving layouts

Most sub-level caving layouts are planned today from a knowledge of the basic draw characteristics in conjunction with practical and economical considerations. These factors and the resultant layout designs are now discussed briefly.

Vertical interval between extraction openings

Model tests carried out in different countries suggest an optimum vertical interval of from 50 to 60 ft between levels, depending on the other parameters involved. This vertical interval is not the height of the fan or ore column and depending on layout geometry the latter could be twice that of the mining vertical interval. At Mufulira model tests using a 6-ft burden angled at 70° suggested that the optimum vertical interval should be 60 ft. However, owing to the present difficulties in drilling blast-holes accurately over distances greater than 75 ft without using expensive and time-consuming aids, the consequent blasting and fragmentation difficulties which are magnified by crushed or blocked holes, the need for selectivity in variable-grade orebodies, and in one known case the high cost of drilling in a very abrasive rock, the vertical intervals in most mines vary from 24 to 50 ft. It is known that in the U.S.S.R.[3] vertical intervals greater than 60 ft are being used.

To reduce the length of the centre blast-holes in the fans some mines have extended the fans on either side down to almost horizontal. This has proved unsuccessful because, with the limiting draw angle of approximately 70°, great difficulties have been experienced in breaking the ground below this angle. After retreating a few fans from the strike slot, blasting below 50° becomes ineffective as a complete choke blasting condition occurs. As a result most blast-hole fans are now drilled to a diamond-shape layout, the angles to the horizontal varying from 70° to 50°.

Horizontal interval between extraction openings

The ellipsoid of draw theory suggests that the spacing between crosscuts for optimum recovery and dilution is a pillar width equal to the width of the extraction openings: this would provide a completely vertical draw. Economical and practical considerations, however, prevent the application of this ideal spacing in

most mines. The minimum horizontal spacing between extraction openings is governed by the need to provide sufficient load-bearing capacity to the pillar and also by the cost of development.

The maximum distance is dependent on the width of the draw ellipsoid for recovery purposes and, possibly of more importance, on the limitations of effective blasting and breaking at the extremities of the fans where the pillar is wide, e.g. at Mufulira the initial layout consisted of 14 ft × 14 ft crosscuts at 50-ft horizontal centres, resulting in a 36-ft pillar between the crosscuts. Great difficulty was experienced in achieving and maintaining satisfactory caving conditions. Progressive failure to break the ground to the extremities of the holes at the sides and apices of the fans resulted in arching. This arching and pillar build-up was usually initiated at the sides and was not realized until two or three fans had been blasted and the increasing size and strength of the side pillars finally forced the completion of the arch. These pillars were being formed continually with the initial layout and considerable expense in time and money was required to reinduce caving. Also, ore loss in some cases was quite appreciable in that these pillars affected draw on the levels below. It is expected that the revised layout, with crosscuts at 33-ft horizontal spacing, giving a 19-ft pillar, will greatly improve conditions (see Fig. 5(*a*)).

It is therefore considered that when planning a sub-level caving layout the minimum possible horizontal spacing should be aimed at, even if the extra cost of development is high and apparently unwarranted.

Angle or gradient of blast-hole fans

Research in Sweden has indicated that the optimum fan angle, within the angle limits of 60–120°, is mainly dependent on the ratio of average ore sizes to waste sizes, and has been shown to be between 90 and 110°. Janelid and Kvapil[1] produced a simple table which is reproduced, in a slightly modified form, below.

Ratio of rock sizes	Fan angle, *a*
$K_o/K_w > 1$	Positive, $a < 90$
$K_o/K_w = 1$	Vertical, $a = 90$
$K_o/K_w < 1$	Negative, $a > 90$

where K_o is the average particle size of the ore, K_w is the average particle size of the waste and a is the angle between the horizontal and the fan, away from the direction of retreat (see Fig. 5(*a*)).

The theory is that if the fan is sloping backward away from the direction of the blast the smaller pieces of ore are less likely to be blasted into the voids in the caved waste. This argument is logical, but very difficult to assess in terms of percentages or figures.

Model tests at Mufulira have indicated that the major difference in draw between a 70° and 90° fan is that ore lost at the back of the drawpoint by the vertical gravitation of waste is more with a 70° fan than a 90° fan. These model tests indicated that the 90° fan yielded a slightly better recovery than the 70° fan—an increase of 2 to 3 per cent only. (Model tests could not simulate blasting action, i.e. the expansion of ore particles into the waste voids.)

The angle of fan adopted is, however, also dependent on mining considerations—such as maintaining a good brow and a good drilled footage efficiency. Probably because of these considerations very few, if any, mines have adopted a backward sloping fan angle and most have adopted angles varying from 70 to 90° depending on these conditions. At Mufulira a 70° forward sloping angle has been adopted for the following reasons: (1) because of the 50–70° dip of the orebody, the need for maximum development in ore and the adopted foot-wall to hanging-wall direction of retreat, loss in tons per foot drilled would be significant if 90° fans were adopted; (2) the orebody is divided into several different grade zones parallel to the dip and therefore to the 70° fan angles: this facilitates the calculation of fan reserve grades for control purposes; (3) maintenance of brow; and (4) recovery is expected to be satisfactory.

Fan burden

Tests carried out in Sweden have indicated that the optimum fan burden is 2·1 m (6·9 ft). For a fixed dilution the expected recovery decreases if the burden is greater or less than this. Present blasting and fragmentation performances, however, indicate the most common burden to be 6 ft.

Location of extraction drifts

It is generally accepted that the extraction openings on the different levels should be mined on a staggered system, i.e. the openings on the lower level are placed midway between those on the level above. This permits almost complete recovery of the initial ore loss, which is left on either side of the drawpoints. It is estimated at Mufulira that approximately 20 per cent of the blasted ore is left in the fringe on either side of the drawpoint; however, experiments show that most of this lost ore, which is largely undiluted, will be recovered on the lower level. Figs. 5(*a*)–(*d*) illustrate typical sub-level caving layouts in use or planned.

Control of draw

The complexity of sub-level caving draw as described above has clearly demonstrated the need for a positive control if the best results are to be obtained. When it is intended to introduce and operate an effective draw control the following factors are important: (*a*) the main objective is to increase the profitability of the mine by decreasing the mining cost per ton of concentrate or salable product; (*b*) the draw characteristics of the method, with particular reference to the nature of the fluctuating grade, should be understood as fully as possible; and (*c*) the unavoidable limitations inherent in the control method to be introduced must be appreciated: the method will depend on the availability

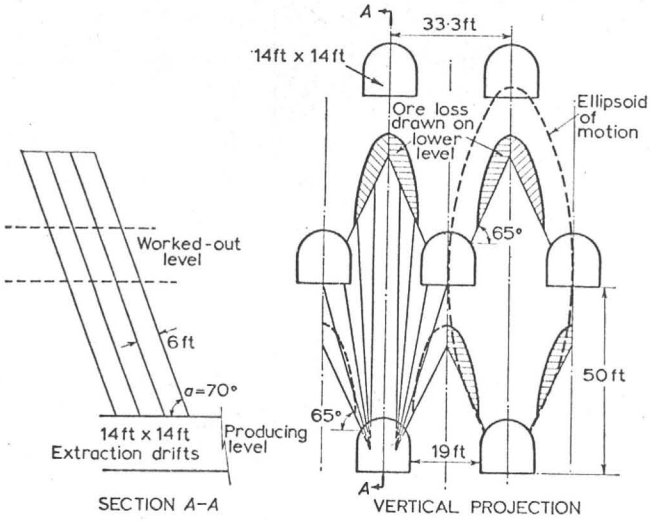

Fig. 5(*a*) Planned sub-level caving layout at Mufulira showing expected ore loss on sides of fans. Estimated ellipsoid of motion shown in broken lines

of men for a draw control department, the resources of the assay laboratory and type of instrumentation to be used.

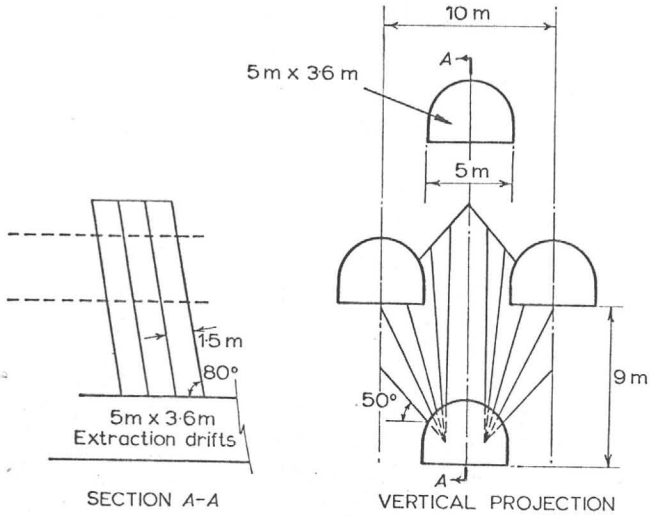

Fig. 5(*b*) Present sub-level caving layout at Kiruna, Sweden. Short vertical interval between levels is necessary to provide selectivity in mining variable-grade ore

Information required for draw control

(1) Reserve grade or the *in situ* grade of the ore: in variable-grade base metal orebodies the grade for individual fans should be obtained, if possible; in massive homogeneous ores, such as ferrous deposits, a block or area grade covering a large section may be acceptable.

(2) The reserve tons to be blasted in each fan.

(3) An accurate record of the extraction tons drawn from each fan: this is usually compiled by recording the number of loads drawn from each drawpoint and reconciling them to tons at a later stage.

(4) A sufficient number of draw grades obtained by sampling the ground in the drawpoint at certain intervals.

From a knowledge of the reserve grade, draw grade, reserve tons and extraction tons the recovery and dilution can be calculated.

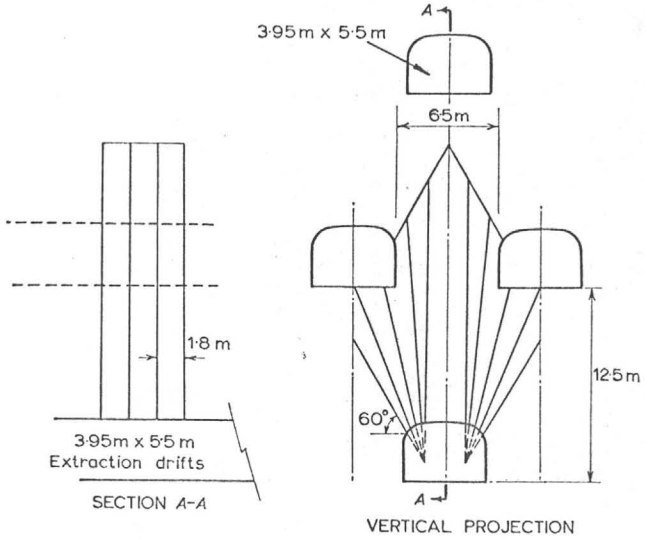

Fig. 5(*c*) Projected sub-level caving layout at Malmberget, Sweden. Present layout, 10 m vertical and 10 m horizontal interval between 3·1 m × 3·0 m extraction drifts

If one or more of the above factors is not known with any degree of certainty the recovery and dilution figures obtained could be erroneous and misleading. In certain cases the errors could completely defeat the main objects of providing a control and give management a costly false picture.

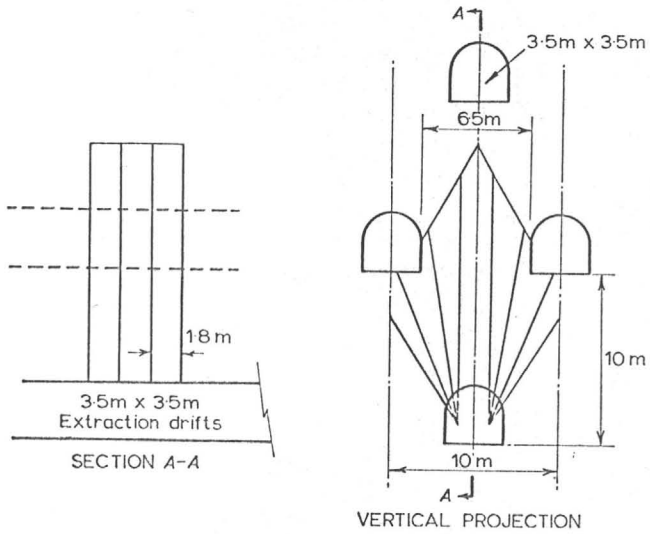

Fig. 5(*d*) Projected sub-level caving layout at Norberg, Sweden. Fan angle, drill pattern and fan burden assumed (not yet finalized by company)

It is intended to describe, first, the present draw control systems in operation in certain base metal and ferrous mines. An outline of the proposed system for Mufulira is then described; it is considered that this system could be applied in both base metal and ferrous mines.

Present draw control systems

Ferrous mines

Most iron ores are in massive form, i.e. iron content 40–70 per cent, are usually easily distinguishable from the country rock and mineral dressing is simple and relatively cheap. It is therefore not surprising that where sub-level caving is used such mines employ what is considered to be a negative control of draw.

After blasting a fan the ore is drawn until there is approximately 50–70 per cent waste present in the muckpile. The percentage waste allowed depends on the reserve grade of the ore. The decision to stop the draw from any fan is made either by the loader operator or the foreman (shift boss) in charge of the section. It is done visually and at the discretion of the men concerned; draw is often continued beyond the laid down cutoff percentage if it is considered that some hung-up ore may still be drawn. It can be argued that experienced miners in sub-level caving have a shrewd idea of what ore has been drawn or left and therefore can judge with a fair degree of accuracy when to stop drawing and to blast the next fan. It is agreed that the men do have a shrewd idea of the draw situation, but it falls well below the control standard desired and made possible by continuous sampling. This has been borne out on many occasions. Proof of this lies in the fact that dilution* in Swedish iron mines varies from 20 to over 30 per cent by weight or, more important from cost considerations, 35 to over 50 per cent by volume.

Most Swedish mines satisfy three of the four draw control requirements. (1) The reserve grade is obtained from core analysis of suitably spaced diamond drill holes in the relatively homogeneous ore. At Kiruna, where the magnetite orebody has zones of differing phosphorus content, horizontal holes are put through the orebody on every level at intervals of 50 m; in addition, the muckpile from each development blast is sampled. It is intended shortly to reduce the interval of diamond drill holes to 25 m and to drill a fan of several holes on each level. It is expected that this, with the development muckpile sampling, will provide a sufficiently accurate iron and phosphorus grade for each blast-hole fan. This will ensure that the ores are split satisfactorily into the different phosphorus grades during stoping. (2) Calculation of the reserve tons is straightforward from a knowledge of the blast-hole fan dimensions and the specific gravity of the ore. (3) In most of the mines the extraction tons are fairly accurately known because of the almost universal use of weighbridges on all main tramming levels. Consequently, the actual tonnage drawn from each orepass is known constantly. From a simple record of the number of loaders or shuttle trucks, tipping x loads from y drawpoints into an orepass, an acceptable record of the extraction tonnage drawn from any particular fan can be kept. Apart from supplying the desired extraction tonnage, the weighbridges are extremely useful in ensuring the maintenance of efficient tramming and facilitating the operation of bonus schemes.

Base metal mines

In base metal ores experience is restricted to Mufulira and therefore information on draw control practice is limited to this single source.

Before discussing the present control system at Mufulira, it is necessary to point out that copper ores, and consequently most other base metal ores, introduce certain difficulties to the control system that are not present in iron ores. (1) The mineral is usually disseminated thinly, often no more than a few per cent, and unevenly through a parent rock which frequently has the same characteristics and texture as the footwall or hanging-wall country rocks. At Mufulira, although the hanging-wall is distinctive, the orebody consists of different grade horizons roughly parallel to the dip, varying from 0·3 to +10 per cent copper. (2) There is little difference in specific gravity between ore and waste.

Under these circumstances a grade control in the form of sampling at intervals during the draw becomes imperative. The four basic requirements for the control are met in the following manner.

(*a*) The reserve tons in each fan are calculated from the mining layout, fan burden and the specific gravity of the ore.

(*b*) A reserve block grade, i.e. a grade for a block measuring 200 ft along strike by 250 ft vertically, with a horizontal width varying from 100 to over 200 ft, is computed by the geological department from diamond drill holes and from channel sampling in certain crosscuts. This block grade, originally designed for the block caving mining method, will in future be modified and extended to suit sub-level caving (see later).

(*c*) The extraction tonnage from each fan is obtained from a record of the number of loads drawn during the life of the fan. As there are no weighing devices underground it is necessary to reconcile the monthly mill tonnage with the total number of cars trammed. The resultant car factor is then applied to the cars filled from each orepass and, subsequently, to the loads drawn from each fan.

(*d*) The draw grade is obtained by periodic sampling of the drawpoint.

Sampling procedure

The following sampling procedure has been evolved from a knowledge of the continually fluctuating grade

*Dilution $= \frac{\text{waste}}{\text{ore + waste}}$

(see Fig. 3), the availability of labour and the Portable Isotope Fluorescence (P.I.F.) analysing technique which has been successfully developed at Mufulira (see Appendix). With an average sub-level caving reserve grade of approximately 3·0 per cent copper the draw is run to a cutoff grade of 0·7 per cent copper.

(i) The ground in a drawpoint is sampled immediately after a fan has been blasted and before loading commences. The operation consists of taking two 4-lb representative samples from the total surface area of the muckpile and averaging the results.

(ii) During most of the draw, and while the mean grade is above 1·5 per cent copper, the sample interval is 50 loads, i.e. 125–150 tons. The average fan pattern of 50 ft × 50 ft contains approximately 1000 reserve tons.

(iii) When the grade drops consistently below 1·5 per cent copper the sample interval is reduced to 25 loads.

(iv) The drawpoint is stopped when three consecutive grades below cutoff are recorded.

Fig. 3 illustrates the importance of the last step to be taken. With this procedure it has been possible to keep the average sub-level caving draw grade at an acceptable figure of 2·3–2·4 per cent copper. The dilution at present is running at approximately 25–30 per cent and recovery in areas where draw conditions are reasonably normal is over 85 per cent.

In order to permit close sampling of drawpoints, without hindering production, it was found necessary to place a Portable Isotope Fluorescence rapid analysis assaying laboratory underground, near the sub-level caving section. The present sub-level caving production is between 70 000 and 80 000 tons per month, which is drawn from up to four drawpoints on each of five separate levels.

Production is on a three-shift basis and the number of double samples analysed on each shift varies from 20 to 30.

Portable Isotope Fluorescence (P.I.F.) assaying procedure at Mufulira

(1) The 4-lb sample is taken to the underground laboratory and crushed in a laboratory jaw crusher to $-\frac{1}{4}$ in.

(2) The sample is then split down to approximately $\frac{1}{2}$ lb in weight on a riffle.

(3) The $\frac{1}{2}$-lb sample, if dry enough, is then ground to approximately 200 mesh in a vibratory grinding mill.

(4) Even if apparently dry, the sample powder is dried for 10–15 min on a heater to ensure that there is no moisture present, as this could affect the results considerably.

(5) The sample is now ready for analysing with the Portable Isotope Fluorescence instrument. The dried powder is heaped in the sample dish and then cut level to the rim of the dish. It was found that this was the best way to obtain consistent sample compaction. Any appreciable variation in the above factor could cause significant error. The probe is then pressed down over the sample and readings taken with both filters. Some few seconds are required to allow the needle to settle down as oscillation is quite severe initially. From the average maximum and minimum readings the difference can be determined. The degree of oscillation depends on how well the instrument has been calibrated; great care should therefore be taken when doing this. Previously it was necessary for the operator to apply constant pressure to depress the safety button, but with the arrival of a stand and certain other modifications the instrument can now be operated in the inverted position without manual pressure (see Figs. 3 and 4 of Appendix).

Performance and limitations of P.I.F. analyser

Advantages

(1) Continuous testing over a period of several months has shown that the accuracy of the instrument with ore samples varying from 0·4 to 4 per cent copper is well within acceptable limits compared with assays checked in the main laboratory on surface. At the critical cutoff draw grade of 0·7 per cent copper and the common draw grades of 1·5 to 3·0 per cent copper the results are usually no more than 0·1 per cent different from the surface analyses, for which an X-ray spectrograph is used. All grades of less than 0·5 or more than 3·5 per cent copper are sent to surface for checking, with one or two in the middle range, every few days.

(2) The condition and accuracy of the instrument can easily be checked against standard samples, and this is usually done once a week.

(3) The instrument has made possible the introduction of a strict grade control in sub-level caving. The capital cost of this laboratory is well within the resources of most mines.

(4) Several samples can be assayed within 30 min, and this has proved to be important from the point of view of minimal interference with production and at the same time ensuring a satisfactory positive control. Before the introduction of the P.I.F. analyser underground it took at least 24 hours to obtain the results of the samples; in a high production sub-level caving operation this was very unsatisfactory.

Disadvantages

(1) The instrument is not as accurate as the X-ray spectrograph on surface, which counts the total random emissions over, say, 10 sec. The P.I.F. analyser indicates the mean emission at any one time and, due to the random emissions, causes the needle to continually oscillate. It is hoped that future models can be developed to reduce the degree of oscillation.

(2) The instrument must be operated by a man of reasonable intelligence and experience to detect immediately any malfunction of the unit.

To date, Mufulira are quite satisfied with the accuracy of the analyser and the draw control has improved

appreciably. (The instrument is also designed for sampling or analysing mineral content *in situ*, but this has not yet been attempted at Mufulira.)

Projected and proposed draw control methods

An outline of the projected method of draw control to be introduced at Mufulira is given here. This control is still in the theoretical stage, but it has been included to promote interest and comment on the method.

Projected tonnage/grade control at Mufulira

The present grade control is working satisfactorily in a section producing approximately 80 000 tons per month. When this production rate is increased—initially to 200 000 tons per month and eventually to 300 000 tons per month—it is expected that a strict grade control will be extremely difficult to operate effectively without increasing the draw control department considerably, and this may prove difficult. In addition to a senior draw control officer to control the whole operation, a team comprising one experienced draw control officer and one staff learner training to become a draw control officer is now required on each shift. The object is therefore to attempt to reduce the number of samples required during the draw of each fan. At present between 10 and 20 samples are taken from each 1000-ton fan. If the extraction tonnage from a fan with a known reserve grade can be reliably estimated, notwithstanding continual draw grade fluctuation, it may be possible to reduce the number of samples required to one at the start of the draw, one during the middle stages of the draw and three or four towards the end. It is hoped that the present field staff, in addition to one or two extra personnel per shift, would then be able to cope with the increased production.

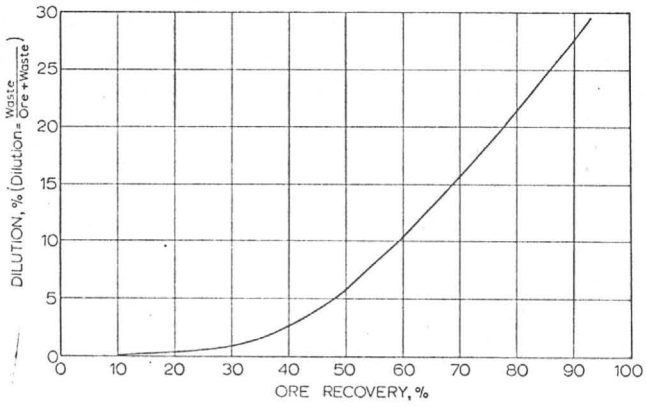

Fig. 6 Sub-level caving recovery–dilution graph for 50 ft × 33·3 ft layout obtained from series of model experiments

A dilution–recovery curve (Fig. 6) was obtained from the results of several model tests carried out on the improved 50 ft × 33 ft sub-level caving layout that is to come into operation shortly. Using this basic curve a reserve grade–draw grade–recovery–extraction graph (Fig. 7) was drawn up; after obtaining a weighted reserve grade and recovery from approximately 30

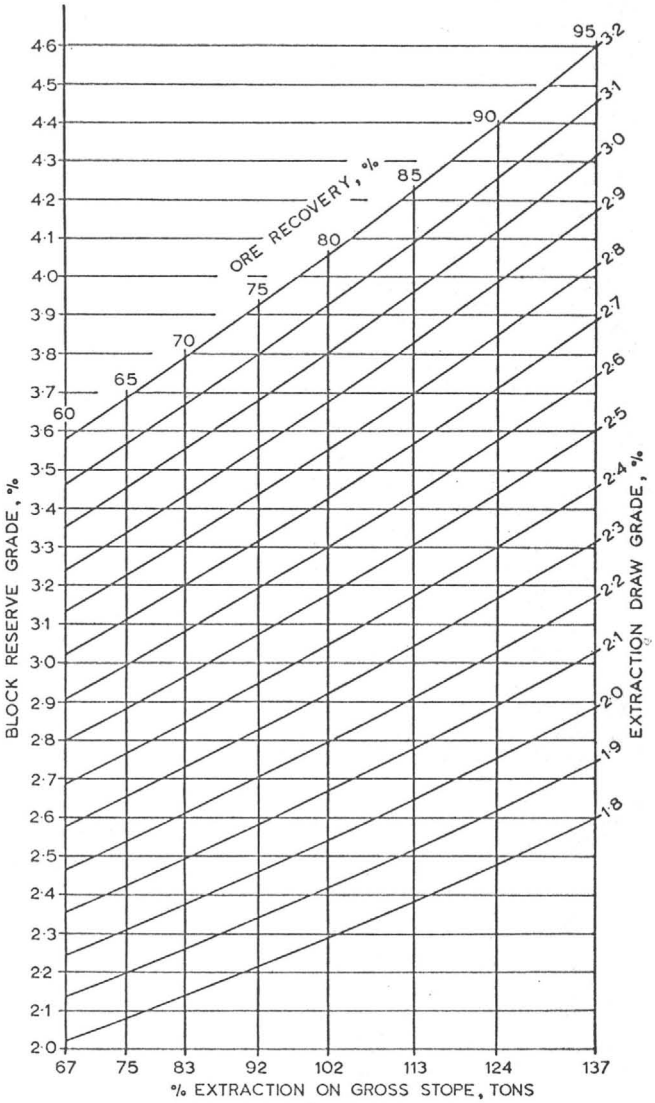

Fig. 7 Sub-level caving recovery–extraction graph

extraction crosscuts a reserve grade–recovery–draw grade curve was computed (Fig. 8). For practical purposes a preliminary draw characteristic graph (Fig. 9) was then prepared to enable the draw control department to estimate the draw grade and extraction of each fan, within allowable limits, from a reliable knowledge of the reserve tons and grade. Because of the fluctuation of the copper content through the orebody this draw characteristic graph could not be applied using the present block reserve grade system. It will be necessary to obtain individual fan reserve grades. One method being investigated is to channel sample all the extraction crosscuts and to average the values of the samples bounding the fan. In the case of the crosscut vertically above the fan, the point is at the projection of the fan vertically to the crosscut. The inclusion of the above crosscut is necessary because of the expected ore loss recovery (see Fig. 5(*a*)). Other methods that may

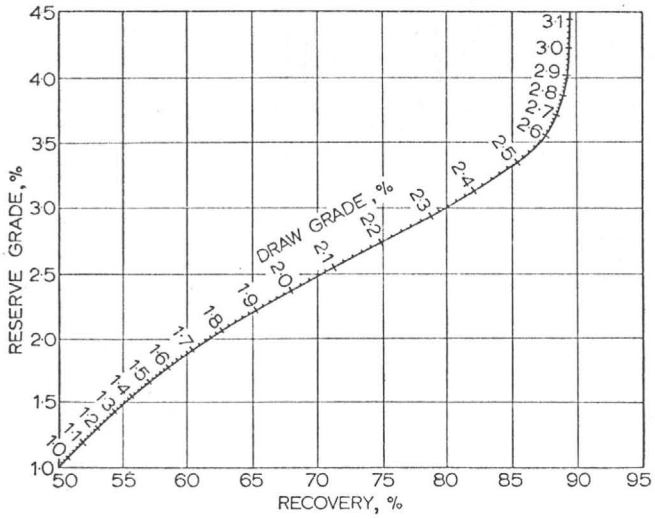

Fig. 8 Recovery–reserve–draw grade graph from recovery–extraction graph (Fig. 7) after obtaining weighted reserve grade and recovery from more than 30 crosscuts

be investigated include sludge sampling of the blastholes. Therefore, with individual fan reserve grades and tonnages, it is hoped that the draw characteristic graph can be used to control the draw in conjunction with the limited number of samples as suggested above. This control is referred to as a tonnage/grade control.

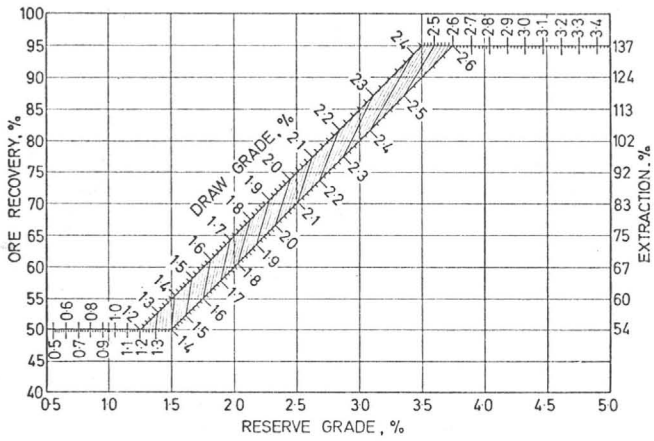

Fig. 9 Sub-level caving draw characteristic graph used to estimate ore recovery, extraction and draw grade from knowledge of fan reserve grade

Because underground conditions could not be completely simulated in the model, and thus several sources of possible error were known to be inherent in the model tests, the operation of the tonnage/grade control will have to be conducted in conjunction with strict grade control for several months. Confidence in applying the draw characteristic graph obtained from the model results is based on the noted close correlation between the draw characteristics observed in the model and those observed in actual conditions underground. The graph and the sample intervals can be adjusted, if necessary, during the trial period. Two important points have to be realized and acted upon if a tonnage/grade control is to have a reasonable chance of success: first, an accurate record of the extraction tonnage must be obtained and, secondly, the control system is nullified if normal draw and caving conditions are lost in a fan under draw.

It must be stated that it is not expected that the tonnage/grade control will prove superior to a full grade control. It is merely suggested as a more practical method of controlling the draw satisfactorily in a high production area, where fans may be blasted in any one extraction crosscut at the rate of one every day.

Summary of basic steps in introducing a tonnage/grade control in mines planning a high production rate

(1) Conduct model tests to obtain a draw characteristic graph as described below.

(2) Obtain individual fan reserve grades.

(3) Calculate the reserve fan tons from the drilling layouts. The grade and reserve ton calculations can be easily computerized. For example, Kiruna are operating a computer program that produces plans giving data of individual fan location, iron and phosphorus grades and reserve tons.

(4) Apply a full grade control, sampling continuously, at the minimum practical interval to obtain the optimum recovery and dilution. Simultaneously introduce the tonnage/grade control with the draw characteristic graph and adjust the curve after comprehensive testing to bring the two controls into line. When control is satisfactory, reduce the sampling to the maximum interval required to maintain the desired recovery and dilution.

Smaller mines could introduce the same tonnage/grade control with some modifications, such as relying on the grade of the development muckpiles in the crosscuts and diamond drill holes spaced at greater intervals if a full channel sampling or diamond drill hole survey is found to be too costly. It should also be possible to reduce the number of men required to operate the control. Any mine introducing a sub-level caving mining method, however, should first investigate fully the possibility of applying a full grade control, as this is considered to be far superior to any other method.

General conclusions

(1) The sub-level caving mining method is being increasingly considered and employed in large orebodies due to the ease of mechanization, the relative safety of the method, the possibility of selective mining in complex orebodies, easier standardization of equipment and fewer labour problems. Referring to the last point, the possibility of almost completely mechanizing underground operations leads to reduced labour requirements and in highly industrialized and sophisticated countries the main problem is to find men willing to work underground. The mines find it easier to recruit personnel to operate mechanized equipment, where

most of the hard work is taken out of the job, than to recruit into mines where hard manual work is still required for most operations.

(2) Extensive research into sub-level caving draw characteristics has indicated that for large orebodies the method can compete in terms of recovery and dilution with other methods, such as block caving.

(3) It is considered that, due to the inherent draw characteristics, a positive draw control is imperative in both base metal and ferrous ores if the maximum benefit is to be obtained from sub-level caving. Apart from reducing dilution and direct operating costs (sub-level loading, main haulage tramming, hoisting and concentration), the following advantages are obtained. (*a*) Management could adjust the recovery and dilution more accurately by varying the draw grade. This is important if marketing the ore became increasingly difficult and a reduction in mining costs became necessary. If, on the other hand, the market improved, it would be possible to control the reduction in draw grade to increase the recovery and dilution to a new acceptable level. Adjustment of recovery and dilution would be very difficult to accomplish if control underground were visual and the efficiencies were based mainly on information obtained from the surface reduction plants. (*b*) A positive draw control could aid the task of increasing the efficiency of mining operations, in that a badly run section can be detected easily from the draw results. (*c*) A positive control would also aid planning engineers considerably in evaluating the present layout designs and therefore assist in the research into improved layouts.

Acknowledgement

The author is indebted to Atlas Copco AB, Sweden, for enabling a young graduate to make an extensive tour of Swedish mines that has proved, and will continue to prove, of great value. Thanks are due to the Swedish Mining Association and to the many Swedish mining companies for their hospitality and cooperation in providing a wealth of information on modern mining practices.

The author is grateful to Mufulira Copper Mines, Ltd., for permission to publish this paper, and to the company personnel who helped in criticizing and suggesting improvements. Special acknowledgement is made to the late W. J. Nixon, whose help and criticism were invaluable during the preliminary stages of preparing the paper.

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Besides the references cited below, unpublished technical reports of Mufulira Copper Mines, Ltd., have been consulted. Illustrations of present and projected mining layouts were obtained from Kiruna and Malmberget of LKAB and from Norberg mine, Sweden.

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Appendix Portable Isotope Fluorescence (P.I.F.) analyser

The P.I.F. analyser is best described in the words of the introduction to the Hilger and Watts instruction manual on the instrument. 'The Portable Isotope Fluorescence Analyser is an X-ray fluorescence instrument which uses a radio-active isotope to excite the X-radiation of the element being analysed. The radiation characteristic of this element is isolated from the general background radiation by the use of two balanced filters which are inserted into the X-ray beam in sequence. One filter (filter α) permits the passage of background radiation only and the second filter (filter β) allows the characteristic element radiation to pass in addition to the background radiation. The measurement of radiation in terms of "counts per second" is shown on a meter. The radiation from the element being analysed is measured by subtracting the reading obtained when using filter α from that obtained when using filter β.

The instrument is calibrated by reference to a set of "standard samples" and analysis is carried out by comparing the readings taken from the sample being analysed with the standard sample readings.

The isotope and filters used are selected to suit the particular element being analysed and by changing the radio-active source and the filters (or in some cases filters only) other elements can be analysed.'

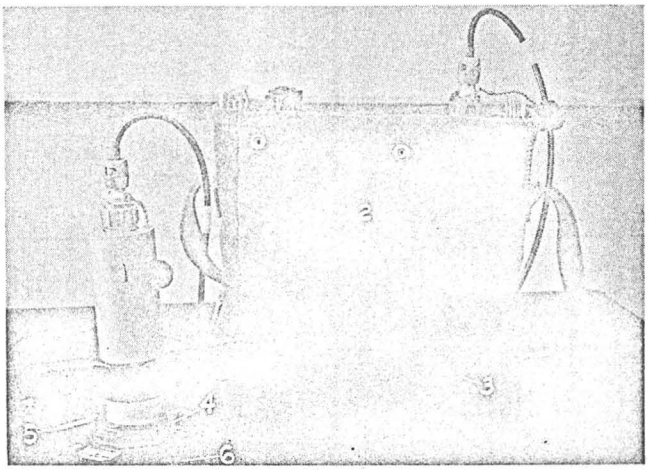

Fig. 1 P.I.F. analyser

The instrument is in two parts—the probe and the electronic measuring unit (see Fig. 1). The probe (1 in Fig. 1) consists of a measuring head, filter tray and detector. The measuring head carries the radioactive source and is supported on three buttons (legs), one of which is a safety device (6 in Fig. 1). The lever operating the safety shutter in front of the source (4 in Fig. 1) cannot be moved until the safety button is depressed. In normal operation the unit is completely safe and presents no danger to the operator. A second lever (5 in Fig. 1) provides the selection for either filter. The filter tray consists of a pair of specially balanced

filters situated between the measuring head and the detector. The filters are absorption-edge filters. The detector unit consists of a NaI (Tl) phosphor mounted in front of an E.M.I. photomultiplier, and the whole assembly is suitably protected against dust and moisture. The electronic measuring unit (2 in Fig. 1) supplies the voltage to the detector in the probe through an extra high tension/signal lead (3 in Fig. 1). It receives and amplifies the signal from the detector and displays it on a meter (Fig. 2). The whole unit is powered by two 6-V dry batteries of the Ever-Ready PP1 type and, with constant use, they usually last 3–4 weeks.

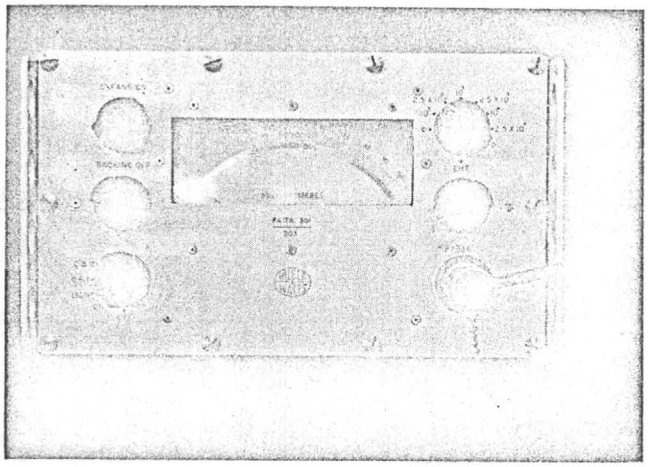

Fig. 2 Control knobs and meter on P.I.F. electronic measuring unit

After adjusting the unit for the type and grade (minimum and maximum) of ore to be analysed a graph is plotted from several samples of known grade, of the grade per cent against the difference in the meter readings, i.e. the filter β–filter α readings. After obtaining the readings from a sample the grade can be obtained directly from the graph. Full details of the instrument can be obtained from the manufacturers.

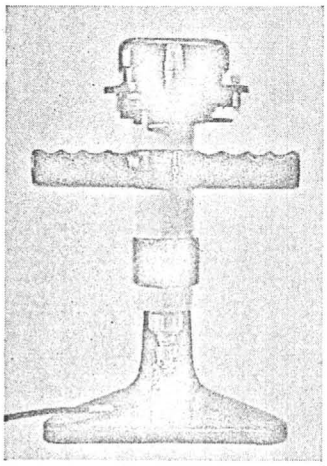

Fig. 3 P.I.F. probe mounted on a stand for use in the inverted position. Without the stand continual manual pressure is required during the analysis

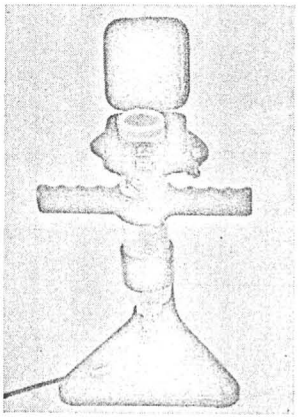

Fig. 4 P.I.F. probe mounted on the stand with lead protection lid open showing sample to be analysed

The stand and arrangement to permit the operation of the instrument in the inverted position is shown in Figs. 3 and 4. This method will rule out the need for the operator to apply continuous manual pressure, and more precise readings should be obtained.

Discussions and contributions

Introduction of the cascade method of continuous-retreat open stoping at Mufulira Copper Mines, Ltd., Zambia

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Author's reply to discussion* on paper published in *Transactions/Section A* (*Mining industry*), vol. 75, July, 1966, pp. A137–46

L. D. Airey In reply, first, to the questions raised by Mr. J. T. M. Taylor (pp. A87–8).

The information already given about the limits of dip within which the cascade method of mining would work successfully was based on the solution of problems encountered in the layout of the extraction opening. A minimum length of loading crosscut is required in which the type of loader selected can operate, considering the shuttling movements necessary. Where the dip is steeper, a drawpoint crosscut would be too short for these movements and, where flatter, the drawpoint crosscut would be excessively long and in waste rock. In addition, with flat dips, an excessive amount of waste coning is necessary, for recovery of stope ore and pillar recovery is poor and in steeper dips the stope and pillar size is considerably reduced unless vertical intervals are increased. In areas of steeper dip, say 50° and over, other methods of sub-level stoping with long back lengths are cheaper, as advantage can be taken of gravity. The loading of ore in this case may still be done mechanically. Fig. 1 illustrates these points.

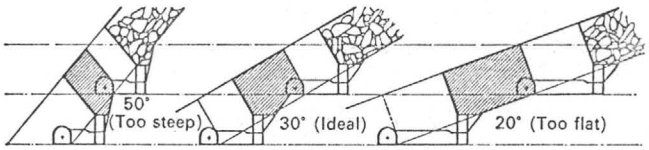

Fig. 1 Sections showing limitations of method with regard to dip

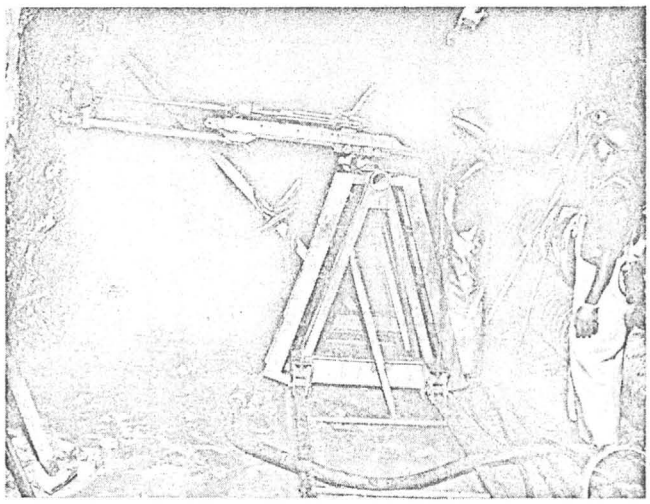

Fig. 2 Drifter on 6-ft cradle with pneumatic rod spotting clamp on redesigned drill rig

*Trans. Instn Min. Metall. (Sect. A: Min. industry). 76, April 1967, A87–8.

The use of elaborate rigs is not justified without the availability of sufficiently skilled labour to operate and maintain them in operating condition at the working place. The lower performance, due to breakdown and damage of complex rigs compared with the cheaper simple rig, has been adequately proved in practice. The best Swedish practice at Kiruna or Strassa, for example, indicates that about 500 ft per shift can be drilled by a skilled driller. At Mufulira an average of 200 ft per shift can be drilled with the equipment illustrated in Fig. 2, which also shows the rod clamp.

The pneumatically operated rod spotting clamp consists of two diametrically opposed pistons forming a coupling clamp on an extensible arm attached to the drill cradle ahead of the drill. The ends of the pistons coming together allow the drill rod to pass through, acting as a rod guide, especially when collaring holes on a rock face some distance away from the end of the drill cradle. The main purpose is to hold the coupling firmly, and hence the rods in a hole, enabling one man to perform the operation of adding or removing rods in a string.

The cleaning of spiral or other inclined roadways is now being done successfully with front-end loaders, but at the time in question no front-end loaders had arrived on the property and the equipment used was what was available.

At one stage difficulty was experienced in charging buttress pillar holes, due to cutoff holes and ground movement, owing to the layout of the fans of drill holes. This was overcome by altering the layout so that each portion of pillar included the relevant draw-point crosscut completely and holes drilled from a subsequent crosscut did not protrude over the crosscut from which the previous portion of pillar was blasted. Fig. 3 shows the revised layout.

The ore breaks to a smooth hanging-wall.

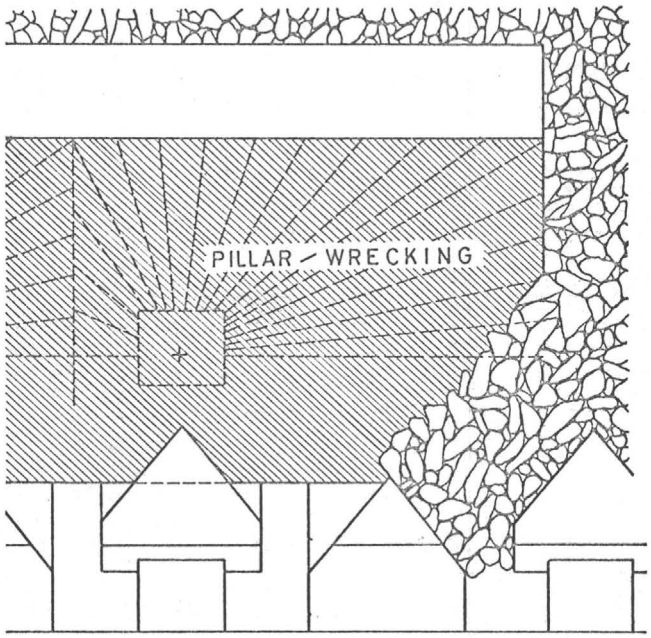

Fig. 3 Vertical longitudinal projection illustrating change in pillar drilling layout

The costs of the transloader and front-end loader are not directly comparable as, although they were on similar duties, they were working in different areas and the front-end loaders had the shorter travelling distance.

Directly comparable costs are still not available, though in the cascade mining area transloader operating costs have remained substantially the same as they were, while in the sub-level caving area three front-end loaders have now between them extracted 663 000 tons in 10 500 total working hours at an operating cost of 1·64s./ton, of which 0·55s./ton is tyre cost. This rate of 63 tons/h is satisfactory and in line with original estimations.

When evaluating extraction machinery, three main factors are considered: total operating cost, production rate and spares availability. Technical features of the machine are of secondary importance. The front-end loader was found to have a production rate at least as good as the transloader, and better on short travelling distances. It is more economical to operate and substantially cheaper to purchase. Choice of a particular make of front-end loader was made on spares availability and the machine delivery times.

It is necessary to pressurize the intakes of the cascade stoping area with auxiliary fans, in addition to using a main exhaust fan, in order to have effective air distribution over the working levels—otherwise the air flow would take the shortest and easiest route, thereby starving some levels. The auxiliary fans with short lengths of ducting are located in the drives on each level off the intake point at the spiral roadway and boost the air flow, acting as regulators themselves, no doors being used.

In reply to Mr. P. Westerberg (p. A88), tyre costs are heavy, being the largest single operating cost. On some particularly onerous duties tyre costs have exceeded all other operating costs put together.

Rudd chains have been tried and did protect the tyres. But the cost of using chains on the hard and abrasive ground at Mufulira equalled the savings made on tyres and as the necessary down-time for chain repairs was a nuisance they are no longer used. In addition, towards the end of the life of the chains large numbers of worn sharp links are scattered freely on the roadways and these do more damage to other tyres.

Pizzuti tracks have also been tried. These are hinged steel plates which envelop the tread and part of the sidewall of the tyre, forming a continuous track. They did not show an overall saving in costs.

An extensive investigation into tyre costs is now being undertaken and one early conclusion is that, insofar as we are aware, no manufacturer has a tyre designed exclusively for use over broken ground at low speeds. All tyres now available are made light enough to avoid heat build-up due to continuous running at speeds very much higher than those possible in mining applications. The mining industry requires a very tough tyre having great depth of rubber and until one becomes available tyre costs will remain higher than they need be. Some international tyre companies are working on this problem.

In reply to Mr. R. J. Clemmow (p. A88), the planes of the rings of drill holes are inclined to the strike of the orebody to aid the breaking of the toes of the rings. The holes subtend a more 'open' angle with the hanging-wall and have the same additional advantages as the inclined holes on open-pit benches, mainly by reducing the effects of 'build-up'.

The drilling rigs illustrated in the paper were originally designed for drilling in the sub-level caving area, where the inclination of holes is generally steeper than in the cascade mining area and the machine would be drilling in a near vertical plane. In this instance it was necessary for the machine to be drilling on its side for most of the time and undue wear was experienced on the slides. The rig has since been redesigned so that the machine drills in a vertical plane. The new rig is shown in Fig. 2.

ROCK MECHANICS OF BLOCK CAVING OPERATIONS

by

Seth D. Woodruff*

INTRODUCTION

This paper deals with the stresses which are involved in the fracturing of ore, and subsidence of blocks, in caving operations and with the pressures which develop on workings underlying a caving block.

Block caving involves the undercutting of large blocks of ore. Stresses induced in the ore after the undercutting cause it to crush and fracture as the broken ore is drawn off at the bottom. As the ore is drawn off the block settles and subsidence eventually extends to the surface.

The sizes of individual blocks which are caved vary with the physical characteristics of the ore and the dimensions of the orebody. Blocks in various mines have varied from about 37 ft x 75 ft in lateral dimensions up to about 250 ft x 250 ft in lateral dimensions. Heights of blocks are generally between 100 ft and 300 ft although blocks up to 500 ft high have been successfully caved in the asbestos mines of Quebec.

The pressure exerted by the subsiding ore on the supports of openings beneath caving blocks varies with the lateral dimensions as well as with the height of a block. A block should be small enough to avoid excessive crushing of supporting timber. On the other hand, a block must be large enough so that stresses created in the ore will be sufficient to fracture the ore.

Some porphyry copper ores will cave when as little as 700 square feet of a block is undercut. On the other hand, in a strong rock, such as limestone, undercuts about 80 x 200 ft have been required to initiate caving. Long and Obert[1] cite an instance in which 35,000 square feet of a block was undercut without producing a cave.

Mechanics of Ore Fracture

Self Supporting Rock Structures

A block of ore which has been undercut has certain self-supporting tendencies. To obtain successful caving it is necessary to create stresses which are sufficiently large to overcome this self-supporting tendency of the ore block.

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A strong, intact rock may support itself over an opening in the manner of a beam. In fractured rock, or rock which has little tensile strength, a beam cannot exist. Under such conditions the rock derives its strength (that is its tendency to support itself) from "arching effect." The simplest form of arch is the "linear arch" which is formed when a beam is cut in the center as shown in Fig. 1. This same beam may be cut transversely in a number of places and as long as the direction of the cuts does not approach the direction of the maximum shear stress the "linear arch" will not be appreciably weakened.

(Note: The term "arching effect" as used herein does not signify the formation of a vaulted or "peaked" roof or back. Rather, the term refers to the stress distribution within the rock. "Arching effect" signifies that the rock tends to support itself because the vertical forces due to the weight of the rock are resolved into diagonal thrust forces. These forces are carried by the rock at the sides of the undercut opening.)

A man-made masonry arch - the voussoir arch - has been used in civil engineering structures since early Roman times. Such a structure consists of a series of shaped rock, or concrete, blocks as shown in Fig. 2. Such a structure tends to form naturally in the rock when an opening is excavated in fractured rock.

In badly fractured rock the roof tends to stabilize itself in the form of a vaulted (arch shaped) roof. Rock tends to fall from the roof until all tensile stresses are eliminated and only compressive stresses exist along the arch line. Thus the rock tends to cave to form a natural voussoir arch which will be self supporting.

In caving operations it is necessary to create stresses which are sufficient to overcome the strength of such naturally formed arch structures.

Caving in Weak Rock

Since the back of an undercut is essentially a flat surface it resembles the bottom of a beam and initial failure of the undercut block occurs at this surface in a manner similar to failure of a beam. Stresses are the greatest at the center of the back and most ore fragments fall from this location. The vaulted opening so formed works its way up until it reaches the natural arch line of the ore. Along this line stresses are compressive and parallel to the arched rock surface. This caving process is illustrated in Fig. 3. The back has arched up until stresses parallel to the back are compressive. Under such conditions shear failure of the rock under compressive stress becomes important in causing breakdown of the structure. Shear stresses are maximum at the ends, or abutments, of the arched back.

It is not necessary that any visible opening exist between the pile of broken ore and the ore which is supporting itself in place. In weak rock, or in a granular material, there may be no visible plane of discontinuity between the de-coupled ore and that which still appears to be in place. A partial removal of the supporting pressure of de-coupled ore may be brought about by sub-

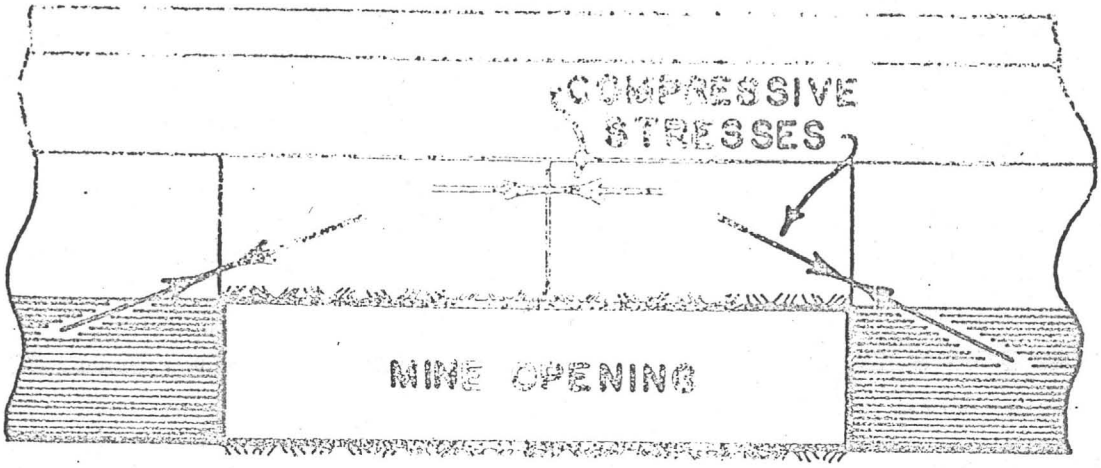

FIGURE 1. A Simple "Linear Arch" Formed by Fracture of the Roof Beam

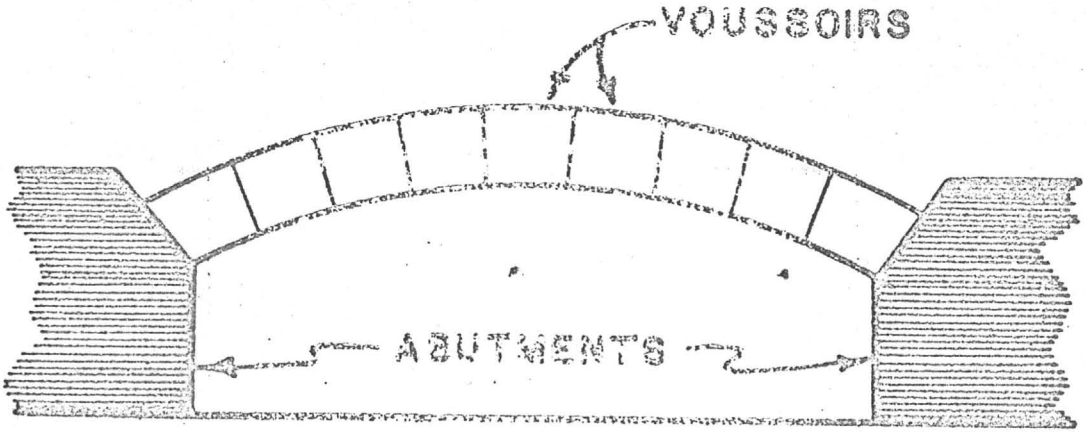

FIGURE 2. A Typical Voussoir Arch

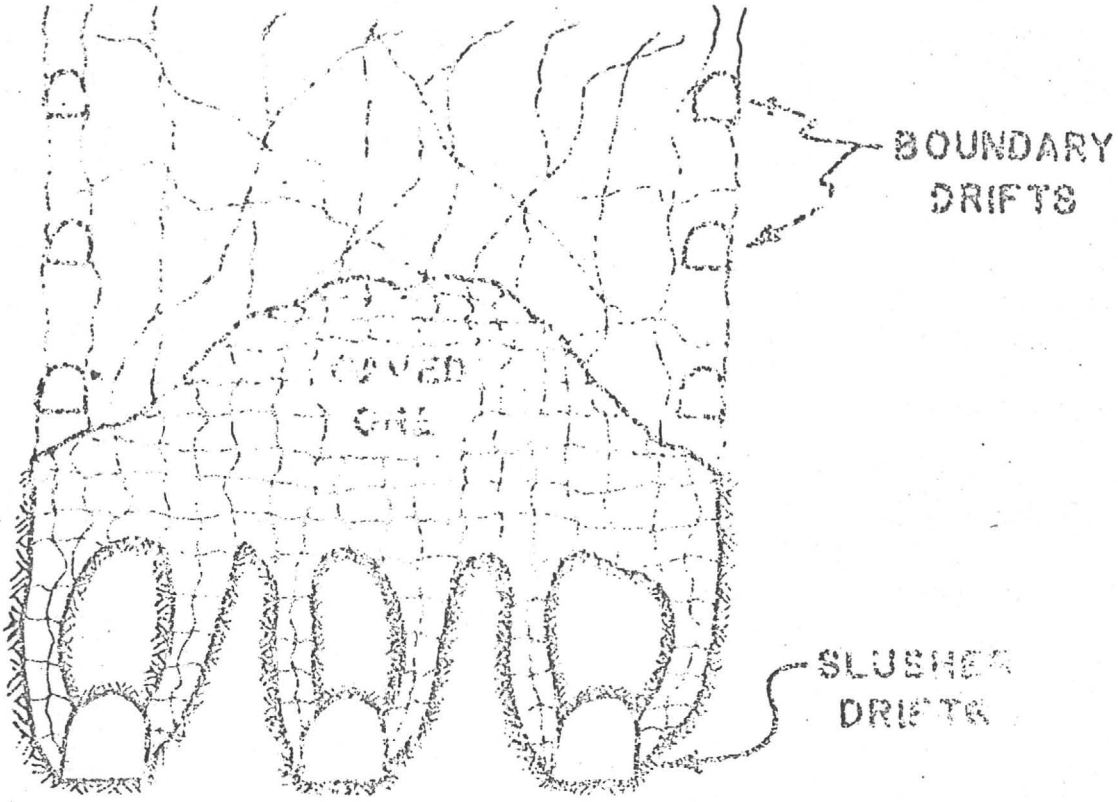

FIGURE 3. Block Caving Action in Weak Ground

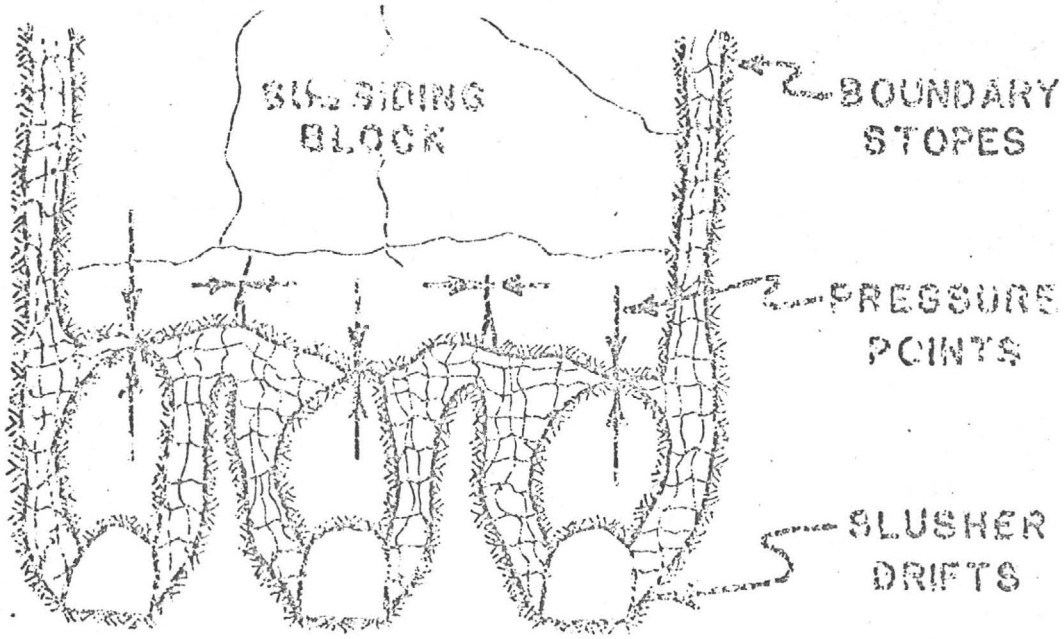

FIGURE 4. Fracturing Produced by Block Subsidence in Strong Rock

sidence of the de-coupled ore. Even a slight subsidence may be sufficient to cause the disaggregation of more of the overlying ore while all ore fragments are still in contact with each other.

Caving in Strong Rock

When the ore to be caved is strong in tension then the cave line does not form a vaulted arch but the ore subsides instead as an integral block and fracturing takes place under compressive stress in the lower portions. The mode of fracture is illustrated in Fig. 4. As the block settles after undercutting is completed it contacts the pillars at random points and the concentration of compressive stresses at these points cause the rock to fracture by shearing and crushing. Other points of high compressive stress occur where individual large ore blocks contact each other in forming crude linear arches. Each of these contact points is eroded away by progressive shear fracturing until other random support points are established. These in turn are reduced by shear induced by excessive compression with the process repeating itself as long as the resultant broken material is taken from the appropriate drawpoints. If the broken material is not drawn off as it forms it may be so compacted by the weight of the overlying block that it will be impossible to draw.

Example - Caving in Strong Rock vs. Caving in Weak Rock

Block caving operations are usually confined to massive deposits of structurally weak ores. An exception to this was the block caving operation in limestone at the Crestmore Mine of the Riverside Portland Cement Co., at Riverside, Calif.[17] A limestone bed with a thickness of about 300 ft was mined. The rock had a compressive strength of about 7000 to 11000 psi. In order to secure subsidence of a block the following measures were used.

1. The blocks were completely cut off on all sides by either vertical shrinkage stopes or caved areas. In block caving in weak rocks only portions of block boundaries are cut by boundary drifts.

2. The area of the undercut required to start a block caving was much greater than that in weaker ores. An undercut area of about 16,000 square feet was generally required to start caving and as much as 35,000 square feet were undercut in one instance without causing a cave. In contrast to this in some porphyry copper ores caving can be induced by undercutting an area as small as 625 square feet.

3. In normal block caving the most widely accepted practice is to pull equal tonnage from each drawpoint according to a pre-determined daily schedule. At Crestmore rock could only be pulled from those drawpoints which were situated under those points which were highly stressed enough to fracture the rock. Such points were indicated by the relatively copious amount of powdered rock in the drawpoint chutes. Pulls were made and continued only from drawpoints where the powdered rock was evident until a substantial void, perhaps 75 ft in diameter, had been created. The weight of the block was thus transmitted to other local-

ized areas around the perimeter of the void and pulling was shifted to drawpoints where the presence of powdered rock in the chutes indicated a high stress area. This arching to form a void was in contrast with the usual action in porphyry copper ores where sustained draw from one set of drawpoints usually results in funneling, or piping, up to the surface. Distribution of maximum pressure points could not be predicted in advance and such points could only be located by searching to find the locations where rock powder had accumulated in the chutes.

Stresses in Broken Rock at Drawpoints

As the broken ore converges on a drawhole transverse stresses are brought into play. These are due to "arching effect" as the fragments crowd together in entering the drawhole. As shown in Fig. 5 a few large pieces can form an arch across the entrance to obstruct the drawhole. Similar stresses exist in the fragmented ore even when it is flowing freely through the raise. However, the raise will not clog because a stable arch cannot form as long as the shear stresses in the broken ore exceed the shear strength of the mass.

Whether or not broken ore will form an arch to obstruct a drawhole depends upon the relative size of the fragments, the coefficient of friction between fragments, and the vertical pressure to which the fragmented ore is subjected.

Pressures on Support Due to "Arching Effect"

Severe local pressures on drift support may develop where the ore breaks into relatively large fragments, or where it contains a percentage of large strong inclusions. Pulling at one drawpoint causes arches to form which have abutments on support for other surrounding drawpoints. Even heavy concrete lining for slusher drifts may be broken by the intense point pressures developed by lateral thrust due to such arching. Fig. 6 illustrates this type of pressure development.

Boundaries of Caving Blocks

Each block is ordinarily separated from surrounding ground by planes of weakness created by boundary shrinkage stopes or by driving a series of drifts, one above the other, along the selected boundary. These boundaries ordinarily suffice to provide a barrier along which broken ore subsides and which served to prevent most exterior rock from mixing with the ore in the block.

Even when caving of a block progresses upward into the overlying capping the limiting planes of subsidence usually remain essentially vertical if the cross-sectional area of the block is relatively small as compared with its depth beneath the ground surface.

Fig. 7 shows the theoretical stress conditions at the boundaries of a caving block. In this case the lateral dimensions of the block are small as compared to its depth. The narrower the width of a block relative to its depth the more does the subsiding ore tend to arch across. This "arching effect"

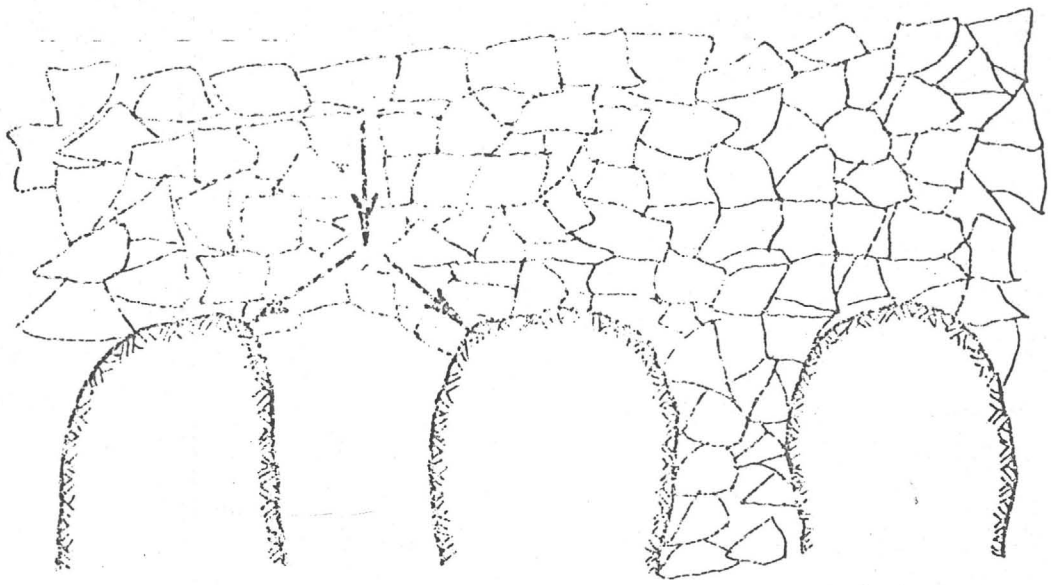

FIGURE 5. Drawpoint Stoppage by Arching.

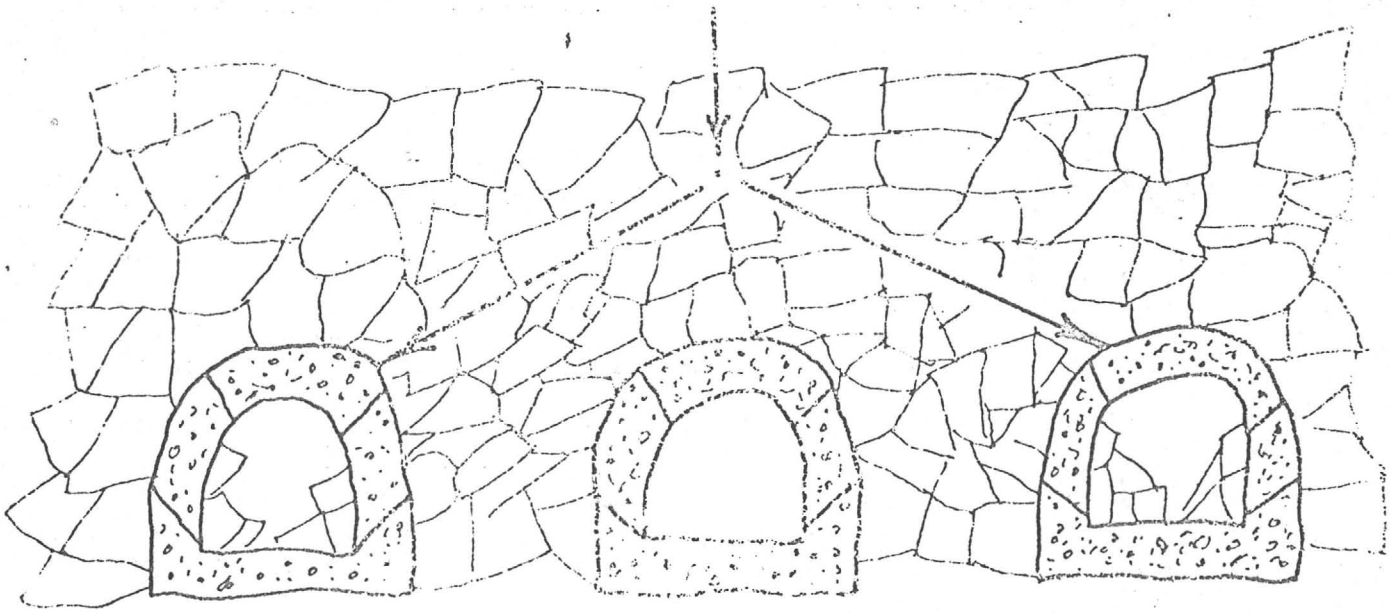

FIGURE 6. Destructive Localized Pressures Due to Arching Effect in Coarsely Broken Rock

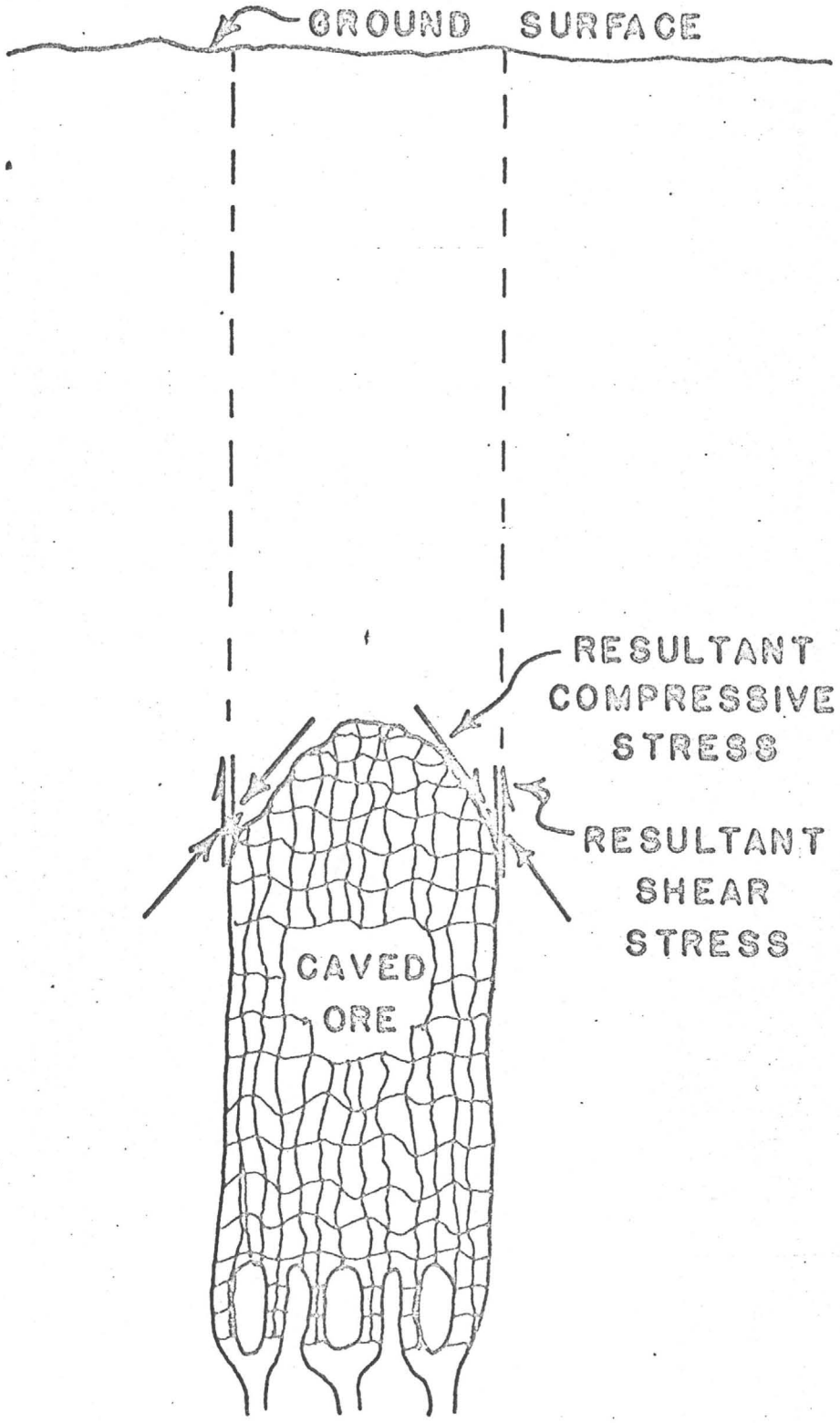

FIGURE 7. Caving Action in a Narrow Block in Fractured Rock. The Boundaries Are Steeply Inclined, or Vertical.

results in diagonal thrust stresses against the adjacent rock. The resultant principal compressive stress is inclined rather than vertical, and the resultant fracture occurs along shear planes inclined at $45^\circ - \frac{\phi}{2}$ to this principal compressive stress. The resultant planes of movement will be steeply inclined, or vertical, as shown in Fig. 7. (ϕ = the angle of internal friction of the ore)

Examples of "Chimney Caving"

Fig. 8 is taken from a curve plotted by Marr[2/] which shows the "angle of draw" observed over mined out areas in British coal mines. (Note: Marr measured the "angle of draw" from a vertical line drawn through the edge of the mined out area rather than from a horizontal plane.) Marr observed that the angle of draw decreased as the ratio of the width of the mined-out area to its depth increased. That is, the boundaries of the subsiding block approach the vertical as the width/depth ratio of the mined-out area decreases. Such subsidence progressing upward between nearly vertical planes has been termed "chimney caving" or "piping".

An example of chimney caving has been cited by Rice[3/]. In this instance an opening 14 ft wide and 28 ft long was made in Hanbury slates in order to obtain filling material by caving. The caving proceeded, filling material being drawn off, until after about a year's time the caving had worked through to the surface in the form of a vertical chimney similar in cross section to the opening 900 ft below. The dip of the slates was about 60° from the horizontal but the caving cut diagonally across the bedding.

Still another example of chimney caving is shown in Fig. 9. In this case the subsidence of the block (along nearly vertical planes) of jasper capping nearly 1900 ft thick resulted from the extraction of a block of ore by top slicing.

Arching Effect in Horizontal Planes

When a block of ground is of limited length, as well as of limited width, the effect of "arching effect" or "ring action" in a horizontal plane must be considered. This tends to prevent movement of ground toward an excavation and is probably also a factor in causing subsidence to follow nearly vertical planes. The ground surrounding the subsiding block may be considered as being a horizontal ring in compression.

Landslide-Type Subsidence

Once vertical subsidence has broken through to the surface vertical banks may form temporarily. When these become high enough a "landslide-type" of slope slump occurs. Non-cohesive material cannot form vertical banks but if the material possesses some cohesion then high vertical banks may form before the weight of the material causes a landslide to develop. The progression of sub-

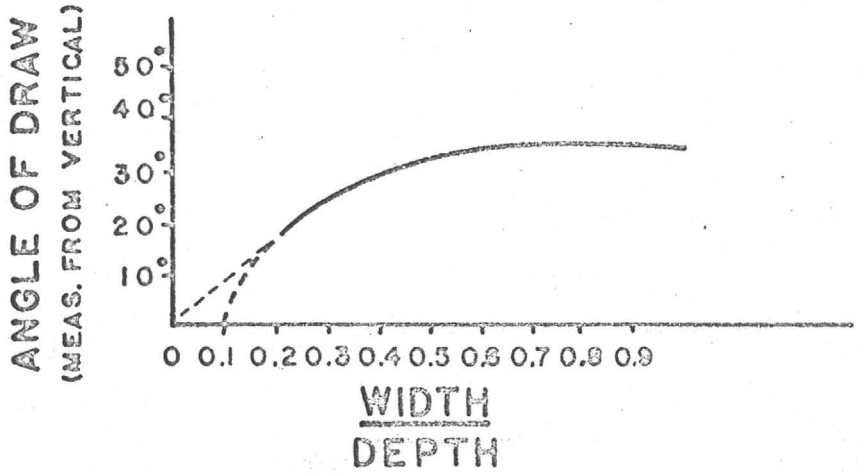

FIGURE 8. Angle of Draw vs. the Width/Depth Ratio. (after Marr)

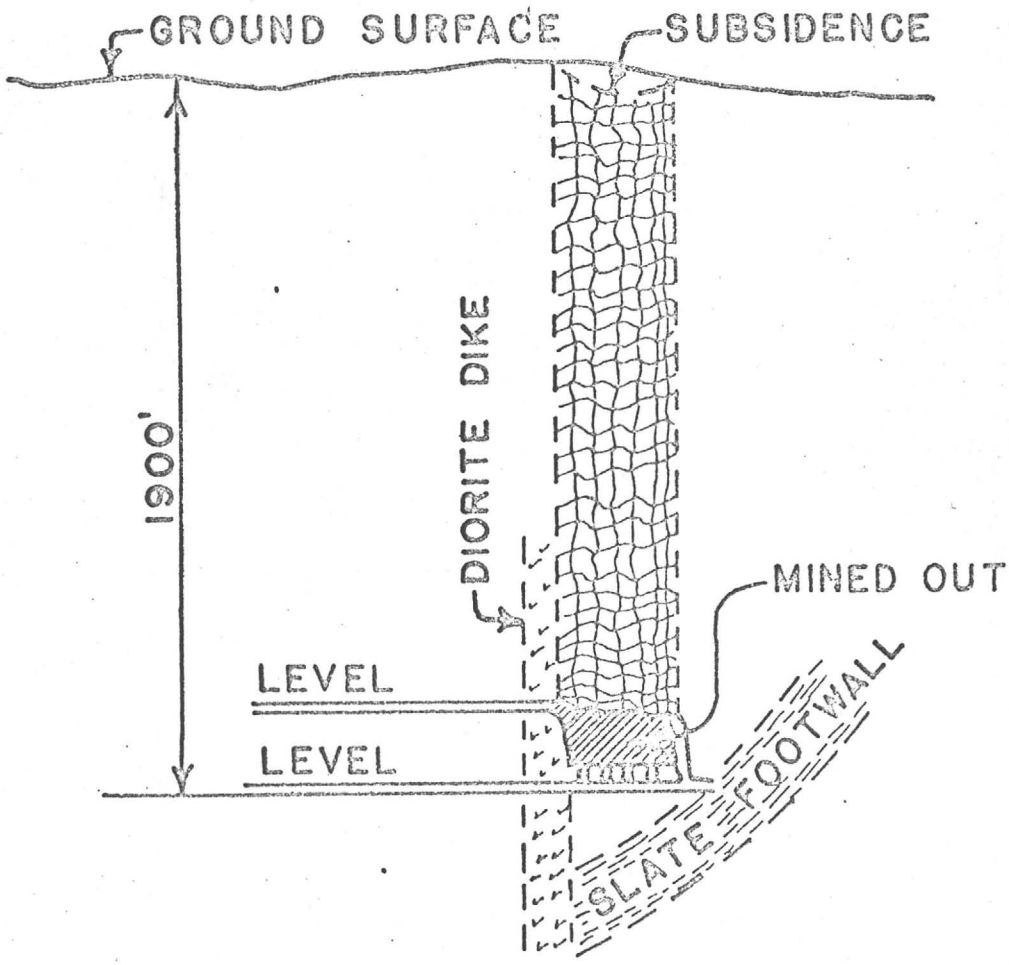

FIGURE 9. An Example of "chimney caving" at the Athens Mine, Negaunee, Mich. (after C. W. Allen)

sidence to the surface and the slumping toward the surface depression are shown in Fig. 10.

Pressures at the Bottom of a Caving Block

As shown in Fig. 11, the subsidence of an isolated block of fractured ore may be likened to the drawing off of a granular material from a bin. The boundaries of the block represent the sides of the bin and the drawholes would correspond to bin hoppers. It is of interest to calculate average pressures at the bottom of such a block by means of Jannsen's formula. This was derived for the purpose of calculating pressures of granular materials on the bottoms of bins. This formula has been modified by Terzaghi for application in the field of soil mechanics.

The example which follows Fig. 11 assumes that the material being caved is a fragmented material and that an equal amount of ore is drawn from each point so that uniform subsidence takes place. In the example the average unit vertical pressure at the bottom of the ore is computed to be equivalent to the weight of a depth of 44 ft of rock, or to about 22% of the actual rock depth. The remainder of the weight of the rock block is transferred by "arching effect" to the rock at the boundaries of the caving block.

Pressure Variations

The foregoing computations do not give any information as to the actual variation of pressure across the bottom of the block. Rather the computed result is an average pressure across the entire area of the block bottom.

Fig. 12 shows the pressure distribution across the bottom of an experimental model circular bin 12 inches in diameter and 24 inches deep filled with dry sand.[4] The average measured vertical pressure on the bin bottom was about 0.73 psi or equivalent to 57% of the total weight of the sand. The remaining 43% of the weight of the sand was transferred to the walls of the bin by friction and "arching effect". Maximum vertical pressure occurred at the center of the bin. A similar pattern of pressure distribution might be expected at the bottom of a caving block of similar dimensions.

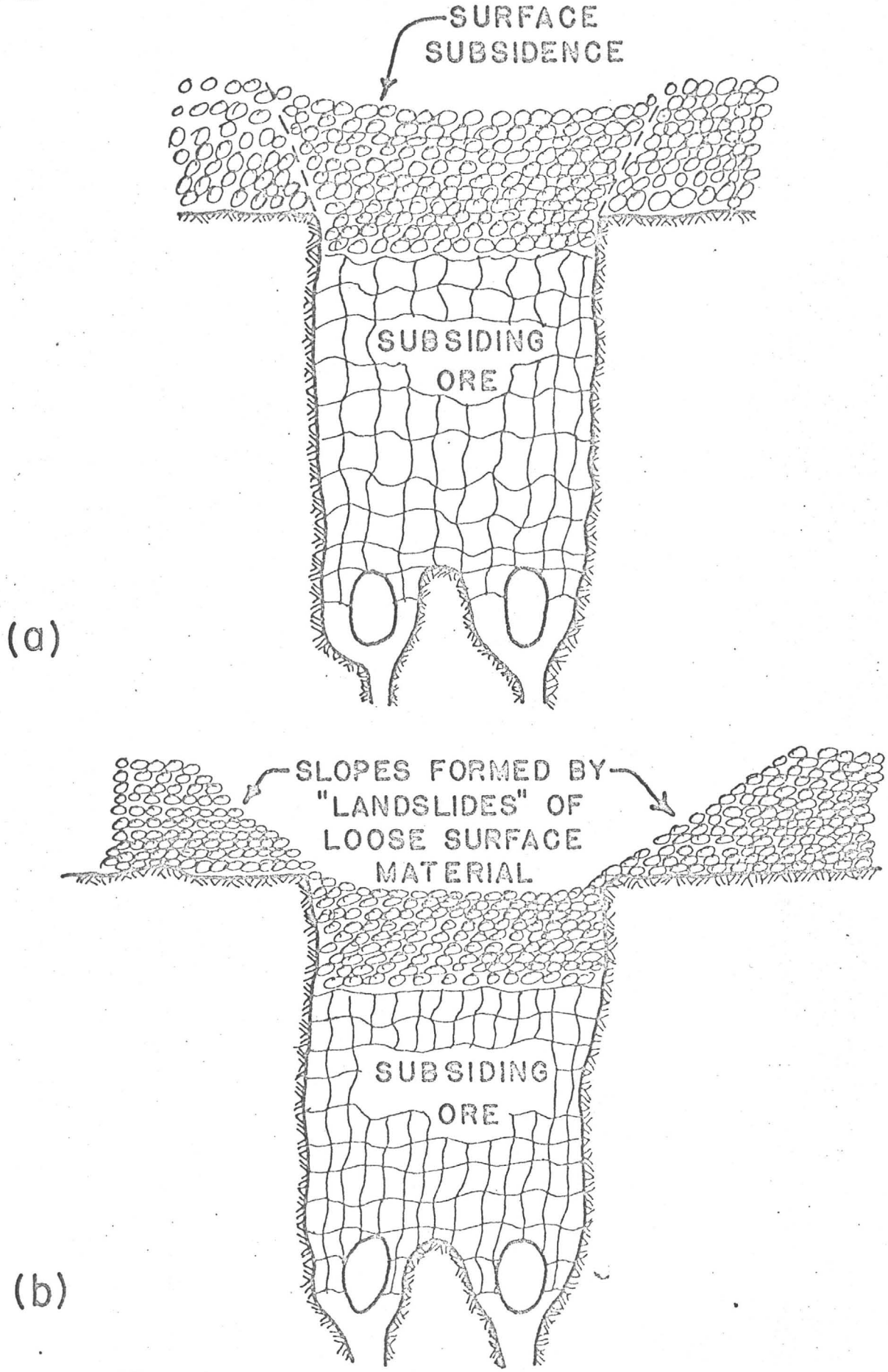

FIGURE 10. Subsidence of Surface over a Caving Block.

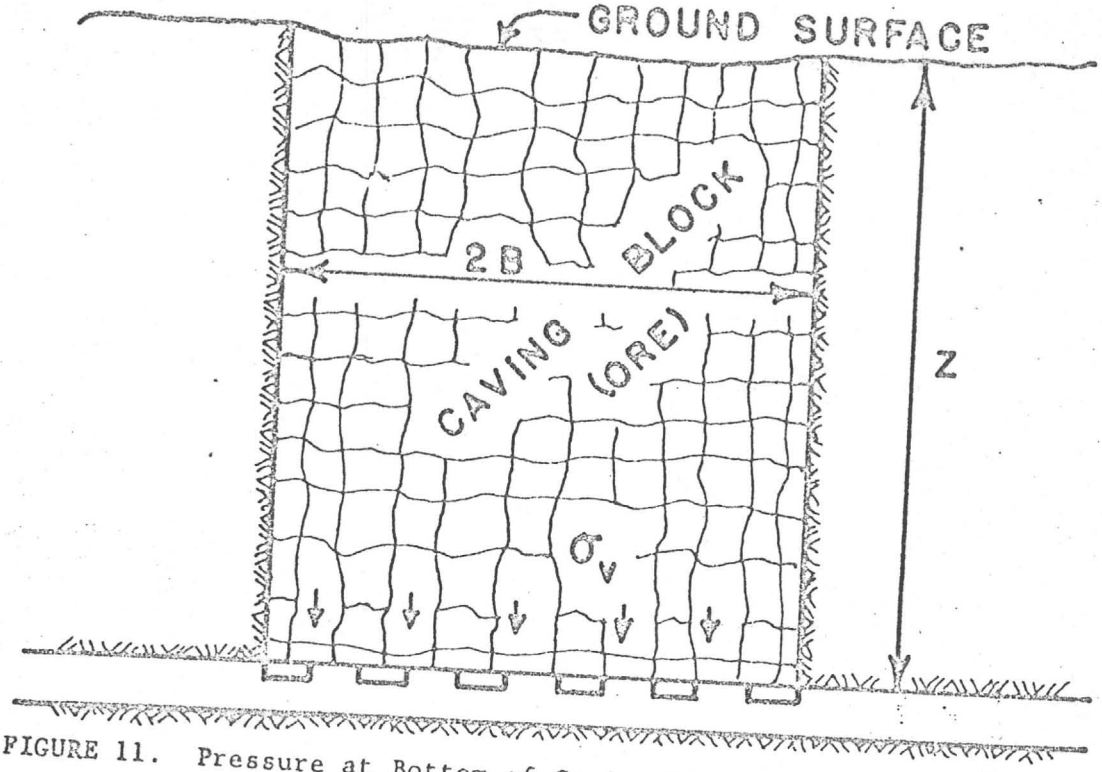

FIGURE 11. Pressure at Bottom of Caving Block

$$\sigma_v = \frac{\gamma R}{i \tan \theta} \left(1 - \epsilon^{-i \tan \theta \frac{Z}{R}}\right)$$

where:

σ_v = Average unit pressure at bottom of caving block.

γ = Density of ore in pcf.

$i = \frac{1 - \sin\phi}{1 + \sin\phi}$

ϕ = Angle of internal friction of the fractured ore.

Z = Depth from surface to undercut level.

R = "Hydraulic radius" of the caving block,

that is: $R = \frac{\text{(horizontal cross-sectional area of block)}}{\text{(perimeter of block)}}$

$\tan\theta$ = Coefficient of friction between the caving block and adjoining walls.

Assuming a caving block 150 ft square and 200 ft high and taking γ = 170 pcf, ϕ = 45; and $\tan\theta$ = 0.90 then:

$$R = \frac{150 \times 150}{4 \times 150} = 37.5$$

$$\sigma_v = \frac{(170)(37.5)}{(.17)(.90)} \left[1 - \epsilon^{-(.17)(.90)\left(\frac{200}{37.5}\right)}\right]$$

$= (41,600)(1 - .82) = 7500$ psf which is the average vertical pressure.

The average vertical pressure at the bottom of the block is equal to a depth of 44 ft of rock or to about 22% of the full depth pressure.

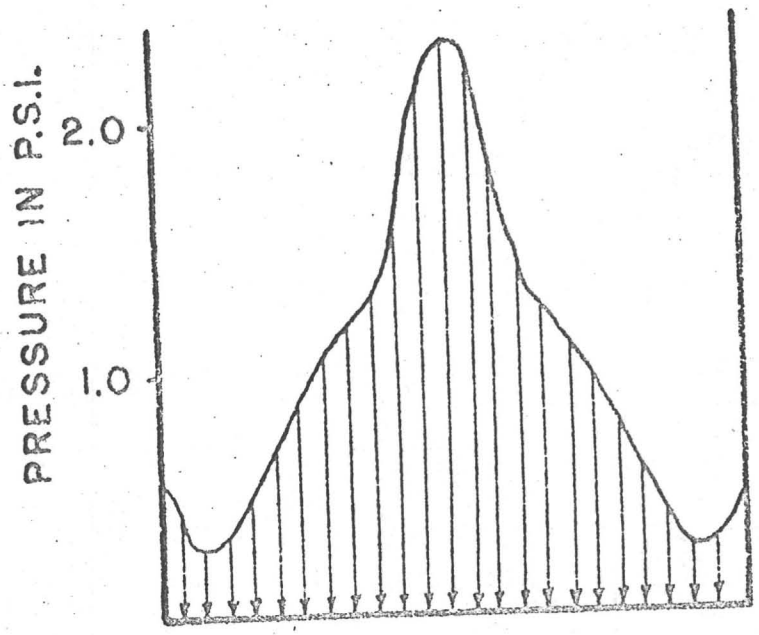

FIGURE 12. Vertical pressure distribution on the bottom of a full non-flowing bin (after Lucas & Verner).

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