CONTACT INFORMATION
Mining Records Curator
Arizona Geological Survey
1520 West Adams St.
Phoenix, AZ 85007
602-771-1601
http://www.azgs.az.gov
inquiries@azgs.az.gov

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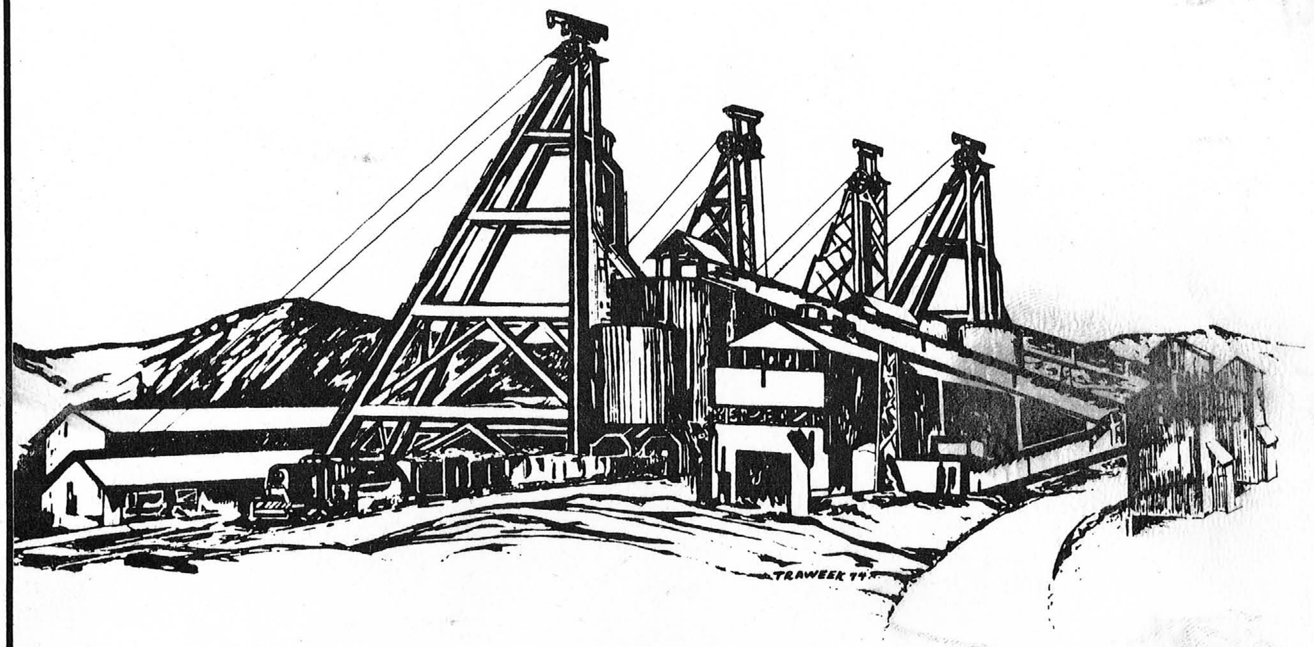
MAGMA COPPER COMPANY



1984 SPRING SESSION
MINING GEOLOGY DIVISION
ARIZONA CONFERENCE OF AIME

HOSTED BY:
MAGMA COPPER COMPANY

A Subsidiary of Newmont Mining Corporation

San Manuel, Division

San Manuel, Arizona

MAGMA COPPER COMPANY
A SUBSIDIARY OF NEWMONT MINING CORPORATION
SAN MANUEL DIVISION
P. O. Box M, San Manuel, Arizona 85631 (602) 385-2201

Mining Geology at San Manuel

Presented To
Mining Geology Spring Meeting, 1984
Arizona Conference——A.I.M.E.

JUNE 1, 1984

David A. Baker - Chairman

PROGRAM

1. Registration at the San Manuel High School Auditorium, San Manuel.

2. Introduction

3. Technical Session -- San Manuel High School.

   Speakers:

   - L. A. Thomas — A Summary of the Development of the San Manuel Mine Subsidence Area
   - D. A. Baker — A General Description of the San Manuel Oxide Deposit
   - L. A. Sandbak — Continuous Miner Drift Excavation and Geomechanical Rock Classification at San Manuel
   - R. L. Hockett — Tiger: A Brief Look at Current Utilization and Potential.

4. Luncheon at the San Manuel Country Club.

5. Surface tour of the San Manuel subsidence zone, the San Manuel oxide outcrop, and the Tiger silica quarry.

History and Development of the San Manuel Mine

An Excerpt From a Paper By:

J. F. Buchanan

F. H. Buchella

Location and History

The San Manuel copper deposit is located about 45 miles northeast of Tucson. The concentrator, smelter, administration building, and other plant facilities are located about seven miles southeast of the mine area at the new town of San Manuel as shown on Figure 1 (sketch).

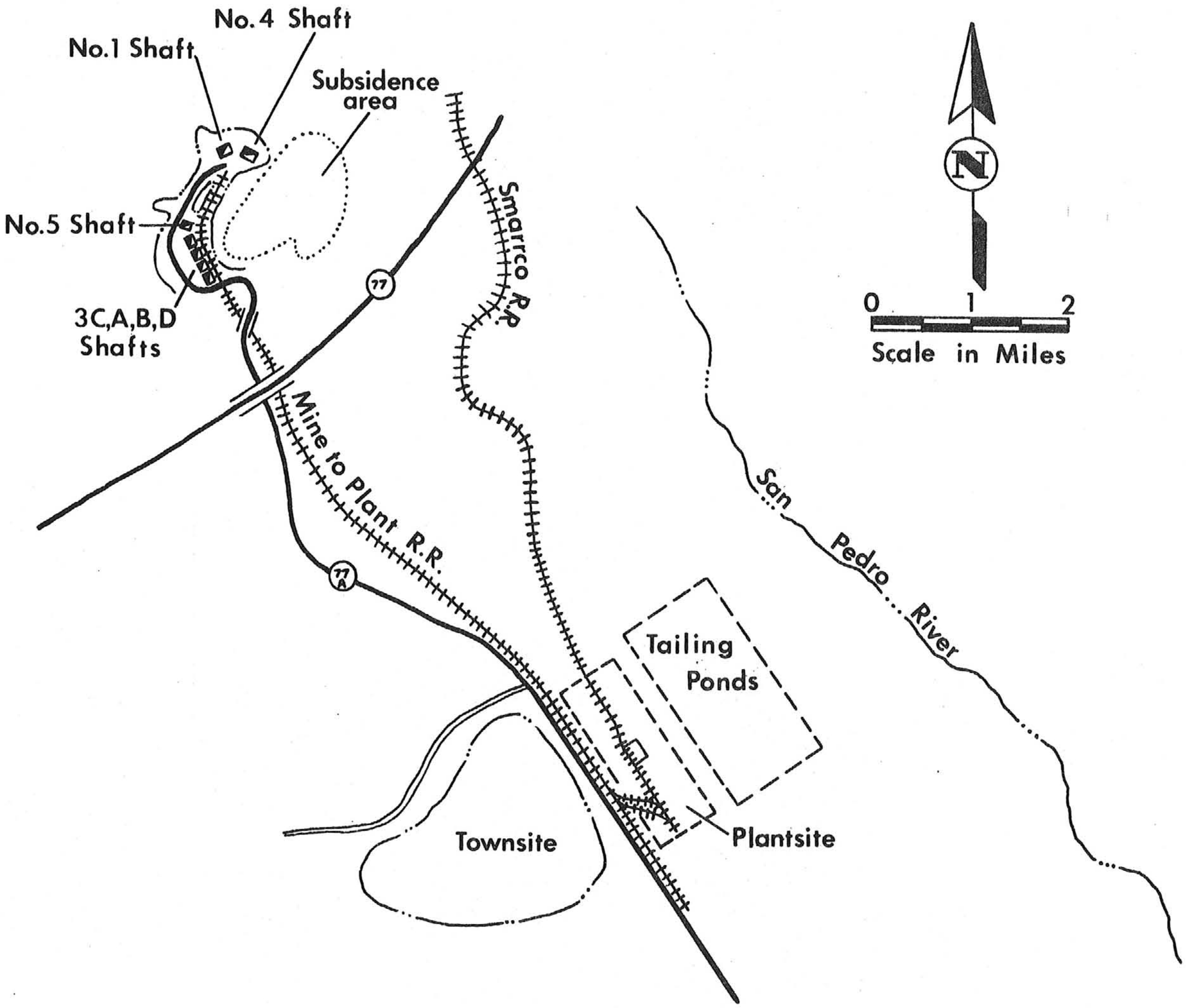


Figure 1

The chief producer in the area formerly was the Mammoth-St. Anthony Mining Company, operating the Mammoth and Mohawk Mines; located a mile north of the San Manuel Mine. These mines were worked intermittently from the 1800's to 1952. The Mammoth-St. Anthony property is now owned by the San Manuel Copper Corporation.

Interest was attracted to the prominent surface exposures of the San Manuel deposit because of its proximity to an operating mine. There are records of claims on the property to 1900, but these early claims were soon abandoned.

The first exploration work of consequence on the San Manuel property occurred in 1916, when several churn drill holes were put down by a group of mining men who had obtained an option. One of these holes intersected what is now considered oxidized ore, but at that time there was little interest in low grade copper deposits and the option was allowed to expire.

The San Manuel group of claims, which have been maintained continuously, and from which the mine obtained its name, were staked in 1925. These claims eventually were acquired by four men who actively promoted the property. As a result of their efforts, engineers from the United States Bureau of Mines examined the property in March of 1943 and recommended a limited amount of test drilling, which was started in November of that year. The results were encouraging and drilling was continued by the bureau until February

1945. At that time, 17 churn drill holes had been put down with a total of 15,844 ft. drilled. This exploration disclosed substantial tonnages of ore which was mainly oxidized, but which conformed in grade to what was subsequently found to be the average of the deposit. This was too low-grade for a small-scale operation, but the possibility that further drilling would extend the discovery created considerable interest in the property.

In 1944 Magma Copper Company obtained an option to purchase the property and started churn drilling in December of that year. Magma exercised its option on September 17, 1945, and also purchased and located additional adjoining claims. All of the property so acquired by Magma was deeded to San Manuel Copper Corporation, which was organized in 1945 as an operating subsidiary of Magma Copper Company.

The churn drill exploration was essentially completed on March 17, 1948, when No. 1 Shaft was collared, and major mine development started in January 1953. Mine production began about three years later when the undercutting of the first block was started. This block was completely undercut on January 23, 1956.

Exploration--Churn Drilling

The churn drill program initiated by the Bureau of Mines was expanded into a grid with holes spaced on 100-ft. centers along the long axis of the deposit, and on 200-ft. centers at right angles to the long axis. As drilling progressed into the southeastern area of the deposit, the vertical ore section was found to be relatively higher and of greater width than the ore in the northern area, and the churn drill grid was expanded to a spacing of holes of 400-ft. centers both along the length and width of the ore deposit. Sludge samples for assay and panned concentrates for mineral identification were collected at 5-ft. intervals through the mineralized zone.

The churn drill exploration program was terminated soon after sufficient ore had been outlined to proceed with the underground exploration of the ore deposit. In all, 109 holes were completed, and a total of 205,536 ft. drilled at a cost of over two and one-half million dollars.

Exploration--Underground

To confirm the grade of ore indicated by churn drilling and to gain first-hand information about the mining and metallurgical characteristics of the ore, a timbered shaft No. 2 was sunk to a depth of 2,068 ft. in the center of the sulfide ore body. The shaft also served for the development of the first lift prepared for mining.

An underground exploration program was undertaken for the purpose of determining the mining limits of the Southeast Ore Body. The 1285 exploration level, located 105 ft. above the planned undercut level, was developed to the northeast and southwest of No. 2 Shaft by driving 2,784 ft. of drift approximately through the center of the south ore zone. The ore zone was then ring-drilled on 210-ft. spacings along the long axis of the ore body. The drill core (AX)--about 3/4 inch--was broken into 5-ft. samples and the whole core used for assay. Sludge samples were taken from the first holes drilled but were not found to be as reliable as core samples, and this practice was discontinued. Cut samples, car samples, and surface truck samples checked with the diamond drill core samples. Churn drill assays were found to run about 0.02% lower than other samples, probably due to dilution of the drill cuttings.

Description of the Deposit

Based on an assay cutoff of 0.5% copper, the ore blocked out by churn drilling, plus the initial underground exploration, consisted of 367,624,000 tons of sulfide ore at 0.785% copper and 111,876,000 tons of oxidized ore at 0.717% copper, or a total reserve of 478,500,000 tons averaging 0.769% copper.

In cross section, the ore body is neary U-shaped, with the "U" leaning to the northwest. Consequently, the upper part of the ore zone is split into two branches which, for convenience, are called the North Ore Body and the South Ore Body. With the exception of a small oxidized outcrop, the ore and most of the surrounding mineralized rock are overlain by post-mineral conglomerate that varies from a feather edge to more than 1,900 ft. in depth. The contact of the conglomerate with the underlying mineralized rock is a fault of regional scope, the San Manuel fault, and forms the most important structure in the area (see Figures 2 and 3).

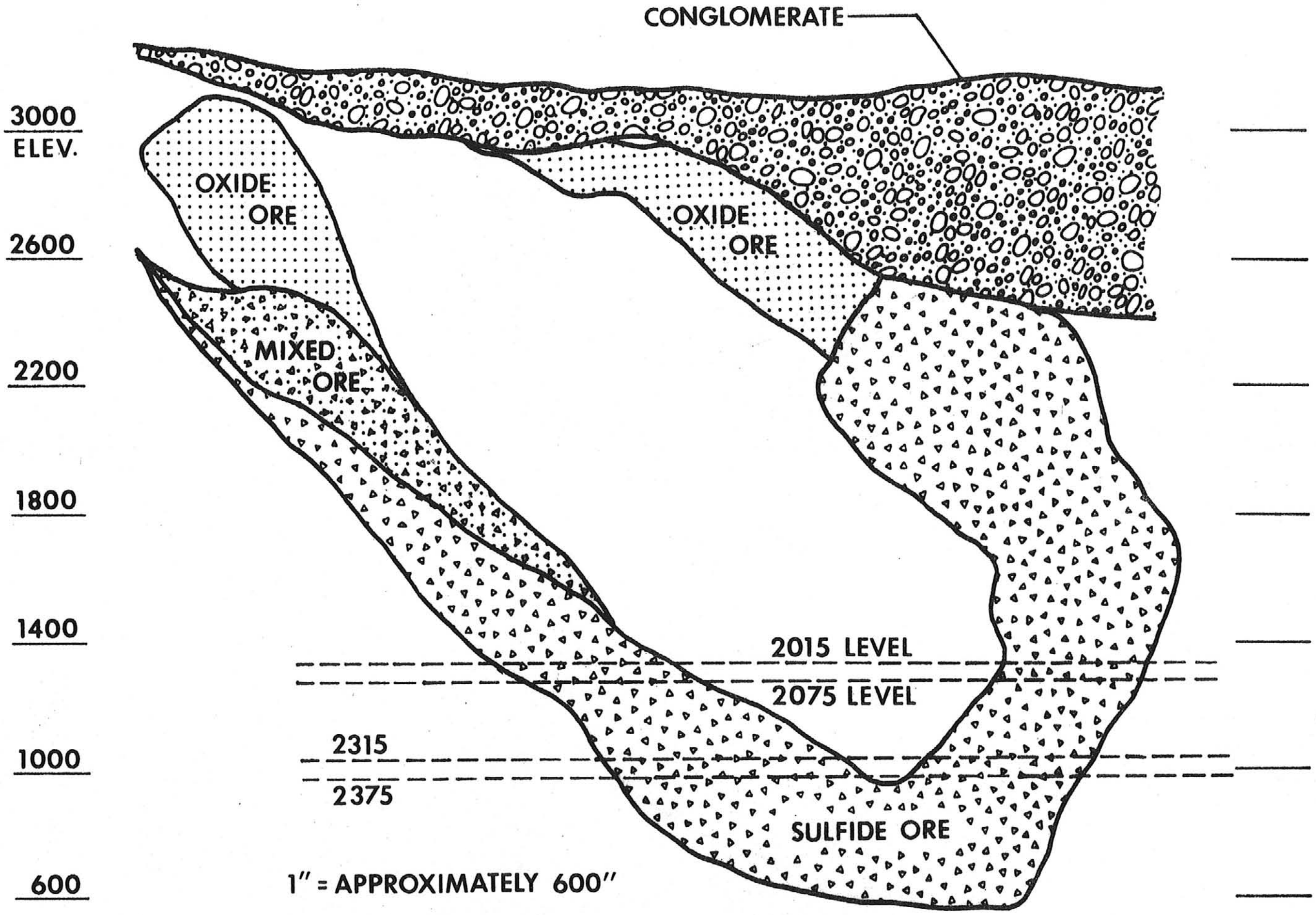


Figure 2: Cross section through ore body, N 46° W, looking northeast.

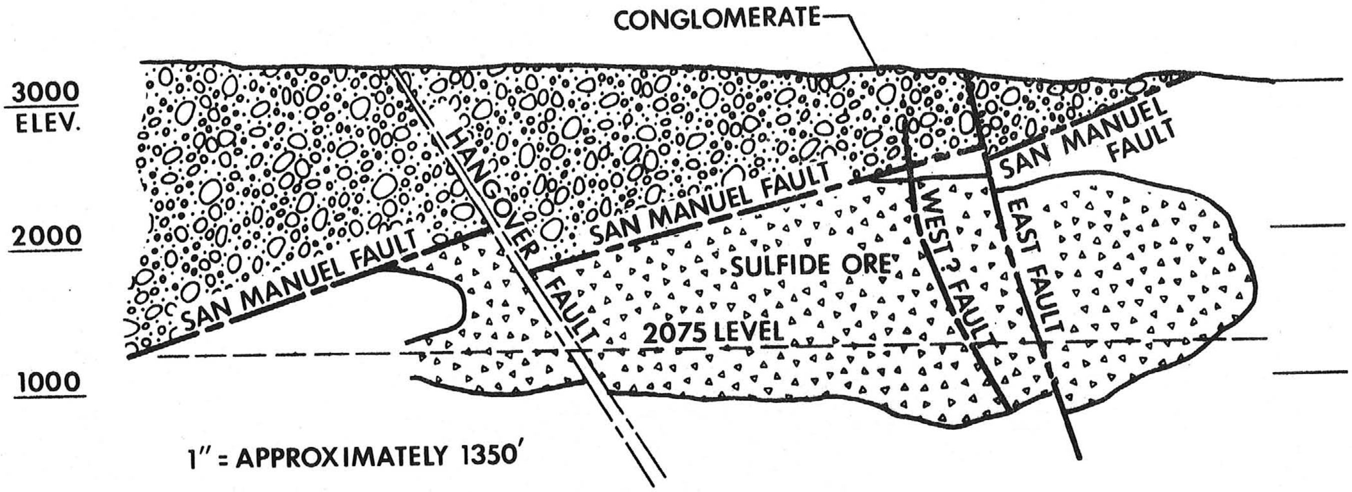


Figure 3: Longitudinal section through Nos. 2 and 3B Shafts, looking northwest.

The ore consists essentially of disseminated chalcopyrite in quartz monzonite and monzonite porphyry. Some chalcocite enrichment is present but tonnage-wise is minor.

The sulfide ore ranges in depth from 475 ft. to 2,665 ft. Much of the upper part of the ore body has been oxidized in varying degrees, mainly to chrysocolla. The bottom of oxidation is irregular and extends from depth of about 400 ft. to over 1,600 ft.

(1) - "History and Development of the San Manuel Mine," J. F. Buchanan and F. H. Buchella, A.I.M.E. Transactions, Vo. 217, 1960.

MAGMA COPPER COMPANY

President and Chief Executive Officer: Gordon R. Parker.

General Manager, San Manuel Division: John W. Goss

General Manager, Superior Division: Frank M. Florez.

Chairman Emeritus: Wesley P. Goss

Chairman of the Board: Wayne H. Burt

Founded: 1910 - Superior, Arizona

Merged: 1969 - Newmont Mining Corporation

1983 Production: 110,000,000 short tons copper
3,140 tons MoS_2
26,000 ounces gold
466,000 ounces silver
plus minor platinum
palladium
selenium
rhenium

Employees: 3,600

Current Ore Reserves:

San Manuel/Kalamazoo:	±684 million tons at average grade of 0.7% copper, 0.03% molybdenum.
Superior:	±4.4 million tons at average grade of 5.7% copper.

Miscellaneous

Largest electric power user on APS system.

Produces up to 20% of power needs with steam turbines operating from smelter waste heat.

Refinery uses power equal to 500 homes.

Water Usage: 28,000 gallons per minute continuously reclaimed and recycled. Water is pumped from mine and supplemented from deep (2,000 ft.) artesian wells; no water is discharged from mine or plant.

Seven Labor Unions -- major one is Steelworkers.

San Manuel Arizona Railroad Company, subsidiary
ICC licensed shortline connects with Southern Pacific at Hayden.

Magma Arizona Railroad Company connecting Superior and Florence Junction.

Both railroads are commercial freight haulers.

Customers: Wire, cable, brass, and copper fabricators.

Land Holdings: About 30,500 acres deeded land.
About 33,000 acres leased land.
Total is about 1.5% of Pinal County.

San Manuel Community

Population: Approximately 5,500.

Elevation: 3,500 ft.

Founded: 1953.

Original Contractor: Del E. Webb.

Government: Unincorporated under Pinal Board of Supervisors.

Utilities: Arizona Public Service
Arizona Water Company
Mountain Bell Telphone
Magma furnishes sewage and garbage service.

Private Ownership: All businesses; about 200 homes.

Company Rentals: About 75% below Phoenix-Tucson averages.

Churches: Seven.

Schools: 2 elementary; 1 middle school; 1 high school.

Parks and Playgrounds: Provided by Magma.

Golf Course: Private 9-hole golf club.

Hospital: General hospital (3 doctors, 40 beds) provided by Magma. Open to public at cost of services furnished.

Demography: About 35% of Magma's employees live here. About 40% commute from Tucson and the rest live in Oracle, Mammoth, and Catalina.

History

1879	Mines started in Old Hat District around Mammoth (gold, silver mines).
1896	Post Office opens for Schultz, Arizona.
1915-19	Molybdenum and Vanadium campaigns.
1934	All mines consolidated by St. Anthony Mining Company.
1939	Schultz changed to Tiger.
1935-43	Renewal of activity; lead and zinc campaigns.
1942	War Production Board investigates area for copper.
1943-45	Exploration drilling by U.S. Bureau of Mines.
1944	Magma consolidates and purchases claims, begins additional exploration and drilling.
1948	Underground exploration and development begins.
1952	Federal loan of $94,000,000 made to Magma to begin mining copper.
1953	Construction begins on surface plant.
1956	First stopes undercut and first smelting beings.
1965	Expansion from 30,000 tons of ore per day to 40,000 TPD.
1968	Purchase of adjacent ore body doubles size of reserves.
1969	Merges with Newmont Mining Corporation.
1972	Expansion from 40,000 to 60,000 TPD ore processing. Operation of electrolytic refinery and continuous rod casting begins.
1973	Installation of air quality control systems and sulfuric acid plant.
1978	First tests of Tiger flux for smelter.
1982	Superior Division placed on standby.
1983	Commenced testing to lead to production from oxide ores.

A Summary of the Development of the San Manuel Mine Subsidence Area

L.A. Thomas
Chief Planning & Geological Engineer

ABSTRACT

The subsidence area over the San Manuel Mine is the result of mining approximately 400,000,000 tons of ore over a 28-year production span. Four hundred million tons have been withdrawn from three major caving levels located at depths of 1,415, 2,015, and 2,615 ft., plus subsidiary levels at depths of 1,715 and 2,315 ft. In plan view the subsidence takes the shape of a recumbent letter "A" which has its top to the NE and its legs to the SW, reflecting the shape of a plan view of the underlying ore body. The maximum dimensions of the area within the major scarp line are 6,500 ft. along the NE axis and 3,600 ft. along the NW axis. By contrast, the maximum plan view dimensions of the ore body which was undercut to produce this cave are 5,800 ft. by 2,800 ft.; hence, the perimeter of the major scarp line is somewhat greater than the composite undercut perimeter.

Maximum depth of the interior of the caved ground with respect to original topography is 700 ft. located along a northeasterly trending trough which is positioned over the south limb of the ore body. This trough represents a zone from which 60% of the original 1,800-ft. height of the rock mass above undercut has been withdrawn.

The development of the cave has been monitored by Magma ever since startup of the operation and monitoring continues to this day. Hence, a large body of information on the genesis of the cave exists in Company files and highlights of this data will be described.

REGIONAL SETTING

Regionally, the San Manuel Ore Body consists of an elliptical shaped ore shell of 300-1,000 ft. width which occupies the general contact zone between a central core comprised of Laramide porphyry and an outer pre-Cambrian porphyritic quartz monzonite. The latter rock has long been referred to as the "Oracle Granite." The entire mineralized system is capped by a wedge of intermontane conglomerate locally termed "Gila," which has a thickness varying from tens of ft. at the NE end of the axis to hundreds of ft. at the SW end.

Both igneous host rocks are closely fractured and strongly enough altered to cause the rock mass to have low overall strength characteristics. Consequently, when undercut, the mass fails quickly and caving action is easy to propagate upwards. By contrast, the sedimentary Gila capping is a much tougher rock, relatively unfractured except on major post-Gila structural zones which normally trend about N 30° W. Hence, the initiation and propagation of failure within the Gila is much more difficult than in the underlying igneous mass. The Gila forms prominent vertical escarpments, especially where it is several hundred ft. thick.

A General Description of the San Manuel Oxide Deposit

D. A. Baker
Senior Geologist

Introduction

The San Manuel oxide mineral deposit was explored during the mid 1940's while churn drilling what is now known as the San Manuel porphyry copper deposit. The initial churn drilling was done based on a 200-ft. drill grid with no attempt to fill in all the theoretical intersections of that grid. This drilling did clearly delineate the oxide deposit as being the oxidized upper portion of the San Manuel sulfide ore shell. Subsequent to this initial exploration, the oxide zone has been probed utilizing underground diamond drilling to define the complex sulfide-oxide interrelationship for mine planning purposes, and more recently down-the-hole hammer, reverse circulation drilling has been utilized to embellish the early churn drilling results pertaining to grade lensing and mineralization. The cumulative drilling has blocked out many tens of millions of tons of oxide material.

A brief understanding of the general geological relationships at San Manuel will be useful in discussing the oxide deposit geology.

The plutonic Precambrian quartz monzonite was intruded by monzonite porphyry during the Laramide Orogeny. Thus, the monzonite porphyry formed the core of a cylinder approximately 8,000 feet in length with an elliptical cross section of some 5,000 feet by 2,500 feet along the respective major and minor axes. The pulse of copper mineralization and concordant hydrothermal alteration closely followed and was centered around the monzonite porphyry emplacement. The economic ore zone, as defined by a 0.5% total copper assay value, generally occupies the elliptical area along the contact zone between the monzonite porphyry and quartz monzonite. Within the zone bounded by the 0.5% total copper assay boundary exists a poorly mineralized core referred to as the "low grade interior core." Grading outward from the economic ore zone boundary is a decrease in the total copper assay values and a marked increase in the pyrite content of the rock.

Subsequently, the cylinder was tilted to the northeast and concurrently buried beneath thick strata of sedimentary and volcanic origin. The cylinder was bisected by faulting with the upper portion of this tilted cylinder being transported approximately 8,000 ft. to the southwest.

The down faulted portion of this cylinder is known as the Kalamazoo deposit. The portion of the cylinder in the footwall of the fault zone is the San Manuel ore body.

The important primary ore minerals associated with the San Manuel/Kalamazoo system are pyrite, chalcopyrite, and molybdenite.

The Kalamazoo deposit has not undergone significant oxidation or supergene enrichment. The upper portion of the San Manuel deposit has been subjected to oxidation and some supergene enrichment.

The San Manuel Oxide Deposit

The main thrust of oxide mineralization and supergene enrichment has occurred over the north limb of the San Manuel ore body. The North ore body, as it is called, crops out and exposes a mineralized area of approximately 80,000 square feet. The oxide deposit occurs within the same granitic host rocks as the primary sulfide ore and also occurs predominately within the phyllic alteration zone.

The oxide zone exhibits traceable mineralogical and grade lensing patterns similar to grade lensing patterns defined within the primary sulfide system. These patterns within the oxide are, however, less consistent than those defined within the sulfide system. A series of steep northwest trending, easterly dipping, normal Basin and Range age faults have offset both the oxide and sulfide mineralogical and grade lensing patterns, with offsets in the range of fifty to one hundred fifty feet. Additional disruption and displacement of the mineralogical and grade lensing has been the result of the sulfide block cave mining operation, which was initiated without regard to its impact on the oxide deposit above. This portion of the oxide deposit that existed above the primary sulfide draw cones has been rubblized and lies draped down and across the various north limb undercuts on the 1415, 1715, 2015, and the 2315 levels.

This rubblized portion of the deposit is in essence a homogeneous oxide mass with no intrinsic grade or mineral lensing. This mass forms a composite of the previously existing grade lenses. Sulfide mining draw has caused some waste to pipe down into this rubblized mass. Of particular interest is the large subsidence pipe that formed the original surface breakthrough from the 1400 level North ore body sulfide mining. This pipe has pulled approximately 400,000 tons of post ore material well into the oxide zone. In any evaluation of the San Manuel oxide system, the distinction between the oxide material outside of the sulfide mining draw cones and the oxide material within the influence of sulfide mining draw cones is critical.

Mineralization

Chrysocolla is the prominant copper mineral of the oxidized zone. Chrysocolla at San Manuel is generally bluish-green in color, although the colors range from pale blue to sky blue. The large variety of colors is a reflection of the wide compositional range of this mineral.

The most prevalent occurrence of chrysocolla is as coatings on the surface of rocks, including vugs of chrysocolla and fractures partially filled with chrysocolla and ferruginous oxides.

The second most common occurrence is chrysocolla associated with quartz veinlets, cutting into the interior of the rocks.

Chrysocolla also occurs as disseminations and stringers within the rock interiors, generally associated with quartz.

Cuprite is also present within the oxide zone, particularly along the northern flank bounded by the oxidized pyritic zone. The occurrence of the red colored cuprite at San Manuel is very difficult to identify due to the quantity of red iron oxides prevalent throughout the oxide zone. In many drill intercepts there does not appear to be adequate quantities of chrysocolla to account for the soluble copper content. It is probable that a certain portion of the soluble assay can be attributed to cuprite which has been masked by the iron oxides.

Other copper oxide minerals of minor importance within the oxide zone are malachite, azurite, atacamite, dioptase, and native copper. None of these minerals exist in sufficient quantities to be of economic importance.

Mineralogical studies have indicated that some copper within the oxide zone exists as exotic copper bearing minerals. Particularly within the ferruginous oxides and alterations clays' lattice works. This copper can represent a significant portion of the total copper assay value where these minerals are present in major amounts.

Other interesting occurrences of exotic copper are associated with the oxide zone. The post ore, barren andesite and rhyolite dikes, both intensely fractured, have emplaced along these fractures quantities of chrysocolla with assay values to 0.8% total copper.

Portions of the conglomerate within the vicinity of the North ore body outcrop contain quantities of chrysocolla precipitated along the bedding planes and within the matrix of coarser interbeds.

By far the most prevalent oxide minerals within the oxide zone belong to the limonite group. The majority of the iron oxides were derived from the oxidation of pyrite. Zones of fine cellular boxworks and other remnants of pyrite are evident throughout the oxide zone. The distinct red color of the iron oxides is very prominent in the few bedrock surface exposures north of the ore body. This coloration gives rise to the name of one of the prominent landmarks in the mine area, Red Hill. Early studies concluded that the iron oxides present are predominantly goethitie mixed with lesser amounts of hematite, with the finely divided hematite giving the oxides their overall red color.

Supergene Enrichment

The supergene enrichment of copper in the San Manuel ore body has been extremely variable. Supergene enrichment tended to be more prevalent along the footwall portion of the system bounded by the pyritic zone than along the hanging wall of the system bounded by the low grade interior core.

The ratios of pyrite and primary copper sulfides along the footwall of the ore body were good precipitants for the development of the supergene copper zone.

The low grade interior core material associated with the hanging wall was originally metal deficient and was not as intensely fractured as the primary ore shell or the pyritic zone, making the percolation of solutions through the interior core difficult. Thus, the system had neither adequate solutions nor adequate precipitants to form a supergene copper zone.

Chalcocite is the predominant copper mineral within the supergene zone. The chalcocite replaces either pyrite or chalcopyrite, depending on the pre-enrichment pyrite to chalcopyrite ratios. Covellite, unknown as a primary sulfide mineral at San Manuel, is relatively common within the supergene copper zone.

Conclusion

The current oxide deposit at San Manuel is the oxidized upper portion of the San Manuel North sulfide ore body. The block cave sulfide mining has disrupted the mineralogical and grade lensing within the cone of draw forming a homogeneous mass of oxide material without distinct mineral or grade lensing. The oxide zone beyond the sulfide draw cone exhibits traceable mineralogical and grade lenses similar, though less consistent than, grade lenses defined within the primary sulfide ore body.

Geologic conditions favored supergene enrichment along the pyritic zone rather than along the low grade interior core; however, the supergene enrichment has not been of significant extent with respect to the total oxide mass.

Chrysocolla is the predominant copper mineral within the oxide deposit and chalcocite the predominant copper mineral within the supergene enrichment zone.

Considering the significant tonnage of oxide copper material, the similar oxide and sulfide grade lensing, and the modest amount of supergene enrichment, it is evident that the conditions that formed the oxide zone did not favor solution and transportation of the copper for any distance.

Continuous Miner Drift Excavation and Geomechanical Rock Classification at San Manuel

L.A. Sandbak
Geologist

TABLE OF CONTENTS

Abstract

Drift excavation by continuous mining machine methods in the San Manuel and Kalamazoo ore bodies has been very successful. A geomechanical rock classification in conjunction with detailed geologic mapping has been necessary to assess roadheader performance. The modifed geomechanical classification of Bieniawski (1976) has been used to predict roadheader performance in the various rock classes likely to be encountered in the San Manuel Ore Body.

Rapid excavation and less fractured ground are obvious advantages of roadheader drift excavation over conventional drill and blast techniques. Further testing in harder rock classes expected in the lower levels of the San Manuel Ore Body is needed to fully assess future roadheader applications.

LIST OF FIGURES

Introduction

The recent testing of a continuous mining machine, or roadheader, in the Kalamazoo and San Manuel ore bodies has facilitated the introduction of a geomechanical approach to drift mapping and to drill core. The purpose is to introduce a rapid and effective method of evaluating roadheader excavation rates, and to predict roadheader performance on a wide variety of rock types and classes.

History

A Dosco SL-120 continuous miner was first tested on the 2890 level of the Kalamazoo Ore Body. The Kalamazoo site was selected because it was reasonably assured that a boring machine could be effective in areas of altered Laramide porphyry dikes (Cockle, 1979). Approximately 700 ft. was excavated by the roadheader in just under three months in the North Grizzly Fringe Drift on the 2890 level before the machine was moved to the 2375 haulage level of the San Manuel Ore Body. The main goals of the 2890 level test were to determine if the roadheader could cut typical San Manuel rock, and if so, whether it could be cut economically.

2890 Test

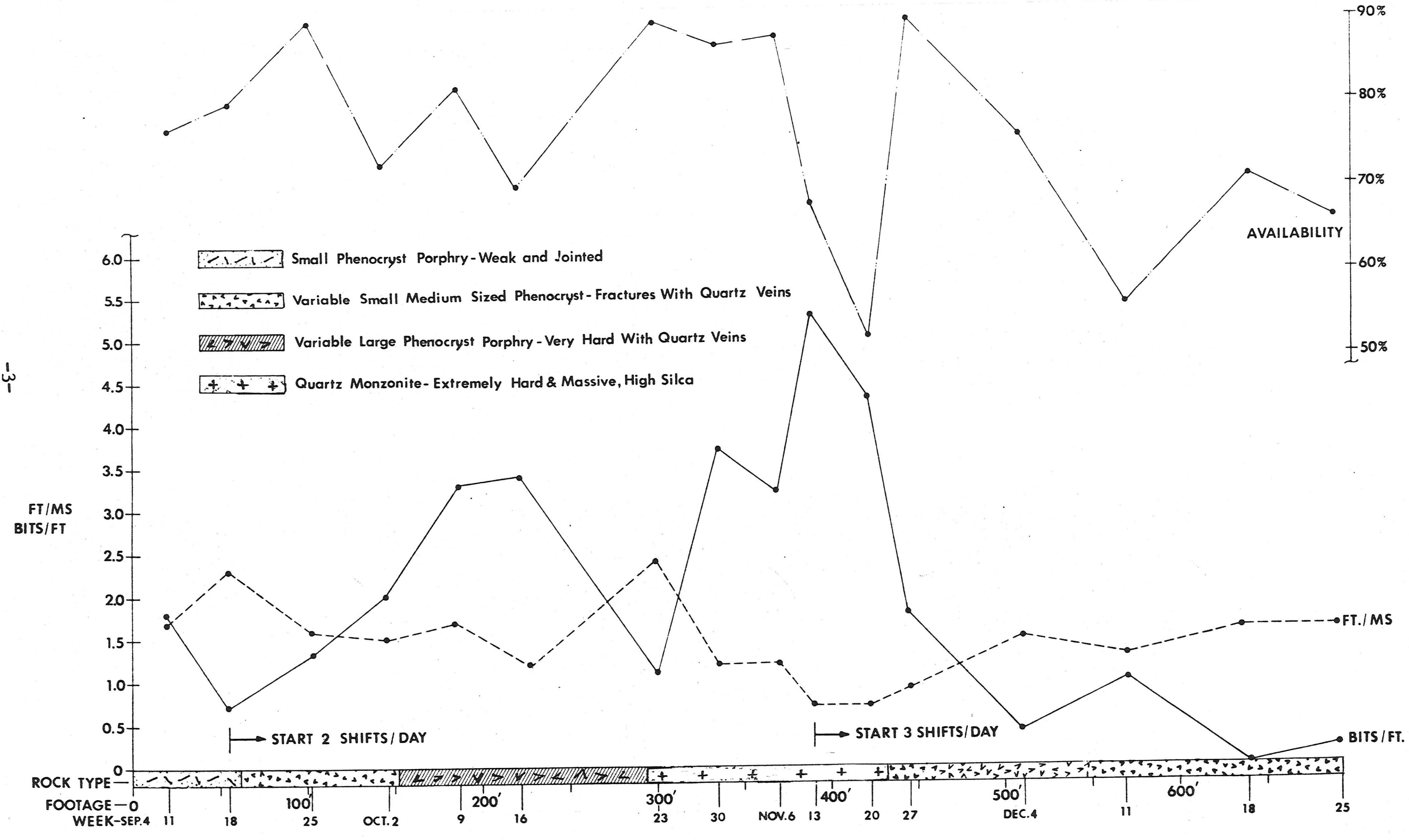
The results of the 2890 level test are graphically illustrated in figure 1, and summarized in table 1. Note that 571 ft., or 82%, of the drift was excavated in relatively weak and jointed dacite and granodiorite porphyries. An average of five ft. per shift was excavated in the porphyries and at the rate of just over one bit per ft.

Approximately 125 ft. or 18% of the drift consisted of a very hard, massive, and potassically altered quartz monzonite (QM). This QM was not expected, as it was not apparent from diamond drilling in the area. The advance in the QM was dramatically slowed to an average of three ft./shift, with a corresponding increase to four bits/ft. of drift excavated. In addition, the QM had to be drilled and blasted to fracture it sufficiently to be cut by the roadheader. The average data collected from the 2890 level test for the quartz monzonite and various porphyries was then used as a guide for typical San Manuel rock.

Panel 21A Prediction

The data from the 2890 test was extrapolated to several likely sites in the San Manuel Ore Body which could be cut by the roadheader. The percentage relationship of basic rock types in the lower levels of the San Manuel Ore Body was reversed from the Kalamazoo Ore Body in that 70-80% of the rock was expected to be

FIGURE 1: 2890 NGFD Test Modified from: (Geyer and others, 1984)



DOSCO-AVAILABILITY, EFFICIENCY, & BIT USAGE

Note: Feet per worker per shift (FT./MS) based on crews of three workers.

-3-

TABLE I: 2890 NGFD DATA SUMMARY

Footage Interval	Rock Type and Characteristics	Thickness (Ft.)	Average Ft./ Shift	Average Bits/ Ft.	% of Total Footage	Cutting Hours per Shift
0- 55'	Small phenocryst porphyry; weak and jointed 4-6" pieces average (D.P.) dacite porphyry	55	6.27	1.11	7.90	2.15
55-147'	Variable--small sized phenocryst granodiorite porphyry (G.P.); fractured and broken 3-6" pieces average. Greater quartz veining and potassic alteration	92	4.60	1.65	13.22	1.84
147-300'	Variable--large phenocryst ore porphyry (G.P.). Very hard and less jointed 4-6" pieces average. Very potassically altered with quartz veins	153	6.02	2.19	21.98	3.17
300-425'	Quartz monzonite (Q.M.); extremely hard and massive 8"-1' blocks or >. High silica and potassic alteration	125	2.91	3.97	17.96	2.19
425-696'	Small--medium phenocryst porphyry (G.P.); very weak and jointed 4-6" pieces average. Breaks easily into less than 2" pieces. Increasingly faulted	271	4.29	0.50*	38.94	1.40
Totals	All Porphyries	571	4.99	1.19	82%	2.02
	Quartz Monzonite	125	2.91	3.97	18%	2.19

* - Note: Decrease from 0.59 (425-570') to 0.1 bits/ft. with carbide button bit (570-696').

quartz monzonite, with 20-30% porphyries. The Panel 21A drift on the 2375 haulage level was selected to test the roadheader because it was predicted to have appreciable percentages of weak porphyries and faulted ground to facilitate cutting.

A conservative prediction report for P.21A and P.21B was distributed just before excavation started (figure 2). General characteristics of rock type and alteration were based on 2315 level mapping and the partially completed 21B drift on the 2375 level.

Very fresh chloritic to slightly phyllically altered quartz monzonite was expected for the first 150 ft. or so of the drift, to be followed by intervals of low grade porphyry dikes and high grade potassically altered quartz monzonite. Difficult drifting was expected 515 to 700 ft. from the starting point, as the potassic QM alone became dominant.

Problem in Search of a Solution

Soon after drifting began in P.21A, it became apparent that the actual performance of the roadheader was going to be much better than predicted, especially in the QM. It was realized that the generalized mapping procedure used at San Manuel was not adequate in defining quantitative measurements which affected roadheader production rates. One could get a feeling of rock strength and faulting effects, but other data could not be quantitatively compared

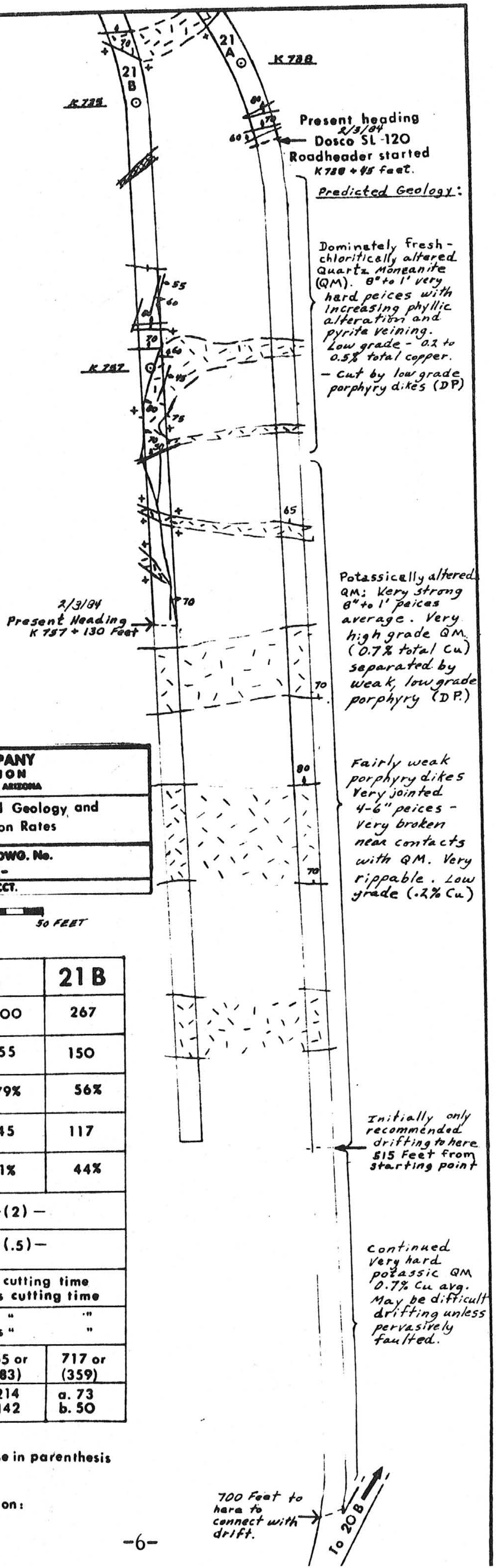


FIGURE 2

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

P. 21A and 21B, 2375 Level — Predicted Geology and Roadheader Excavation Rates

DRAWN LAS	DATE 2/3/84	DWG. No.	X-
CHKD.	DATE		
APPD.	SCALE 1"=50'	ACCT.	

50 0 50 FEET

		21A		21B
Distance to be driven (ft.)		515	700	267
Footage QM		370	555	150
% QM		72%	79%	56%
Footage DP		145	145	117
% DP		28%	21%	44%
Bits / Foot Rate Used	QM	— 4 or (2) —		
	DP	— 1 or (.5) —		
Feet / Shift Rate Used	QM	3 for 2 hours cutting time 4.5 for 3 hours cutting time		
	DP	5 for 2 hours " 7.5 for 3 hours "		
Total Bit Use		1625 or (813)	2365 or (1183)	717 or (359)
Working Shifts to Complete		a. 152 b. 102	a. 214 b. 142	a. 73 b. 50

Bits/Ft Rate Used and Total Bit Use in parenthesis are estimates for Button Bit

Working Shifts to Complete based on:
a. 2 hours machine cutting time
b. 3 hours " "

to other geologists' drift mapping on much more than a quartz monzonite versus porphyry approach or a "weak" versus "strong" ground.

The rock quality designation (RQD) and average compressive strength are helpful in classifying rock, but are misleading in that the classifications are too conservative. Another problem is the variability of rock strengths and fracturing, even across the same heading. A quick and effective classification scheme that would take into account more of the factors dealing with the overall strength of the specific areas of interest was needed.

Geomechanical Classification

The geomechanical classification proposed by Bieniawski (1976), and modified by Laubscher and Taylor (1976) for mining applications was chosen to be used on P.21A. This classification assigns a numerical value to each of the geologic properties that affect the in situ strength of a jointed rock mass. Namely, these parameters are RQD, intact rock strength, joint spacing, condition of joints, and ground water. In addition, the in situ classification can be modified further by the effects of joint strike and dip orientations, weathering, field and induced stresses, and blasting.

In effect, the adjusted geomechanical classification evaluates the worst conditions of the rock mass rather than the average to above average conditions a RQD-strength classification could provide. Each of the five basic parameter ratings are added together for a subtotal rock mass classification ratings number. A rating adjustment for orientations of joints and faults is then subtracted from the subtotal to obtain a total ratings number. This number is then designated to the appropriate rock class description ranging from very poor to very good. These descriptions are based on 5-20 unit divisions from very poor class 5 (rating 0-20), to very good class 1 (rating 80-100).

Roadheader Performance, P.21A

Figure 3 illustrates the geology and geomechanical classification as applied to the P.21A drift. The performance of the roadheader is also profiled with respect to bits/ft., ft./cutting hour, and ft./ manshift.

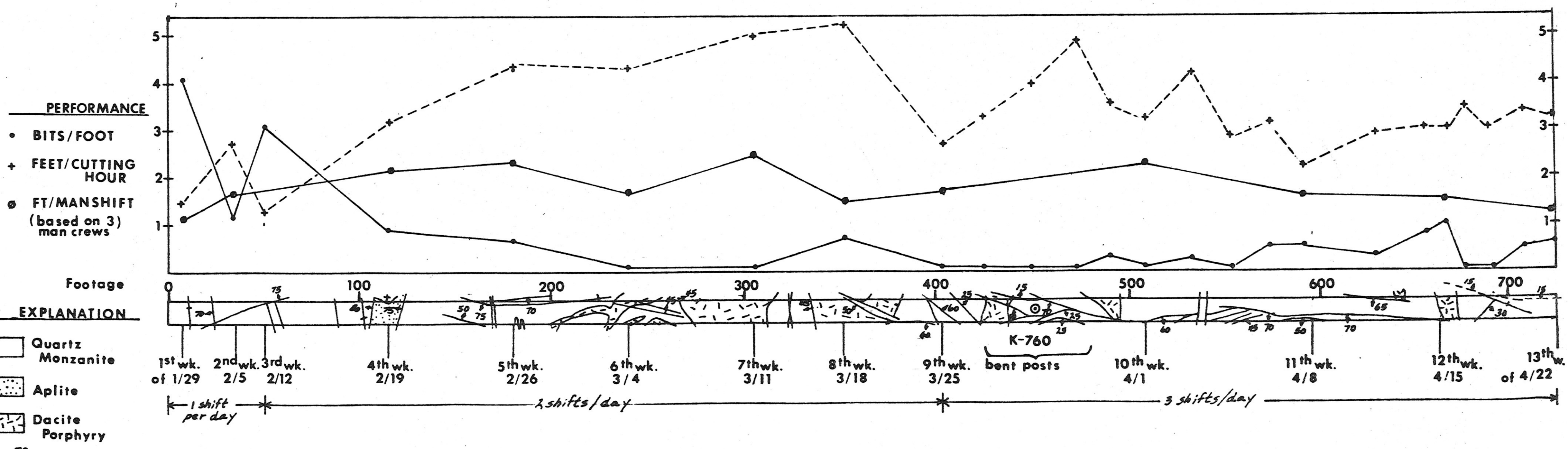
The ft. per hr. of machine cutting time was chosen to more accurately reflect roadheader performance than ft. per manshift. An average of two hrs./shift were used to cut the drift, with four or more hrs. to install supporting posts and caps. The ft./manshift (ft./worker per shift) rate given is one-third of the actual footage driven because the data is based on crews of three workers.

Note in figure 3 the increase in ft./cutting hr. and corresponding decrease in bits/ft. used at approximately 100 ft. from the starting point. This corresponds to the transition from class 2 rock (good) to class 4 rock (poor). This also correlates to the boundary

FIGURE 3

GEOMECHANICAL ROCK CLASSIFICATION AND DOSCO SL-120 PERFORMANCE, P.21A-2375 LEVEL

1	R.Q.D. %	72	70	62	35	50	58	25	38	53	5	18	3	19	28	42	22	39	39	55	18	56	52	63	52	50	7	47	44
	RATING	15	15	13	7	11	13	5	9	11	0	5	0	5	7	9	5	9	9	11	5	13	11	13	11	11	3	11	9
2	I.R.S. (M Pa)	200	170	150	120	137	116	127	172	162	130	100	75	60	88	103	95	90	92	85	90	115	130	157	130	145	70	115	120
	RATING	15	12	12	8	10	8	9	12	12	9	7	5	4	6	7	6	6	6	6	6	8	9	12	9	12	5	8	8
* 3	JOINT SPACING	.6/1.2	.3/.8	.3/.6	.15/.3	.1/.3	.15/.6	.1/.3	.1/.2	.15/.2	.1/.15		.1/.2	.1/.15	.1/.2				.15/.30		.1/.2	.15/.4			15/.30	.15/.38	.1/.15	.15/.3	
	RATING	15	11	11	2	2	6	2	2	2	2	2	2	2	2	2	2	2	2	2	2	6	6	6	2	6	2	2	2
4	CONDITION OF JOINTS	Rough to very rough surfaces. Hard joint wall rock.			Soft wall rock	> slicken. surf.	> gouge and slickened joints. Very broken ground; continuous jointing.				Clay, MoS_2, and calcite slickenslided joints; continuous						Soft joint wall rock. Low angle faulting and clay gouge.					50:50 mix of very hard wall rock and slickenslided faults.			Strong joint wall rock; minor slickened surfaces. Low angle faulting in back.				
	RATING	20	25	20	12	9	6				6						12					16			16		12	16	
5	GROUNDWATER	DRY			MOIST	DRY							Moist	DRY				M.	DRY										
	RATING	10			7	10							7	10				7	10										
SUBTOTAL RATING		75	73	66	36	42	43	36	39	40	27	30	20	27	31	34	35	36	39	40	35	53	52	57	51	55	34	47	45
ROCK CLASS		2A		2B	4A	3B		4A			4B		5A	4B	4A							3A					4A	3B	
6	JOINT ORIENTATION	FAIR		FAVORABLE		FAIR	VERY UNFAVORABLE						FAIR	VERY FAVORABLE			UNFAVORABLE				FAIR TO VERY UNFAVORABLE				FAIR TO UNFV.		V. FAV	FAIR-UNFV.	
	RATING	-5		-2		-5	-12						-5	-0			-10				-8				-7		-0	-7	
TOTAL RATING		70	68	64	34	37	31	24	27	28	15	18	15	27	31	34	25	26	29	30	27	45	44	49	44	48	34	40	38
ADJUSTED ROCK CLASS		2B			4A			4B			5A			4B	4A		4B					3B					4A		
DESCRIPTION		GOOD ROCK			POOR ROCK						VERY POOR			POOR ROCK								FAIR ROCK					POOR ROCK		



*Note: 1st number intermediate joint spacing; 2cd maximum spacing (in meters).

between the low grade and relatively unaltered zone to the higher grade 0.5 to 0.7% total copper and potassically altered ore zone. The class 2 (good) rock averaged only 2.25 ft./cutting hr., and consumed more than 40% of the total bits used in over 727 ft. of drifting.

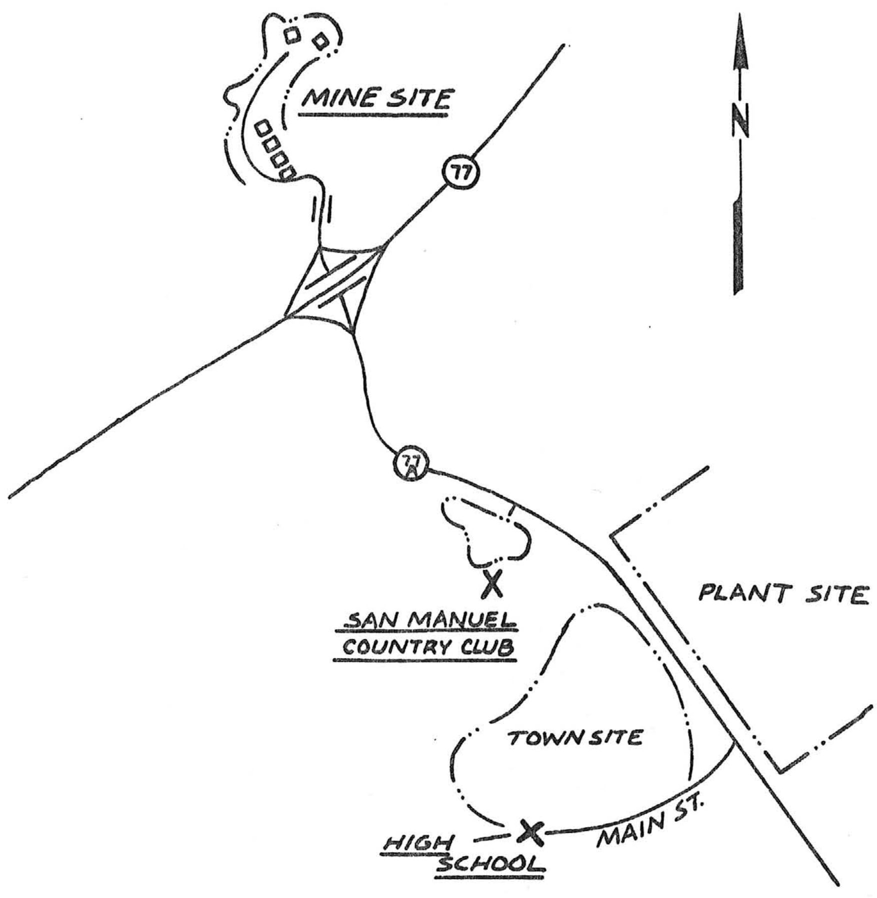
The majority of the drift (66%) continued in poor to very poor rock with a mix of potassically altered QM and low grade porphyry dikes. The dramatic effects of low angle faulting and faulting parallel to the drift are evident in the adjusted rock classes.

The rock class changed at 500 ft. of drifting from class 4 (poor) to class 3 (fair) when the majority of the porphyry dikes disappeared, and drifting was confined to dominantly potassic QM alone. This was anticipated in a vaque sense in figure 2. The ft./cutting hr. rate decreased, and the bits/ft. used correspondingly increased. The week of April 22, 1984 was driven in 4A (poor) rock. The bits/ft. rate was lowered, partially due to the connection to the 21 South Ladder Drift, and due to a weak porphyry dike.

The ft./manshift rate stayed around two for most of the drift, dipping to 1-1.5 for the good rock, and just below two for the fair rock. It is also interesting to note that the ft./manshift rate also dropped in the class 5 (very poor) rock when extra cribbing and support were needed in the porphyries, and toward the end of the drift when additional posts and supports were required for the curved turnout.

BT 10-11

Correction: The 1983 production figure should read 110,000 short tons.


MINE SITE
77
N
77
PLANT SITE
SAN MANUEL
COUNTRY CLUB
TOWN SITE
MAIN ST.
HIGH
SCHOOL

Evaluation of Geomechanics Data

A graphical evaluation of roadheader performance in each rock class (20 point interval) and subclass A or B (10 point interval) is shown in figure 4. Each adjusted rock class rating number from figure 3 was plotted against the ft./cutting hr. and bits/ft. used for that particular interval. The data for the 1B (very good) rock class was provided by the very hard quartz monzonite encountered in the 2890 NGFD test. Figure 5 is a generalized expectation sheet and guide for rock classes that also gives a % summary of each rock class encountered in P.21A.

The bits/ft. rate is very low (less than 0.1 bit/ft.) from class 5 to class 4A. The bits/ft. rate steadily increases to class 2B, and then rapidly increases to over 4 bits/ft. from 2B to 1B.

The ft./cutting hr. rate graph is rather straight until the class 4 (poor) rock, where it rapidly increases into the class 5 (very poor) rock. This great variability in ft./cutting hr. in the poor and very poor classes is reflected by the smaller joint spacing and block sizes of less than six inches. The fair to very good rock classes reflect a greater percentage of the block sizes greater than one ft., and a more direct relation to roadheader production rates. The lack of data in the stronger rock classes 3A through 1B indicate that additional testing is necessary to predict roadheader performance. The adjacent P.21B drift has already been started as of May 15, 1984, using the roadheader.

FIGURE 4: P.21A ROADHEADER PERFORMANCE

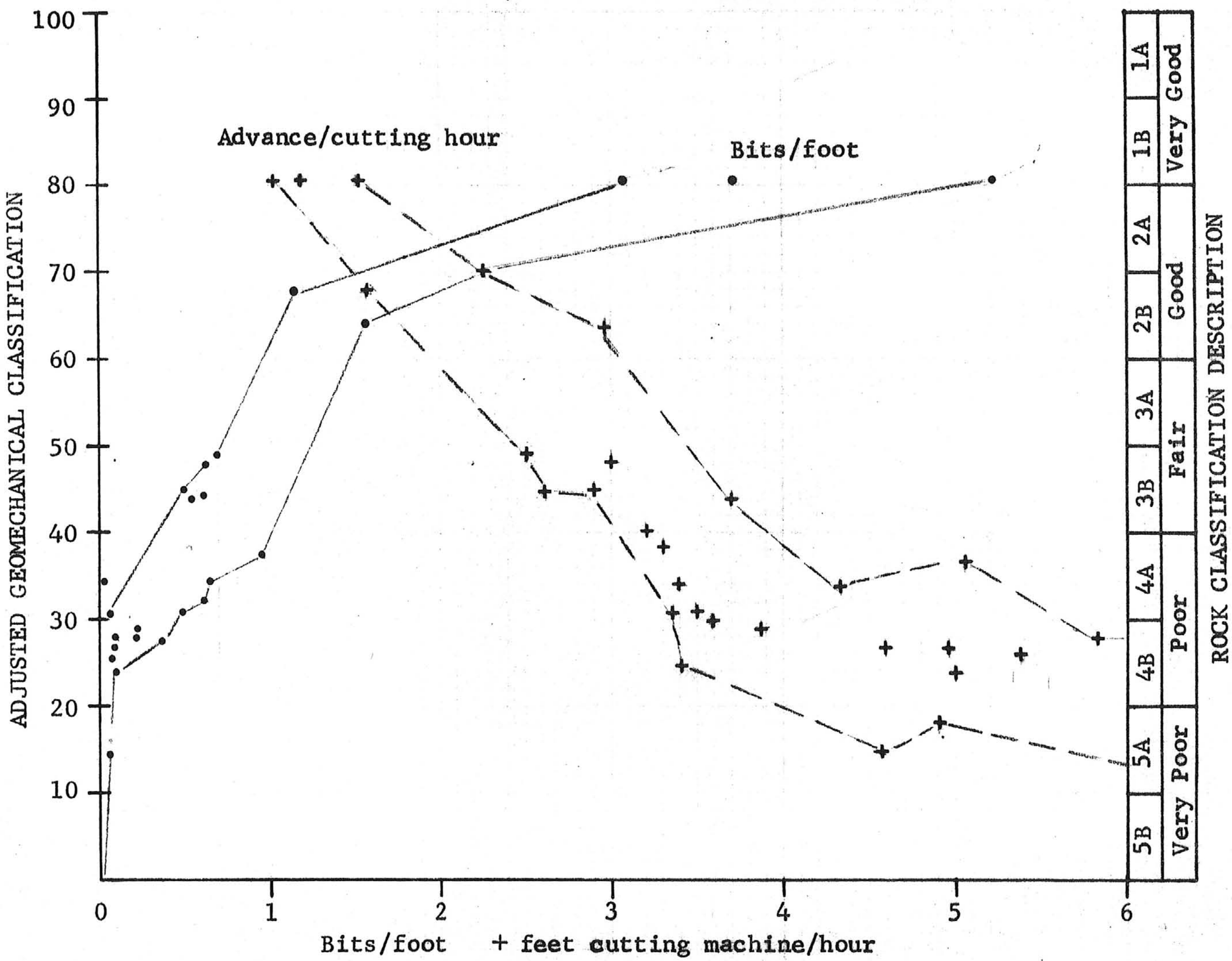


FIGURE 5
Predicted Roadheader Performance
Based on Rock Classification

Rock Class		Bits/ Ft.	Ft./Cutting hour	Characteristic Rock Types	Expectations	% Rock Class P.21A
1	1A	5 or less	N/A	Perhaps massive diabase	Drill & blast	N/A
	1B	3-5	1-1.5	Remobilized massive QM	Blast to fracture	N/A
2	2A	2-3	1.5-2	Fresh chloritic to slightly phyllic QM	Difficult drifting (blast if necessary)	N/A
	2B	1.5-2	2-3		Moderate drifting with ideal ground support--very few overbreaks	12%
3	3A	.8-1.5	2½-3	Potassically altered,--QM alone		N/A
	3B	.6-1.1	3-3½			22%
4	4A	.2-.7	3-4	Average porphyries	Excellent drifting but less ground support & cribbing requirements	27%
	4B	.1-.3	4½-5	Porphyries and porphyry and potassic-QM mix		29%
5	5A	.1 or less			Slowed drifting with overbreaks & cribbing. Major ground support	10%
	5B		N/A	Major faults & shattered ground		N/A

P.21B Prediction

The geomechanical data supplied by the P.21A roadheader test was extrapolated to the remaining 380 ft. left to be driven in P.21B. Figure 6 is a geologic and roadheader prediction sheet for P.21B, in which the geomechanical classification of P.21A is also plotted for comparison.

P.21B is expected to be stronger than P.21A, with less low angle faulting. The figures are based on actual machine performance. P.21B is expected to be a good test of continuity of rock class in the potassic ore zone.

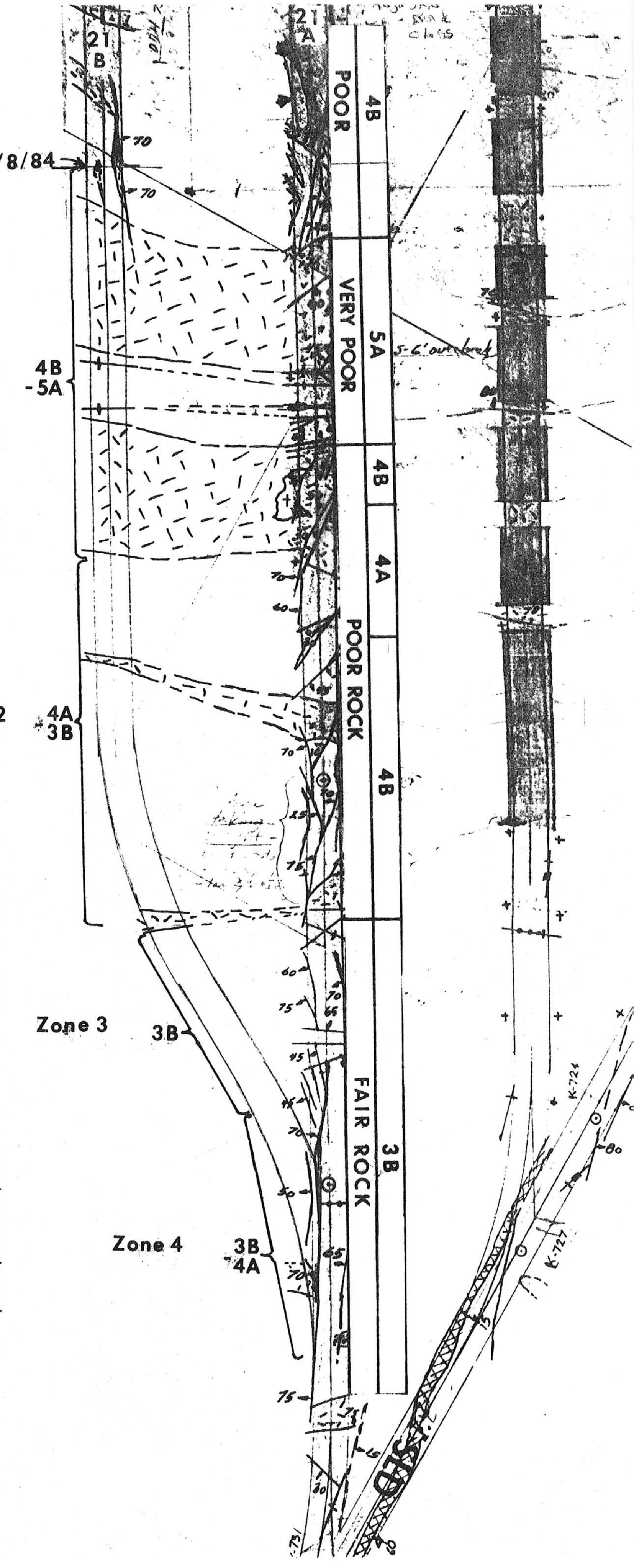
Future Roadheader Tests

The next major test for the roadheaders will be in the more massive and phyllically altered QM on the 2675 haulage level. These low grade, phyllically altered rocks underlie the northern and southern ends of the 2600 level blocks. If the roadheader can cut these areas economically, then it would have proven itself in the majority of the rock classes expected in the San Manuel Ore Body.

The roadheaders are also being suggested for use in driving grizzly drifts on the 2315 level. The potential for very rapid excavation of up to four times the footage achieved in the haulage drifts is expected. This is partly because the excavation cross sectional

FIGURE 6

Geologic Plan Map and Roadheader Prediction, P.21B, 2375 Level



Explanation

Quartz Monzonite

Dacite Porphyry

Fault, showing dip

Scale 1"=50'

50 0 50 FEET

	Zone 1	Zone 2	Zone 3	Zone 4	Zones 1-4
Rock class	4B to 5A	4A to 3B	3B	3B to 4A	5A to 3B
Footage to drive	130'	125'	75'	50'	380'
Bits/ft. rate	.1 to .3	.3 to .5	.5 to .8	.3 to .5	N/A
Ft./cutting hr.	3-5	3-4	3-3½	3-4	N/A
Total bit use	13 to 39	38 to 63	38 to 60	11 to 18	100 to 180
* Working shifts to complete	13 to 22	16 to 21	11 to 13	7.5 to 10	48 to 66
Weeks to complete based on 3 shifts/day, 5 days/week				3.2 to 4.5 weeks	

* - Note: Data based on 2 cutting hrs./shift

area of the grizzly drift varies from 66 to 76 sq.ft., nearly half the 130 to 150 sq.ft. excavated in the P.21A haulage drift. In addition, the hours of actual cutting time/shift could increase from an average of two in P.21A to perhaps four hrs. or more. This is because excavated rock can be dropped through the transfer raises as the drift is being driven instead of mucking and transporting the rock in mine cars. The only limitation so far has been space to maneuver the machines in tight corners.

Roadheader Excavation Advantages

The use of a roadheader for drift excavation has several distinct advantages over conventional drill and blast techniques. For one thing, the roadheader does not introduce fracturing or promote over-breaks and side breaks, as blasting does. The roadheader can cut just enough rock to suit the cross sectional area required for each particular drift. Conventional blasting techniques expose an average of 30% or more of the area than is needed, depending on the ground conditions. This means more cribbing and side lagging support, as well as increased concrete costs. Roadheader excavated drifts would be stronger with a more uniform cross sectional area, and thus less ground support would be needed.

Another probably more important advantage of roadheader excavation is the rapid excavation, especially in fair to very poor rock classes. The roadheader has achieved an excavation rate of two to three times faster than conventional methods on haulage drifts. This means less cost and a quicker turn-around time for development drifting. The excavated rock is also being mucked at the same time the machine is cutting. There is no need to move the machine out of the way for mechanical muckers. Improvements in the machine cutting time per shift, availability of the machine, muck handling techniques, and bit technology may make the roadheader an even more attractive alternative to conventional drifting.

Conclusions and Comments

The continuous mining machine, with its modifications since the 2890 test, has achieved rapid excavation with an average of six ft./shift, or three ft. per cutting hour. This has primarily been in a mix of potassically altered quartz monzonite and low grade porphyry dikes. In addition, low angle faulting and faulting parallel to the drift have lowered the in situ classification in several instances. Evaluation of roadheader performance in the fair to very good rock classes expected in the lower levels of the San Manuel Ore body is needed to assess its full potential.

The modified geomechanical classification of Bieniawski (1973) can be very site specific and can provide a quick and effective quantitative addition to drift mapping. Continued use of the system, modified for possible stress changes, will build a data base for minimum ground support requirements. Standardization with other underground mines could allow comparisons for cost effective ground support. Research into current split set and shotcrete support versus rock class is being conducted.

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Tiger: A Brief Look at Current Utilization and Potential

R.L. Hockett
Senior Geologist

Introduction

The mines of Tiger, or Schultz, as it once was known, are famous. Not, perhaps, for the value of their production (other mines have exceeded them) nor for the variety of metals produced (significant production of Au, Ag, Pb, Zn, Cu, V, and Mo), but largely for the diversity and beauty of the mineral specimens found here. Many of the seventy or so minerals have been found in few other deposits. Specimens from Tiger are found in most of the major mineral collections of the world.

Tiger has a colorful history going back to 1878, when Frank Schultz staked his first claim on the Collins vein. We are not going to dwell on the history or the details of the deposit. For these, please refer to the publications listed in the bibliography.

Today's paper will dwell primarily with Magma's current utilization of the deposit, examine some of the evidence regarding its origin and geologic history, then consider its potential and future if time allows.

By the time Magma had acquired the San Manuel deposit and commenced development in the early fifties, the Mammoth-St. Anthony Company was reaching the end of economic production. The final years of operation were in the sulfide zone deep in the Collins vein. Gold and silver content was low and lead and zinc prices were not high enough to justify continued mining. Magma acquired the property in exchange for stock in the new operation.

Early utilization was restricted to using the Mohawk Shaft as a supply of potable water for the new mine. By 1959 the water table was lowered below the old workings and the pumps in the Mohawk Shaft were shut off. Efforts were made to close all access to the mine to prevent injury to the curious and persistent rock collectors.

Current Utilization

Decontrol of gold changed the economic perspective. In the mid seventies an examination suggested that there was auriferous silicification in the mineralization peripheral to the primary veins that could provide siliceous flux for the smelter with sufficient metal credits to pay the cost of mining. A shallow diamond drilling program and surface sampling was conducted.

An open pit was started over the Mammoth Mine. Through 1978, over 100,000 tons were produced and used as converter flux by the smelter. Drilling had suggested a usable zone 80 ft. wide, centered on the Mammoth vein. The procedure was to mine this zone and combine it with silicified rhyolite to yield the desired volume of material with acceptable assays. Gold assays should have approximated 0.03 opt.

The results were disappointing. It was impossible to maintain sufficiently uniform silica content and the gold values were too low--0.011 opt. This application was discontinued in early 1979.

Tests conducted on stockpiled vein material suggested that, by crushing and screening to +3/8-in. minus two-in. size, the gold and silica content could be upgraded. In addition, the stockpiling operation would perform a blending function to stabilize grade fluctuations. While the siliceous flux (quartzite) in use then was high silica, it must be hauled 30 miles and incurs a royalty payment to the State. The cost is on the order of $10/ton, with no metal credits. Some of this material is still used to raise the silica content for fluxing.

In April 1983 a contractor, quarrying quartzite at our Camp Grant quarry, was invited to mine and process 20,000 tons of Tiger rock. Economics precluded extensive additional drilling, but the information available and visual examination extended the mining width for the test to 120 ft. Results were startling. The grade achieved for the minus two-in. rock was 73.7% silica and 0.044 opt. Au. Mining cost was approximately $5 per ton and the fines are usable as reverberatory flux.

Accordingly, a contract was let to produce 100,000 tons of minus two-in. Tiger flux at $5.45/ton. Silica content was 74.7% (75% of which is available silica) and metal credits were 0.048 opt. Au and 0.25 opt. Ag. Mining width was 250 ft. The metal content was bolstered by values contained in old dumps, stope filling, and pillars. The major advantage of this utilization is that there is no direct milling cost in recovering the gold values. They report to the slimes in the refinery.

Mining practice to date has been to drill and blast the approximate tonnage desired and make up any shortfall by ripping and dozing.

Origin and Geologic History

Probably no geologist has worked in the San Manuel area without speculating on the relationship between Tiger and the porphyry deposit. Is the spatial relationship causal in nature or coincidental? There have been many questions unanswered regarding time relationships of the various events recorded in the lithology of the area.

This year, with the known details of Tiger foremost in our thoughts because of the flux mining campaign, serendipity became an active force. Several pieces of information came to our attention in fortuitous sequence. One of the AGS sessions dealt with the mode of transport of epithermal gold. At this meeting I chanced to be seated by Jim Loghry, who logged core for Dave Lowell on the Kalamazoo project. He had some comments on gold distribution perceived in that project that were thought provoking. We happened to gain access to Weibel's thesis on the Cloudburst formation west of the mine with some pertinent age dating.

Before we proceed, let us consider a thumbnail sketch of the lithology of the two mines.

The San Manuel/Kalamazoo deposits are the faulted halves of mineralization emplaced following the intrusion of Laramide porphyry some 70 million years ago. The formerly economic mineralization created a near vertical elliptical cylinder with minor and major diameters of 2,500 and 5,000 ft. It probably had some primary tilt to the NE and was at least 8,000 ft. on the long axis. Mineralization occurs in both the porphyry and the pre-Cambrian host, Oracle granite, but is not necessarily symmetrical with regard to the intrusion. Subsequently, some sequence of faulting and tilting displaced the SW half down 8,000 ft. and laid the system over to the NE. The San Manuel fault which split the ore body also displaced the Cloudburst conglomerate and the younger Gila conglomerate. The only significant post mineral intrusion is rhyolite, which predates the fault and intrudes the Cloudburst, but not the Gila. Both the Gila and the Cloudburst units normally strike north to northwest and dip 30 to 45° to the east. Basin range faulting has offset the San Manuel fault in several places, usually down to the east, with displacements generally less than 200 ft.

The Tiger system occupies a NNW trending structural zone dipping steeply to the SW. This zone is occupied by or involves rhyolite. Much of the mineralization is along rhyolite contacts or in rhyolite breccia in the Mammoth vein portion. The vein was deeply oxidized and then faulted. The Mammoth fault strikes slightly more north than the vein, but dips steeply east, generally 75°. The originally deeper part of the faulted vein is called the Collins. It has Oracle granite in both walls, but the structure still involves rhyolite and rhyolite breccia.

The Mammoth vein was oxidized full depth to where it was intercepted by the fault between 700 and 800 ft. down. The Collins portion was almost entirely oxidized to the 600-ft. level and from the 700-ft. level down was largely sulfide, predominantly galena and sphalerite. By the time the lowest level, the 1025, was reached, pyrite and chalcopyrite seemed to be increasing.

The general concensus of workers in this area for a number of years regarding the age of the Cloudburst formation was that it was near contemporaneous with the porphyry intrusion a la Sillitoe. We now know that this is not true.

Weibel's thesis on the Cloudburst west of the mine lends several pieces of pertinent data. A rhyolitic welded tuff in the E half of section 33 near Schultz Spring was dated at 22 million years. This fits nicely as a known period of rhyolite activity.

An andesite unit striking NS through the middle of section 32, a mile west of Schultz Spring, dates at 28 million years, indicating normal sequence and position.

Imbrication studies indicate flow in the direction of dip, so there has been no reversal of dip. There are, however, any number of inferred or observed structures crossing Magma's property between Schultz Spring and Tiger. Much study is needed here.

Several working hypotheses have been developed or reinforced this year relating to local problems. Some of these are here briefly stated.

There has often been a problem in distinguishing the conglomerates of the area. It appears likely that the disconformity between the Gila and the Cloudburst was brief, but was marked by the emplacement of the closely related rhyolites and rhyodacites of the area. Conglomerate intruded by rhyolite or rhyodacite is Cloudburst, but if it contains clasts of these materials it is Gila.

Rhyodacite is marked by biotite plates, while the rhyolite is distinguished by quartz or feldspar phenocrysts. The tuff of Schultz Spring probably represents the Cloudburst surface at the time of rhyolite emplacement.

Now that Weibel's work places the Cloudburst formation in the Oligocene, it appears that there was ample time for erosion to reduce the cover over the porphyry system and allow oxidation to make the necessary ions available for redistribution of elements. The rhyolite event provided the trigger and added lead and zinc.

The rhyolites entered the area from a position that is now below the hoisting shafts. In many places in the shaft pillar we find evidence of lead-zinc mineralization spatially related to the rhyolites. The mineralogical sequence is quartz, specular hematite, galena, sphalerite, minor pyrite and chalcopyrite, and pink barite. The sulfides are discontinuous and are enclosed in gray quartz making their precise paragenesis vague. The alteration envelope is characterized by chlorite and is usually narrow, a foot or so. Gold and silver levels are low. The volume of material available for study has never been large and these occurrences are not economic.

References other than Weibel include Creasey's comments on multiple oxidation periods, the older one being related to an old erosion surface now tilted to the north. Also, Chaffee's trace element study indicated lower than background levels of lead-zinc in the porphyry system.

Tiger's Potential and Future

It is certain that we will continue to use some 50,000 tons per year of Tiger rock for flux. This usage will be proportional to smelter throughput. There are some options available to us that can alter the economics of this application.

The most recent mining contract was for 100,000 tons. Costs can be reduced by mining and stockpiling larger tonnages through the use of larger and more efficient equipment.

If gold prices move up and hold, it is possible to heap leach the stockpile and move some of the gold values forward in the cash flow.

Even though preliminary reports suggest low recovery from a heap leach, this is no drawback when the residual values will be recovered in the smelter.

The low leaching recovery has caused another series of tests to be requested. The recovery is low because a large percentage of the gold is micron-sized and locked in or between quartz grains.

If gold prices move up as some economists predict, Tiger potentially will once again become a mine in its own right. To this end it is prudent to establish some parameters for metallurgical treatment. One procedure suggested by the gold occurrence is a quartz float. The cationic collectors used in this process should also recover the molybdates and vanadinates.

Gold production at Tiger was listed by Creasey as 397,201 ounces through 1947. How much gold remains? A gross calculation involving the Mammoth vein alone gives a clue. Known length is 2,000 ft. Minimum depth is 700 ft. Using a 100-ft. width for the zone, we have a volume of 140 million cu.ft. Using a bulk density of 14 cu.ft. per ton to allow for mine openings and filled stopes would indicate 10 million tons. At 0.04 ounces per ton, there is a suggested potential of 400,000 ounces, or something equivalent to past production.

The current pit was designed to provide siliceous flux with no stripping. Planned depth is to 3,100 ft. elevation, or just below the old 100-ft. level. This will allow recovery of the pillars and sills. Gold cutoff grade is mining cost, or 0.015 ounces per ton. Internal waste is processed. It usually is higher than average in silica. What would have been waste in the high NE wall was also processed because of a high silica content. Planned gold content was 0.04 ounces per ton, but this has been persistently bettered. There are 700,000 tons remaining, but this is open ended to the SE.

Additional open pit mining would involve considerable stripping. There is one other feasibility study that needs to be conducted that was suggested by an event in 1901. The early stopes were supported by square sets and were not filled. On the night of April 15, 1901, the stopes north of the Mammoth Shaft caved from the 750 level to the surface. No one was injured nor was the surface plant or shaft damaged. The cave followed the vein to the surface where there was 25 ft. or more of subsidence.

Should economics suggest that maximum production were desirable, the action of that cave suggests that a slot caving procedure could be successful. It may be some time before we let go of the Tiger's tail.

Tiger Mines' Chronology

Collins	Mammoth	Mohawk	Other
1879			
		1881	
Schultz	1882 Schultz		
	1885 Fletcher (shaft 500')		
	1889 Mamm. G.M. ltd.		
		1892 Mohawk GM Co.	
	1893 (caved (200-400)		
1894 Johnson Bernhart Collins			
	1896 Mam.G.Mng. Co.		
1898 Mam.Gm.Ltd.	1898 (tramway) Mam. Coll. M&M Co. caved 1901		1899 Generator installed
	1902		Tailings reworked (Au) St. Louis Gold Rec. Co. about 1900
		1912	
	1913 Great West. Cu Co.		
			1914 Tailings reworked Az. Rare Met. Co.
	1915 Mam. Dev. Co.		
	1917 Fire Mammoth Shaft		
	1918 St.Anth.M&D Co.		
	1919		
			1926 New Year Shaft Houghton
			1932 No prod.
		1933 Moly. GM Ca	
	1934 Mamm.St.Ant.Ltd.		
	1944 Fire-Mohawk Shaft		
			1948 Magma started development
	1952		1953

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