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Gold Note Reprints 1983 - ~~[illegible]~~

1. Oro Fino, La Paz Co. AZ. Jan '83
2. McLaughlin, CA.
3. Quesnel River Au - CANADA.
4. Sampling of Gold Deposits
5. Lacana Exploration, NV.
6. Golden Sunlight, MT
7. Anaconda's Talk — Summitville Proj., Colo.
8. Ore Sample Preparation
9. Relief Canyon Au - NV.
10. Carlin Type Au — Have they All Been Discovered? Feb '83
11. Gold Thoughts for Environmentalists
12. Abitibi Gold Deposits.
13. Geol. of Jerritt Canyon
14. Round Mtn, NV
15. Ball Pen Gold, NV
16. KCC ~~[illegible]~~ Tintic Div., UT
17. Fondaway Proj. NV
18. US~~MX~~ — Basic Prospectors.
19. Telfer Mine, Australia
(include Gold Dep of Seward, Alaska)
20. Heap Leaching Gold by RLO-FTG Jan '83
21. Golden Sunlight, MT
22. Gold Exploration in L-203 Areas Jan '83
23. Leaching & precipitating Au-Ag
24. Carlin Gold - Newmont
25. 5th World Gold Conf. Notes
26. Regional Metamorphic Ore Deposits

27. Table of Contents Special Report 13 "The Role of Heat" - - - -

~~28~~. a). Epithermal Au/Ag Dep. by H F Bonham

~~29~~. ~~Contents: OF 82-1795 A Compilation of Mineral Deposit Occurrences~~

28. Contents: OF 83-450 Fluid inclusion etc in epithermal deposits.

29. Canadians Looking for Epithermal Gold Deposits in Great Basin, USA. June '83

30 ~~26~~. What is the Value of Gold?

31 ~~27~~. GSA Abstracts: Alligator Ridge System below Silverton, Colo.
Seminitville Au-Cu Deposit

32 ~~28~~. Queenstake Resources Project June '83

33 ~~29~~. Felmont's sale of Round Mtn, NV.

34 ~~30~~ Getty dedicates Mercur Mine w/ Fact Sheet.

35 ~~31~~ Noranda's Golden Giant (Hemlo) Gold Project.

36 ~~32~~. PGE in Magnetite. JMC Oct '83

37 ~~33~~. Mesquite Up-date.

38 ~~34~~. Gold Production in Superior Province - Canada.

39 ~~35~~. Seminar Proposal by SW Discoverie 1983

40 ~~36~~. McLaughlin Gold Deposit, CA.

41 ~~37~~. Exploration for Epithermal Au-Ag Dep. by Dines.

42 ~~38~~. Open-file 83-710 Comparative Analysis of Epithermal Precious & Base-Metal Districts Hosted by Volcanic Rocks.

43 ~~39~~. Tobin Springs, NV. by R. Hardesty Dec 2, '83

44 ~~40~~. Relief Canyon, NV. (page 31)

ASARCO

Southwestern Exploration Division

JDS

January 13, 1983

File Memorandum

Re: Summary of Field Incidents encountered January 5 & 6, 1983 during the course of an examination of a portion of the SP Claim Group; Middle Camp (Oro Fino) Mining District; LaPaz (formerly Yuma) County, Arizona

Introduction

The subject claim group owned by West World and the adjacent (West) S.E. (Sleeping Elephant) claim group owned by Heinrichs GEOEXploration Inc., Jim D. Loghry, W. C. Hirt and Wombat Mining Co. (R. Lundin) was brought to the attention of ASARCO's J. D. Sell on Nov. 30th by Richard "Rich" J. Lundin, President of Wallaby Enterprises Inc. See enclosed Plate for location of claim groups. Lundin had told Sell that he has a zone of gold occurrence measuring 1500 x 6000 feet and ranging 0.1 to 3 ppm gold. Also the gold is believed to occur in pyrite within a fragmental, quartzite felsic tuff. This rather large gold anomaly zone is coincident with the "old" (early 1920's) Kerr-McGee surface Mo anomaly and a portion of the strong, but shallow Kerr McGee I.P. anomaly (see Plate 1, Feb. 12, 1979 ASARCO Exploration Record Sheet; Kerr-McGee Corp.'s Sugarloaf Peak Project, Middle Camp (Oro Fino) Mining District, Yuma County, Ariz. by G. J. Stathis).

I talked with Lundin over the phone on Nov. 31st and once again he mentioned that he had a 1500 x 6000 foot gold zone that averaged better than 0.5 ppm Au. On December 6th Lundin delivered to J. D. Sell a 3 p. summary of the SE group property and a 500 scale claim map of the SE group with geochem assays shown on both the SE and SP (Sugarloaf Peak) claim areas. I had just missed Lundin's visit to the ASARCO office on the morning of December 6th as I was on my way to the Annual AIME Tucson Meeting at the Marriott Hotel. The prospect was examined January 5 & 6, 1983 with special attention given to the main gold zone which lies almost entirely (+90%) within West World's S.P. claim area (see enclosed Plate 1). The following is a summary of the events of January 5th & 6th.

Events of January 5th

1. Parked the company Blazer at Stop #1 and began sampling in the A.M.

2. Two vehicles pull up around 11:00 A.M. and park on hillside just SW of Stop #1. Three men get out and stand and watch me.

3. Continue sampling and return to vehicle for lunch around 12:00 Noon. Three men had left before this. Considerable "tourist" vehicle traffic in the area.

4. Walk over to get sample on small knoll at Stop #2 when same 3 men show up again.

5. Men introduce themselves as Mr. Jerome (Jerry) Tramp of Big Tramp Mining Co. who has 5 unpatented mining claims in the area (see Plate 1). Mr. Tramp is a large man, in his mid to possible late 50's, about 6', 240 lbs or more. His address is 14240 Riviera Drive; Blythe, CA 9225 , telephone (619) 922-4365. I failed to write down the name of Tramp's 2 associates but one of them was 45-50 years old, approx. 5'8" and the other 60-65 years old, approx. 5'10". This latter individual did not participate in any of the ensuing conversation.

6. Mr. Tramp and his middle age associate informed me that I was on their ground and they had 5 claims in the area surrounding the Princess Mine; they would appreciate it if I would not collect any more "test samples." All of this was said to me in a friendly manner. I gave these men my name, but I did not tell them I was with ASARCO as Lundin and the claim map he gave us did not indicate any other adverse claimants in the area. I did note a claim post of Mr. Tramp and a cardboard sign along the road warning of valid claims by Big Tramp Mining Co. in the area and no removal of mineral allowed, on my return to the vehicle for lunch. I told these men I was a consultant from Tucson examining mining claims submitted to a client by a Mr. Lundin of Wallaby Enterprises. These men never heard of Lundin or Wallaby and upon further questioning had no deal with Lundin or anyone. I didn't think, at the time, to ask these men if they had heard of West World. The younger associate of Mr. Tramp told me he surveyed in the claims with a $1200 sextant! and then sketched in their outline on the 7½' topog map I had in my clipboard.

   I was told they had run into Mr. Wilkins (Gulf Minerals) and Newmont people on their ground in the past. Also, Big Tramp Mining Co. had a camp site (location "A" on Plate 1) and was currently sinking a shaft which currently was down 180 feet. A Mr. Jim J. Sands and his wife lived (trailer) at the Princess Mine shaft area (location "B" on Plate 1) and that he was known to shoot at trespassers on occasion. The Princess claim is unpatented. The men added that they observed me walking and sampling just NW of their property and that this ground was currently in litigation between a Mr. Scott of Yuma, AZ and a Mr. West of San Diego, CA. They claimed I probably would have been in some kind of trouble, if either one of the two men had seen me. Again I did not see any claim notices of Scott or West when I was in the area. I shook hands with Tramp and his associates and told them I was driving into Quartzsite immediately to phone Mr. Lundin and perhaps I would be working over to the east near Sugarloaf Peak later that afternoon.

7. Left the area around 1 P.M. and drove to a pay phone at Quartzsite where I reached Lundin at Mountain States Engineering. I told Rich what had happened and he advised me that there was "no problem," that West World had a deal with both Tramp and Sands. Lundin didn't

know anything about Scott and West, but he admitted they have had "some problems" with placer claimants to the north. Lundin told me that either Norman Gossinger or John Lacy had made the deals for West World with Tramp and Sands. I bought this explanation and the conversation ended with Lundin telling me to call him if any more problems arose. Returned to the field around 1:30 PM and parked at Stops 3 & 4 and worked until dark with no further interruptions.

Events of January 6th

1. Left for the field early morning and parked off gravel road at Stop #5. Noted Big Tramp Mining Co. warning sign between Stops 4 & 5 which was outside of Tramp's claim block. This main access road services many small mining claim operations in the area.

2. I climbed to top of ridge with elevation marked 1432 (Plate 1) located SE of Stop #5. Noticed a pickup parked beside my company vehicle with horn honking. Two individuals got out but made no attempt to climb up and meet me. I had planned a SE traverse along ridge crests towards Sugarloaf Peak and away, as far as I knew, from any adverse claim situation as pointed out to me the day before. I later found out that the two individuals were Mr. & Mrs. Sands.

3. I commenced on my SE sampling traverse and noted a blue? pickup around noon time parked behind my vehicle. I didn't have my binoculars with me at the time.

4. I returned to the vehicle a little after 2 PM and found that the air had been let out of all 4 tires and the inner tube valve caps gone. Ate my lunch and started to reinflate the tires with a portable air compressor.

5. Tramp's associate, "the surveyor", drives by in a pickup similar to what I had seen parked at noon time alongside my vehicle. I attempted to flag him down to talk to him but he headed north to the interstate.

6. Mr. & Mrs. Sands drive by shortly thereafter and stop and sarcastically inquire if there is something "wrong" with my tires.

7. I ask Sands if he has a deal with West World and he says no. I also asked him if he has heard of Messrs. Gossinger and Lacy and he says no again. Mr. Sands, who is probably in his 70's, informed me that the road I am parked nearby goes south a short distance to his Princess Mine. I could drive up to this point, but the road which continues due south of his ground is washed out (not so, as I observed vehicular movement all morning in this area) and if I attempted to go beyond (south) he would shoot over my head.

8. Mr. Tramp and his wife? arrive in a station wagon while I am talking to Mr. & Mrs. Sands and he is considerably agitated. He tells me:

a) Didn't I see the sign on the road showing that I was on his property and didn't he tell me to leave the area the day before.

b) When I tried to ask him about the West World agreement, he heatedly replied that as he told me the day before he had no agreement with anyone and that he never heard of Gossinger or Lacy.

c) Twice during the course of this "heated" discussion Tramp threatens to "do me in" the next time he catches me out here.

d) He denied letting the air out of my tires and replied that he instead would have done the job "right" by slashing them into ribbons. I have no doubt that Tramp's "surveyor" associate let the air out with the full knowledge of Messrs. Tramp and Sands.

e) Tramp showed me a recently marked piece of pink flagging he found on his property that morning and accused me of it being either mine or that I had a second associate working in the area. I denied both accusations.

f) Tramp by this time was getting quite agitated. He threatened to shoot me the next time I showed up on "his ground". By this time I was starting to reinflate the 4th tire when he informed me he was in a hurry to get into town (Blythe) and I should get out at once and could finish the inflation alongside the highway. This was done.

9. Returned to the Motel around 3:30 or 4 PM and immediately got ahold of Lundin at Mountain States Engineering. I told Rich what had happened. All of a sudden he started to verbally "back track" by telling me that in his Nov. 30th meeting with Jim Sell, he stressed the importance of covering all the gold bets on the SE ground. Specifically he meant 3 altered areas which existed in this western claim block area, e.g. a) "Circular" area, b) "CuOx" area in Hancock Wash, and c) an area I don't recall now. See Plate 1 for location of first two areas. Lundin told me he has an interest in the SE ground and nothing, other than having done consulting work on the SP ground. He hoped I would continue with examination of the SE ground. The tone of the conversation implied I shouldn't have been on the SP ground in the first place! Of course in his initial contact with J. D. Sell and myself he emphasized the so-called 1500 x 6000' target which is almost entirely on SP ground. The delineation of this anomalous zone is based on extensive sampling by Felmont Oil, Newmont Corp., Utah Mtl., Gulf Minerals and AMAX and is coincided with pervasive alteration and pyrite dissemination. This zone is the logical first step evaluation of the entire area.

10. Lundin finally admitted they have had trouble with Mr. Tramp in the past and that he has had to post a Peace Bond(?) which as Lundin put, "has cost him (Tramp) money". Lundin was especially interested in hearing I was stopped on Jan. 5th (Stop #2) at SE claim #108 which is "Lundin's ground".

Conclusions, Afterthoughts

Though I consider Mr. Tramp's actions reprehensible, my chief gripe is Mr. Lundin. I consider his conduct completely unethical. At this juncture I am not sure whether Lundin is a calculated liar or a complete incompetent masquerading as a consultant to the mining industry. The truth probably lies somewhere in between. It is recommended that ASARCO have nothing to do with Mr. Lundin or those currently associated with him in a business sense.

Finally, I received a call from Jim Loghry yesterday afternoon, Jan. 12th. After making several unflattering comments and observations on Mr. Lundin and some on West World managment, he made the following points:

a) Loghry and Heinrich's GEOEXploration did not know that ASARCO was approached by Lundin.

b) Lundin has no business peddling the SE claim group in the first place. I didn't pursue this point with Jim. As I perceive it, Loghry et al. are "covering their tracks".

c) West World got in trouble with Tramp a couple of years ago when they found he had not kept up his claims, via annual assessment work and sent him a letter to that effect.

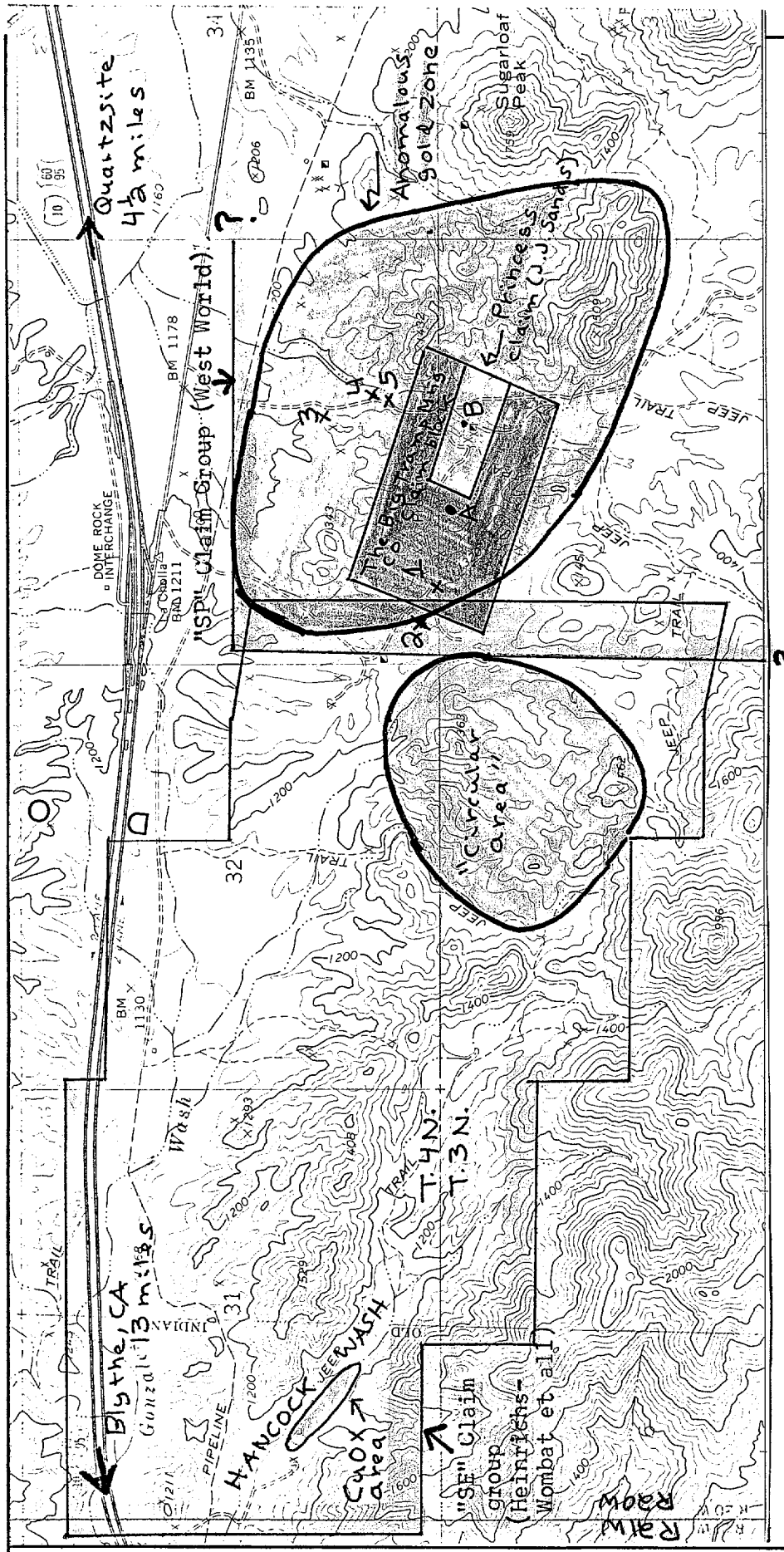
George J. Stathis

George J. Stathis

GJS/cg

cc: RBCrist
WLKurtz
JDSell

Attachment



x = stop (see text)

A = reference point (see text)

base: 7½' USGS Middle Camp Mtn. topog. quad.

ASARCO INCORPORATED

area: Sugarloaf Peak Area, Dome Rock Mountains	data by	state: Arizona	township-range	revisions-date
		county: LaPaz	map number	PLATE 1
title: Location Map	drawn by: GJS	2000 1000 0 2000 ft.		
mining district: Middle Camp (Oro Fino)	date: 1/13/83	contour interval: 40 ft.		

Skillings' Mining Review

Jan. 22, 1983
Vol. 72 No. 4
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Looking northwest along strike of orebody at Homestake Mining Co.'s McLaughlin gold project in California.

Homestake Proceeding With Its McLaughlin

Homestake Proceeding With Its McLaughlin Gold Project

$200 million project for 1985 start up featuring open pit mining/new technology

by DAVID N. SKILLINGS Jr.

IN NORTHERN CALIFORNIA, Homestake Mining Co., which is the largest gold producer in North America, is proceeding with development of a major new operation known as the McLaughlin gold project. During the past year, emphasis was on engineering and permitting at this property situated in the remote Knoxville mining district, 70 air miles northwest of the state capital of Sacramento and about the same distance northeast of San Francisco, in which Homestake has headquarters.

About the second quarter of 1985, the McLaughlin project, featuring open pit mining and new technology in its processing facilities, is scheduled for start up with an annual production estimated at 216,000 oz. of gold. An expenditure totaling about $200 million including the costs of land and exploration is required for the McLaughlin project, which ranks as one of the larger gold discoveries of the 20th century in the U.S.

At the rated capacity of 3000 tons of ore per day, the McLaughlin project ranks as the same size as the Mercur gold mine scheduled to be started up in July 1983 by Getty Mining Co. in Utah and is larger than Freeport Gold Co.'s Jerritt Canyon mine completed in July 1981 in Nevada with a capacity of 2750 tons per day. Although providing for 3000 tons of the hardest ore, the McLaughlin plant is being designed for future modification to handle a larger tonnage of softer ore.

RESERVE CONTAINING 3,200,000 OZ. GOLD

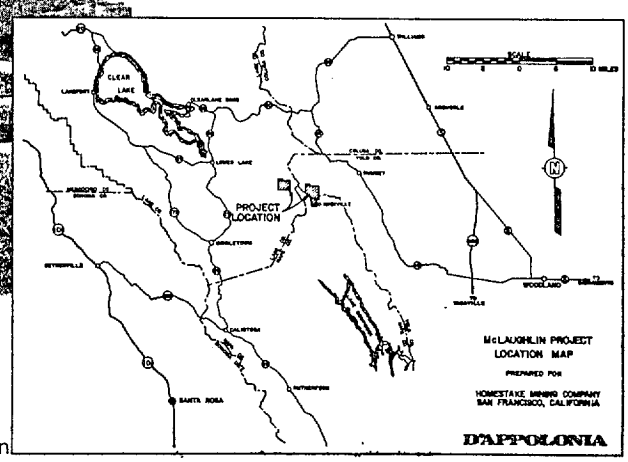
On the basis of initial drilling results, an announcement was issued on Aug. 27, 1980, by Homestake of discovery of a disseminated gold deposit totaling over 6,000,000 tons averaging 0.17 oz. of gold per ton for a total of 1,000,000 oz. Following additional drilling, an increased tonnage was announced at the annual meeting of the company's shareholders on May 12, 1981, at 20,000,000 tons averaging 0.16 oz. of gold per ton containing 3,200,000 oz.

Because the gold is in finely disseminated particles that cannot be panned and are not visible to the eye, prospectors during the 1849 California gold rush bypassed the area as unpromising. As a result, the Knoxville mining district that has been active for more than 100 years in production of mercury never was mined for gold.

Presently potential of mining deeper ore by the underground method at the McLaughlin project is being studied. Reserves of the McLaughlin deposit combined with

(Continued on page 4)

Scene looking south toward area containing future tailing basin at McLaughlin gold project.



Homestake Proceeding With Its McLaughlin Project

(Continued from page 3)

those of the Homestake mine in Lead, S.D., maintain Homestake's position as having the largest published gold reserves of any company in North America. The McLaughlin mine has a life estimated at 25 years, and the open pit will reach a length of one mile with a width of one-half mile.

In northern California, the nearest major gold operations to the McLaughlin project are Yuba Goldfields Inc.'s dredging operation north of Marysville and the Grey Eagle mine being placed in production at a site near the Oregon border by Noranda Mining Inc.

During Nov. 1978, management of Homestake decided to intensify its exploration program to find new gold deposits to retain its rank as leading gold producer in North America, and a budget for increased exploration was approved by the company's board of directors. By Jan. 1979, geologic models of known gold deposits had been developed by Homestake geologists, who adapted exploration techniques used previously in searching for non-gold deposits and began field research, principally in the western U.S.

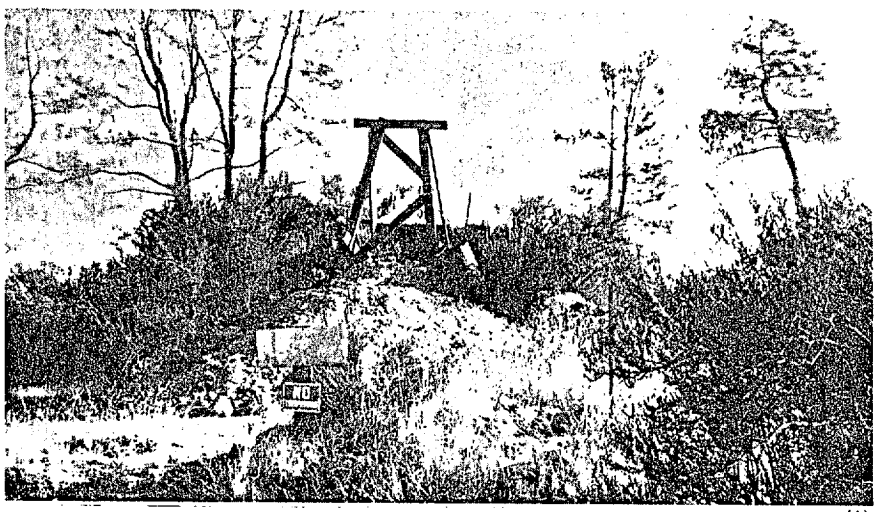
DISCOVERY OF ORE DEPOSIT IN MAY 1979

In March 1979, a small gold deposit was found by Homestake in northern California, and although this proved to be uneconomic it indicated that the model had been built correctly. At a site 15 miles south in Napa county, the old Manhattan mercury mine, known as the One Shot mine, was recognized as having common denominators. During May 1979, outcropping gold-bearing mineralization was identified by Donald L. Gustafson, geologist for Homestake, who presently is manager of U.S. resource development for the company.

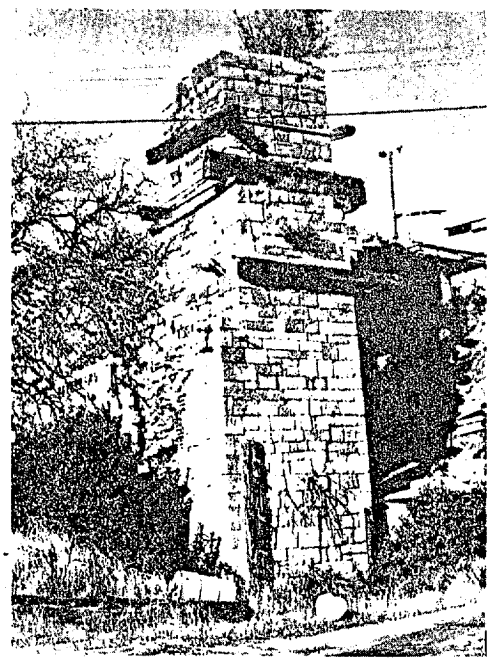
In Sept. 1979, negotiations were commenced with Bill Wilder, owner of the One Shot mine, and exploration drilling was started. During Aug. 1980, the project was named in honor of Dr. Donald H. McLaughlin, who served with Homestake for nearly 56 years and currently is chairman emeritus. After drilling and preliminary engineering, the McLaughlin project was transferred in 1982 to operations.

The McLaughlin project is situated at an elevation of 2000 ft. in a sparsely inhabited area of the coast mountain range, midway between Clear Lake, which is natural, and Lake Berryessa, which is artificial. The project straddles the county line, with about 80% in Napa county and 20% including the plant site in Lake county. The McLaughlin project is expected to help relieve high unemployment in this area. Personnel will reside principally in the recently-incorporated city of Clear Lake, 18 miles northwest of the project.

The Knoxville mining district covers 50 square miles including several underground mining properties with prior production. The Manhattan mine, which is the site of



(1)

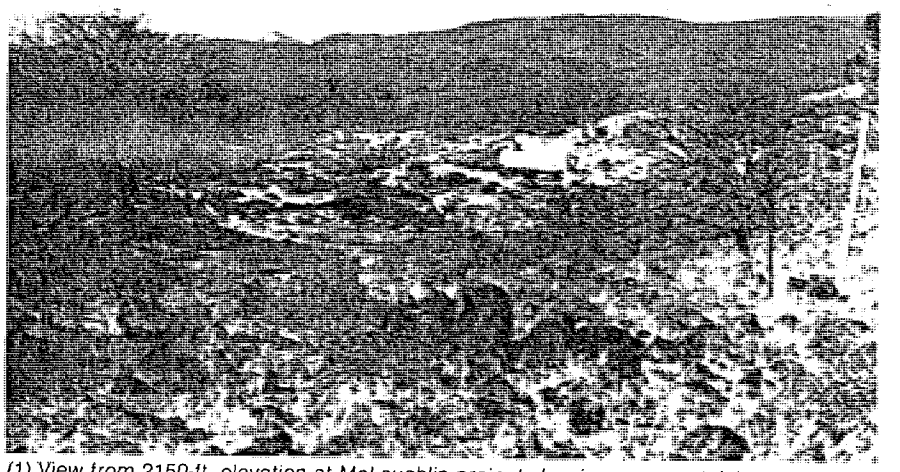


(2)

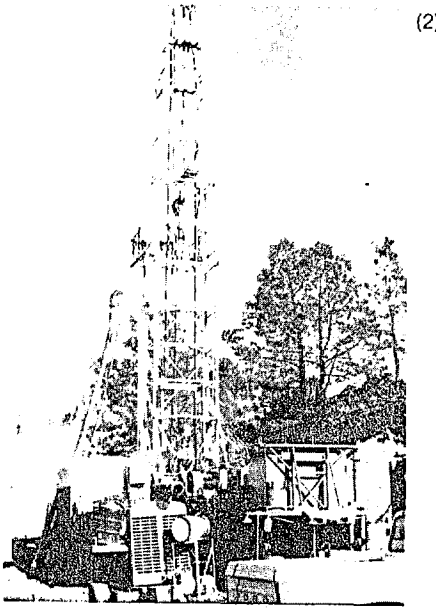
(3)

(1) Remnants of old mercury mining on property of McLaughlin project in Knoxville mining district. (2) Retort in which mercury was produced in Knoxville mining district. (3) Scene of the original Manhattan underground mine operated intermittently for mercury for more than 100 years.

(1)



(2)



(1) View from 2150-ft. elevation at McLaughlin project showing area containing future open pit mine, with coast range in background. (2) Rig drilling 17½-in. diameter well for water supply at McLaughlin project.

the McLaughlin ore deposit, operated intermittently for mercury beginning in the 1860s and up to as recently as the 1960s and 1970s. At the peak of activity during the 1860s and 1870s, about 300 persons were employed in the Knoxville mining district at mines including the Boston, XLCR (Excelsior), Redington and Knoxville, all of which were worked for quicksilver.

One Shot Mining Co., which held the Manhattan mine property, was acquired by Homestake from Bill Wilder, and the other major landowner was Bob Hoffman, both of whom retain a royalty interest. A lease is held by Homestake on the Harrison mercury mine, on which some drilling has been performed.

FIRST DRILL HOLE INTERSECTS OREBODY

In this area, 9000 acres are controlled by Homestake that include fee lands, lands held under lease, and mining and mill site claims on lands under jurisdiction of the U.S. Bureau of Land Management. Of this total, 1700 acres will be used for the McLaughlin project. Because this has been a cattle grazing area, a range surveyor has been employed by Homestake, which will lease portions for grazing. Currently the company has two major lessees for cattle grazing, which is helpful in keeping the grass down eliminating need for controlled burning.

At the McLaughlin project, the #1 hole was drilled from Sept. 5 to Oct. 15, 1979, in the center of the orebody to a 497-ft. depth and intersected 380.5 ft. of mineralization averaging 0.12 oz. of gold per ton. The #2 hole intersected 315 ft. at 0.268 oz. of gold per ton. In the program, which was principally diamond drilling with some rotary drilling, 408 holes were completed of HQ size and averaged 350 ft. deep with the deepest to 1000 ft. Six 6-in. diameter core holes were drilled to obtain samples for metallurgical tests.

The ore was crushed, a cut taken, prepared and sampled for checking grades and for metallurgical testing. Three adits were driven with a total of 1500 ft. of drifting completed proving continuity of the ore zone, and 20,000 tons of ore and waste were excavated from the center of the orebody. Most of the ore was shipped to Sherritt Gordon Mines Ltd., which has had experience with processing this type of ore in Canada. Tests included oxidation, CCD washing, leaching, carbon-in-pulp and tailing treatment as well as pilot plant work. Results of pilot plant investigations have indicated that the estimated recovery of 90% is correct and that recovery of 92.6% can be reached with a flowsheet handling different types of minerals.

Although the orebody is considered nearly drilled out, additional drilling work will be conducted on the north side of the property and also in a higher grade zone at depth that is too deep for open pit mining. Exploration also is being continued in the district.

The engineering contract for processing facilities at the McLaughlin project has been awarded by Homestake to Davy McKee Corp., San Francisco, which also designed and built ore treatment complexes at the Jerritt Canyon and Mercur mines. Homestake is conducting mine design and handling the equipment.

CONSTRUCTION BEGINNING IN JULY 1983

While engineering continues, construction is expected to commence in July 1983, depending on receipt of permits, and roads will be built. Following about 18 months of work, the ore processing facilities are scheduled for start up in the second quarter of 1985.

In Jan. or Feb. 1984, the premine stripping program will begin at the McLaughlin project for removal of a total of 9,000,000 tons of waste material. Over the life of the orebody, 108,000,000 tons of waste will be removed from the pit having a waste-to-ore ratio of 5.2-to-1.

At the McLaughlin project, mining will be conducted on a schedule of three shifts, five days per week, with milling on a continuous seven-day per week basis. On a daily schedule, 30,000 tons of total material will be moved at the McLaughlin mine. Tight ore control will be required similar to that used at the gold properties in Nevada.

In the pit, developed on 20-ft. high benches, three rotary drills are expected to be used in drilling 6 or 7-in. diameter holes on 25-ft. passes normally and occasionally to 40 ft. Blasting will be conducted with

(Continued on page 6)

Homestake Proceeding With Its McLaughlin Project

(Continued from page 5)

ANFO and incorporate a non-electric system. Loading will be with three or four shovels or loaders including 11-cubic yard hydraulic units or 13 to 15-cubic yard front-end loaders, and haulage will be in a fleet of ten 85-ton trucks.

The ore either will be fed directly from trucks or will be reclaimed from stockpile with a front-end loader and dumped onto a grizzly above a pan feeder feeding a jaw crusher for reduction to minus 8 in. in a single crushing stage. Ore will be conveyed to a radial stacker feeding a crushed ore stockpile providing some blending.

From stockpile, ore will be reclaimed with a front-end loader and will pass through a small grizzly. The ore will be fed to a semi-autogenous mill using balls as grinding media, with additional grinding to be conducted in a ball mill. The product of the ginding section consisting of 80% minus 200 mesh will go to a thickener, and the thickened material will be pumped to slurry storage.

From storage, slurry consisting of about 50% solids will be transferred to an 8 to 10-in. diameter pipeline for transportation a distance of five miles to the mill, at which it will be received in storage tanks. In the mill, situated on a hillside providing gravity flow, slurry will be directed to a pre-oxidation thickener and then will be pumped to two autoclaves for oxidation at high temperatures and high pressures. This will mark the first application of autoclaves in gold processing worldwide, having previously been used in production of alumina, cobalt, nickel and uranium.

FIRST AUTOCLAVES IN GOLD PROCESSING

The autoclaves are expected to have dimensions of about 14½ ft. diameter by 54 ft. long with four or six compartments. At 160° to 170° F., pure oxygen will be pumped into the vessel with a pressure of 180 p.s.i. for reaction with sulfur to generate heat. Because of the mineralogy of the ore, this chemical process is required to provide a gold recovery of over 90%.

The remainder of the flowsheet includes two thickener/washing stages, two carbon-in-leach stages, and eight carbon-in-pulp stages. The solids will be discharged to tailing, whereas the pregnant carbon will go to a standard pressure/stripper circuit followed by electrowinning on steel wool. After being retorted to recover mercury, it will be refined and cast into dore bars.

In addition to gold and silver, which are about on a 1-to-1 basis, mercury will be extracted at the rate of about 1 flask per day.

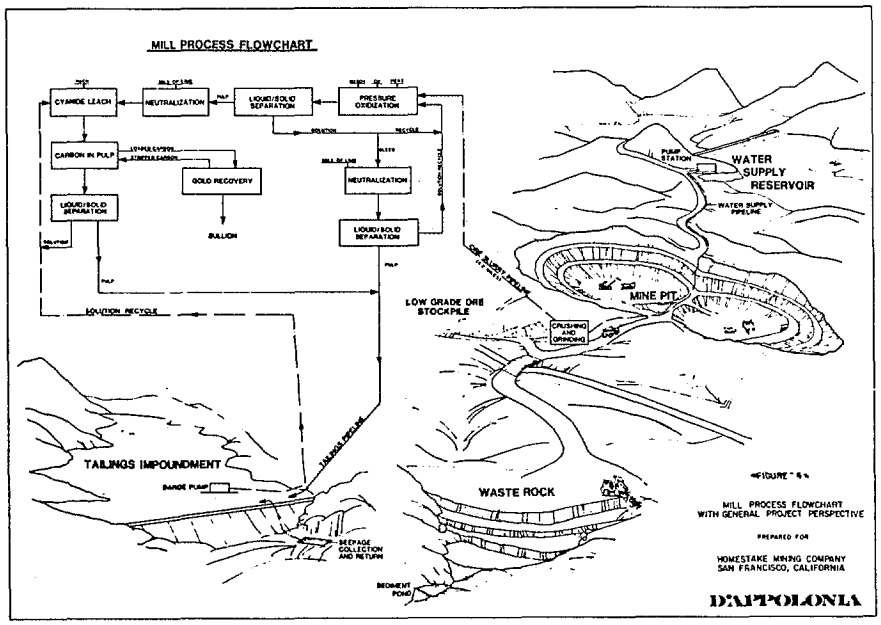
D'Appolonia Consulting Engineers Inc., Irvine, Calif., is environmental consultant to Homestake at the McLaughlin project, and Steffen, Robertson & Kirsten, Denver, Colo., is performing design of the tailing dam, which will provide a capacity for the first nine years of operation in the initial stage and the remainder of the life of the mine in the second stage.

Jack E. Thompson Jr., who previously served as resident manager for the Dawn Mining Co. at Ford, Wash., joined Homestake in 1981 as general manager of the McLaughlin project. His staff includes Micheal P. Attaway, mine superintendent, and Joseph L. Young, mill superintendent.

At the McLaughlin project, from 230 to 240 personnel will be employed by Homestake.

(1)

(2)



(1) Office of One Shot Mining Co., which operated the Manhattan mine in recent years for production of mercury. (2) Jack E. Thompson Jr., general manager of McLaughlin project, holding drill core.

What's Going on in WORLD MINING

UNITED STATES

Phelps Dodge idles 300,000 tons of Cu capacity

he second largest producer of copper i this country, **Phelps Dodge Corpo-** ...ended copper production all of its domestic mines and concen- ators, as well as its three smelters in the te of Arizona. The operations affect- are at Ajo, Douglas, and Morenci, izona, and Tyrone, New Mexico.

At normal operating rates, the Phelps dge mines and precipitation plants capable of producing 350,000 tons of per annually, of which 50,000 tons in the form of cement copper from ching operations. Leaching and pre- itation production will continue and company will also continue to oper- its Hidalgo smelter in New Mexico. smelter will treat concentrates other Southwestern copper mines, ing **Amoco Minerals Company's** ad operation and **Cyprus Pima** ny's Pima mine, both in Arizona. he shutdown is planned to last at until June 1; at press time, no an- ements of production resumption been made. Phelps Dodge, like United States producers, has been with production costs exceeding arket price of copper, which is at its level in real terms in 30 years.

hile the Phelps Dodge closures re- nt the largest single action to re- copper production in this country, al other companies have either re- production levels or closed mines. o Inc.'s Silver Bell operation in ona continues shut; **Kennecott Min-** Company's Ray Mines Division in ona has recently been closed; **Duval** oration only recently resumed pro- on at its Sierrita, Arizona oper- , while the Esperanza and Mineral properties remain shut down. In ion, **Magma Copper Corporation** ly reduced work hours at its San el, Arizona Division and laid off 330 employees there—this follow- 10 percent reduction in the com- y's work force at its Superior, Ari- division.

The net result of these closures is an all reduction in United States cop- production of about 40 to 50 percent. atements by the various producers, all have indicated that copper production would resume when copper prices increase to a level where the companies can once again operate the mines at a profit.

In contrast to United States producers, foreign and particularly Latin American mines continue to produce at capacity. Both the social impact as well as the effects that production reductions would have on foreign currency earning power are key reasons for not lowering capacity or closing mines. In fact, **Compañía Minera de Cananea** in Mexico has recently informed WORLD MINING that it has no plans to cancel or scale down its plans for expanding the Cananea mine operations. The company will build a new 50,000-ton-per-day concentrator to process increased ore tonnages from the Kino and Colorada Veta open pits. **Corporación del Cobre** (Codelco) in Chile recently considered shutting down its El Salvador Division, which had been losing money, but later decided against this because of the social impact of staff layoffs.

Arizona copper mines also generally mine much lower grade ore than their foreign counterparts. Many of these producers have been able to continue profitable operations during past copper price slumps because of the offsetting earnings from byproduct molybdenum. But these days, with moly prices also low, these producers have had no option but to close down until economic conditions are more favorable.

UNITED STATES

McLaughlin gold permits sought

Homestake Mining Company is continuing its base line environmental studies at its McLaughlin disseminated gold deposit north of San Francisco, California. It is continuing its engineering and metallurgical studies, is working with county and state agencies, and is planning to file for operating permits necessary to develop an open pit mine, and build a cyanide plant. The company controls the mineral rights across 40 square miles around the ore body, which is part of and contiguous to the old Knoxville mercury mine in Napa County. This mine produced between 40,000 and 50,000 flasks of mercury in about 100 years of operation and was last mined for that metal in 1974. Gold production target is 1985.

The company has completed exploration drilling, 400 holes with NC 2½-inch diameter core, with a total length of 160,000 feet. Proven reserves total 20,000,000 tons assaying 0.16 ounce gold per ton.

Three adits and one decline have been driven to confirm drilling results and obtain bulk samples for metallurgical testing. Each development found was crushed in a large crushing plant, sampled, and assayed. Decline No. 1 had an ore intercept of 600 feet and drifting and cross-cutting was done from the decline.

This bulk sampling and testing indicated there are 10 to 12 metallurgical types of ore here, with an indicated average gold recovery of 90 percent. The ground ore will have to be oxidized before cyanidation.

Exploration has found the deposit to be an epithermal type with a high grade core, or pipe, surrounded by a stockwork of fine veins. Gold is always associated with silicification and pyrite. The ore is considered to be complex and refractory.

Since the discovery, Homestake has torn up almost every hot spring and mercury deposit in the northern California Coast Range and southern Oregon. A promising gold deposit of similar geological type has been reported from southeastern Oregon by Homestake goelogists.

The ore body and the mine to be developed have been named "McLaughlin" in honor of the famed economic geologist, Homestake president, chairman, and chairman emeritus.

World Mining

by JAMES A. ANDERSON
Executive Vice President, Homestake Mining Co.

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Gold—its history and role in the U.S. economy and the U.S. exploration program of Homestake Mining Co.

Homestake Mining Co. has had a long and cordial relationship with the American Mining Congress, which has represented the mining industry in Washington since 1897—just 20 years after Homestake was first listed on the New York Stock Exchange. Over the many years of this relationship, Homestake and the Mining Congress have witnessed and interacted in various governmental policies and attitudes toward gold. These attitudes are being reexamined today and some of the government policies that evolve from this reexamination could influence the industry's precious metal exploration. The subject of precious metal exploration represents a broad scope of activity. These remarks will, therefore, focus on two topical considerations—the U.S. financial history of gold and, with that as background, Homestake's gold exploration in the United States.

The first part of this discussion shows that the gold price generally acts as a thermometer of the economic ills of the country rather than the cure for those ills. There are several ways of reading the thermometer both in terms of the success and failure of certain federal government policies and in Homestake's attitude toward investing in gold exploration in this country. Consequently, the second part of the discussion summarizes Homestake's exploration program and investment commitment to discover new gold deposits.

Homestake Mining Co. is the largest gold producer in the United States and holds the largest known gold reserves in North America. Its position as the premier domestic gold mining company has for years been based on the production from the Homestake mine in Lead, S.D. More than 33 million ounces of gold have been extracted from this single location during the last 100 years.

Homestake's position was strengthened in 1980 with the discovery of another major gold deposit, the McLaughlin deposit, in a remote part of Napa County, Calif. That discovery was the result of an aggressive, new-technology exploration program begun in 1978. Homestake now estimates that the McLaughlin deposit contains 3.2 million oz of gold—at least 20 million tons of ore with an average grade of 0.16 oz of gold in each ton. The McLaughlin project is expected to be in operation as a full-scale mine in 1984.

With the Homestake mine, the McLaughlin discovery and a continuing long-term gold exploration program, Homestake expects to remain the leading domestic gold producer in the coming years.

Free market providing incentive for gold mining

Major increases in world gold market prices during the free-market system of the past ten years have created attractive operating margins, and thus new incentives for gold exploration, development and mining in the United States. The magnitude of this renewed profit incentive is depicted in the plot of gold price against the Consumer Price Index in fig. 1.

The federal government has "fixed" the U.S. currency value of gold under various forms of the gold standard for 160 of the last 190 years. Conversely, free-market forces have established the value of gold on just two occasions during the same period, spanning a total of 30 years.

The dollar value of gold was first maintained at a fixed price by the 1792 Mint Act which established a Bimetallic Standard for U.S. currency. The standard was based principally on silver in which the U.S. dollar was equated to 0.774 oz of silver, with the mint required to maintain a fixed silver/gold ratio of 15:1. This established a fixed price of gold at $19.38 per ounce which was maintained until 1834 when the silver/gold ratio was changed to 16:1, elevating the fixed price of gold to $20.67 per ounce.

This fixed price for gold persisted until 1861 when a free-market system called the "Greenback Era" prevailed during the events surrounding the Civil War.

Price of gold increased to $35 in 1934

The deflation following the Civil War caused prices to fall. In 1879, the U.S. government reestablished the fixed price of gold at the pre-war value of $20.67 per ounce, establishing parity with the value of the dollar as indicated by the Consumer Price Index. This fixed price was maintained through the inflation of World War I and the depths of the Great Depression. But in

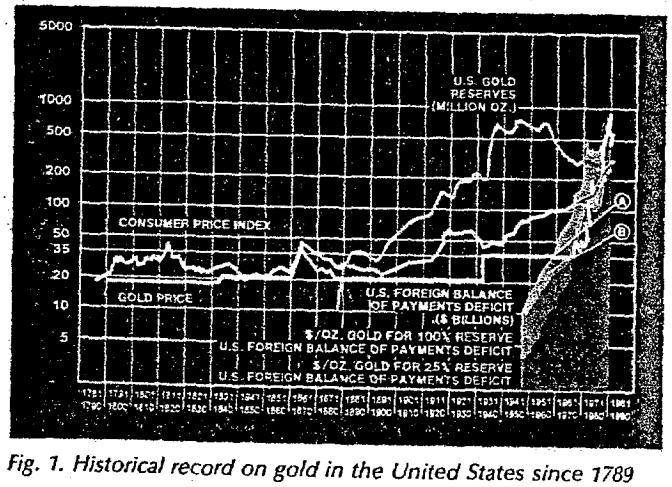

Fig. 1. Historical record on gold in the United States since 1789

1934, the government finally devalued the dollar by increasing the price of gold to $35 an ounce, restoring gold's purchasing power to near parity with U.S. currency.

The $35 an ounce gold price was retained by the 1944 Bretton Woods Agreement, which adopted a gold-dollar standard using the U.S. dollar as the official reserve currency for the newly established International Monetary Fund. Under this agreement, the U.S. dollar was defined as being 1/35 of an ounce of gold. The U.S. agreed to settle all foreign accounts with gold bullion payments and receipts. Dollars and dollar credits held by foreigners were to be counted as reserves equal to gold. This established the gold-backed U.S. dollar as a world reserve currency and stabilized international currencies until the 1960s, when various factors reduced the world's confidence in the ultimate ability of the United States to redeem its currency and debts in gold.

Some of the factors that undermined world confidence in the U.S. dollar were:

1) The rapidly increasing U.S. balance-of-payments deficits during the 1950s (see fig. 1);
2) The rapid growth of gold-backed U.S. dollars as international reserves (Eurodollar reserves) and resulting major extensions of bank credit (Eurodollar loan liabilities);
3) Persistent inflation in the U.S. (decrease in real purchasing power of outstanding money balances as indicated by the CPI).
4) Mounting U.S. federal debt.

U.S. government reserves halved in eight years

This decreasing confidence in the dollar forced the London market price of gold to reach $40 (price on the London Free Market for Gold, established for convenience in 1954) an ounce in October 1960, when the U.S. foreign balance-of-payments deficit exceeded the total value of U.S. government gold reserves at the fixed price of $35 an ounce. As a result, foreigners accelerated their conversion of dollars to gold, reducing the U.S. government reserves from approximately 600 million oz to about 300 million oz between 1960 and 1968 (see fig. 1). During 1968, the outstanding U.S. foreign balance-of-payments deficit exceeded the U.S. gold reserve, valued at $35 an ounce, by four times. In other words, the deficit was backed by less than 25 percent gold reserves. Rather than devalue the dollar to maintain at least a 25 percent gold reserve backing to foreigners, the U.S. government stubbornly clung to the "official" $35 an ounce value of gold but implemented a policy to confine its gold sales to foreign central banks.

Under this policy, the U.S. Treasury exchanged gold for dollars only with foreign central banks and at the "official $35 an ounce value." For other transactions, the U.S. government recognized the free-market dollar value, but it did not exchange gold at that price. (U.S. citizens were still not allowed by law to own gold bullion). This policy established a "two-tier" gold pricing system in the U.S.

The continuing weakness of the U.S. dollar as an international currency led to the establishment of a new international reserve system in January 1970, comprised of Special Drawing Rights (SDR's—paper gold), gold, dollars and foreign currencies. But foreign central banks continued converting U.S. dollars into gold at the "official" fixed price of $35 an ounce. By early 1971, the U.S. government faced another critical point when U.S. currency in domestic circulation also exceeded the 25 percent gold reserve backing at $35 per ounce (actually, Congress had revoked the requirement for a 25 percent gold backing of U.S. currency as early as 1968). Rather than devalue the dollar, the U.S. government stopped the conversion privilege of dollars to gold in August 1971 and allowed the value of the dollar to "float" on international money markets in December 1971.

Dollar devalued in 1971 and again in 1973

For only the second time in our history, the government had abandoned the gold standard (a fixed dollar value for gold). The previous instance was during the "Greenback Era" (1861–1879). For the purpose of establishing its International Monetary Fund reserve positions, the U.S., in an agreement called the Smithsonian Accord, accepted devaluation of the U.S. dollar to $38 an ounce in December 1971. Again, for IMF purposes, the dollar was devalued to $42.22 an ounce in 1973. But the international pressures on the value of the U.S. dollar have not ceased. Since the Organization of Petroleum Exporting Countries' (OPEC) oil embargo in 1973, the United States' balance-of-payment deficits, federal debt and domestic inflation rates have worsened markedly. To put this into some kind of perspective, were the U.S. to settle just one of its outstanding monetary obligations, its foreign balance-of-payments deficit, with its current gold reserves, gold would have to be valued at $830 an ounce.

World political tensions and uncertainties—such as the OPEC oil embargo—do affect free-market gold prices as seen by wide fluctuations in the free-market value. The exile of the Shah of Iran and major increases in OPEC oil prices during 1979 drove the price of gold from $230 an ounce to $498 an ounce by mid-December. When in late December 1979 the U.S. government froze Iranian assets in U.S. banks, the gold price for the first time exceeded $500 an ounce. By the time of the events surrounding the silver commodity market speculation debacle in March 1980, gold prices had soared to $850 an ounce. The question remains as to whether there is a correlation between that top dollar value for gold and the corresponding balance-of-payments deficit value of $830 an ounce mentioned earlier. Whatever the correlation, when the Iranian tensions eased, oil became more plentiful, and the U.S. Congress passed the Reagan economic package to curb domestic inflation, gold prices dropped back to between $385 and $435 an ounce in August 1981.

Incentives for gold exploration and mining identified

Homestake Mining Co. has been producing gold at Lead, S.D. for 104 years, 94 of which have been under some form of a gold standard. During this time, there have been periods when gold prices have been equivalent to cost inflation, providing incentives to find, develop and mine gold. On the other hand, periods of inflation and fixed prices have produced inflationary

cost squeezes, often creating unfavorable climates for mining gold.

The mining industry recognizes there have been times, historically, when investment in gold exploration and mining has been more reasonable than at others. The incentives for such investment are essentially: a) the ability to make a profit on the investment; b) improvements in exploration and mining techniques that make once-marginal deposits economical to process; and c) favorable prospects for the future purchasing power of gold.

A number of periods in our nation's history have provided profit incentives to find, develop and mine gold; on the other hand, fixed-price, inflationary-cost squeezes have often created disincentives to invest in gold mining. The 190-year relationship between U.S. gold prices and the U.S. Consumer Price Index (which, in a general way, reflects mining-cost inflation) provides a means to identify favorable and unfavorable periods in the history of U.S. gold mining. In general,

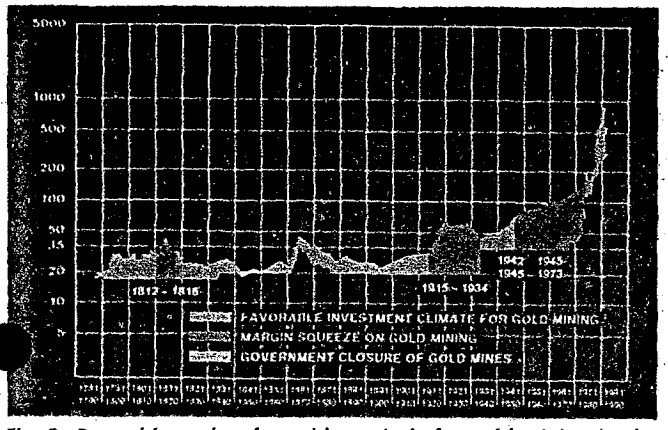

Fig. 2. Favorable and unfavorable periods for gold mining in the United States

except for a brief inflationary period between 1812 and 1816, an attractive investment climate existed for U.S. gold mining during the 123-year period from 1792 until 1915 because the long-term trend in costs generally paralleled changes in the gold price (fig. 2).

Strong inflation beginning in 1915 reversed the favorable price-cost relationship, since the price of gold remained fixed. This created an unfavorable climate for gold investments until 1934, when the official price of gold was increased (dollar devalued) to $35 an ounce. The higher price provided profit incentives for gold mining until 1942, when War Production Board Order L-208 closed U.S. gold mines during World War II. This order was rescinded in 1945, but many mines did not reopen. Many of those that did reopen were forced out of business by 1962–1963 due to high postwar inflation which pushed mining costs up while gold's price remained fixed. A major exception was the Homestake mine, which, except for the three-year government closure, has operated continuously since 1877. Other important exceptions include the Carlin and Cortez discoveries in Nevada which were placed production in the 1960s because of relatively low operating costs attributable to bulk mining by open-pit methods. Overall, however, the investment climate for gold mining remained unfavorable from 1943 until early 1974, two and one half years after the U.S. abandoned the fixed dollar value of gold.

Mine operating costs rising with price of gold

The free-market value of gold provided strong profit incentives for investment, when, in January 1974, the price reached $140 an ounce, establishing parity with the purchasing power of a depreciated U.S. currency. Since 1978, the incentives have been even greater because gold prices have exceeded U.S. currency depreciation for the first time in our history. In summary, favorable gold investment climates existed from 1792 to 1915, 1934 to 1943, and from 1974 to the present; unfavorable climates persisted from 1915 to 1974, with the exception of the period 1934–1943. Although the gold price has risen substantially, operating costs for the gold mining business have also increased. The average free world cost of production of gold is estimated at $215 an ounce as of Dec. 31, 1980.

Exploration technology improvements finally applied to gold

Major advances in exploration technology that have been achieved in the last 35 years were not generally applied to gold because of the unfavorable investment climate that persisted. These advances include new geologic concepts, geologic models, and improved exploration methods, which have been used successfully in the search for porphyry copper, molybdenum, lead/zinc, uranium, and for other mineral deposits. With an improved investment climate, these advances can now be applied with great advantage in gold exploration programs.

Other factors further enhance prospects for profitable gold investment in the United States. New types of gold deposits, such as at Carlin, have been recognized and provide new exploration targets. Improved processing methods, such as heap leaching and carbon-in-pulp recovery methods, may also provide lower operating costs and therefore improve the economic potential of some new projects.

Price a measure of gold's purchasing power

The dollar price of gold is the measure of gold's purchasing power in the United States. Judgments concerning future dollar values of gold are germane considerations when assessing profit incentives to justify investments in exploration, development and mining of gold.

One approach to estimating gold's purchasing power in the United States is referred to as the "commodity value." This approach assumes that gold is simply a commodity and that its U.S. purchasing power is a function of the value of the dollar, as measured by an index such as the Consumer Price Index. Based on this index (with 1967 = 100), the U.S. "commodity" value of gold should be $275 an ounce, the current CPI (fig. 1).

Other values come into play when considering purchasing power since gold represents wealth and can be used as a reserve in support of paper currencies and government debt. Another method, therefore, of estimating the dollar value of gold can be conveniently termed the "U.S. monetary value," implying the U.S. purchasing power of gold is related to such economic variables as the U.S. money supply, federal debt,

balance-of-payments deficit, etc. These "monetary values" of gold can be calculated by dividing the quantity of each by the gold reserves held in the Treasury. These values will obviously change almost daily as the Federal Reserve and Treasury positions change.

Assuming only a 25 percent backing by gold reserves, the current "money supply (M1B) value of gold" would be $407 an ounce; the "federal debt value," $923; and the "U.S. balance-of-payments deficit value," $207 as given below:

U.S. Treasury gold supply	264.2 million oz	$/oz gold	
		25% reserve	100% reserve
Federal debt*	$976 billion	$923	$3,694
Money supply (M1B)	$430 billion	$407	$1,627
U.S. foreign balance of payments deficit	$219 billion	$207	$ 829
Money Supply + U.S. BOPD	$649 billion	$614	$2,456

*as of Sept. 1, 1981

If these monetary values were to require 100 percent gold reserve backing, gold prices in U.S. dollars for each category would be four times greater.

During the last nine years the base gold price has exceeded the "U.S. balance-of-payments deficit value" of gold, assuming the deficit is backed by a 25 percent gold reserve. On the other hand, the highest free-market price for gold ($850 an ounce), achieved in 1980, correlates to the "U.S. balance-of-payments deficit value" of $829, assuming backing by a 100 percent gold reserve. At 25 percent coverage, the "federal debt value" of gold of $923 per ounce somewhat exceeds the highest free-market price for gold attained to date.

If, as some economists assume, a minimum responsible gold coverage would be a 25 percent backing for outstanding money supply (M1B) plus our balance-of-payments deficit, the current purchasing power of gold in the U.S. would be about $614 an ounce. Should confidence in the U.S. dollar totally collapse, the 100 percent backing for the money supply and balance-of-payments deficit value of gold alone would approximate $2450 an ounce. Similar considerations apply to various U.S. government commitments and guarantees, which substantially exceed the official federal debt of $976 billion as of Sept. 1, 1981. Of course, similar calculations for the purchasing power of gold for other countries would result in different values, based on their reserves and monetary liabilities, so that gold values continually vary from currency to currency.

Return to gold standard under discussion

Another financial indicator for the purchasing power of gold might be a comparison of the price performance of gold and of the stock market in the United States (fig. 3). Historically, the stock market has established a long-term growth rate, increasing at approximately 3 percent a year. Gold prices in dollars per ounce were higher than stock market averages from 1792 to 1830, overlapped the market from 1830 to 1890, were lower than the market from 1890 to 1980, and, finally, briefly overlapped the market in the spring of 1980.

These methods of estimating the purchasing power of gold are not just an abstract economic exercise. Since precious metal mining is a very capital intensive industry, we at Homestake use these types of analyses, and others, to arrive at our estimate for the future purchasing power of gold. We have found the outlook sufficiently optimistic to commit a substantial investment in gold exploration, development and mining.

Recently, economic discussions have keyed on the possibility of returning to the gold standard to cure the country's economic ills. The U.S. Federal Reserve defines the "gold standard" as the commitment to establish a fixed relationship between currency and a specified amount of gold. By this definition, the U.S. has been on the "gold standard" for 160 of the last 190 years. The country experienced relatively stable prices for the first 123 years of its history. The long-term trend lines for inflation rates averaged 0.2 percent a year from 1792 until 1915. This stability changed dramatically beginning in 1915, with inflation increasing at an average rate of 3.3 percent a year for the 66 years from 1915 to 1981. This dramatic change in inflation rates took place when the country was on the gold standard, which, according to proponents of gold standard as a cure for inflation, should not have happened. The gold standard, clearly, did not prevent this major long-range change in the rate of inflation. Why should a return to the gold standard cure inflation in the 1980s?

One immediate and obvious association with high inflation rates commencing in 1915 is World War I and the years of war and cold war since then. While wars, and the cost of wars, certainly have contributed to inflation and to actions resulting in inflationary federal policies, it appears that the policies themselves are the cause of persistent long-term inflation. What government policies coincided with World War II that could continue to cause inflation?

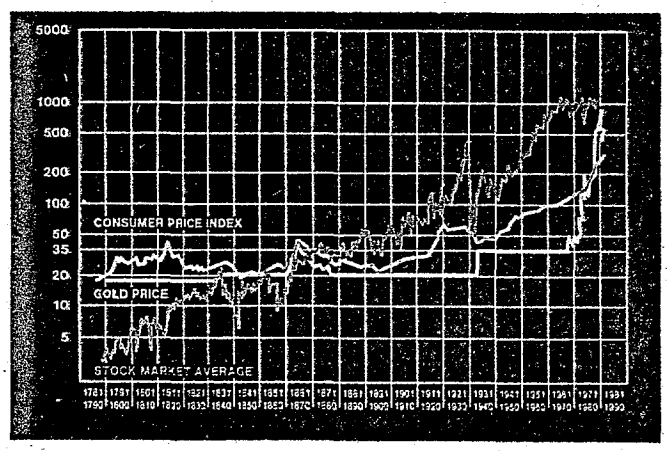

Fig. 3. Price performance of gold and stock market averages

To consider this question, it is enlightening to examine the events surrounding the major increase in the rate of inflation commencing in 1915.

Historical evidence suggests that long-term inflation rates respond to major governmental policies and actions that cannot be prevented by a gold standard. One of the actions was the creation of the Federal Reserve System in 1914, which marked the beginning of permanent, accelerating federal debt, increasing at an average long-term rate of 7.0 percent a year be-

tween 1914 and 1981. Economists agree that excessive government spending, creating federal deficits, results in inordinate demands for goods and consequently forces increases in prices for those goods. Another government policy that directly influenced inflationary pressures was that of federal taxation of personal and corporate income initiated in 1913. This taxation results in higher prices for products, services and labor to maintain investment returns and income at acceptable levels. Nonetheless, the rate of taxation has increased steadily since the inception of taxation. These two government policies clearly influence rates of inflation—influences that would not be prevented by reestablishment of a gold standard.

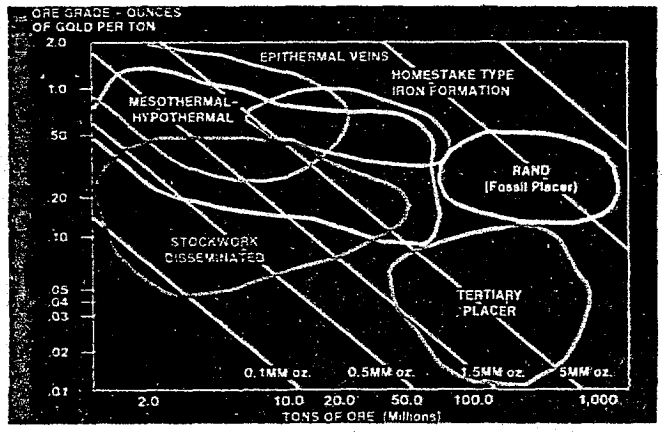

Fig. 4. Types of major world gold deposits

The initial maximum tax rate was 7 percent, without distinction between capital gains and earned income. By 1981, the top tax rate on personal earned income was 50 percent; unearned income, 70 percent; and corporate income, 48 percent. U.S. individual and corporate income tax rates have increased at a rate of 3.4 and 4.2 percent a year, respectively, since 1913. Actual federal income taxes collected have increased at a rate of 6.5 percent a year since 1913. U.S. history has shown that the gold standard does not prevent or cure inflation. Responsible government monetary and fiscal policies and actions, together with minimum government regulation of a free-market system, are required to insure stability of the economy and prices. The free-market price for gold does, however, appear to be a thermometer that reflects the success of the government's efforts to control or cure inflation.

New growth plan set by Homestake in 1978

Homestake's long-term historical business emphasis and commitment to profitable gold exploration, development and mining were reinforced during the development of a new corporate growth plan in 1978. Our optimistic estimates of the long-term future purchasing power of gold, new profit incentives, combined with significant geological and technological advances in exploration and processing opportunities justified accelerating our exploration program to capitalize on Homestake's existing expertise in a timely manner.

A number of criteria were considered in formulating our U.S. gold exploration objectives. Included among them are:

1) Consideration of size and economic parameters;
2) Geology and geologic types of world gold deposits;
3) Appropriate levels of exploration efforts;
4) Budgeting and scheduling of an effective exploration program;
5) Projected exploration risks and rewards.

Homestake's corporate plan emphasizes maintaining and strengthening its position as the premier U.S. gold mining company. We therefore concentrated our efforts toward discovery of large deposits that would substantially enhance our U.S. gold production and earnings. The major objectives of the exploration program were to discover:

1) Gold reserves containing 3 million oz (a major deposit) within ten years;
2) Gold reserves containing 1.5 million oz (a substantial deposit) within five years;
3) Gold reserves containing a minimum of 0.5 million oz (an acceptable deposit discovered by "fallout" from the search for larger reserves).

Internal rate of return considerations (minimum of 15 percent) require that we find relatively low-cost mines (in the lower half of the cost curve) and relatively high-grade deposits suitable for bulk surface or underground mining. For open pit mining, our objectives focus on deposits with ore grades exceeding 0.15 oz per ton; for underground mining, grades exceeding 0.3 oz per ton.

Geologic types of gold deposits studied

Homestake geologists studied major world gold deposits, classifying the known deposits according to their geology, size and grade, and from this data they determined the desirability of each for our exploration programs in the United States. The geological types of major gold deposits identified in the study are depicted in fig. 4.

The contour lines on fig. 4 represent total contained ounces of gold in a deposit and provide a quick reference to the overall size and importance of each geologic type of gold deposit. The largest gold deposits of the world are the Witwatersrand type Fossil Placer Deposits of South Africa, which contain grades ranging from 0.10 to 0.50 oz per ton, with ore tonnages ranging from 100 million to over 1 billion tons.

After studying the economics, size and grade objectives and the geological occurrences of the various gold deposits likely to occur in the United States, it was decided to search for Homestake-type and Stockwork-disseminated deposits. We also decided to conduct geologic research to develop new geologic concepts that would lead to the discovery of new types of gold deposits of substantial size and grade to meet our corporate objectives.

Exploration success examined empirically

The next step was to formulate the exploration program and determine the extent of the exploration effort in terms of budget and manpower. To accomplish this, we analyzed cost data and probability factors for each of the recognized sequential stages of mineral development. The stages include reconnaissance, target exploration, deposit development, plant construction/mine development and production.

The costs of each activity within each stage of exploration and development were estimated from

Homestake's exploration experience and records. In order to determine probable success ratios to apply to a gold exploration program, it was decided to use the empirical approach based on prior experience. Published and privately developed data on the success ratios of metal exploration programs in the United States and Canada from 1880 to 1976, representing a wide range of success ratios, were compiled (table 1).

	AMC Survey of 41 Large Mining Companies (6 Years, 1970–75)		Private Survey of 9 Major Firms (Pre-1958 Thru 1966)		OTA Survey of 12 Major Firms (10 Years, 1970–76)	
	Number	%	Number	%	Number	%
Reconnaisance	5,718		732		?	
Target Exploration (Drilling Project)	536		165		451	
Deposit Development	83	15	19	12	32	8
Substantial Economic Deposit	?		3	2	19	4

% = % of targets advancing beyond drilling stage

Table 1. U.S. metal exploration success rates

qualified personnel, and the competitive situation.

Formulation of Homestake's exploration program included all of the above factors in determining its U.S. gold exploration program. Projected probabilities, costs and timing of the company's exploration program are summarized in fig. 6 and table 4. These graphics show that substantial expenditures must be made to test a large number of targets to achieve the high probability of success in gold exploration that was desired. Homestake's projected gold exploration expenditures and projected returns (i.e., risk/reward ratio), based on achieving the five-year program objectives, are summarized in fig. 8.

Exploration team reorganized to manage by function

To take maximum advantage of the company's in-house expertise to achieve the major objectives of the new growth plan, the exploration team was reorganized in November 1978 to manage by exploration function rather than by geographic location. The spe-

	50% CERTAINTY			75% CERTAINTY		
	Contained Ounces Gold			Contained Ounces Gold		
	500,000	1,500,000	3,000,000	500,000	1,500,000	3,000,000
	Number of Targets			Number of Targets		
Reconnaissance Examinations	56	280	560	100	500	1,000
Target Evaluations	8	40	80	10	50	100
Ore Target Evaluations	2	10	20	2	10	20
Deposit Development	1	1	1	1	1	1

Table 2. Gold exploration probability levels

Table 3. Gold exploration program exploration budget (in millions of dollars)

Ten-Year Production (Million Oz)	Certainty Level 75%	50%	30%
.5	$ 7	$ 6	$ 5
1.5	$30	$22	$16
3.0	$42	$36	$31
6.0	$78	$65	$57

All Production From One Property

Probability analyses were then conducted on the data to provide the statistical framework to develop Homestake's empirical probability models for the number of activities that would be required in each exploration stage to accomplish the discovery of an economic deposit of a given size. The results of two such analyses are summarized in table 2.

Once the probability model for a given size of deposit was available, the cost estimates for each exploration activity were used to determine the cost of exploration programs for a wide range of probabilities for a given deposit size (table 3). For example, we estimated it would cost $30 million to search for and find a deposit that would produce 150,000 ounces of gold a year for ten years; that was for a probability of 75 percent. To put that figure in perspective, we estimated it would cost $22 million for a 50 percent probability of success, and $16 million for a 30 percent probability of success.

It is important to note that for a given budget expenditure, exploring for a single metal, such as gold, has a higher probability of success of a discovery than does the probability of finding gold in a multi-metal search with the same budget (fig. 5). It follows that similar studies be conducted to develop exploration programs for each metal in a proposed multi-metal search effort. The timing of exploration expenditures must also be considered, both from the standpoint of the resources of the exploring company, availability of

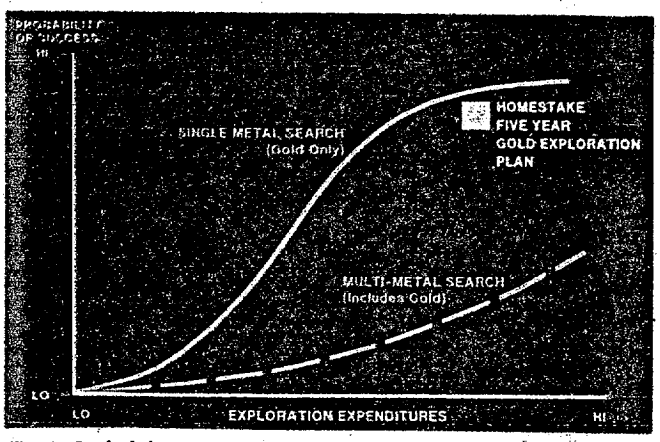

Fig. 5. Probability model for gold discovery

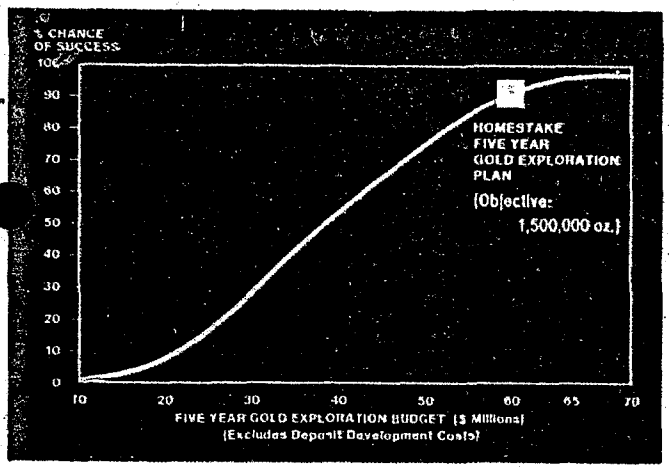

Fig. 6. Gold exploration budget

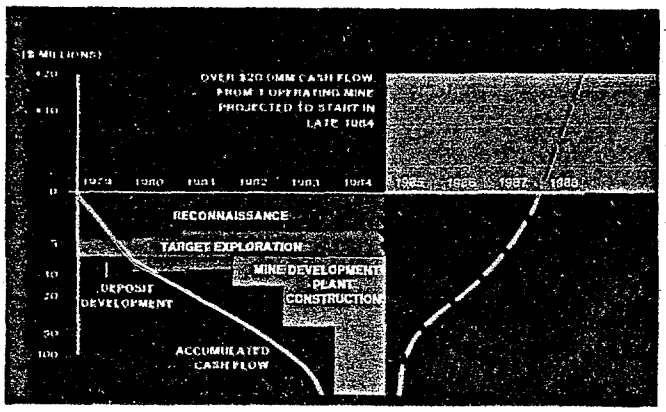

Fig. 7. Estimated gold expenditure and income—five year plan

Specific functional exploration activities include reconnaissance, target exploration, deposit development, and land acquisition-administration. The managers of these activities are responsible for developing plans and budgets, quality and quantity standards, programs and organization to achieve the desired results for their functional areas, regardless of the geographic location. For instance:

1) The gold reconnaissance manager is responsible for the search and selection of favorable areas, detailed appraisals of favorable areas, and the selection and prioritization of a queue of target areas, as well as land acquisition to control target lands. High priority targets are transferred to the target evaluation manager for further evaluation. Lesser priority properties are retained in a queue for future consideration by reconnaissance.
2) The target exploration manager is responsible for initial exploration testing to evaluate specific target areas. If preliminary results and economic estimates are encouraging, this manager is responsible for conducting additional exploration to delineate the deposit and to conduct preliminary engineering, metallurgical testing and economic appraisals. High priority properties are transferred to the deposit development manager. Again, lesser priority properties are retained in a target queue.
3) The deposit development manager is responsible for bulk sampling, pilot plant testing, metallurgical and engineering appraisals, conceptual mine and plant design and for preparation of a preliminary feasibility study. High priority properties with acceptable size and economics are transferred to the Homestake operations management for mine development, plant construction and production.
4) The land acquisition-administration manager is responsible for acquiring land control and administering land contracts. This manager provides these services to the other exploration managers in coordination with corporate policy.

Conceptual approach to reconnaissance used

The conceptual approach to reconnaissance has been utilized in place of the more conventional geographical approach. Under this system, reconnaissance teams pursue specific types of gold deposits in any favorable area. Applied geologic research is conducted to: refine geologic models and search areas with potential for well known types of gold deposits; develop geologic models and search areas for poorly known or poorly understood gold deposits; develop theoretical concepts and techniques to search for new types of gold deposits; and improve performance and decision-making at each functional stage of exploration.

Development of a prioritized queue of prospects for each stage of exploration has been an essential element of Homestake's exploration program since this system provides an opportunity to identify "fast-track" high priority projects, leaving lower priority prospects in an inventory classification for future consideration.

Priority ratings are assigned to properties by estimating the geologic probability of occurrence of commercial gold deposits which conform to the objective of the exploration program. Criteria used to make these priority evaluations include: a) the presence of a geologic trap (favorable structural setting or favorable permeable host rock), b) evidence of a gold source (presence of ore grade or anomalous gold values, or in some cases, pathfinder elements), and c) the presence of favorable alteration systems (alteration assemblage

	Number of Projects Strategic Objectives	
	5-year plan	10-year plan
Reconnaissance		
• Reconnaissance Exams	2000	4000
• Detailed Evaluations	200	400
Target Exploration		
• Target Evaluations	65	130
• Ore Target Evaluations	16	32
Resource Development		
• Deposit Inventory	8	16
• Deposit Development	4	8
Mine Discoveries		
• Economic (0.5 million oz.)	3	5
• Substantial (1.5 million oz.)	1	2
• Major (3 million oz.)	0	1

Exploration Expenditures
• 5-year plan: $60 million

Table 4. Exploration probability statistics—U.S. gold program, Homestake Mining Co.

	Number of Projects 1979–1981 Program	
	Plan to Date	Results to Date
Reconnaissance		
• Reconnaissance Exams	1011	1111
• Detailed Evaluations	58	55
Target Exploration		
• Target Evaluations	17	14
• Ore Target Evaluations	2	3
Resource Development		
• Deposit Inventory	3	3
• Deposit Development	0	0
Mine Discoveries		
• Economic (0.5 million oz.)	0	0
• Substantial (1.5 million oz.)	0	0
• Major (3 million oz.)	0	1

Exploration Expenditures to Date (2.5 years): $28 million

Table 5. Exploration statistics—U.S. gold program, Homestake Mining Co.

related to a geological model for a given type of gold deposit).

After the geologic probability assessment for each prospect is estimated, the cost of arriving at the next decision point for each prospect is determined. Risk factors and present value estimates are then determined for each property and a priority established. By updating priorities at each decision point for a project, we are able to continuously prioritize projects at all stages of the exploration sequence.

During the first year of exploration under this system, Homestake emphasized the development of a reconnaissance queue of prospects. At the end of the first year, we had identified enough high-priority properties to begin our target exploration programs. Target evaluation during the second year generated a deposit development project and also resulted in identification of several targets requiring intensified target evaluation efforts. Deposit development continued on a high-priority project during the third year. In addition, we continued fine-tuning of our reconnaissance, target exploration and management systems.

Payoff came within first two years of program

Homestake has been fortunate early in its exploration program to realize the soundness of the theoretical concepts. The company discovered and began development of a major discovery within the first two years of the exploration program. This compares quite favorably with our hopes and expectations.

The McLaughlin gold deposit, located in a remote area of northern Napa County, Calif. near Knoxville, was discovered during the second year of our exploration program. This deposit contains announced reserves of 20 million tons at 0.16 oz of gold per ton and is obviously a major addition to Homestake's gold reserve portfolio. This discovery achieved two of our exploration program goals:

1) The deposit meets the objective of our ten-year program, i.e., the discovery of an ore deposit containing 3 million oz in gold reserves
2) Discovery of a new geologic type of gold deposit, using new geological models and concepts.

The McLaughlin deposit may best be described as a Stockwork gold deposit in volcanic and sedimentary rock, suitable for open pit mining. In addition, we foresee the possibility for an underground mine operation on higher-grade gold ore below the estimated pit limits. The cost to discover and develop the McLaughlin property, together with establishing a queue of more than 60 properties in various stages of evaluation, amounts to $28 million to date, or approximately $9 per ounce of announced gold reserves at the McLaughlin deposit.

An analysis of published open pit gold reserves in 19 deposits in the United States illustrates that a large number of deposits have been discovered, expanded and resurrected during the last five years. U.S. open pit-type gold reserves as of Sept. 1, 1981 are estimated at 242 million tons of ore at a grade of 0.076 oz of gold per ton or 18.5 million oz of gold. Grades range from 0.03 to 0.22 oz of gold per ton, with tonnages ranging from 1 million to 65 million. The average deposit is estimated to contain 12.7 million tons of ore at 0.076 oz of gold per ton, or approximately 1 million oz of gold.

Success was partly luck

The profit potential for gold mining investments due to the free-market prices of the last decade—after nearly 65 years of adverse fixed-price, high-inflation margin squeezes—has provided new incentives for gold exploration, development and mining.

Major advances in exploration technology that have been developed and successfully used in other metals and minerals exploration since World War II have not been intensively applied to gold exploration because of the adverse investment climate for gold. To take advantage of these new economic and technological incentives, Homestake developed its current exploration strategy and objectives in 1978.

A major exploration program based on new geologic concepts, probability analyses, and a functional approach to exploration management was implemented in 1979. The objectives included discovery of a major 3.0-million-oz gold deposit within ten years and a 1.5-million-oz deposit within five years. Our board of directors committed to an exploration budget of approximately $12 million a year for five years, a major commitment for Homestake at a time when corporate earnings were at $31 million a year (1978). This was a daring commitment, but the board expressed its confidence—in dollars and cents—in the expertise Homestake had developed in exploring, developing and mining for gold during its 100 years as the nation's largest gold producer.

The program met with early success in the fall of 1980 with the discovery of the 3.2-million-oz McLaughlin gold deposit in California. The deposit has been named after Dr. Donald H. McLaughlin, the 89-year-old retired chairman (and still a director) of Homestake Mining Co. It was McLaughlin, who, as a geologist in the 1920s, played a key role in expanding the known gold reserves at the Homestake mine in South Dakota. The McLaughlin discovery has to be attributed to the efforts of a fine, dedicated professional exploration team. It achieved at the end of the second year Homestake's ten-year exploration program goal. This, we admit, was partly luck but also a result of Homestake's excellent and extensive geological files which aided in rapid development of new geologic concepts.

The long-term commitment to new exploration ideas and approaches by Homestake's executive management and directors has provided excellent support for our exploration efforts. Our early success and our ongoing search for new deposits will keep the company in the forefront of gold production and expertise in the United States for years to come.

James A. Anderson is currently executive vice president of exploration and business development for Homestake Mining Co., which he joined in 1975. Previously he was with Occidental Minerals Corp. from 1968 to 1975, lastly as vice president of U.S. exploration, and Kennecott Copper Corp. from 1960 to 1968, initially as a research geologist and later as an exploration geologist. He has an MS in geological engineering and a PhD in economic geology.

Occy Minerals has a 50 percent interest in the Alligator Ridge gold mine near Ely, Nevada and a controlling interest in the Candelaria silver and gold mine near Hawthorne, Nevada. Nerco is the mining and resource development subsidiary of Pacific Power and Light Company of Portland, Oregon.

CALIFORNIA

Homestake Mining Company has awarded Davy McKee Corporation the engineering procurement and construction contract for the McLaughlin gold project (see WORLD MINING, January 1983, page 86). The McLaughlin deposit is located 90 miles north of San Francisco and has estimated reserves of 3,200,000 ounces of gold. Initial production from an open pit mine is planned at 3,000 tons of ore per day, beginning in 1985.

Davy McKee will build mine shop and support facilities, a grinding plant, a five-mile slurry transport line, and a processing plant, which will include autoclaves and a carbon-in-pulp circuit.

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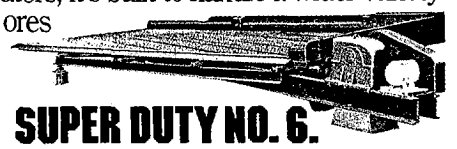

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CARIBBEAN

JAMAICA

UNITED STATES MAKES ITS SECOND BIG BAUXITE BUY

The United States General Services Administration (GSA) has agreed to buy 1,000,000 long dry tons of bauxite from Jamaica this year. Details are still being worked out, but Jamaican prime minister Edward Seaga said that the shipments may begin in the next two to three months. Jamaica sold 1,600,000 tons of bauxite to the GSA in February 1982, in exchange for cash and dairy products.

Impetus for the GSA purchase may have been the failure of United States president Ronald Reagan's Caribbean Basin Initiative, a financial aid plan for Caribbean countries.

CUBA

NICKEL-COBALT PILOT PLANT IS UNDER CONSTRUCTION

A 20-ton-per-day pilot plant for nickel-cobalt laterite ores is reportedly being built at the construction site of the Punta Gorda nickel plant. The pilot plant is expected to be completed in mid-1983. Cuba now has two operating nickel plants, at Moa Bay and Nicaro.

JAMAICA

The Soviet Union will purchase 1,000,000 tons of bauxite per year from Jamaica over a seven-year period starting in 1984. A 200,000-ton advance shipment will be delivered in 1983. The new contract replaces an earlier agreement for 250,000 tons of alumina per year for seven years.

AFRICA

ZAMBIA

CHAMBISHI COBALT PLANT HAS BEEN COMMISSIONED

The cobalt vacuum refining plant at **Zambia Consolidated Copper Mines Ltd.**'s (ZCCM) Chibuluma Division (see WORLD MINING, November 1980, page 7) has now been commissioned. The plant is actually located at Chambishi, where a roast-leach-electrowinning (RLE) plant recovers cobalt from concentrates received from the Chibuluma and Luanshya properties.

The purpose of the vacuum plant is to increase the purity of the finished cobalt by reducing the trace quantities of the main impurities, lead, zinc, and cadmium.

Vol. 36, no. 2

F.T.G.
FEB 4 1983

RECEIVED
FEB 3 1983
EXPLORATION DEPT.

JANUARY 28, 1983

MEMORANDUM TO: F. T. GRAYBEAL

FROM: R. E. GALE

British Columbia
Quesnel River Au
Dome Mines Ltd.
93 A

The accompanying memo by Dave Fletcher on Mark Rebagliatti's data clears up the location of Dome's 1 million ton 6.8 gm/ton Au deposit which I previously believed was the Kwun Lake deposit. The deposit in question is in fact the Quesnel River Au deposit situated 20 - 30 miles N.W. of Kwun Lake. I will correct this fact in our files here.

The data that Dave obtained from Mark gives us more insight into this somewhat unique type of deposit here. I wonder if the same belt of Alkalai intrusions could extend down into Washington State i.e. Kennecott's Glacier Peak Cu deposit might carry Au?

We are seeking deposits of this type and Rebagliatti offered a possible deposit of this type to us, but his deal is complicated and tough and we want to see the area this summer.

R E Gale

R. E. Gale

RECEIVED
FEB 11 1983
EXPLORATION DEPARTMENT

January 27, 1983

R.E.G.

JAN 28 1983

MEMORANDUM TO: R. E. GALE

FROM: D. M. FLETCHER

GOLD IN ALKALINE INTRUSIVES

Please find attached a paper by Mark Rebagliati (ex-manager of Saskatchewan Mining Development Corporation at Vancouver, currently a consulting geologist) describing in fair detail his genetic concepts pertaining to gold deposits associated with synvolcanic alkaline intrusives occurring in the Intermontane Belt. The background for Rebagliati's model which is limited spatially (See Figure 1) and in time (upper Triassic and lower Jurassic) is gleaned from Paper 36 titled The Alkaline Suite Porphyry Deposits - A Summary, CIMM Special Volume 15, Porphyry Deposits of the Canadian Cordillera, which is also attached.

Rebagliati's model is specific as it links gold mineralization to significant copper deposits at Afton, Cariboo Bell, Copper Mountain, Stikine Copper, Gnat Lake and Lorraine. The linear distribution of alkaline and related intrusions located in regional faults, fault intersections and breccias postioned in contact with volcanoclastic rocks stresses the importance of structural controls. Hypogene sulfide and precious metal systems can be delineated utilizing conventional geochemical and geophysical techniques.

D. M. Fletcher

A DESCRIPTION AND GENETIC MODEL
for the
SYNVOLCANIC ALKALINE INTRUSIVE ASSOCIATED
QUESNEL RIVER GOLD DEPOSIT
Cariboo Region, British Columbia

The Quesnel River gold deposit is located in the Upper Triassic Quesnel Trough, a linear belt of alkaline and calc-alkaline volcanic rocks and derived sediments that extends from the U.S. border to the Yukon.

At Quesnel River (QR in Figure 2), the volcanic Pile is comprised of a thick assemblage of interbedded augite (±hornblende) porphyry basalts and their autobrecciated and pyroclastic equivalents. This volcanic assemblage is overlain by or interdigitated with flanking siltstone, greywacke and various volcanic sandstones. A few augite porphyry basalt flows are interbedded with the siltstones near the base of the sedimentary assemblage. The youngest unit in the area is an inwardly zoned diorite-monzonite-syenite alkalic stock. The stock is comagmatic with the volcanic pile, but in the vicinity of the gold deposit has intruded all units. Although they do not intrude the stock, a number of basalt dykes occur in the area, and are the most prevalent within the augite porphyry basalts. These dykes are probably high-level equivalents of the stock, and are pre-, syn- and post- gold mineralization. Narrow hornblende porphyry dykes intrude all units including the stock. The period of mineralization appears to have taken place at the culmination of volcanic activity simultaneous to the onset of deposition of the overlying siltstones. The superimposition of typical porphyry alteration assemblages upon the deposit and altered host rock indicates that gold deposition took place before the stock rose through the volcanic pile and intruded the sediments.

The augite (±hornblende) porphyry basalt flows and breccias host the mineralization. In the footwall of the deposit, the basalts are pervasively carbonitized and are essentially a carbonate rock cut by calcite veinlets. Only a few primary textures and relic augite phenocrysts are preserved. The intensity of the carbonitization and the pervasive finely disseminated pyrite decreases gradually away from the deposit. Conversely the intensity of the carbonitization increases upsection until it terminates abruptly against a propylitic zone. It is this propylitic zone that hosts the gold. The intensity of the epidote-pyrite alteration

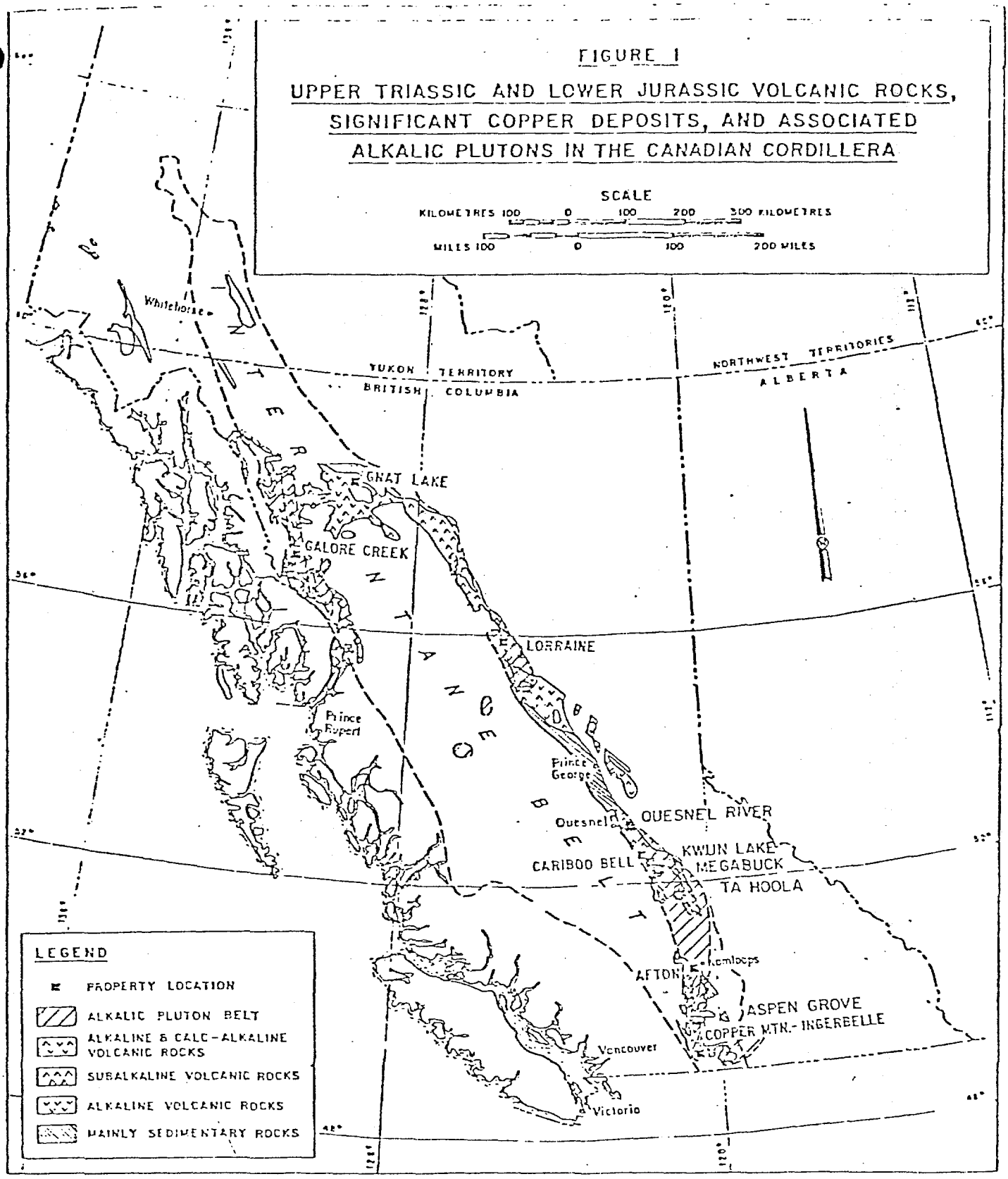


FIGURE 1 — Upper Triassic and Lower Jurassic volcanic rocks, significant copper deposits and associated alkalic plutons in the Canadian Cordillera.

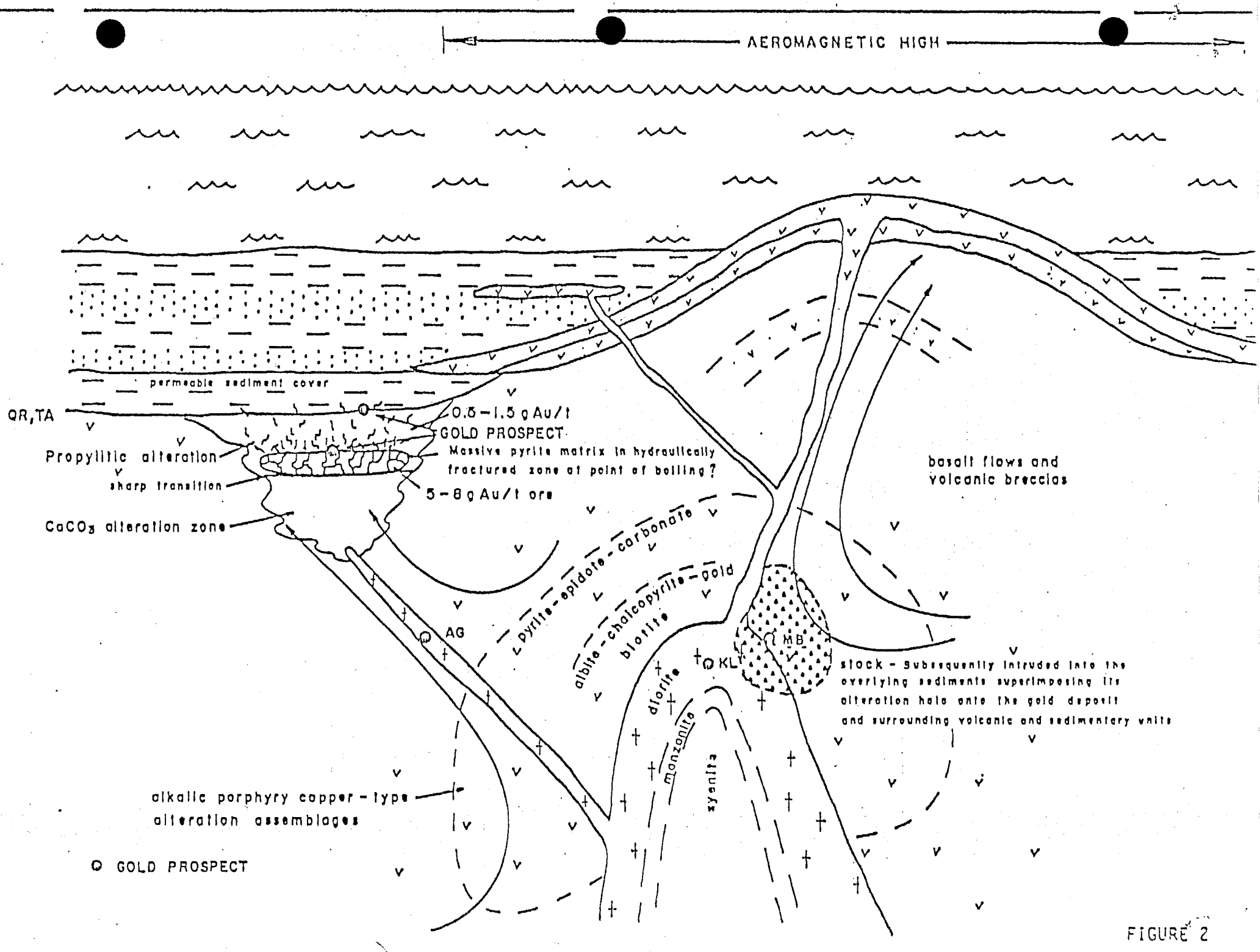


FIGURE 2

THE QUESNEL RIVER GOLD DEPOSIT--A GENETIC MODEL & GEOLOGICAL CAPTION

is so strong that the rock can be called a propylite. The propylite is comprised of 20-80% epidote and 10-70% pyrite. The epidotization of the silicate minerals is pervasive and is consistent in its nature, whereas pyrite occurs in three forms, namely as very finely-disseminated grains, coarse aggregates and as coarse-grained veinlets. It is the very finely disseminated "dusty" pyrite that is thought to be auriferous. In the propylite, at or very near the transition between the carbonate and propylite alteration zones, massive to semi-massive pyrite occurs as a matrix in brecciated augite porphyry flows and pyroclastics. Upwards from the contact the intensity of the fracturing decreases as does the intensity of the epidotization and pyritization. The gold content near the carbonate-propylite contact averages about 7 ppm, and ranges from 2 to 45 ppm. The overlying propylite contains in the order of 0.50 to 1.50 ppm gold. It appears that the hydrothermal fumarolic system deposited the gold slightly below the volcanic-sedimentary interface while the overlying siltstones were being deposited. This is evidenced by weak epidote and pyrite alteration and anomalous gold contents in the lowermost siltstone beds overlying the deposit. Immediately adjacent to the deposit the siltstones contain in the order of 200 ppb gold but grades diminish rapidly away from the contact. In the basalts, away from the deposit, minor epidote-pyrite veinlets or epidote-pyrite coated joints occasionally contain anomalous concentrations of gold.

The auriferous propylite zone is approximately 500 m long and up to 200 m wide. The ore zone adjacent to the carbonate-propylite contact is approximately 300 m long and up to 30 wide. To date reserves in the order of 1 million tonnes grading 6.8 grams of gold have been identified within a larger body of approximately 10 million tonnes grading in the order of 1 gram. The deposit is terminated by faults at depth and in both directions along strike. Reserves will undoubtedly be extended once the fault riddle has been solved.

The ore has a specific gravity of 3.2. Metallurgical work indicates that, with cyanidation of a pyrite floatation concentrate, a high recovery rate can be achieved. The associated chalcopyrite is coarse-grained, but may not be sufficiently abundant to economically recover.

Results from exploration undertaken in other areas of the Quesnel Trough, principally near Ta Hoola Lake (TA) suggest that (semi) conformable gold-bearing zones may be underlain by discordant zones of epithermal mineralization. Whereas, at Kwun Lake (KL) a complex syenodiorite-monzonite stock hosts a significant stockwork prospect of auriferous anhydrite-pyrite veinlets.

Types and setting for gold deposition that have been identified include:

- Stockwork vein systems in the subvolcanic intrusives (KL)

- High level intrusive or contact breccias (MB)

- Discordant zones in the overlying volcanic assemblage (TA)

- Semi-conformable accumulations deposited in coarse-grained permeable pyroclastic units at or near the sea floor (QR)

- Syngenetic mineralization in siltstones and volcaniclastic sediments deposited contemporaneously with geothermal exhalative hotspring activity (TA)

Thus as illustrated in Figure 2, in the synvolcanic alkaline intrusive environment gold deposition can take place over a broad vertical range in diverse geological settings.

The Alkaline Suite Porphyry Deposits — A Summary

D. A. Barr,
Du Pont of Canada Exploration Limited,
Vancouver, B.C.

P. E. Fox,
Fox Geological Consultants,
Kamloops, B.C.

K. E. Northcote and V. A. Preto,
B.C. Department of Mines and Petroleum Resources,
Victoria, B.C.

Abstract

Alkaline suite porphyry deposits and related intrusions occur in the Intermontane Zone. They are spatially and genetically related to the Upper Triassic Nicola-Takla-Stuhini volcanic assemblages and comagmatic alkaline plutons. The latter are associated with linear structures of regional extent and range from plugs and stocks to small batholiths. They have chemical compositions identical to their volcanic host rocks and were evidently emplaced in volcanic centers from which most of the nearby volcanic material was derived.

Porphyry deposits are coeval with associated plutons and enclosing volcanic host rocks. The deposits occur in zones of intense faulting, fracturing, brecciation and hydrothermal alteration. The dominant alteration products related to mineralized zones are potash feldspar and biotite; propylitic zones commonly fringe the deposits. Hypogene sulphides, which have formed contemporaneously with hydrothermal alteration of host rocks, comprise pyrite, chalcopyrite, bornite, chalcocite and pyrrhotite in decreasing order of abundance. Molybdenite is a minor constituent, but gold and silver are present in significant amounts. Stockwork and fracture-controlled deposits are most common, but pyrometasomatic deposits are also abundant. Supergene deposits, although rare in the Canadian Cordillera, are important in some of the alkaline suite deposits — native copper, chalcocite and copper oxides form a significant part of supergene ores.

Future discovery of alkaline suite porphyry deposits will depend on utilizing combined geological-geochemical-geophysical techniques and considerable exploratory drilling.

Introduction

THE PURPOSE of this paper is to outline the distribution, main characteristics and structural setting of the alkaline suite porphyry deposits. These deposits, which represent an important class of porphyry copper deposits of the Canadian Cordillera, are the "alkaline", "syenite", "volcanic" or "diorite" deposits of Sutherland Brown (1969, 1971, this volume), Preto (1974), Hollister (1974) and Fox (1975). They should not be confused with deposits associated with calc-alkaline intrusions, because the former deposits generally pose a very different challenge both to the mining engineer and exploration geologist. Compared with calc-alkaline deposits, porphyry deposits of the alkaline suite commonly grade into pyrometasomatic or skarn deposits, they lack appreciable amounts of molybdenite, and are usually richer in gold and silver. The deposits are invariably associated with small, complex, alkaline plutons that are comagmatic with the enclosing volcanic rocks.

Distribution and Age

Figure 1 is a generalized outline of the Intermontane Zone of British Columbia showing the distribution of porphyry deposits and related alkaline intrusions. All deposits and alkalic plutons occur within the Intermontane Zone and coincide with the outcrop area of Upper Triassic Nicola-Takla-Stuhini volcanic assemblages. The isotopic age of these deposits falls in the 175-198 m y range (Christopher and Carter, this volume), which agrees well with the Upper Triassic to Lower Jurassic age of the volcanic host rocks.

The plutons characteristically form linear belts several hundred kilometers or more in length. They are most abundant in the Quesnel Trough in central and southern British Columbia and in southerly parts of the Whitehorse Trough in northwestern British Columbia (see Douglas, 1970; Souther and Armstrong, 1966, for use of terms). Smaller belts are known north of Thutade Lake and near the headwaters of Stikine River.

Structural Setting

Sutherland Brown (this volume) has included the porphyry deposits of the alkaline suite in the volcanic subclass, in which, among other characteristics, ore zones and intrusions are spatially related to faults, many of regional extent. Such relationships have long been known in southern British Columbia in the Copper Mountain and Iron Mask areas. Both these plutons are composite intrusions elongate in a north to northwesterly direction. The Iron Mask batholith is situated along a regional northwest fracture zone and is cut by numerous northwesterly faults. The Copper Mountain stock lies along a north-trending fault system (Boundary - Summers Creek faults) of regional magnitude and has northwest faults along its northern edge. East- and northeast-striking faults are also prominent in both plutons, or in their immediate vicinity, and are spatially related to orebodies.

In the Stikine River district, two north-trending belts have been traced for lengths of 100 kilometers and 250 kilometers respectively; the longest belt comprises at least 15 alkalic plutons. Many are sulphide-bearing porphyries and several, notably occurrences at Galore Creek (Stikine Copper), contain important deposits. The two belts, individual plutons and associated copper deposits are controlled by faults, intersection of faults, fracture zones and breccias respectively. Similarly, alkaline plutons of the Quesnel Trough are controlled by north to northwest fault systems. Such faults can be traced from the Boundary - Summers Creek fault near Copper Mountain north to the North Thompson - Quesnel fault system near Cariboo Bell. Spatial relationships indicate that major faults influenced the emplacement of alkalic porphyry complexes and were active over a prolonged period, possibly from Upper Triassic to Recent time.

Volcanic Rocks

DISTRIBUTION AND LITHOLOGY

Thick sequences of Upper Triassic and Lower Jurassic volcanic rocks are the hosts of porphyry deposits of the alkaline suite. These are the well-known Nicola-Takla-Stuhini groups. The volcanic rocks have a wide range of chemical composition. In many areas, thick successions of volcanogenic and epiclastic sedimentary rocks are intercalated. Rice (1960) described the Nicola Group in southern British Columbia as a bewildering array of volcanic types that defied attempts to subdivide the group. Work by Schau (1970), Preto (1972, 1974), Campbell and Tipper (1970, 1971), Roddick *et al.* (1967), Souther and Armstrong (1966), and much unpublished information, has permitted subdivision and documentation of many of the general aspects of these rocks.

Figure 1 shows the main volcanic groups and as-

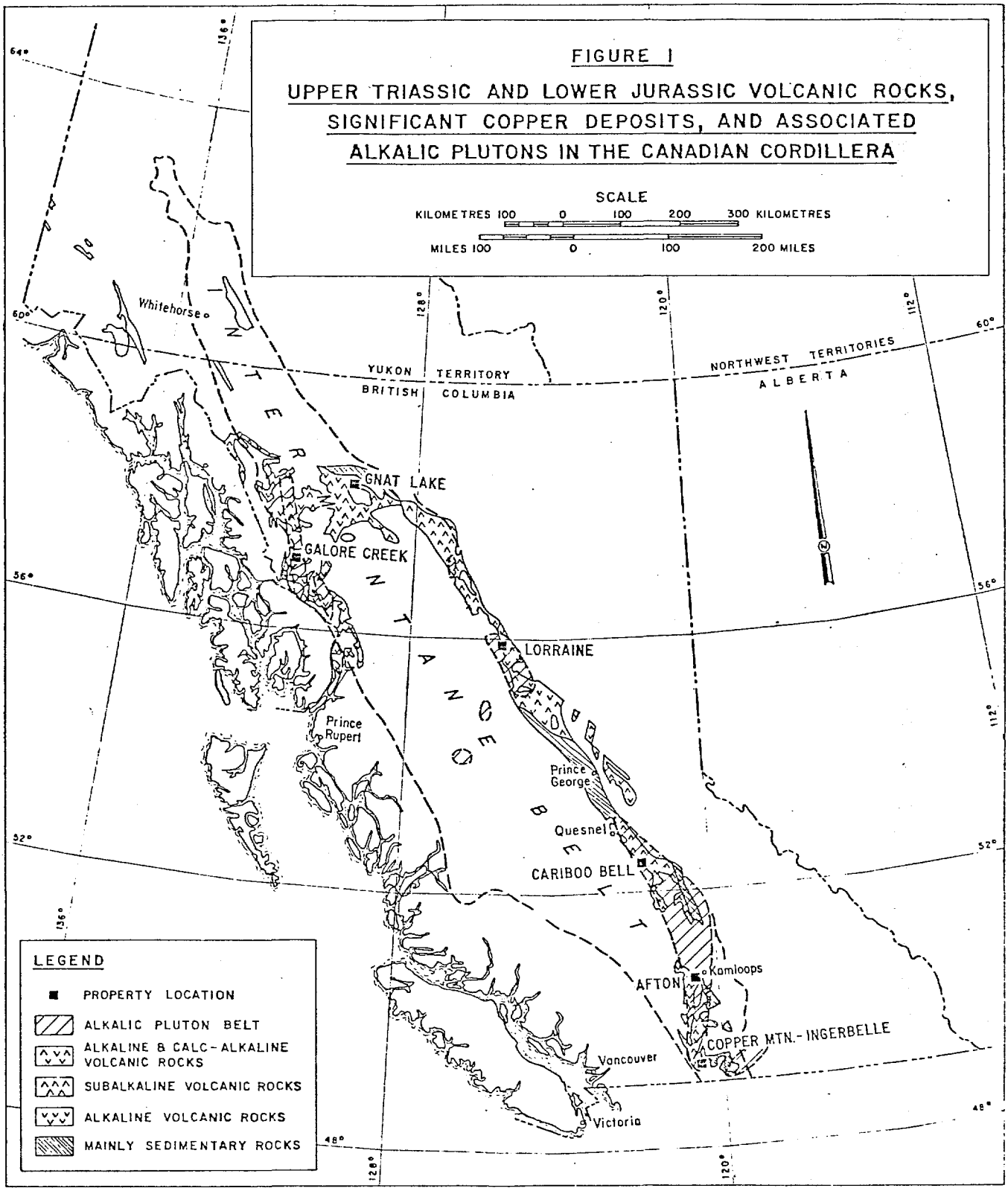


FIGURE 1 — Upper Triassic and Lower Jurassic volcanic rocks, significant copper deposits and associated alkalic plutons in the Canadian Cordillera.

sociated sedimentary basins. The succession is divided into four major groups: a southern part, from the International Border to Quesnel, consisting of mixed alkaline and calc-alkaline rocks in the south and alkaline rocks farther north; a northern part, which stretches from north of Prince George almost to the [illegible]on, that consists from south to north of mixed alkaline and calc-alkaline, subalkaline, and alkaline and calc-alkaline rocks; a region in the Stikine area near Galore Creek that appears to consist entirely of alkaline rocks; and a region near Prince George that is underlain by flyschoid sediments consisting of greywacke, shale, argillite and tuff. Elsewhere, thick sedimentary sequences flank volcanic belts to the east and west. The volcanic and interlayered sedimentary strata form a succession approximately 6500 meters thick.

The volcanic rocks consist of augite- and plagioclase-bearing andesitic and basaltic porphyries. Most contain phenocrysts of pale green augite and plagioclase in a groundmass of augite, plagioclase, alkali-feldspar and small amounts of brownish glass. Trachybasalts, typical of alkaline volcanics, contain large phenocrysts of sanidine and analcite. Volcanic strata comprise massive flows and pillow lavas, thick beds of monolithologic andesitic breccia, reworked lithic and hyaloclastic material, poorly bedded felsic breccias, tuff and conglomerate. Intercalated greywacke, argillite, fissile siltstone and black shale are also common. Cognate plutonic fragments are common in breccia beds of the volcanic assemblage and are most abundant near alkalic intrusions of the same composition.

PETROCHEMISTRY

The volcanic rocks have a wide range of chemical composition. Chemical data given below and in the following section are from Preto (1972, 1974), Schau (1968), Morton (per. comm., 1975), Procyshyn (per. comm., 1975), Bailey (per. comm., 1975), Allen and Panteleyev (this volume), Hodgson (this volume) and Findlay (1969). Figure 2 is a plot of total alkalis and silica content. The boundary separating alkaline from subalkaline rocks is from MacDonald (1968). Most analyses are of volcanic rocks from the southern part of the Intermontane Belt; only seven analyses are available from the northern part, and these are from the Stikine River area. Samples plot in both the alkaline and subalkaline fields, but most lie within the former. Samples plotting in the alkaline field include both sodic and potassic types. Sodic volcanic rocks, which are much rarer than potassic types, range from hawaiite to mugearite and trachyte. Most rocks are potassium-rich trachybasalt and tristanite (terminology from Irvine and Baragar, 1971). These potassic and sodic volcanic rocks form the widespread flows and thick unstratified breccias that comprise the bulk of the volcanic successions of the Quesnel and Whitehorse troughs.

The subalkaline volcanic rocks (Fig. 1) are typical of the calc-alkaline magma series. They are aluminous types relatively rich in potassium and lack iron enrichment typical of rocks of the tholeiitic suite. Rock types vary from basalt to rhyolite, tholeiitic andesite and dacite being the most common. Rhyolite and many of the dacites are relatively rich in sodium and are better classified as sodic rhyolite and dacite.

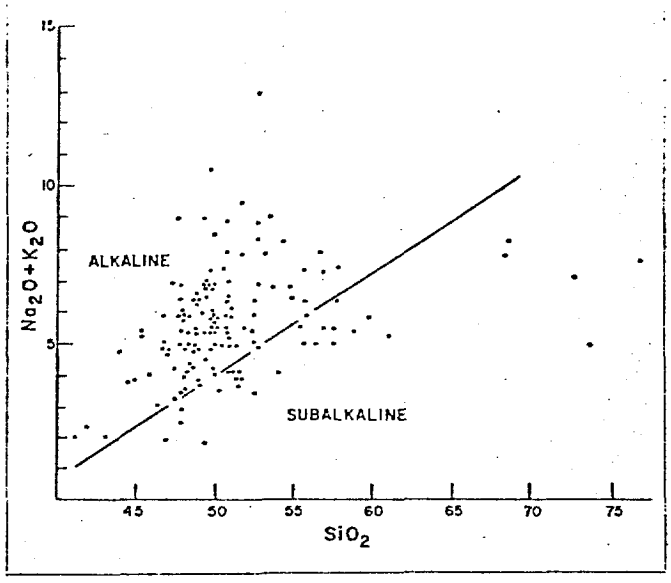


FIGURE 2 — Alkalis-silica plot for volcanic rocks of the Quesnel Trough (wt. %).

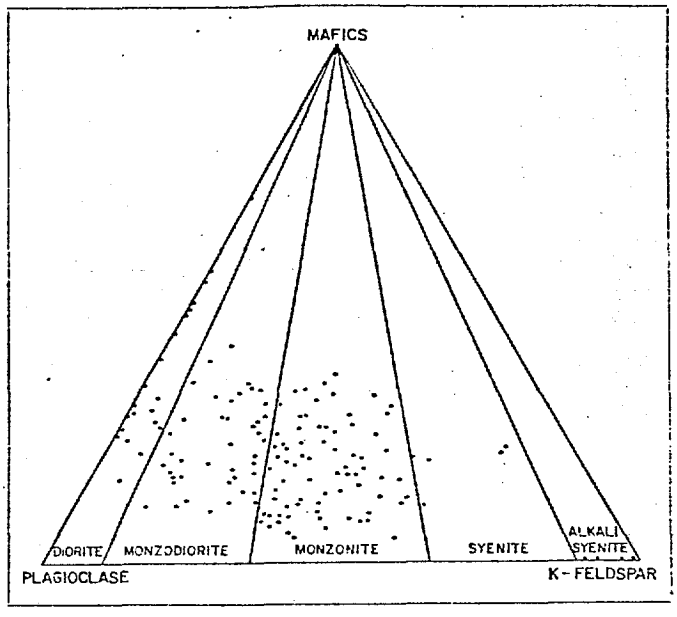


FIGURE 3 — K-feldspar - total mafics - plagioclase contents of the alkaline plutonic rocks.

Plutonic Rocks

DISTRIBUTION AND LITHOLOGY

Plutonic rocks of the alkaline porphyry suite comprise a group of intrusions ranging from syenogabbro to alkali syenite. Most are small plugs and stocks rarely more than a few kilometers in diameter, although a few attain dimensions of small batholiths. Host rocks are usually marine volcanic breccias comprising thick accumulations of unstratified, monolithologic felsic and basic autobreccias and slump deposits. Figure 3 is a plot of modal plagioclase, alkali feldspar and total mafics of various rocks from the Copper Mountain area (after Montgomery, 1967; Preto, 1972; Rice, 1960). Medium-grained equigranular monzodiorite and porphyritic monzonite comprise the bulk of the plutonic rocks. Syenite, alkali syenite and nepheline syenite are relatively rare and are confined to small late-stage intrusions and pegmatitic dykes.

However, stocks of porphyritic syenite up to 3 kilometers in diameter form the bulk of intrusive rocks at the Galore Creek deposit. The most common mafic minerals in the alkaline plutons are pale green augite and biotite. Plagioclase and alkali feldspar form about 60 per cent of the rock. Accessory amounts of apatite and magnetite are also characteristic of this group.

PETROCHEMISTRY

Figure 4 is an alkalis-silica plot for various plutons. Rocks of the alkaline suite show marked alkali enrichment relative to silica — the most alkali-rich samples contain considerable nepheline and alkali feldspar. The plutonic suite has the same alkalis-silica ratios as the associated alkalic volcanic host rocks and is clearly distinct from the extrusive calc-alkaline rocks. Most of the plutons, like the enclosing volcanic rocks, belong to the potassic suite — none have strongly sodic compositions. Syenitic porphyries from the Stikine area are notably more potassic than most of the other plutons.

Plutons of the alkalic suite have chemical compositions identical to the enclosing volcanic strata; they are of the same Upper Triassic - Lower Jurassic age and are confined to the alkali volcanic assemblage. Cognate plutonic fragments identical to nearby alkaline intrusions are abundant in some of the unstratified breccias within the enclosing volcanic sequence. All evidence points to the conclusion that the alkaline plutons were emplaced in volcanic centers from which most of the nearby volcanic material was derived.

Mineral Deposits

GENERAL CHARACTERISTICS

Alkaline-suite porphyry copper deposits have characteristics of exploration significance that distinguish them from their counterparts in the calc-alkaline class. The deposits are localized in fault, fracture or rift zones cutting volcanic rocks of similar age and in zones of brecciation within subvolcanic plutons. All plutons dated are in the 175-201 my range. In contrast to the deposits of the calc-alkaline suite, molybdenite is rare, but gold and silver are usually present in sufficient quantity to constitute a significant smelter credit. Related pyrometasomatic deposits are more abundant than in the calc-alkaline class. Alteration products directly related to hypogene sulphides are potash feldspar and secondary biotite.

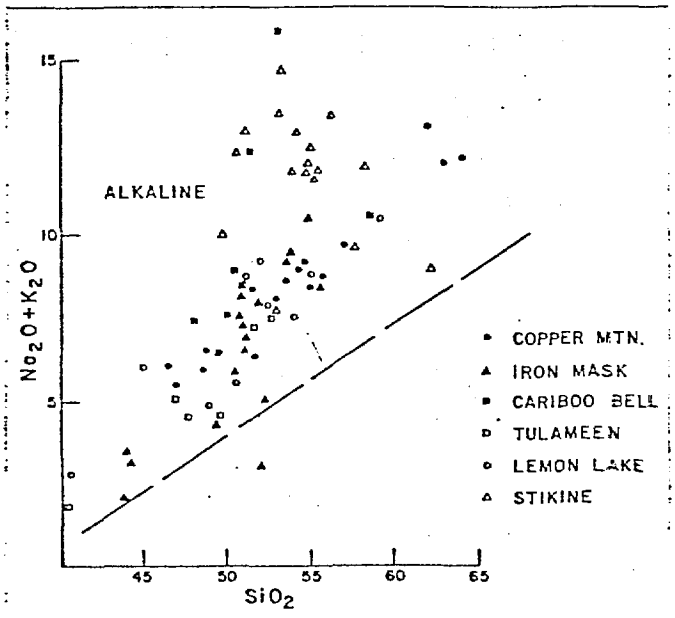


FIGURE 4 — Alkalis-silica diagram for plutonic rocks (wt. %).

ALTERATION

Alteration products and sulphides are spatially coextensive and contemporaneous, or nearly so. Classic zoning patterns evident in calc-alkaline porphyries do not normally apply, because phyllic and argillic zones are notably absent or poorly developed. Alteration zones are usually arranged peripheral to the alkaline pluton(s) and not to associated orebodies. Ore zones occur in all alteration units, although most, as at Galore Creek, are associated with a potassic zone close to the associated alkaline pluton. Alteration zoning and sulphide distribution are complicated by cross-cutting dykes, intrusive breccias and small stocks; telescoping and overlapping of various alteration zones are common. Primary rock compositions have a marked control on the alteration assemblages produced.

Potassic Alteration

Potassic alteration, consisting of biotite and potash feldspar, is directly associated with hypogene copper sulphides in all important deposits. Where bornite is present, it is associated with chalcopyrite in this zone. At Galore Creek, biotite forms fine-grained aggregates in the groundmass of syenite porphyries and is locally present in amounts of up to 50 per cent by volume (Barr, 1966). Biotite also occurs with potassic feldspar in breccias, veins and on fractures in many of the deposits. Potash feldspar generally occurs as a pink alteration product on fractures, in veinlets and fracture envelopes, and as widespread disseminated aggregates. Much of the potash feldspar forms by the replacement of sodic plagioclase.

Potassic alteration at Ingerbelle, which is poorly developed, is not centered on the orebodies, but is controlled by proximity to the nearby Lost Horse intrusions. Although potassic alteration is prominent at Lorraine, no zoning sequence has been recognized to date (James, per. comm., 1975).

Propylitic Alteration

A propylitic zone, consisting of chlorite, epidote and albite, is invariably present peripheral to a more centralized potassic zone. The degree of alteration varies from fracture-plane envelopes, coatings on fractures and weak pervasive alteration of the host rock to bodies of rock that have been intensely recrystallized and metasomatized. Compared to propylitic zones associated with calc-alkaline porphyry deposits, those in the alkaline suite have a much greater degree of hydrothermal alteration. Orebodies, such as the Ingerbelle deposit, occur within the propylitic zone.

The common silicate assemblage is albite + epidote + chlorite with associated garnet, scapolite (at Inberbelle), zeolites and carbonate. Where well developed, minerals of this assemblage form a mottled pink and green rock characteristic of this alteration type. Cross-cutting veinlets of scapolite (at Ingerbelle), carbonate, various zeolites and alkali feldspar are common features. Associated sulphides are pyrite and chalcopyrite,

E 1 Summary of Geology and Exploration Data Significant Copper Deposits Associated with Alkalic Plutons in nadian Cordillera

.hip	Afton (Teck. Corp.)	Cariboo Bell (Teck Corp.)	Copper Mountain/Ingerbelle (Similkameen Mining)		Galore Creek (Stikine Copper)	Gnat Lake (H.B.M. & S.)	Lorraine (Granby-Kennco)	
(	Diorite, diorite phy, syenite phy.	Syenodiorite, monz. phy.	Syenite-monzonite-diorite complex		Syenite phy. complex	Syenite porphyry, quartz monzonite	Syenite migmatite, leucocratic syenite, pyroxenite	
	198 ± 6 m y	184 ± 7 m y	U. Triassic (193 ± 8 m y)		L. Jurassic (198 ± 7 m y)	L. Jurassic	175 ± 5 m y	
Rock	U. Triassic tuffs, aggloms., flows	L. Jurassic tuffs, breccias, flows	U. Triassic volcs., minor seds.		U. Triassic metavolcs, minor seds.	U. Triassic andesites, dacites, rhyolites	Mesocratic phase of migmatite	
ation zoning ence from er	Potassic-propylitic	Potassic-propylitic	Wk. potassic-propylitic		Potassic-propylitic	None recognized	None recognized	
URE	Stockwork, breccia	Breccia	Dissem., stockwork		Disseminated stockwork, breccia	Disseminated stockwork	Disseminated	
ZONE meters) ne %	850 × 300 0 — 10	4500 × 600 5 (avg)	Undetermined 0 — 1	1200 × 850 2 — 10	1500 × 600 2 — 10	Not defined 1 — 3	None Less than 1	
ERALIZATION %							Upper Zone	Lower Zone
opyri	9	74	50	100	90	100	30	50
te	10	—	50	—	9	—	30	40
ocite	22	26	—	—	1	—	20	10
e Copper	56					—	—	—
Secondary	3					—	20	—
TITE %	Not available	3 (avg)	0 — 1	0 — 1	2 — 10	0 — 20	0 — 5	
ICANT ATION								
dspar	x	x	x		x	x	x	
e	x	x	x	x	x		x	
t		x			x			
drite					x			
ite	x	x		x	x	x		
nate	x	x		x	x	x		
e	x	x		x		x		
otite				x				
te	x	x		x	x			
naline						x		
ES & PAST JCTION							Upper Zone	Lower Zone
(× 10⁶)	31 (indicated)	25 (indicated)	32 (past production)	65 (pre-production)	125 (indicated & inferred)	25 (drill inferred)	5 (3)	5 (3)
	1.0	0.49	1.08	0.53	1.06	0.44	0.75	0.60
/t	0.10	0.04	0.12	0.02(2)	0.25	Undetermined	Undetermined	
/t	0.015	0.025	0.005	0.005(2)	0.013	Undetermined	0.01	0.003
ABLE VERY METHODS of Discovery ods(1)	1970 Prosp. (IP, Gc)	1964 Magnetics (Prosp., Gc)	1884 Prosp.	1966 Prosp. (IP, Rock Gc)	1955 Prosp. (Gc, IP, Mag)	1960 Prosp. (IP. Gc, Mag)	1945 Prosp.	1970 Prosp. (Gc)

l discovery methods with applicable or helpful methods in brackets.
ered 1972-73 production.
ated potential.

ations:
= Metric tons
= Induced Polarization
= Geochemical

Prosp. = Prospecting
Mag = Magnetic

Wk = Weak
phy = porphyry
monz = monzonite

often in massive lenses and pods. Both hematite and magnetite are usually present. Exsolved blades of hematite in albite produce the pinkish colour typical of this alteration type.

Other Alteration

Individual deposits have important alteration products that may be lacking in others, as indicated in Table 1. Alkalic plutons in the Stikine River belt contain significant amounts of garnet, which also occurs in adjacent hornfels derived from altered Triassic volcanic rocks. At Galore Creek, garnet varies from andradite to grossularite, occurs as granular replacements of feldspar phenocrysts in syenite porphyry, and is frequently associated with chalcopyrite-bearing gypsum and anhydrite veins. The Ingerbelle deposit contains moderate to abundant scapolite, which has not been recognized at other deposits. The Gnat Lake deposit contains an average of about 5 per cent tourmaline as vein-fillings and massive replacements.

HYPOGENE SULPHIDES

Hypogene sulphides, which have formed contemporaneously with products of hydrothermal alteration, include the following in order of decreasing abundance: pyrite, chalcopyrite, bornite, chalcocite and pyrrhotite. Sulphide content ranges from 2 to 15 per cent, by volume, and averages about 5 per cent. Pyrite is the most abundant sulphide, although where abundant it occurs peripheral to the main mineralized zone, as at Galore Creek and Cariboo Bell. Sulphides occur as fracture fillings, disseminated grains, massive lenses and pods, and in breccias. Sulphide and magnetite zones may coincide, but this is not always the case. In the central zone at Galore Creek, magnetite occurs on three sides of the deposit close to the syenitic porphyries, and is associated with rocks having high bornite-to-chalcopyrite ratios. Hypogene sulphide zoning across the deposits ranges from low pyrite:chalcopyrite on the west near the contact to high pyrite: chalcopyrite farther east. A similar relationship exists at Cariboo Bell, but magnetite appears to have formed early, followed by chalcopyrite and pyrite. At Ingerbelle, chalcopyrite is predominant in the north part of the deposit and pyrite is more abundant to the south, east and west, farthest from the nearby Lost Horse intrusions.

SUPERGENE DEPOSITS

Significant supergene deposits are rare in the Canadian Cordillera, although they occur at several deposits both of the calc-alkaline and alkaline classes. Although oxidized, only a few deposits are actually enriched relative to the protore. The most notable is the Afton deposit, in which native copper, chalcocite and copper oxides form a significant part of the ore. Supergene minerals occur on fractures and in fault zones to a depth of 450 meters. At Cariboo Bell, copper oxides account for about 25 per cent of the total copper content. At the Lorraine prospect, copper carbonates account for about 20 per cent of the upper deposit, which is well exposed; the lower deposit, which is mostly concealed, contains less than 1 per cent copper oxides. Supergene copper minerals at other deposits are relatively insignificant, being less than 3 per cent of the indicated reserves.

SIZE OF DEPOSITS

The configuration of various deposits and related induced polarization anomalies are shown in plan form in Figure 5, and the outline of deposits and results of induced polarization, magnetic and geochemical surveys are shown in section in Figure 6. Because of the irregular shape of the deposits, a direct comparison of reserves will not be obvious from the plan view. The deposit with the greatest copper reserves is the central zone at Galore Creek, which is 1800 meters long, 500 meters wide and extends to a depth of at least 300 meters. A comparison of reserve data shown on Table 1 indicates that most of the orebodies are in the 25 to 35 million tonne class and tenor ranges from 0.40 to 1.1 per cent copper. Many deposits, such as Ingerbelle, consist of several ore zones. The Galore Creek deposit, which consists of 125 million tonnes of 1.1 per cent copper, is anomalously large. At the Afton deposit, persistence of good grade material to a depth of 600 meters suggests excellent possibilities for significantly increasing the published reserves.

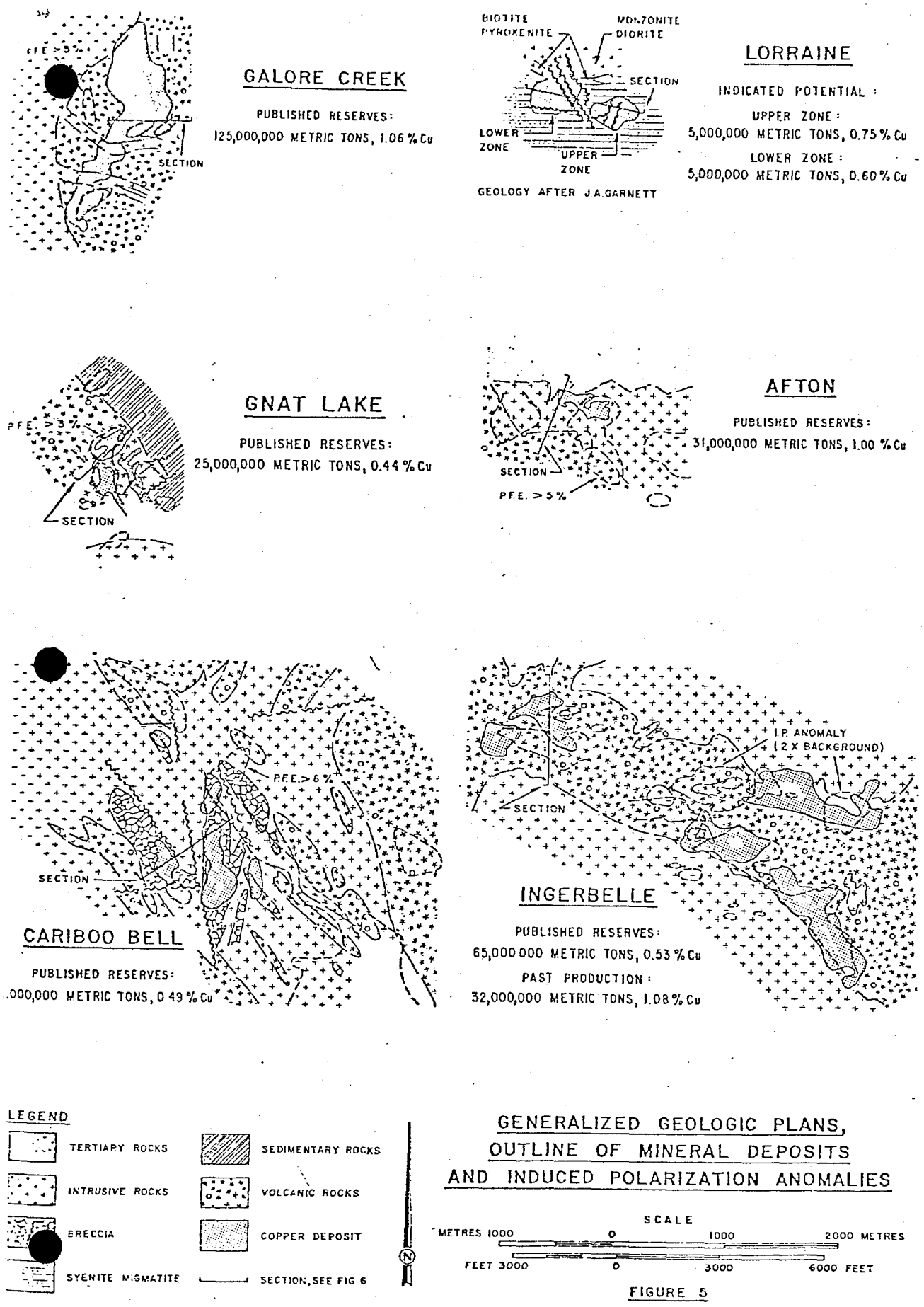
Exploration Techniques

The date of discovery of each of the mines or important prospects is shown in Table 1. Also noted are principal exploration techniques and other applicable exploration methods used.

All of the deposits possess surface exposures. Cariboo Bell was discovered during investigation of a magnetic anomaly, and Galore Creek has one small vividly malachite-stained outcrop of high-grade material — the significance of the deposit was not established until completion of reconnaissance diamond drilling. Numerous prospects similar to Afton, but of lower grade, were known in the Iron Mask pluton within 2 kilometers of the deposit. Discovery resulted from persistent drilling by various exploration groups over a number of years.

Historically, prospecting has been the principal exploration method leading to the discovery of mineable or potentially mineable deposits. Trenching during early exploration stages played an important role, and no production decision would have been possible without extensive diamond drilling. Geophysical and geochemical methods have also been important in defining target areas. In many cases, induced polarization anomalies coincide with underlying copper sulphides, but most reflect pyritic zones marginal to the deposit, as at Cariboo Bell, Afton and Galore Creek (Fig. 6).

Future exploration for alkaline suite porphyry deposits will become increasingly difficult because of the extensive glacial deposits and other overburden materials that are present in potentially productive regions. Geological mapping of the volcanic succession is essential in predicting the location of mineralized alkalic plutons. Case histories, such as Copper Mountain and Afton, have shown that geological mapping will help to outline favourable zones. The common occurrence of magnetite with alkalic intrusions suggests that magnetometer surveys will aid in defining target areas. The linear distribution of alkaline intrusions, regional faults, fault intersections and zones of brecciation are important structural controls that will aid exploration for these deposits. Geochemical surveys may also provide targets in areas of requisite physiography. Although induced polarization surveys will probably be important in providing local drill targets, a considerable amount of drilling to test geological targets is envisaged.



URE 5 — Generalized geologic plans, outline of mineral deposits and induced polarization anomalies.

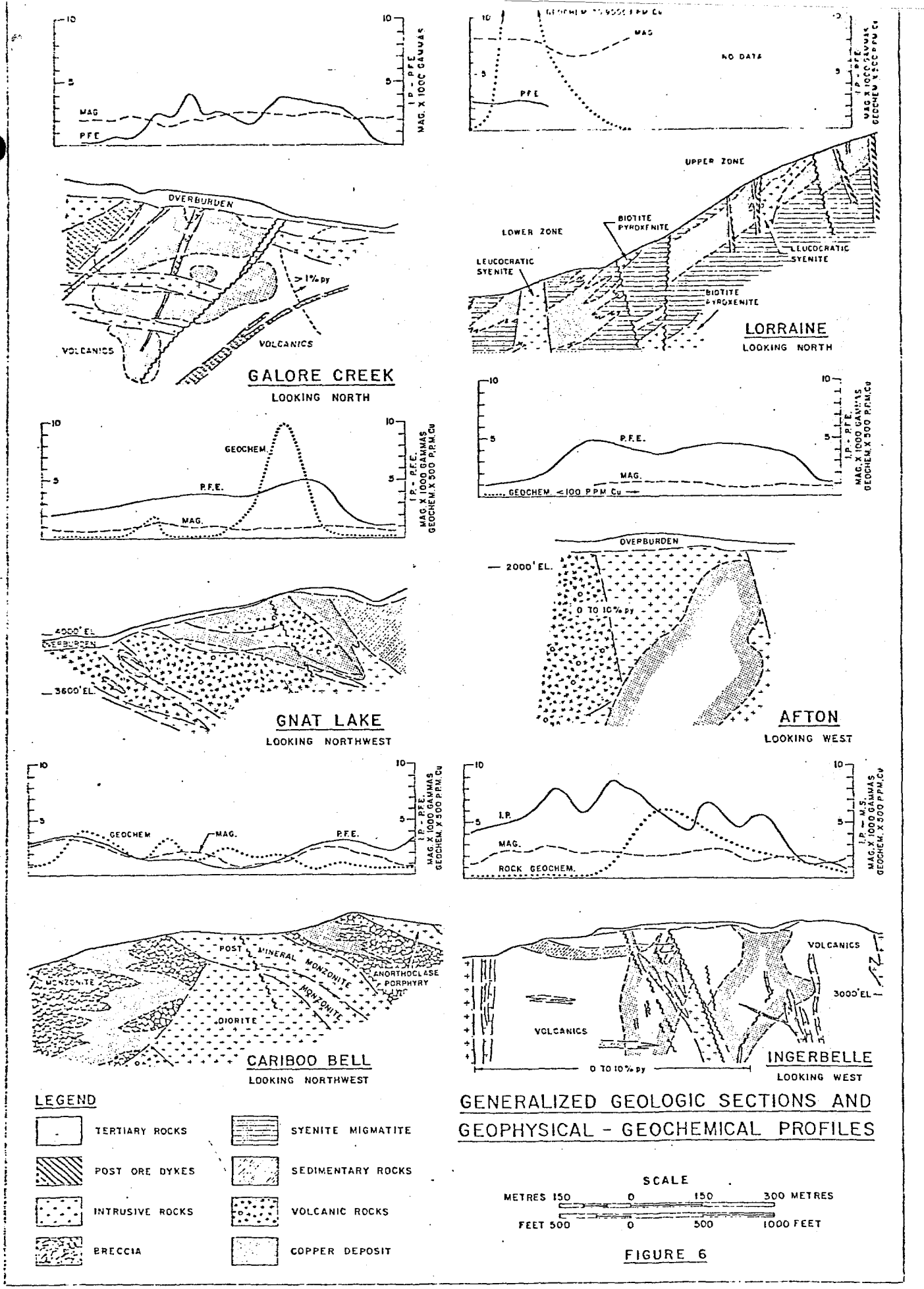


FIGURE 6 — Generalized geologic sections and geophysical-geochemical profiles.

Acknowledgments

The authors extend their thanks to the many individuals and organizations who provided generous assistance and time in the preparation of this paper. Stikine Copper Ltd. permitted publication of previously published gold and silver contents of the Galore Creek deposit and provided other data that have been incorporated in this paper. R. A. Freberg of Hudson Bay Exploration and Development Company Limited provided information on the Gnat Lake deposit. J. M. Carr and A. J. Reed of Teck Corporation Ltd. provided considerable information on the Afton and Cariboo Bell deposits. D. H. James of Granby Mining Company provided data on the indicated potential of the upper and lower zones of the Lorraine prospect. J. A. Garnett of the B.C. Department of Mines and Petroleum Resources also contributed data on the Lorraine deposit from his co-authored paper in this volume. T. N. Macauley of Newmont Mining Corporation provided information from his co-authored paper in this volume. R. L. Morton, E. Procyshyn and D. G. Bailey provided unpublished chemical analyses — their contribution is gratefully acknowledged. Thanks are also extended to K. L. Jones of Du Pont of Canada Exploration Ltd. for drafting services and to Miss A. T. Puusaari of the same company for typing the manuscript.

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File

ASARCO

Northwestern Exploration Division
John C. Balla
Manager

January 31, 1983

Mr. W.L. Kurtz, Manager
Western U.S.A.
ASARCO Incorporated
P.O. Box 5747
Tucson, AZ 85703

Sampling of Gold Deposits

Dear Mr. Kurtz:

Attached is a copy of a memorandum prepared by Bob Garwood of Placer Amex on sampling of low grade gold deposits. The paper was prepared as an internal report. The report is being distributed as it may be of passing interest to our other offices.

Very truly yours,

John C Balla

John C. Balla

JCB/dt
Enclosure

cc: R.L. Brown with enclosure
R.E. Gale " "
J.D. Sell " "
D.M. Smith " "
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QUALITY CONTROL FOR SAMPLING LOW GRADE GOLD DEPOSITS

R. G. Garwood
Regional Geologist

December 29, 1981

Introduction

Numerous papers have been written during the past 25 years concerning sampling and quality control for porphyry Cu-Mo deposits. Many of these papers stressed the importance of quality control and proper sample processing techniques. I believe the state of the art of sampling and sample processing of low grade gold deposits has progressed to about the state of the art of porphyry Cu deposits of about 25 years ago. Several papers have recently been prepared concerning sampling of lower grade gold deposits, and many of these papers attack the problem through a statistical approach. Instead of a statistical approach, I would like to present a practical sample preparation method which has given acceptable results, particularly at the Golden Sunlight property.

It is my experience that numerous geologists or samplers use considerable care to sample a gold mineralized exposure. They then send the sample to an assayer without instruction concerning sample preparation or handling of the sample. Generally, the only instruction given to the assayer is the metals to be determined.

Proper sample preparation is as important as proper sampling. When a drilling program is undertaken, sample preparation may be the most important phase of properly evaluating a low grade, large tonnage gold deposit.

Sample Preparation

Figure 1 is a list of Au, Ag and Mo assays from DDH No. 1 on the Haile gold property, South Carolina. The core was split with a Diamond

FIGURE 1

HAILE

Hole 1

Sample No.	Interval	Footage	Au oz/ton CMS	Au oz/ton Skyline	Au oz/ton Union	Ag oz/ton CMS	Ag oz/ton Skyline	Mo ppm CMS	Mo ppm Skyline
74-03-2546	10 - 20	10	.144	.097	0.070	.04	<.01	75	40
2547	20 - 30	10	.033	.014	0.020	.05	<.01	35	18
2548	30 - 40	10	.156	.11	0.070	.42	.18	45	26
2549	40 - 50	10	.138	.079	0.090	.60	.50	80	55
2550	50 - 60	10	.099	.056	0.070	.10	<.01	70	55
2551	60 - 70	10	.156	.14	0.100	.30	.22	60	36
2552	70 - 80	10	.051	.048	0.030	.13	<.01	65	50
2553	80 - 90	10	.020	.017	0.020	.05	<.01	60	40
2566	90 - 100	10	.051	.048		.15	<.01	75	70
2567	100 - 110	10	.021	.026		.13	<.01	45	34
2568	110 - 120	10	.019	.021		.19	<.01	85	80
2569	120 - 130	10	.023	.013		.10	<.01	60	44
2570	130 - 140	10	.014	.020		.04	<.01	35	26
2571	140 - 150	10	.014	.015		.05	<.01	50	38
2572	150 - 160	10	.020	.016		.08	<.01	70	55
2573	160 - 170	10	.008	.016		.02	<.01	18	14
2574	170 - 180	10	.002	.007		.02	<.01	10	6
2575	180 - 190	10	<.001	<.005		.02	<.01	5	4
2576	190 - 200	10	<.001	.014		.01	.06	3	2
2577	200 - 210	10	.001	<.005		.01	<.01	3	< 2
2578	210 - 220	10	.021	.018		.02	<.01	3	2
2579	220 - 230	10	<.001	<.005		.02	<.01	3	2
2580	230 - 240	10	<.001	<.005		.03	<.01	3	2
2581	240 - 250	10	<.001	<.005		.04	<.01	4	2
2582	250 - 260	10	<.001	<.005		.02	<.01	2	2
2583	260 - 270	10	<.001	<.005		.03	<.01	3	2
2584	270 - 280	10	<.001	<.005		.02	<.01	4	4
2585	280 - 290	10	<.001	<.005		.02	<.01	2	2
2586	290 - 300	10	.002	<.005		.02	<.01	2	2
2587	300 - 310	10	.003	<.005		.03	<.01	2	2
2588	310 - 320	10	<.001	<.005		.02	<.01	2	2
2589	320 - 330	10	.002	<.005		.01	<.01	3	2
2590	330 - 340	10	.001	<.005		.02	<.01	3	2
2591	340 - 350	10	<.001	<.005		.02	<.01	5	4
2592	350 - 360	10	.001	<.005		.01	<.01	1	2
2593	360 - 370	10	.002	<.005		.01	<.01	3	4
2594	370 - 380	10	.002	<.005		.02	<.01	2	2
2595	380 - 390	10	.003	<.005		.02	<.01	3	2
2596	390 - 400	10	.001	<.005		.02	<.01	5	4
2597	400 - 410	10	<.001	<.005		.01	<.01	3	2
2598	410 - 420	10	<.001	<.005		.01	<.01	4	< 2

saw at the company's central sample preparation facilities. One-half of the core was crushed through a lab size jaw crusher and a split fraction was pulverized through a Braun pulverizer. The pulp was split through a riffle-type pulp splitter into four separate pulps. One pulp was retained and the remaining pulps were sent to three assayers. It is not known if each assayer dried and rolled the pulp prior to weighing the sample for assay, or if the assayer weighed the sample by taking it directly from the bag. At first glance at Figure 1, one would surmise the sample has a large nugget effect caused by free gold. One could question the reliability of the assayer. Generally, one would not wonder if the sample was properly prepared. Prior experience had demonstrated the sample prep lab was excellent for Cu, Mo, Pb, and Zn.

Figure 2 is a list of Au and Mo assays from DDH 17 on the Haile gold property. By the time Hole 17 was drilled, a sample preparation lab had been established near the property, and the pulverized sample was rolled on a rubber sheet at least 60 times prior to splitting the pulp. A study of Figure 2 shows better correlation between Au assayers.

From Figure 1, Mo values show fair to good correlation. Au and Ag values are highly erratic. If the sample had been sent only to CMS, the averaged Au value would be 90 ft. of 0.0942 oz Au and .20 oz Ag from 10 to 100 ft. If only Union Assay Office had been used, the averaged value would be 80 ft. of 0.0588 oz Au from 10 to 90 ft. CMS results are definitely encouraging while Union results are only moderately encouraging. Which assayer reported the correct results? In this case (assuming all performed correct assay procedures) none of the assayers reported the true value of gold intersected by the drill core because they did not have a representative sample.

In the case of Hole No. 1, the pulp was ground too coarse, and it was not properly mixed prior to splitting into fractions for assay. One is not sure the crushed core was crushed properly or that proper splits of the crushed core were made.

A tour of sample preparation facilities in a commercial assay lab would show an orderly arrangement of sample crushing, splitting, drying and pulverizing. The arrangement is designed to rapidly process samples because commercial labs depend upon volume work to enhance profits. Special care is not given to any sample unless specifically agreed upon and for which an extra charge is assessed. Most labs established their sample prep room and assay procedure when the price of gold was $35/oz and 0.01 oz Au represented $.35 (an insignificant value). At an Au price of $500/oz, .01 oz Au is worth $5 (a significant value). Commercial labs seldom reassay a second split of the pulp and rarely pulverize and assay a second split of the crushed sample.

Because of a variety of samples run in a commercial lab from all oxide to total sulfide and from high silica to low silica, the fire assayer develops a skill at examining the pulp to determine a proper flux to add to the sample to obtain a proper slag for fire assay. Only the better and most dedicated commercial labs will rerun a gold assay which shows improper fluxing.

FIGURE 2

BAILE

Hole 17

Sample No.	Interval	Footage	Au oz/ton CMS	Au oz/ton Cyprus	Au oz/ton Union	Mo/ppm CMS	Mo/ppm Cyprus	Mo/ppm Union
76-03-3291	10- 20	10	.073	.060	.080	75	60	
3292	20- 30	10	.080	.066	.080	35	30	
3293	30- 40	10	.026	.022	.030	17	10	
3294	40- 50	10	.042	.036	.040	40	30	
3295	50- 60	10	.050	.050	.070	85	60	
3296	60- 70	10	.047	.050	.060	60	50	
3297	70- 80	10	.085	.106	.100	110	90	
3298	80- 90	10	.026	.030	.040	85	80	
3299	90-100	10	.018	.018	.040	100	100	
3300	133-140	7	.003	.01		5	10	
3301	140-150	10	.001	.01		4	10	
3302	150-160	10	<.001	.01		3	10	
3303	160-170	10	.002	.01		4	10	
3304	170-180	10	.001	.01		1	10	
3305	180-190	10	<.001	.01		5	10	
3306	190-200	10	.002	.01		2	10	
3307	200-205	5	<.001	.01		2	10	

Although assay procedures do, from time to time, give bad results, and sample mix-ups do occur, my experience has been that poor sample preparation is the greatest cause of assay discrepancies from the same sample. Figure 3 is a flow sheet of the sample preparation procedure used for the Golden Sunlight project. This procedure has been demonstrated to be reliable with generally acceptable check assays.

Comments concerning this procedure are:

Crushing

The 4" x 6" jaw crusher is used to rapidly crush large rock fragments or large pieces of core which cannot be handled by the smaller chipmunk crusher. The chipmunk crusher is maintained to produce a minus one-fourth inch product.

Splitting

Discharge from the chipmunk crusher is mixed and split through a hopper splitter. The hopper splitter gives a more uniform split of the sample than does a standard Jones splitter. Because most of our samples weigh 15 to 20 pounds, three-fourths of the sample may be rejected, bagged, and stored. This reject is a good sample for metallurgical studies. If the sample is smaller, all the sample should be placed in the cone crusher.

Crushing

The cone crusher is designed to crush all the sample to a uniform size, so that one dimension of a rock fragment is not larger than another.

Splitting

Thorough mixing of the crushed sample from the cone crusher prior to splitting through the hopper splitter is essential.

In practice, the first split is dried, pulverized, and assayed. The second split is stored. If the assay of the first split is 0.02 oz Au or higher, the assayer instructs the lab to dry and pulverize the second split. Note the second split is a check assay of the minus 6 mesh material. If the difference between assays is 0.015 oz Au or higher, generally both pulps are reassayed to check for assay errors. Both assays are reported on the drill log and their results averaged for the assay of the sample interval. This practice also guards against sample mix-ups in the assay office.

Pulverizing

The T.M. Vibratory Ring Pulverizer gives a more uniform and consistent grind over the Braun pulverizer and most important, materially reduces loss of the very fine material (dust) as compared to the

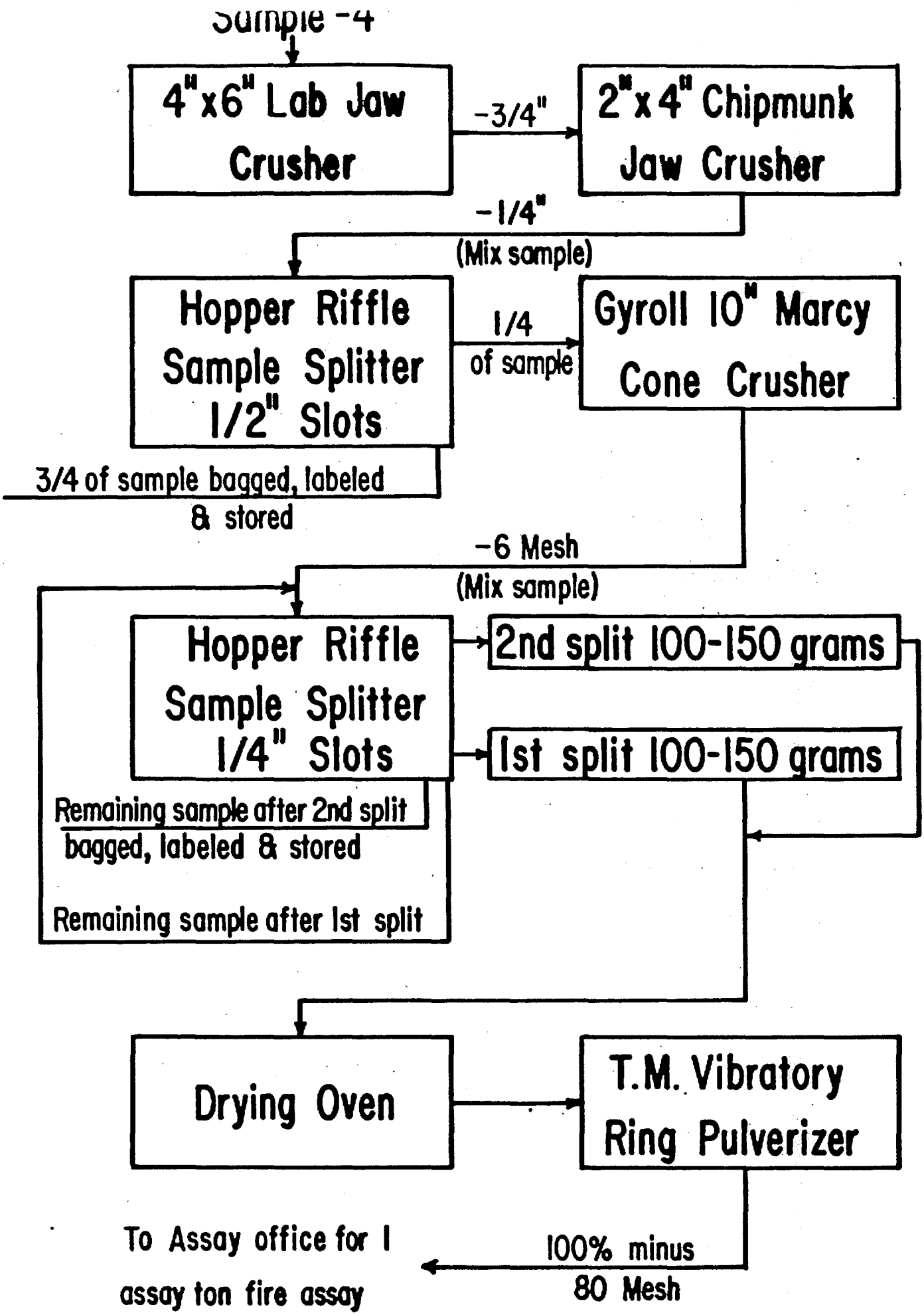

Sample -4
4"x6" Lab Jaw Crusher
-3/4"
2"x4" Chipmunk Jaw Crusher
-1/4"
(Mix sample)
Hopper Riffle Sample Splitter 1/2" Slots
1/4 of sample
Gyroll 10" Marcy Cone Crusher
3/4 of sample bagged, labeled & stored
-6 Mesh
(Mix sample)
Hopper Riffle Sample Splitter 1/4" Slots
2nd split 100-150 grams
1st split 100-150 grams
Remaining sample after 2nd split bagged, labeled & stored
Remaining sample after 1st split
Drying Oven
T.M. Vibratory Ring Pulverizer
To Assay office for 1 assay ton fire assay
100% minus 80 Mesh

Figure 3

Braun pulverizer. As a general rule, fines from Au deposits of the Cortez type or oxide Au deposits do not assay higher than coarse material. Sulfide deposits such as the Golden Sunlight generally are opposite. Screen tests show higher gold values in the fines.

It is a common practice to screen the pulp (discharge from the pulverizer) to insure a proper grind has been achieved and to check any oversize material for free gold. If free gold is believed to occur in any sample, instructions should be given to screen the pulp and check the oversize for free gold. In normal sample preparation procedures, far too often the oversize, if a minor fraction of the sample, is discarded.

Assaying

The standard gold assay is a fire assay. AA may be used and is acceptable. Other wet methods have been used but their assay results generally cannot be checked by fire assay. Gold assays by methods other than fire assay or AA should be suspect.

A majority of commercial assay labs will use a one-half assay ton charge (14.583 grams of sample) to fire assay a sample because it is cheaper for them to run the assay. Cortez, Golden Sunlight and other non-profit assay labs always use a one assay ton charge (29.166 grams). I strongly recommend a one assay ton charge because of the larger sample and greater accuracy. With equipment available today and a one assay ton charge, a fire assay may be made to an accuracy of 0.0025 oz Au/ton (.07775 PPM).

Sample weight for AA gold determinations range from 3 to 5 grams. During 1975-76, assays of over one thousand sample pulps were run by the Sunlight lab with fire assay and by Placer Development Ltd. geochem lab using AA and a 10 gram sample weight. An evaluation of the results of this work by standard computer comparison concludes the gold results from AA are 20% lower than the fire assay results. Perhaps the geochem lab did not get all the gold in solution, but a second factor would be that only one-third of the sample weight was used for AA. The difference between assays increased with increased gold in the sample.

Silver is always added to a fire assay to insure a proper silver to gold weight to part the gold from the dore button. Generally, one or more Hermann inquarts may be added to the sample. Experience has shown the silver in the Hermann inquart may be the equivalent of 0.1 oz Ag in error. If it is important to use silver values as an indicator for metal distribution, assayers should be instructed to use liquid inquarts with a known silver value instead of Hermann inquarts.

As we all recognize with an interesting gold prospect, gold which may be recovered by standard milling practices is more important to an evaluation than gold in the rock. A rapid method to approximate gold or

gold/silver recoveries in a normal cyanide mill is to assay the sample by AA using a cold cyanide digestion. Practice at Cortez is to use a cold cyanide solution equivalent to 5 pounds cyanide per ton of solution. The Au results by AA cyanide digestion divided by fire assay of the same pulp should indicate Au recovery. If further work is indicated and bottle roll tests are run, the metallurgist should be advised of the AA cyanide digestion results.

Sampling

Sampling of low grade gold deposits, particularly in the examination or early exploration stage, is a more difficult and frustrating experience even for a geologist experienced with low grade gold deposits. For the inexperienced geologist, it is possible the mineralized outcrops will not be sampled because the geologist did not see sufficient evidence of mineralization to warrant taking time to sample the outcrop. During 1981, the northwest regional staff of Placer Amex took over 400 samples, each containing at least 10 ft. of rock exposure, upon one property; upon receipt of the assay evaluation results, we found the gold deposit was only two-thirds sampled. On another deposit, we have check sampled and confirmed a .5 oz Au assay of the least mineralized and "fresh" andesite I have seen in years. If geologic evidence indicates the property contains gold mineralization, time should be utilized to map and sample every rock outcrop.

Surface Leaching

Surface leaching of gold deposits is a long standing controversy. Surface leaching within oxide outcrops of low grade gold deposits containing 20 micron size gold, or less, in fractures, or disseminated micron gold in the rock may occur. Leaching may be chemical, or physical. I have seen too many low grade gold deposits with depleted Au values in the top 5 to 20 ft. of the deposit for me not to believe in surface depletion of gold. EVERY GEOLOGIST EXAMINING A POSSIBLE LOW GRADE GOLD DEPOSIT SHOULD BE AWARE OF THE POSSIBILITY GOLD VALUES HAVE BEEN LEACHED FROM SURFACE EXPOSURES. If you believe leaching has occurred and sample the deposit with negative Au values, I am not prepared to advise you how to get exploration funds to test the deposit to a depth of 50 feet or less. Fracture sampling may be the only indication of Au values at shallow depths.

Initial Sampling

Prior to considering an initial sampling program of a low grade gold deposit, a geologist should spend the necessary time to conduct a preliminary geologic study of the area. Map structure and structural trends; note areas of silicification (not quartz stringers); note possible potash enrichment; note changes in iron oxide color in fractures; and, if possible, determine if mineralization may be relatively flat

lying or near vertical. After the initial geologic evaluation, a sample program should be programmed to sample across structures and along strike of structures wherever possible.

Usually outcrops are not available to conduct uniform bulk sampling across structures. A good sample technique is to sample fractures at intervals along their length for a measured distance normal to the strike of the fracture. Figure 4 is a generalized diagram to explain the method. Each fracture would be sampled and its width recorded. The sample from each fracture is combined into one sample over a measured interval and the total width of fractures recorded. A second chip sample of rock between the fractures is taken. This sample may be a random sample but an effort is made to obtain a representative sample of the host rock. An averaged gold content may be calculated for the interval (width x assay + width x assay divided by the interval or total width). The object of fracture sampling is to sample any vuggy quartz or oxidized sulfides in or on the fracture. Calcite fractures generally do not carry Au values. Based upon my experience, hematite red or "favorable" dark brown does not indicate gold mineralization. Jarosite brown or yellow may indicate gold mineralization. If both oxide colors are present, I would sample the fractures separately. Cortez is an excellent example of fracture sampling leading to the discovery of the deposit.

If good exposure of a face or trench is available, a bulk sample program consisting of taking a uniform sample of rock from one to two pounds per foot over a measured interval is appropriate.

Drilling

Except at Zortman, Montana, where core drilling, down-the-hole hammer drilling, rotary, and reverse circulation down-the-hole hammer drilling were tried, I have had good results establishing ore grade with drilling. At Zortman, I failed by any drilling method to establish the grade I was confident was in the rock. Poor core recovery and lack of sludge return is my explanation for failure of core drilling. Slow penetration rates and inadequate air pressure may explain why standard rotary or reverse circulation did not work.

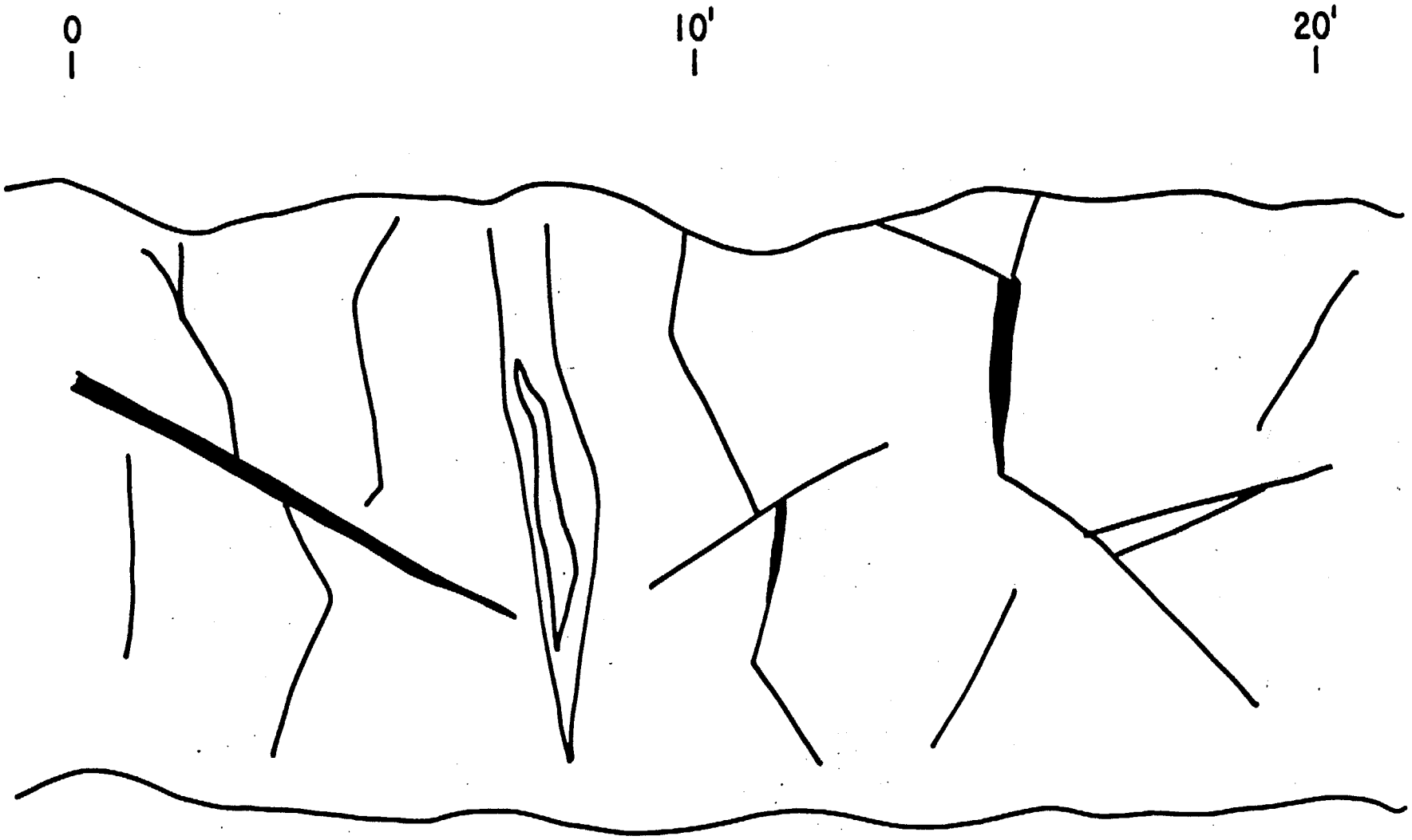
Following are comments concerning drilling methods:

Percussion Drilling

I have only drilled a few holes with a percussion drill, and then I was not satisfied with the results. This may be because I had a poor operator with inadequate sample collection equipment.

Rotary or Down-the-Hole Hammer Drilling

Cortez, Horse Canyon and other generally flat lying gold deposits were drilled by vertical rotary drills with very acceptable results.



Mineralized Outcrop

Figure 4

The Golden Sunlight and German Gulch were, in part, drilled with angle down-the-hole hammer drilling with acceptable results, particularly above the water table. Below the water table, the sample may have been down-graded due to loss of fine material.

Rotary drilling is good where rapid penetration rates may be achieved, when adequate high pressure air is available, when an experienced operator is operating the drill, and when adequate sample collection equipment is used.

Figure 5 is a drawing of a simple and adequate sample collection device for rotary chips. A high percentage of the chips are blown through the flexible hose into the cyclone collector where they are contained until the bottom door is opened. Fine dust is blown out the top of the collector, but a majority of the fines are caught with the sample. At every two to three feet of hole, the cuttings are dumped into the sample pan and split once through a large Jones sample splitter. This process is repeated until usually 10 ft. of hole is drilled. At that time, the hole is blown clean of cuttings and the sample collector is cleaned. The sample is then split through the Jones splitter until a sample of 15 to 20 pounds is split. This sample is then bagged and sent to the sample prep room for processing.

I have used a power driven automatic sample splitter which split one-eighth of the sample. I believe this method causes too high a loss of fines and dust to be acceptable for sampling low grade gold deposits of the Golden Sunlight type (high sulfide, low grade gold deposits).

Core Drilling

Because most low grade gold deposits consist of gold on or in fractures and micro-fractures, core recovery is the key to developing confidence in the gold content of the core. A core size should be specified which will give the best possible core recovery. At the Golden Sunlight, we believe core recovery of 95 plus percent was necessary to be confident of the grade of the deposit. In ore zones, the Au assay decreased with decreased core recovery.

At the Golden Sunlight, re-useable wooden core boxes which hold 20 ft. of NQ or 16 ft. of HQ core were used to collect core at the drill and transport it to the core shack. Each box was lined with 6 mil plastic sheets. Core was placed directly into the core box from the wire-line tube. At the core shack the core was logged and then split using a chisel-type hydraulic core splitter. Half the core was placed in cardboard core boxes for storage and half was placed in a large sample bag. At the end of each 10 ft. sample interval, the plastic used to line the core box was cut and small core pieces left on the plastic were split through a Jones sample splitter; half went to the sample and half to the stored core box. We believe this method retained the fine material which otherwise may have been lost.

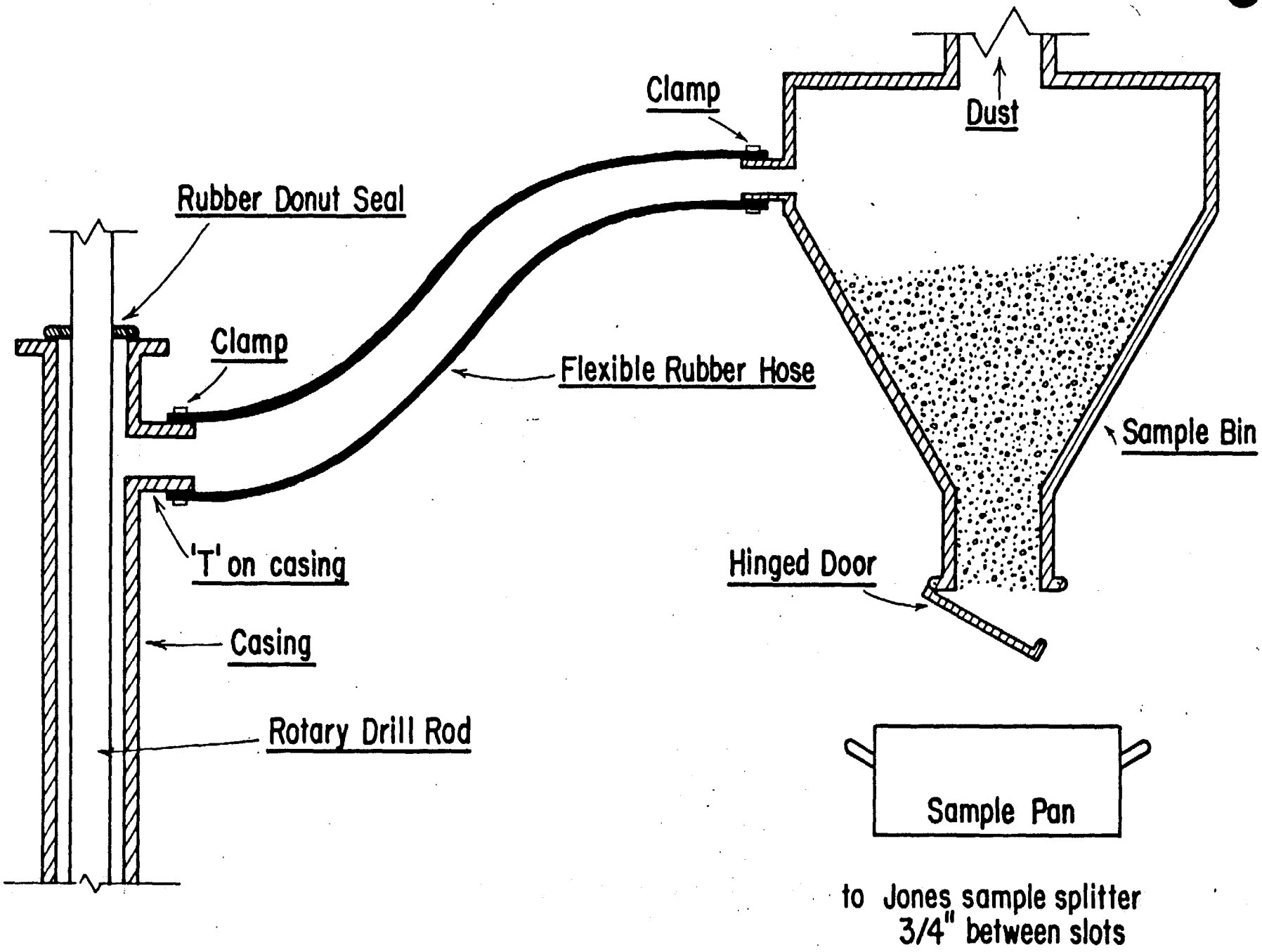

Clamp
Dust
Rubber Donut Seal
Clamp
Flexible Rubber Hose
Sample Bin
'T' on casing
Hinged Door
Casing
Rotary Drill Rod
Sample Pan
to Jones sample splitter
3/4" between slots

Figure 5

Comment

Throughout this paper, I have stressed the fact that exploration for large tonnage, low grade gold deposits requires common sense and meticulous attention to all phases of sample collection, sample preparation and assays. Geologists who establish a system and do not review the system from time to time, or a geologist who allows minor sloppy procedures will develop problems.

At $400/oz Au, an 0.001 oz Au assay is worth $.40, or nearly equal to an 0.01 oz Au assay at $35. Carrying this thought further, one must be ten times more careful and thorough when examining a gold deposit today, than when the gold price was $35/oz.

Lacana Exploration of Major New Gold Property in Nevada

Relief Canyon potential open pit/heap leaching operation

Lacana Mining Inc., 100%-owned U.S. subsidiary of Lacana Mining Corp., commenced on Feb. 1 a major drilling program of about 40 holes totaling 10,000 ft. on its Relief Canyon gold property, 100 miles northeast of Reno, Nev. The program is expected to take three months and if successful will be followed up by a second stage program of 150 holes totaling 40,000 ft. of drilling. Exploration work completed on the property last year and previous wide-spread drilling by a large U.S. corporation has indicated large volumes of disseminated low grade gold mineralization on the order of 5,000,000 to 10,000,000 tons grading 0.04 to 0.06 oz. of gold per ton.

The property is well situated with nearby water, power and labor. The attitude of the mineralized zones and the topography is favorable for an open pit mining operation with a low stripping ratio.

Preliminary metallurgical test work indicates that the gold is amenable to extraction by conventional cyanide milling or by heap leaching.

Lacana expects to spend over $1 million on this phase of the development program and will have earned an approximate 90% interest in the property. Falcon Cement Inc., a wholly-owned subsidiary of Falcon Construction Inc., Auburn, Calif., holds a 10% interest.

Southern Peru Corp. Strike Ends

Southern Peru Copper Corp.'s striking workers returned to the company's Toquepala and Cuajone mines and Ilo smelter Jan. 24 and 25. The work stoppages began on Jan. 6 and were declared illegal by the Peruvian government's Labor Ministry.

SPCC is owned by Asarco Inc. (52.3%), subsidiaries of The Marmon Group Inc. (20.7%), Phelps Dodge Overseas Capital Corp. (16.3%), and Newmont Mining Corp. (10.7%).

IOC's Final Iron Ore Shipment to Great Lakes Region Dec. 9

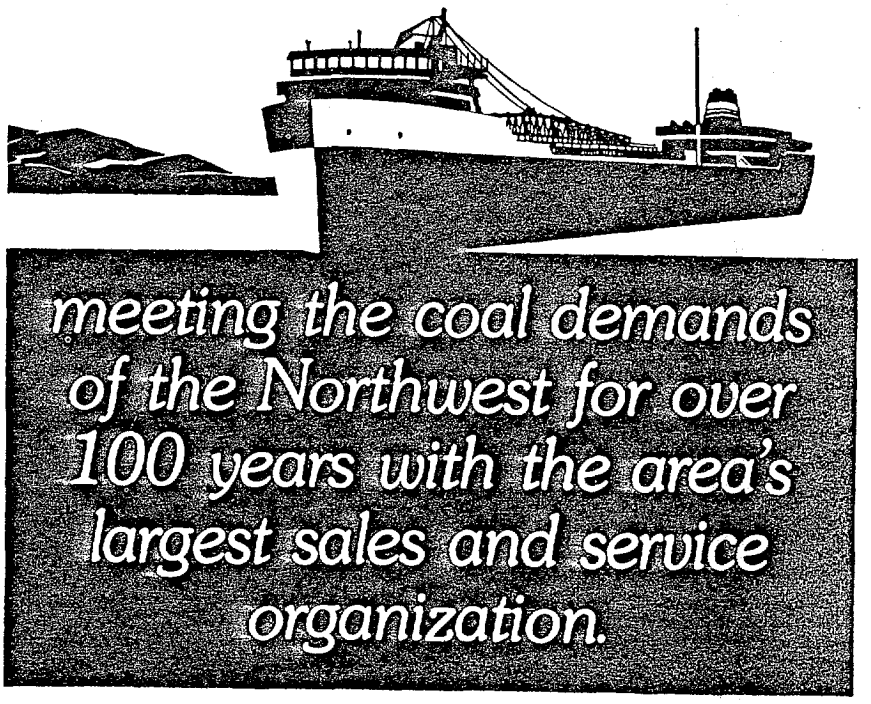
On Dec. 9, Iron Ore Co. of Canada completed the 1982 iron ore shipping season to the Great Lakes region by handling the final vessel cargo destined for transportation through the St. Lawrence Seaway. This last shipment amounted to 25,950 metric tons of pellets produced at IOC's Carol project at Labrador City, Nfld., that was loaded into the mv. *Algolake* of the Algoma Central Ry. fleet for delivery to Ashtabula, Ohio, consigned to Wheeling-Pittsburgh Steel Corp.

J&L's Iron Ore Production in Final Year at McKinley Mine

Output of beneficiated natural ore in 1982 at 616,750 G.T.

In 1982, Jones & Laughlin Steel Corp. produced 616,750 gross tons of beneficiated natural ore in the final year of operation of the McKinley mine by its northwest ore division at McKinley, Minn., on the Mesabi iron range. This tonnage was railed to the port of Duluth for loading into vessels.

During the previous year, 463,136 gross tons of fines and coarse ore were shipped by J&L from the McKinley mine.



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Placer Amex Starts Up Its Golden Sunlight Gold Mine

Project of $50 million completed five months ahead of schedule

Placer Amex Inc. has announced that processing of ore at its new 100% owned Golden Sunlight open pit gold mine near Whitehall, Mont., commenced on Jan. 19. The mine, owned through a wholly-owned subsidiary, Golden Sunlight Mines Inc., was brought into production about five months ahead of schedule at a cost of about $50 million, substantially less than original estimates.

Ore reserves are estimated at 25,800,000 tons with an average grade of 0.05 troy ounces per ton. The orebody will be mined over 13 years, at an average strip ratio of two tons of waste and low grade to one ton of ore. The mill has a daily throughput capacity of 5,000 tons and is expected to yield an average of 72,300 troy ounces of gold annually.

Placer Amex, with head offices in San Francisco, is a wholly-owned subsidiary of Placer Development Ltd., Vancouver, B.C.

Tosco's Oil Shale Activities Undergo Further Consolidation

Tosco Corp. has laid off 63 employees in a further consolidation of its oil share activities.

The employees affected are situated primarily in Tosco's Colorado research center and offices and, to a lesser degree, in its California and Utah locations. Tosco's Vernal, Utah, field office will be closed and its activities transferred to Colorado.

"Our continuing Sand Wash oil shale project in Utah will not require substantial field activity this year, and this consolidation will facilitate our project planning and related activities," said Morton M. Winston, president and chief executive officer of Tosco.

Lead Price Increased By Asarco to 21.50 Cents

Asarco Inc. announced Jan. 27, effective immediately, an increase in its base spot sales price of refined lead by one-half cent per pound. The new price is 21.50 cents per pound, f.o.b. delivered in carload lots.

Metal Prices

Gold, per oz.	$508.50
Copper electrolytic, per lb., delivered Valley	79-80c
Lead, per lb.	21½-22½c
Nickel, electrolytic cathodes, per lb.	$3.29
Aluminum, virgin (99.5% to 99.9%) per lb.	76c
Iridium, per oz.	$400.00
Tin, per lb.	$6.3709
Rhodium, per oz.	$310.00
Platinum, refined per oz.	$475.00
Quicksilver, spot per flask	$350.00
Cadmium, per lb.	65c
Silver, foreign and domestic, per oz.	$14.335
Zinc, per lb.	40c
Molybdenum, per lb., dealer oxide	$2.35
Bismuth, per lb.	$1.35
U_3O_8 Concentrates, per lb.	$20.30
Tungsten, per S.T.U. WO3	$ 76.00

Lake Superior Iron Ore Prices

(Per gross ton, 51.50% iron natural, at rail of vessel lower lake port.)

Mesabi Non-Bessemer	$32.25-32.53
Old Range Non-Bessemer	$32.78
Manganiferous	$32.78

Lake Superior Pellet Prices

Per Iron Natural Unit	86.90c
Per Iron Natural Unit	80.50c

Direct Reduced Iron Pellets

(Per metric ton)

F.O.B. Contrecoeur	$115.00
F.O.B. Georgetown	$125.00-135.00
F.O.B. Point Lisas	$120.00

"Exploration at the Crossroads?" Theme of Prospectors and Developers Assoc. Convention

Featuring sessions on mineral exploration, financing the future and new discoveries

The Prospectors and Developers Association's annual convention will be held March 6-9 at the Royal York Hotel in Toronto, Canada. This year's theme will be "Exploration at the Crossroads?"

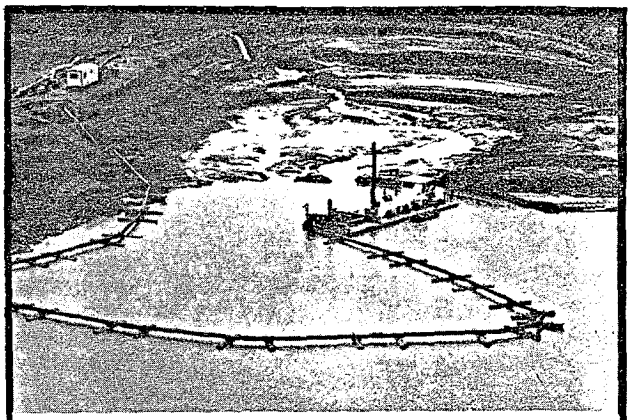
The PDA convention, with about 3,000 expected to attend, is of interest to prospectors, mine developers, geologists, mining engineers, mining company executives, students and associates in the drilling, banking, investment and many other commercial fields related to the mining industry.

Various sessions to be offered at the conference include: Is mineral exploration in Canada at a crossroad?, ore or waste, financing the future, provincial up-date, new discoveries and geophysical and geochemical classes and workshops.

Organization of the technical program is under the chairmanship of Dr. John Hansuld of Amax Minerals Exploration.

Keynote luncheon speakers include: the Honorable Grant Devine, Premier of Saskatchewan; Harold T. Fargey, president of the Mining Association of Canada and executive vice president of Cominco Ltd.; and Adam H. Zimmerman, president and chief operating officer of Noranda Mines Ltd.

For further details on the conference, contact Bill Griffith, general manager, or Greta Firth, executive secretary at the Prospectors and Developers Association, Suite 219—159 Bay Street, Toronto, Ontario, Canada M5J 1J7 or phone (416) 362-1969.



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ASARCO

Southwestern Exploration Division

February 15, 1983

To: J. D. Sell

From: F. R. Koutz

Anaconda Talk - AGS
Summitville Project
Rio Grande County, CO

The following are notes from a talk by R. M. Perkins and Bill Neiman on "Epithermal Gold Mineralization at Summitville, Colorado" presented at the Arizona Geological Society Meeting on February 1. In preparation for this talk I briefly reviewed the Western Exploration Department files in Tucson, especially 2 interim reports by D. J. Krasowski (March 6, 1978) and by C. E. Beverly and D. J. Krasowski (Feb. 1976). Some of the early portion of the talk was on regional/district structure and geology by the USGS and is not reported in these notes. As usual Anaconda has split out several phases of the igneous units on the basis of texture and alteration. A complete 55 minute casette tape of the talk and discussion was made and I will preserve it until their paper comes out.

History: Mike Perkins, B.S. & M.S.-Univ. Colo., Cities Serv., Bendix, Anaconda - 4 years as Summitville Project Geologist - now with Arco Oil & Gas. Bill Neiman, B.S.-U of A, M.S.-U of Wash., Anaconda 1980 on - much time (still) at Summitville. Talk will be published about April 1983 as part of the DREGS Symposium: Mineral Deposits of the Rocky Mtns. They credit: C. E. Beverly with promoting relationship of epithermal mineralization to a potential porphyry copper at depth; Stevens & Ratte: S. Mtn. volc. dome & adv. argillic alteration relationship to Au-Ag mineraliz.

Scenery Slides: Lots of green trees, snow and jarosite. Summitville is in SE portion of S.J. Volc. field in Summitville & Platoro Calderas. Most old gold mines on South Mtn (12553') ± at treeline, 400" snow/yr. Work season 3-5 months. Townsite (11300').

- 1870's placer, 1873 Au° in outcrop at South Mtn.
- 1873-1942: +300,000 oz Au + Ag, Cu, Pb, Zn.
- Mid 1960's-Early 1970's: Cleveland Cliffs, U.P. R.R., & W.S. Moore after Cu-Au-Ag veins. Built buildings and explored Missionary Vein (silicif.-en-py at base of mtn.).
- Early 1970's: ASARCO after bulk tonnage open-pittable Au deposit on S. Mtn. + deeper porph. Cu system.
- 1978: ASARCO released epithermal part of district to property owners and Anaconda "jumped on it". ASARCO continued deep exploration for porphyry.
- 1982: Anaconda consolidated district - controls mineral rights for entire district.

Geology: -35-40 m.y.: widely spaced andesitic flows and BX (aprons).
-30 m.y. - on: pyroclastics & ash flow sheets of San Juans-calderas.
-Summitville caldera is in NW/4 of Earlier Platoro Caldera.
-Intracaldera-fill andesites ± basalt & rhyolite flows - in part overflow rim.
-Shallow intrusives (e.g. Cat Creek in core of early andesite volcanos) include
-South Mtn. Volcanic Dome - Host to epithermal mineralization - on major (Platoro) NW fracture trend. 4 phases of qtz. latite porphyry (QLP) of S. Mtn. volc. dome split out on basis of % round. angular qtz eyes, size of anhedral-euhedral Fs phenocrysts (esp. sanadines - to 5 cm), hexagonal biotites, aphanitic groundmass, and %-intensity of alteration.
-Can walk in alteration continuously S to Alamosa River Stock & to Jasper. Progressive Alteration: propylitic-argillic-adv. arg.-silicic. (Local volcanic stratigraphy discussion).

QLP: 4 Phases
1. Core of volcanic dome, x.g. sanadines (>2 cm), intense alt. (Au° min. host).
2. Flanks of dome, m.g. sanadines (euhed. < 2 cm) (on W & North).
3. Late mineral., x.g. sanadines (2-4 cm) - exterior phase QLP (on E & S), weak to mod. alt.
- Note: Dome has flow laminations and is intrusive (vertical folia.) at center, horizontal at margin where extrusive.
4. Post-mineral. dikes, vitrophyric margins - up to 30' wide - discontinuous.

K/Ar Ages: Summitville area 29-19½ m.y., Dome: 23-22 m.y.
21-19½ Post-mineral Cropsy Mtn. Rhyol. with basal vitrophyre.

Structure: Volcanic dome at intersection of NW(?) margin of Caldera complex and NW-trending Platoro Fx zone. Imposes NW struct. grain + Fx on dome and controls intrusions.
2 Dist. scale faults: Missionary Flt. - doesn't OC - strongly inferred from UG & DDH. South Mtn. Flt zone - up to 200' wide - part of Platoro Fx zone.
"Coherent vein structures are totally absent on present level of exposures of volcanic dome". NW zones of intense BX, alt, mineraliz. along near surf. dilatent fractures by hydrothermal fluid pressures. Silica & alt. minerals destroy evidence of fractures.

Alteration: 6 major alteration assemblages with epithermal mineralization on map.
- Generally outward: Silica, alunite, clays, propylitic, fresh (±). Contacts irregular & gradational - but sometimes sharp.

Map Units
1. Pure silica.
2. Silicic and advanced argillic alteration: irregular pipes, pods and tabular masses. - zoning (selvages) about bodies mimics district scale zoning. (Sometimes several intensities of this unit mapped).
3. Grey argillic alteration.
4. Propylitic with very weak argillic.
5. Propylitic.
6. Fresh or v. weak alteration.

Propylitic: Lt. green but often bleached white by supergene; groundmass to chlorite and montmorillonite, sanadines fresh, biot to mgt ± chl. ± py. 1-3 vol. % py., Fx surf. stained by limonite + MnOx.

Argillic: Grey to lt. green. White to yellow bleached (goeth-jarosite? FRK). If weak illite - kaolinite gets everything except sanadines, if intense: sanadines go also. 3-7+% pyr. (unoxid.).

Advanced Argillic: 1-3 vol. % pyrite - 2 subtypes.
1. Qtz + clay ± alunite: groundmass f.g. microcrystalline qtz, illite + kaolinite + alunite in feldspars.
2. Qtz + alunite ± clay: Feldspars replaced by pink to white coarse-grained alunite and minor clay. Generally more pyrite in this subtype. (f.g. silica in groundmass) x.g. hypogene alunite: 4 dates, 2 USGS, 2 A.M.C. (23 ± 1 m.y.) Simultaneous with volcanic dome formation.

Intense Silica: 2 types.
1. Massive silica, volumetrically insignificant & econ. unimport: tabular shaped bodies - caps to BX pipes & masses - paragenetically late.
2. (Au°-$) Vuggy silica ± advanced argillic. Ore Host. Pencil line contacts with adv. argillic alteration. >10% vugs, casts of feldspar phenos., qtz eyes left, rest of rock replaced by microcrystalline qtz. Cavities contain: drusy qtz, S°, en, cv, py, barite, Au°. Vuggy silica forms irregular pipes, pods and steeply dipping tabular masses. One to several tens of feet wide. Extend over +1000' vertical. Below 200' contain abund. sulfides: Cu sulfides

and sulfosalts as vug and fracture fillings and dissemin. in BX matrix. Above ±200' contains thick encrusting FeOx w/o sulfides - same textures.
Intense BX of vuggy silica over all of vertical extent = hydrothermal conduits but very discontinuous & restricted (laterally) within dome. Very good correlation in vuggy silica of % FeOx and Au° content. Very resistant silica-alunite outcrops; argillic is nonresistant & white to yellow.

Breccias: Lots of BX: pebble BX, rock flour dikes, BX pipes & masses - all associated with intense alt/mineral. Fragments: angular to subrounded often in FeOx matrix.
-Homogeneous BX: 1. alt type in fragments = well mineralized vuggy silica + FeOx in matrix.
-Heterogeneous BX: 2 or more alt. types in alunite ± clay matrix, poorly mineralized, leached, prob. late in last gasps of system during collapse.

Mineralization: Lots of >1 mm Au° but spotty. Poor photos of Gold Boulder, fair of pannings. But Au° averages 1-10 microns; most old production free milling Au° from vuggy silica & breccias - near surface oxides. Au° + barite very important association: highest grades in pipes with lots of barite. 30 pounds of Au in Gold Boulder to Denver Mus. Nat. Hist. & Asarco N.Y. offices. (30 x 12 = 360 oz x $500 = $180,000.)
- deeper sulfide zone: Au° with vuggy silica with copper sulfides and sulfosalts: >10% sulfides: with Au: cv, cc, digenite: also en, luz, py, marc., fam., polybas., cp, sl, gn.
- decrease in Au/Ag & Au/Cu ratios with depth in sulfide zone. Gradation of en to cv-cpy as dominant sulfide with depth but en extends below cv-cpy in pendants or roots.
- Au°: averages 950 fine - a little higher in sulfide zone vs. oxide zone (!!?).

Surf. Geochem: Outcrop & shallow trenches & drilling.
Central As-Au-Pb-Ag zone.
W&S flanks anom. Ag, Au, Pb, Hg.
Halo (esp. on east side) Mn: >1500 ppm Mn (<50 ppm Mn inside dome).
Volcanic dome enrich. in Bi, Sb, Tl, Te - about coincides to Au.
Depleted in Ba, F, Mn.

Genetic Model : Close to surface during Fm. (500' removed by glaciation - would not have found w/o glaciation). Important Points:

1. Volcanic dome with multiple intrusive events - effected chemistry and hydrodynamics of hydrothermal fluids.
2. Major struct. intersection controls implacement of dome, sympathetic Fx, zones of near surf. dilatent Fx guided BX, Alt, mineraliz. - most at hydrothermal conduits.
3. Most of dome has paragen. early prop. and argillic assembl. - later adv. argillic - still later vuggy silica. pH 3-4, T.= 200-300°C (before end of vuggy silica & before mineraliz.). Boiling of fluids. Much alunite contemp. with host QLP.
4. Both high level oxide-barite and deep sulfide (Cu sulfide & Sulfosalt) Au mineraliz. with vuggy silica alteration.

To explain (history):

1. Intrusion of x.g. core phase & m.g. phase of QLP.
2. Early hydrothermal fluids invade dome: prop. & arg. alt.
3. Boiling: hydrostatic > lithostatic: heat from late stage QLP or some other event. (e.g. release of press. - FRK).
4. Fluid reservoir stratified, vap/liq. dominated: Boiling causes BX pipes, H_2S & volatiles partitions to vapor & alters wall rock to alunite. - Liquid system alkaline and ppt sulfides.
5. Vuggy silica: very acid solutions through conduits, low pH alunite but alkaline solutions in late stage collapse (cool oxygenated waters) may have leached alunite and ppt. FeOx, barite & Au° high up while sulfides and sulfosalts in lower alkaline parts of system.

(No mention of sulfide/sulfate equilibria - jarosite or any sulfide or Au paragenesis. Au° apparently does not replace sulfides-sulfosalts).

Perkins compares (without detail) Summitville to Red Mtn., Colorado, Goldfield, Nevada, and El Indio, Chile in struct. setting, BX, caldron margins.

Discussion from the Crowd

FeOx mostly hypogene (doesn't mention FeOx mineralogy) but minor supergene. FeOx hypogene because of breccia textures, encrustations but zones into sulfides at depth - oxidizing/boiling level about at present sulfide/oxide interface.

No changes in FeOx "type" with depth - just Au° - Fe oxide relationship. "Not much" fluid inclusion work (a lie!). Says Skip Cunningham (USGS) got 110-120°C on The Boulder.

No grade or tons given but St. Joe geologist next to me says Anaconda'a property/JV saleswoman advertizes 2½ m.t. each of oxide and sulfide ore with 0.15-0.18 Au in each plus an oz or so Ag/ton in sulfide ores and several hundred thousand tons of +0.2 oz Au/T within 50 feet of surface to pay for mill. Apparently St. Joe didn't believe it.

When pressed Perkins says all of South Mountain was mineralized with Au, but argillic material between pipes nowhere near economic values. Au is "structurally" controlled in vuggy silica and bx - outside Au drops dramatically. Will not say how much area of vuggy silica there was on map (looked like 15-20%, 30-40%:silica + advanced argillic) or if all was economically mineralized (some of silica doesn't outcrop) but mentions "significant" - silica and gold over 2-4000' length by 1000' width - no depth given but says significant gold doesn't extend much more than 1000' depth. "Below that ASARCO did not find a copper porphyry and they drilled pretty deep." (3000' x 1000' x 100' ÷ 12.5 = 24 m.t. x 15% = 3.6 m.t.)

Doesn't recall threshold grades used to construct geochem. maps (Neiman mentioned to me Mo anomaly to 50 ppm, large areas of +100 ppm As & Cu). Can't tell if more than one stage of gold: "dynamic system". On exact morphology of vuggy silica pods pleads: "Rotary holes." (Several slides show low angle (30-45°) reverse circulation rotary rigs set up - sometimes drilling parallel to very steep slopes on South Mountain - comments from crowd on high-grading).

Afterward, Bill Neiman mentioned to me that Anaconda - U.S.G.S. are financing a Brimhall student at Berkeley who is doing lots of fluid inclusion work. (Also sounds like lots of Anaconda time was spent high grading - hunting for Gold Boulders.) Rumor also has it that 2 former Dick Beane students have done Summitville fluid inclusion work for Anaconda: Jim Reynolds (a petrographic consultant in Denver now) and Bob Bodnar a Penn State student. They think they have a magic fluid inclusion method of finding ore based on some new CO_2-salinity data from experimental work - at least Anaconda thinks its great. Bodnar had a number of "DDH" series core samples from Summitville obtained via Beane and D. Norton from Greg McKelvey of Cominco in 1977-78 and has recently obtained additional core samples from the USGS.

Overall a good talk but contained little ASARCO didn't already know.

F. R. Koutz

F. R. Koutz

FRK/cg

cc: WLK
PGV
DMS
FTG
JHC

ORE SAMPLE PREPARATION

by Mark F. Lewis
Manager/Metallurgist
Legend Metallurgical Laboratory

The different types of metallurgical tests, as well as geochemical analysis and fire assaying, all have the same first step—sample preparation. Sample preparation cannot by itself ensure the quality or accuracy of the entire test. But improper sample preparation can introduce errors, which can make any analysis or assay totally useless, or worse, cause a result that is misleading. Care taken in sample preparation can ensure quality results that can be reproduced again and again when combined with quality analytical methods.

A sample is defined as "a small amount which contains all the components in the proportions in which they occur in the original lot." The word "sample" is taken to mean that fraction that is taken to represent the whole. This is an ideal situation that can never be achieved in a practical way. The only way to obtain a *true* sample is to use the entire amount of material. In the case of an ore sample, it obviously is not reasonable.

It is difficult to take an ore sample because of: (1) the large variety of minerals in the rock, (2) the lack of uniformity in the distribution of the minerals throughout the material, and (3) the lack of uniformity in the particle size of the material. Methods must be used that will allow every particle of the nonhomogeneous ore mixture to have the same chance of being included in the sample.

Many sampling methods have been devised to achieve accurate sampling, although they differ in important respects. Most sampling methods stem from random selection, selection by rule, or mixing and cutting. Random selection and selection by rule are forms of "grab sampling," which is only satisfactory for uniform ores. When exact samples of precious metal, or nonhomogeneous ores are needed, only mixing and cutting, either by manual or mechanical means should be employed. Mixing and cutting, when properly done, follow statistical rules and obtain mathematical precision according to the grind of the sample (this grind will depend on the particular ore).

The normal procedure of ore sample preparation consists of three separate operations: crushing, mixing, and splitting. These operations do not necessarily follow in this exact order, and steps may be deleted or repeated numerous times to obtain a sample of the necessary size.

At the Legend Metallurgical Laboratory, a normal five-pound rock sample undergoes the following procedure:

Step 1: The entire sample is crushed to ½-in. through the primary 4-in. by 6-in. crusher.

Step 2: The entire sample is crushed to ¼-in. through the BICO secondary crusher.

Step 3: The entire sample is split through a 1-in. Jones riffle.

Step 4: The remaining sample is split repeatedly through a ¾-in. Jones riffle until a 300-gram sample is obtained.

Step 5: This 300-gram sample is pulverized in a 3 hp. BICO pulverizer to obtain a 100-mesh pulp.

Step 6: The 100-mesh pulp is rolled 25 times per corner (100 times total) for thorough homogenization.

Step 7: This sample is put into a plastic moisture-proof pulp bag for further testing, assaying, or trace analysis.

Oftentimes customers have special sample preparation procedures that are more extensive and thus more effective on problem ores. The sample preparation process can become very extensive and thus very expensive. Ores are never perfectly uniform in composition, and a certain amount of crushing, mixing, and splitting will be necessary in most cases. The greater the difference in the grade of the different minerals present in the ore, the finer the ore must be crushed before a given sample can be taken.

It is important to consider the economic side of this process: How accurate can I afford to be? How accurate do I need to be for this particular situation? The higher the grade of ore, the more effect an error will have. Small percentage errors may be acceptable in low grade deposits or exploration samples. However, when accuracy and precision are necessary, the process must be extensive enough to obtain a sample that comes within an allowable limit of error of the ideal or true figure.

A successful sample will come within a certain allowable limit of error and will be obtained without undue delay at a reasonable cost.

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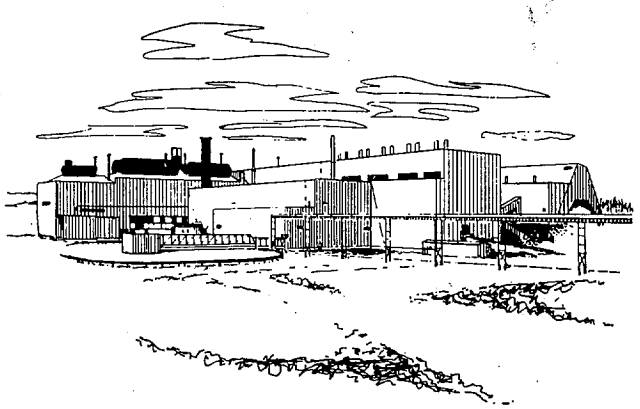

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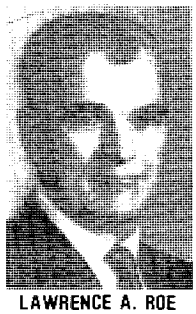
Trade Notes

CF&I Steel Corp. has announced the appointment of **John D. Foster Jr.** as manager, rolled and mining sales product department and **Mrs. Kathleen A. Cara** as assistant manager. Both Mr. Foster and Mrs. Cara will be situated at the Pueblo, Colo., plant.

Goulds Pumps Inc. has appointed **Michael J. Baxter** to the position of slurry pump specialist at its Chicago area office, which is in Lisle, Ill. Most recently he was process engineer for slurry pumps with **Process Systems Inc.**, Jacksonville, Fla., and previously was with **Warman International Inc.**, Madison, Wis. In 1972, Mr. Baxter was graduated from University of Wisconsin, Madison, with a B.S. degree.

Asarco Inc. has issued a new four-color, four-page brochure describing its shaft melting furnace, according to an announcement by **Dr. William P. Roe**, vice president-research. The brochure describes advantages of the furnace and the technical assistance available from the company and also details important features of the furnace's technology. Copies of the brochure may be obtained from Asarco, Center Research Department, 901 Oak Tree Rd., South Plainfield, N.J. 07080.

Lawrence A. Roe, president of **Roeco Inc.**, has announced formation of a technical publishing division that will handle the second book issued by the firm titled "Industrial Waste Flotation." The latest book, published on Feb. 15, describes the growth in use of flotation processes in treatment of industrial chemicals and wastes. Included are developments in wastepaper deinking, oil recovery from water and oil and tar sands, sewage flotation and descriptions of zero discharge plants. Additional information is available by contacting Roeco at 1080 39th St., Downers Grove, Ill. 60515 or by phone at (312) 852-7153.


LAWRENCE A. ROE

Donovan J. Culliton has joined **Borchert-Ingersoll Inc.** as a sales representative, responsible for sales in northeastern Minnesota, with offices at the company's branch in Hibbing, Minn. Most recently Mr. Culliton sold equipment for **International Harvester Co.** in northern Minnesota, serving earlier with **Hall Equipment** as a sales representative and a parts manager and sales representative for **Phillippi Equipment.**

Jarvis Clark Co. Ltd., Canadian manufacturer of underground mining equipment, will consolidate all manufacturing and international parts distribution activities at its plant and head office in Burlington, Ont., and will upgrade facilities at its Sudbury maintenance, overhaul and rebuild complex. Manufacturing operations and engineering activities will be amalgamated into the Burlington operations. Transfer of the company's international parts center at Bramalea, Ont., that serves overseas customers, to the Burlington site is presently under way. In his announcement, **Rick Gray**, president of Jarvis Clark, said that the transfer from North Bay to Burlington will be completed by mid-May.

Hitachi Construction Machinery Co. Ltd., America office, Houston, Texas, manufacturer of hydraulic shovels and backhoes for mining and excavation, has announced appointment of **John E. Mahoney** as marketing manager. Mr. Mahoney has over 30 years of experience with hydraulic machine application. Prior to his move to Hitachi, he served as hydraulic mining machine product manager and manager of mining for other major manufacturers of hydraulic equipment. Mr. Mahoney is currently vice president-elect of the Society of Mining Engineers and a constituent of professional mining engineers of the AIME. Mr. Mahoney's responsibilities will include various aspects of marketing Hitachi hydraulic mining machines.

JOHN E. MAHONEY

ASARCO

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Great Basin Exploration Division

February 22, 1983

MEMORANDUM FOR FILES

Relief Canyon Au
mineralization, Pershing Co., Nv.

Jim McKee (?), a geologist with Duval in Reno, Nevada, volunteered the following information on the Relief Canyon gold mineralization (Section 20, 21(?), T27N, R34E) that has lately made rounds in the journals (e.g. Northern Miner 2/3/83):

The mineralization, which occurs in Natchez Pass carbonate rocks, was originally staked (apparently not leased) for cement by Falcon Cement, Inc. Duval optioned the claims from Falcon after identifying anomalous gold in fault breccia within the Natchez Pass. McKee found the mineralization by sampling USGS-mapped "thrusts" in Mesozoic strata of the Humboldt Range. Duval first examined these thrusts in the vicinity of the Standard Mine (along with Freeport, Cordex, et. al) during the summer hordes days of 1981. Finding no worthwhile mineralization there lead Duval to the area south of Buffalo Mountain, and elsewhere.

The mineralization has drill-estimated reserves of 10 m.t. @ 0.05 oz. Au/ton with no silver. It occurs in fault breccia and underlying carbonaceous shalely limestone and is up to 100 (?) feet thick. Stripping ratio is envisioned as 1:1 and mineralization continues at depth beneath the range of open-pit mining. Grade is uniform with little rock exceeding 0.1 oz. Au/ton. Mineralization is entirely oxidized and leaches "well", although later in the conversation McKee mentioned possible recovery problems.

Duval apparently "handed" Lacana the drilled reserve, which retains 10% interest by Falcon, and kept only 4% NSR for themselves. Southern Pacific land apparently contains ~10% of the reserve and that company has been drilling along the section line that transects the mineralization.

ASARCO examined and sampled the thrust fault zones between Humboldt Canyon and the Standard Mine in 1981 in the course of range margin exploration and failed to substantiate the interest in this area shown by Freeport and Duval. Also in 1981 ASARCO sampled the range margin in the vicinity of Relief Canyon but did not encounter anomalous Au. In 1982 ASARCO

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heavily sampled the Antelope Springs Hg district immediately south of Relief Canyon. Widespread anomalous Au, Sb and Hg were recognized in complexly folded and faulted Mesozoic rocks, but no targets were identifiable. Since then Homestake and FMC have staked all non SP land in that district.

P. G. Vikre

Peter G. Vikre

PGV/jn

cc: Mr. W. L. Kurtz
Mr. F. T. Graybeal

MINING Week

Lacana Finds More Gold in Nevada

LACANA MINING Corp., the Toronto-based company with mining and oil/gas interests in N. America and Mexico, has concentrated most of its exploration efforts in recent years on gold and silver, a sector where it has achieved considerable success to date. The company carries out exploration both on its own account and as joint venturer with other parties. In the U.S., it has an equal one third interest (via its wholly owned U.S. subsidiary Lacana Mining Inc.) in the Cordex IV Syndicate together with Dome Exploration and Rayrock Mines. This consortium has been evaluating the Boulder Creek gold property for the past two years. This low grade, near-surface deposit is located some 85 km east of the Pinson gold mine in Nevada, in which Lacana has a 26.15% interest (*MJ*, November 5, 1982, p.331).

Some 90 km to the south-west of Pinson, Lacana Mining Inc. has been investigating a further occurrence of gold mineralization, a deposit known as Relief Canyon. This has previously been the subject of a programme of widely-spaced drilling carried out by a major U.S. mining company. Lacana has now announced that a 3 month programme to drill 40 additional holes, totalling some 3,000 m, had commenced on February 1 in order that the property might be more fully evaluated. If this programme proves successful, a second stage, entailing an additional 150 holes totalling some 12,200 m of drilling, will follow on.

Work carried out at Relief Canyon by Lacana in 1982, together with the previous drilling of the property, had indicated the presence of a large amount (5-10 Mt) of low grade, disseminated gold mineralization, with average values in the 1.3-1.9 g/t range.

The orebody was amenable to open pit mining with a low stripping ratio, whilst preliminary metallurgical test work had indicated that the gold could be extracted by conventional cyanide leaching or by heap leaching.

Lacana expects to spend about $1.0 million on this exploration programme at Relief Canyon, and by so doing it will earn an approximate 90% interest in the property. The balance is held by Falcon Cement Inc., a subsidiary of Falcon Construction of Auburn, California.

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BRAZIL

Albras Smelter Progress

IN DECEMBER we reported that the construction of the Albras alumina refinery near Belem in northern Brazil had been delayed for three years but construction of the primary smelter would continue (*MJ*, December 24, 1982, p.449). Partners in Albras Aluminio Brasileiro SA, are the government-owned Companhia Vale do Rio Doce (CVRD) with 51%, and a consortium of Japanese companies comprising Nippon Amazon Aluminio Co. (NALCO) with 49%.

It has now been learned that talks are to be re-opened between the Japanese and Brazilian partners to review the alumina refinery project. In the meantime, the aluminium smelter, which will utilize pre-bake technology, is now due to come into operation in the second half of 1984. In its first phase it will produce 160,000 t/y of aluminium, increasing to 320,000 t/y in the subsequent final phase. At the smelter complex there is also provision for the construction of a green anode factory to produce the carbon electrodes needed for the production of aluminium from alumina.

The contract, worth some DM60 million, for the construction of the green anode factory has been placed with a consortium comprising Zaninin SA and Otto Deutz SA of Sao Paulo and KHD Humboldt Wedag AG of Cologne, West Germany. The order from Albras covers engineering, construction and delivery of the factory from raw material feeding to anode moulding. The contract also provides for the provision for a plant for returning and processing the anode butts.

The anodes are to be moulded by a process involving a Humboldt Wedag slide-type shaking machine (VAW system) under vacuum conditions. This produces anode blocks of high homogeneity and density and guarantees an optimal flow of current as well as a uniform and low consumption. The production of 27 green anodes/h will be possible, each with a block weight of 823 kg. A freely-programmable control permits a largely automated production process.

AUSTRALIA

Commercial Production at Argyle

COMMERCIAL production of diamonds from the Upper Smoke Creek alluvial deposits of the Argyle Diamond Mines Joint Venture in W. Australia began on January 1, following the approval of the venture's mining and marketing proposals by the state government on December 21, 1982 (*MJ*, January 7, p.10). This marks the completion of the initial phase of this major diamond venture, which could eventually become the largest single diamond producer (in terms of carats) in the market economy countries.

During the final quarter of 1982, the evaluation of both the Upper Smoke Creek alluvial material and the AK-1 kimberlite pipe continued. However, the sampling results for this period are not truly representative for several reasons. Some of the sampling of AK-1 (in the latest period) was carried out in selected areas known to be high grade zones. Furthermore, large quantities of very small stones were recovered from the reprocessing of material stockpiled in previous quarters. The recovery of these small diamonds had not been assigned a high priority at an earlier stage, as such production results were not required as input for Joint Venture feasibility studies.

Nevertheless, it is worthy of note that in the December quarter sampling of the AK-1 pipe yielded a 14.34 ct industrial quality stone from surface material, a 2.89 ct stone from large diameter core drilling and an 8 ct gem quality stone from material extracted from small sampling shafts. In the case of the alluvial sampling, the largest stone

ASARCO

Exploration Department
Western USA

~~FRK~~
~~LHGK~~
JDS

February 28, 1983

File Memorandum

"Carlin Type" Gold

Last year about this time I circulated a memo attaching a page from the 1981 Newmont Annual Report pointing out the striking alignment of the Newmont deposits, including the then newly discovered Rain deposit.

You guessed it - another deposit has recently been announced along this alignment. Attached is a page from a recent Northern Miner. Have they all been discovered?

W. L. Kurtz

WLK/cg

Attachment

cc: RLBrown
PGVikre
JDSell
GJStathis

Exploration Co., stresses to the Northern Miner that "this is just a proposal. It could change." He adds that although the company continues to study the feasibility of production at its Geco mill 35 miles north at Manitouwadge, "a mill on the site looks more favorable than the Geco mill at the present time."

The proposed size of the mill relates only to the Golden Giant deposit, and does not include capacity that may be added through custom arrangements with other companies in the area (see separate story).

Mr. Harvey also says that no capital costs have been estimated by Noranda and have yet to be worked

See Page 20

for small size, mill capacity

A deal has been struck between three of the major players in the Hemlo camp that will see ore from the property held by International Corona Resources and Teck Corp. put through a mill built and operated by Noranda Mines. In exchange for the mill capacity, Noranda has been allowed access to a quarter-claim on the Corona ground where it plans to sink a shaft to mine the Golden Giant deposit.

Teck has the Corona property under option and can earn a 55% interest upon putting it into production.

Details of the agreement include:

• Noranda must explore the quarter-claim (which has never been drilled at depth) to establish its reserve potential. If reserves are outlined on the small piece of ground, Noranda will allot up to 500 tonnes-per-day hoisting and milling capacity, and obtain a 50% net profit interest. If no reserves are discovered, then Teck/Corona will have the right to utilize the unused mill capacity for ore taken from other parts of the property, and not be charged capital expenses of the shaft or mill.

• Teck/Corona can call on Noranda during the next six months to add up to 500 tonnes-per-day additional capacity to its mill and pay the incremental costs of adding the capacity. **This means the Teck/Corona property could see production of up to 1,000 tonnes-per-day through a Noranda-built mill.**

• The construction of the shaft must begin within six months.

The shaft could be used for the exploration and mining of the Corona west zone, according to

See Page 20

pany would like to brin ductive operation into year.

So far, the goal has be target for this Soquer First it was the limestone the eastern shores of L

What's ahe for metals

Improvement in de prices for precious metals alike will lead dian metal and minin out of its 3-year-long ary spell. And, as th experts generally se recovery will begin to this year, leading into ous 1984.

Next week's featur The Northern Miner d metals' demand and pr tions, in articles writ cially for the issue by of leading national an tional experts in the fi

10m. tons indicated

Lacana in Nevada gold bet with 2-stage drill program

By Allan Jones

Ground in Nevada that was originally staked for its potential use in the making of cement may turn out to be a gold bonanza for Toronto-based Lacana Mining Corp.

Lacana President E. G. Thompson tells The Northern Miner the company this week, through U.S. subsidiary Lacana Mining Inc. launched a major drill program on its Relief Canyon gold property in Nevada, located about 100 miles southwest of the Pinson gold mine (in which Lacana has a 26% interest).

From previous drilling on the property by Lacana, and by what the company will only describe as a "large U.S. corporation," the Relief Canyon property has indicated large volumes of disseminated low-grade gold mineralization, in the order of 5,000,000-10,000,000 tons, grading from 0.04 to 0.06 oz. gold per ton.

Mr. Thompson says the current drill program, which will consist of about 40 holes totalling 10,000 ft., to further delineate the deposit, will take about three months to complete. "We'll follow this up, if successful, with a second-stage program of 150 holes, for another 40,000 ft. of drilling," he said.

Lacana will spend something over $1 million on both the current drilling and on the second-stage program, and at that point will have earned an approximate 90% interest in the property.

Falcon Cement Inc., a wholly-owned subsidiary of Falcon Construction Corp., of Auburn, Calif., holds a 10% interest. It was Falcon Cement which first staked the ground with the intention of using its contained limestone in cement-making operations.

The Lacana president says the whole Relief Canyon situation, including attitude of the mineralized zone, and the topography of the area, favors an open pit mining operation, with a low stripping ratio.

He said preliminary metallurgical test work indicates that the gold is amenable to extraction by either conventional cyanide milling, or by heap leaching.

If the company goes the latter route, after defining the orebody, Mr. Thompson said, it would likely

See Page 2

Big buyers buying

New peaks for gold, s

By S. Paul Rana

The bulls are in the driver's seat. And the bears, who had gone short in the precious metal markets, are today licking their wounds. Prices of gold, silver and platinum have gone up sharply, of course.

Breaking through another psychological resistance level of $US490 during the review week, gold at presstime was hovering around $US510 per oz. A decisive penetration of this barrier, according to many seasoned analysts, could very well push gold to a new $550 per oz. hurdle.

Silver, which has gone up by 31% since the beginning of this year, has outperformed the yellow metal. At mid-week, silver was trading around $US14.50 on the Toronto spot market with near-term target now placed around $US18 per oz. Silver has reduced its ratio to gold to around 35 to one from about 41 at the beginning of January.

The strongest of all, however, was platinum, which during the review week advanced by $45 compared to a gain of $27.60 for gold. This helped to further close the gap between gold and platinum to around $12 per oz. Traders now believe that platinum will soon recover its traditional price premium over gold which it had lost back in 1981.

All the three precious metals are receiving strong speculative and investor support while silver and platinum, in addition, are being influenced by expectations of strong consumer demand ahead as economies of the western world come out of their worst recession

Bullish sentiments receive strong suppor financial crisis which i ing the international b munity. Despite all th the International Mon and Washington, noth faced so far to suggest for the better.

The crisis, in fact, h further with failure o World oil producing c agree to maintain thei price. The internationa

See Page 13

Equity Silver e move ahead sn

VANCOUVER — E Mines reports that inc volume and improved in the fourth quarter mo set losses sustained in t months. Earnings for the Dec. 31, 1982, were $6 73¢ per share after provi ferred share dividends o In 1981 earnings were $ 52¢ per common share sion for preferred share $5,661,000.

President C. Allen 62% increase in conce was largely offset by l prices during the first ni Lower interest rates a debt resulted in low expense.

The company's share for trading on the To Exchange this week.

CHEAPER AT A PRICE
We may be happy at the gas pumps, and in the home with heating bills, but falling oil prices could also have some very undesirable effects ..... Page 4

MORE HOTTING UP AT HEMLO
Literally dozens of companies are now active in the Hemlo gold area, and developments by many of them can be found on ..... Page 20

GOING, GOING — GONE?
Craigmont Mines plans to auction off its mining assets this summer, as well as its oil and gas properties ..... Page 21

Northern Miner Feb. 3, 83

Wedge Lake venture and the nearby Weedy Lake project, which is operated by Golden Rule.

The breakdown of ownership for the Wedge Lake project is Giant Yellowknife, 25%, Golden Rule, 25%, and SMDC, 50%. For Weedy Lake, ownership is shared 50-50 by Golden Rule and SMDC.

Open pit potential

Recently, Giant Yellowknife completed a 1,350-m drilling program at Wedge Lake to test a complexly folded, banded, sedimentary iron formation that appears to strike for at least 3,500 m and indicates widths of from four to more than 20 m.

The results of the 11-hole Giant Yellowknife program showed gold values ranging from 0.08 oz. per ton over two m to a high of 0.68 oz. per ton over one m — all at relatively shallow depths. Drilling to date has established a mineralized strike length of about 500 m open to the east and west along strike and also to depth.

"We feel there's good potential for opening up a (minable) open pit tonnage . . . we're looking at a possible blend of open pit and underground here," says Mr. Harper, who compares the area's geology with that found in the Val d'Or-Bourlamaque field.

At Weedy Lake, located about two miles from Wedge Lake, Golden Rule says that results of a "first stage" drilling program have indicated the potential for a large-tonnage open pit deposit. Of two zones — A and B — on this property, the B zone is the one commanding the attention of Golden Rule and partner SMDC.

Five holes drilled late in 1982 on the B structure over a strike length of 300 m indicated a mineralized zone of about 25 m wide, with values averaging 0.1 oz. gold per ton. "Within this zone is a higher grade core averaging 0.15 oz. to 0.25 oz. gold per ton over widths of two to 13 m," adds Mr. Harper.

Five holes were also drilled on a high grade surface showing on the A zone, with resulting values ranging from 0.2 oz. gold over seven m to 0.72 oz. gold over 0.25 m.

Three other holes drilled on three separate geochemical anomalies on the Weedy Lake group also returned gold values ranging from 0.09 oz. over two m.

Starting this month, a further program of 2,000 m of drilling will be undertaken at Wedge Lake, while a 1,000-m drilling program is also scheduled to be under way this month at Weedy Lake, with a followup program of drilling likely later in the year.

Phoenix Oil Canada and SMDC are joint partners in the SMDC-operated Star Lake project. Phoenix recently reported that a 1,100-m drilling program is planned for the Star Lake property for early in March "to test for possible extensions of the newly discovered zone of high-grade gold mineralization."

(Phoenix directors have approved the spinning off of the company's interest in the Star Lake property to a new company, Starrex Mining Corp.)

A spokesman for SMDC confirmed to The Northern Miner that drilling at the Star Lake prospect will resume in March. However, he described work under way by SMDC on the large Neyrinck Lake project, which encompasses Star Lake, as being at the "grass roots stage."

Lacana in Nevada program

From Page 1

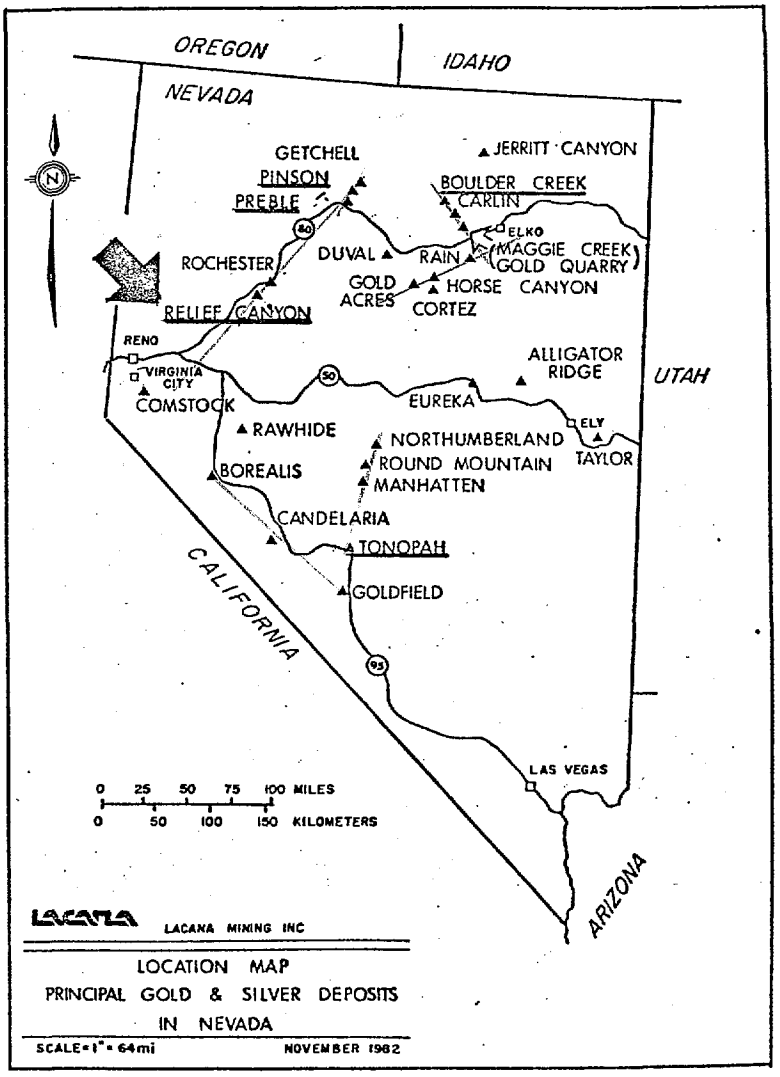
do a 5,000-ton heap leaching test pad, a process which in itself could take some 3-4 months to complete.

"The choice of heap leaching or conventional milling of course simply boils down to a dollars-and-cents decision," he commented.

Other factors in favor of the property, which, he said, "you are going to be hearing a lot about," are its proximity to water, power, and labor. It's about 100 miles northeast of Reno, Nev.

Lacana Mining is currently in a strong working capital position ($8,400,000 as at Sept. 30/82), and is essentially debt free. Mr. Thompson said the work now going on at the new discovery area in Nevada is part of a general expansionary thrust by the company in the U.S., both in mining, and in the oil and gas area.

Lacana has a 29 ⅓ interest, along with Rayrock Resources (29⅓%), and Dome Explorations (29⅓%), in the Cordex IV Syndicate gold discovery in Nevada, to the east of the Pinson mine. The Cordex IV is another open pit minable situation, on which a production decision could come early this year.



Map shows location (arrow) of the new gold discovery by Lacana Mining on its Relief Canyon property in Nevada. The Pinson Mine, in which Lacana holds an interest, is to the north and east, and the gold discovery of the Cordex IV Syndicate, in which Lacana also holds an interest, is at the Boulder Creek property east of the Pinson. Lacana has an interest in the two other underlined properties, Preble and Tonopah.

Iron Ore benefits

Iron Ore Co. of Canada is reported to have instituted a $10 million benefits package for the 600 workers laid off by the closing last November of the company's Schefferville, Que., mine.

The settlement is estimated to cost the company an average of more than $15,000 per man. About 20% of the $10 million is said to cover payments the company is obliged by union contract to give workers permanently laid off.

3/15/83

FROM: F. T. GRAYBEAL

TO: Kurtz, Sell, Vikre, Balla, Perkins, Smith, Harper, Gale, Gray, Nicholls, Courtright, GOLD

If you ever start taking flack from the environmentalists about the uselessness of gold and absence of need to mine it, maybe some of the "new" potential uses can fuel your rebuttals, as described on the attached.

I heard Au was a potentially important component of window glass for thermal insulation, but have seen nothing recently.

RECEIVED
MAR 21 1983
S. W. U. S. EXPL. DIV.

New technological and industrial applications could offset falling gold fabrication demand

The 23% decline in world industrial gold consumption over the past five years may be arrested and even reversed in the next five years through the development and commercialization of several new industrial and technological gold applications, according to Arthur Tasker, general manager of Intergold's US industrial division. Industrial gold usage accounted for about 6.4-million oz, or some 20% of world gold offtake, last year, down from the 8.4-million oz (32% share) consumed industrially in 1978. Tasker blamed much of the falloff in demand on the constraints imposed on industry by recession and gold's price volatility, and on the increased substitution of cheaper metals with properties similar to those of gold. (He excluded the use of gold in jewelry and coinage from the industrial gold usage category.)

Tasker revealed a number of new industrial gold processes for application in the dental, medical, electronics, decorative, and other industrial sectors in a speech given on Mar. 1 to the annual meeting of the Gold Institute in Santo Domingo. Not only would these new uses put a stop to the drop in world industrial consumption of gold, he noted, but they likely would utilize as much as 800,000 oz of gold additionally each year in the US alone, where they would first be applied.

The greatest decline in US industrial gold usage has taken place in the electronic connector industry, which Tasker said used 418,000 oz last year, down 30% from the industry's 1981 offtake. Tasker pointed out that more connectors were produced last year than ever before, which offset to some extent the decline in gold used per unit through miniaturization and substitution. He also foresaw the removal of 70,000 oz of gold from military equipment connectors by 1987, when an ongoing four-step gold reduction program in that sector will be completed.

But up to 350,000 oz of gold annually may soon be used in the manufacture of silver-cadmium batteries. These batteries' performance will at least double if 15% of their content is alloyed with gold, Tasker noted.

And in the medical and dental fields, Tasker dwelt upon two key developments which promise to expand gold use by roughly 200,000 oz annually within the next three years. Smith-Kline Beckman's new drug, Ridaura, contains 29% gold by weight and will be used to combat rheumatoid arthritis once FDA approval, expected by this fall, is granted. A normal dosage for the 6.5-million rheumatoid arthritis sufferers in the US alone would amount to 150,500 oz of gold consumed yearly.

The second development Tasker mentioned was a method of making dental crowns by an electroforming process that uses a thick deposit of electroplated gold. He observed that this innovation could halt the substitution of base metals and of palladium for gold in dentistry, where demand has fallen to 418,250 oz in 1982 from a 780,000-oz peak usage in 1978.

Finally, Tasker presented a new consumer concept: a phonograph record bearing a .995 fine gold coating over its vinyl surface, which he said improves wear resistance by a factor of 10 and is also dustproof, owing to gold's antistatic properties. A leading European industrial concern, funded 50% by Intergold, has developed the record, which has 0.008 oz of gold per disk and which Tasker anticipates will increase gold consumption by 250,000 oz or more annually.

Elsewhere in precious metals . . .

Benguet liquidated 21,000 oz of gold from its hedging stocks last December, leaving only 2,000 oz, as gold prices rose above $450 per oz and the company's primary gold operations returned to "satisfactory profit margins." Benguet president Jaime Ongpin said the gold trading portfolio would be replenished once market levels enable the company to "lock in" at least 50% of its output at a $100-per-oz profit. Ongpin noted that the expansion and renovation of the Balatoc mill had brought primary gold production costs down to $353 per oz by yearend from $449 in December 1981. Benguet produced 238,356 oz of gold and 191,904 oz of silver from its primary and byproduct operations last year, compared with 214,212 oz of gold and 230,520 oz of silver in 1981.

Johnson Matthey USA acknowledged that it has been selling Rustenburg's rhodium and iridium at free market levels since mid-February, as it has sold the South African producer's platinum and palladium since mid-January (MW, *Jan. 24, p6*). "We've merely formalized an existing situation," a spokesman said, adding that current offering prices for rhodium are in a $280- to $290-per-oz range, while those for iridium are between $295 and $305 per oz. Johnson-Matthey and Impala spokesmen concur that a slight recovery in demand for these metals is underway, though manufacturers have adequate stocks to draw on for some time before increased automotive and housing demand filters through to them.

Major Japanese dealers have closed their Soviet palladium contracts covering imports from March through December. As reported earlier (MW, *Feb. 21, p7*), the Soviets did not even hint at any changes from last year's terms.

Bearish oil price news caused precious metals prices to tumble last week, as traders stood by and watched speculators sell. Nearby Comex silver closed at $13.38 per oz on Feb. 24, it lowest level in a month. Comex spot gold toppled $25.70 per oz on Feb. 22, then steadied by Feb. 24 at $469.90 per oz. New York dealer platinum quotes fell $26 from their previous levels to $447-450 per oz, while palladium was sold at $116-119 per oz, down from $127-129. Rhodium firmed to $290-295 per oz, up $6, while ruthenium rose by $1.00 to $26-28 per oz. Iridium and osmium dealer tags remained unchanged.

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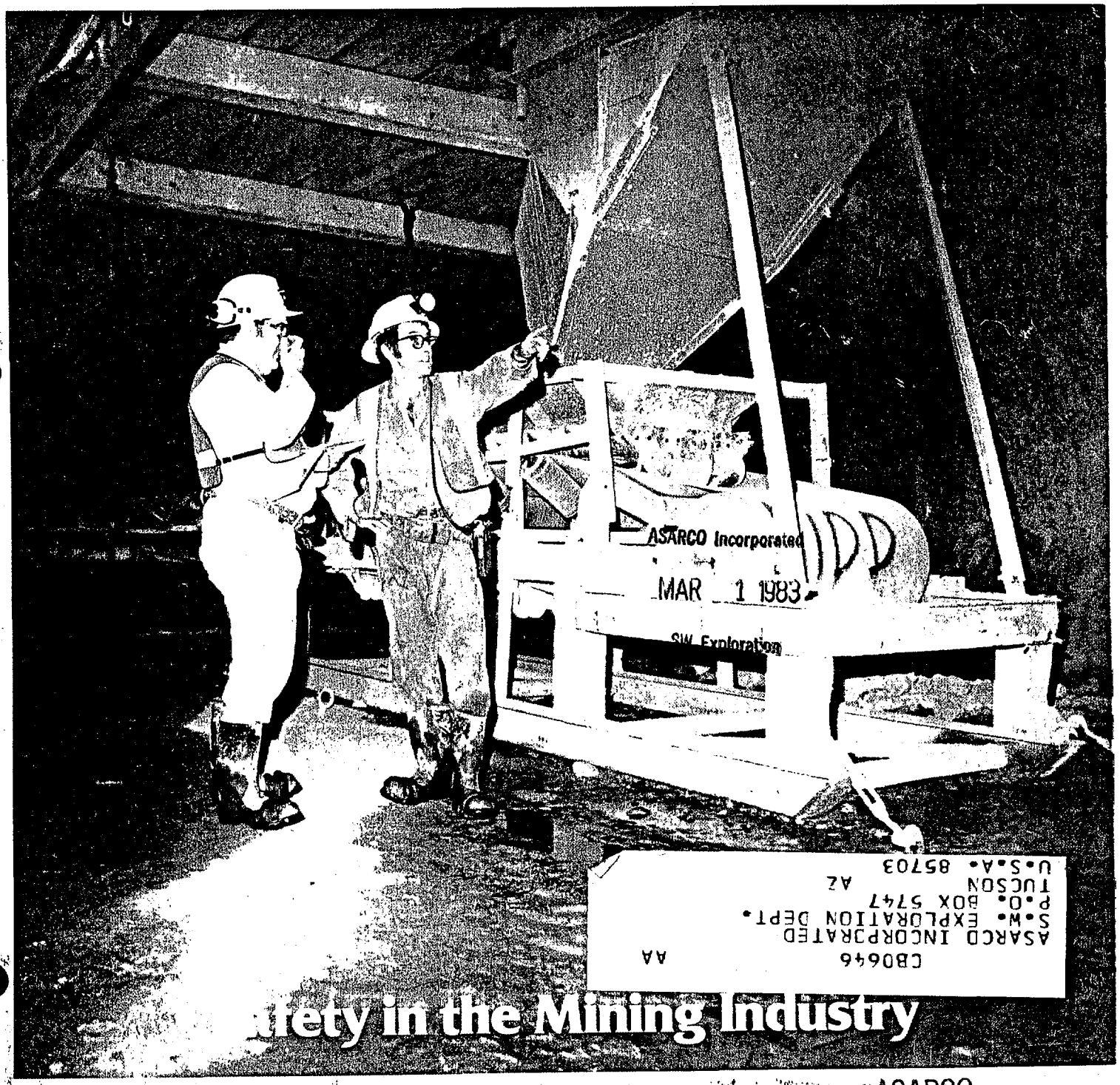
THE CANADIAN MINING AND METALLURGICAL BULLETIN

Abitibi Gold Deposits



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Heap leaching— a low-cost recovery method for small gold deposits in northwestern Ontario

R.C. BEARD
Mineral Resources Coordinator, NW Region
Ontario Ministry of Natural Resources
Kenora, Ontario

ABSTRACT

Heap leaching, currently being applied in the treatment of low-grade ores in the southwestern United States, is shown to have potential for use in northwestern Ontario. The history and operation of the process are described, and cost comparisons are given.

Introduction

Heap leaching is a simple, low-cost method of recovering gold from low-grade ore. Although it is not currently being used in Canada on a production scale, it is being used extensively, and successfully, in the southwestern United States as well as elsewhere in the world. Especially suited for large deposits of very low-grade ore, it also appears to be practical for use on small, high-grade deposits because it avoids the high capital costs of mills. Eveleth (1980) and others suggest that the capital and operating costs required for heap leaching are approximately 20% and 40% respectively of conventional milling methods.

Northwestern Ontario is noted for its large number of undeveloped, small, high-grade gold deposits. Although there are a number of larger and higher-grade deposits in the area as well, it is the relatively small undeveloped deposits which would benefit most from heap leaching, if it could be demonstrated to be a viable extraction method.

To assess the significance of these small deposits, the Ontario Ministry of Natural Resources recently commissioned a consultants' study of the gold reserves in certain parts of northwestern Ontario. This report, titled "Feasibility of Small-Scale Gold Mining in Northwestern Ontario", by J.N. Neilson and R.C.E. Bray (1981, OGS Open File Report 5332), documents 106 known gold deposits having sufficient information to calculate gold reserves. A total of over 7.8 million tons having a weighted average grade of 0.19 oz of gold per ton were compiled in the proven, probable and possible categories, with an additional 8.2 million tons in the speculative category. Many of these deposits could be mined by open-pit methods.

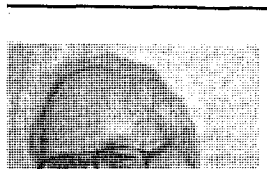
Because of their apparent small size, most of these deposits, despite their relatively high grade, are not being developed at the present time. They simply cannot justify the high capital investments required to install individual mills at each of the sites. If they are ever to be developed, ways must be found to avoid or minimize the high up-front mill costs.

Recognizing this problem, the minerals staff of the Ontario Ministry of Natural Resources has, over the past several years, been actively investigating ways to encourage and assist in their development. One obvious solution to the problem would be to assure the availability of custom milling facilities. The Ministry has recently responded to this need with the GOMILL program, which provides financial assistance to the private sector to establish facilities for bulk sampling purposes and the milling of small lots.

Another concept that might also facilitate the development of small gold deposits is heap leaching.

General Description

Simply stated, heap leaching is a process whereby broken or crushed gold ore is stacked on an impervious pad. Dilute sodium cyanide solution (0.05-0.1% cyanide) is applied to the top of the heap and as it percolates through the heap, the finely disseminated precious metals are dissolved. The pregnant solution, enriched in gold, is collected off the base of the heap through a system of ditches, into a pond. From the pond, the pregnant solution is pumped through a series of carbon tanks or columns where the precious metals are adsorbed on the carbon. The barren solution is then re-circulated back onto the



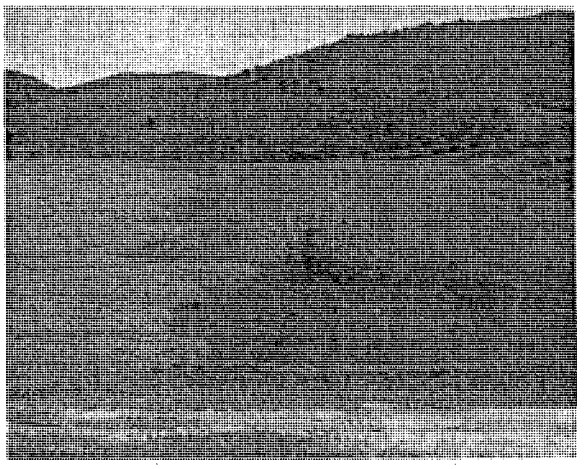
R.C. Beard

Dick Beard obtained his B.Sc. and M.Sc. degrees in geological engineering from Michigan Technological University in 1955 and 1966 respectively. Prior to joining the Ministry of Natural Resources in 1973, he was employed in the private sector in mineral exploration, working throughout eastern Canada and the United States. These positions in exploration included project geologist for Pickands Mather & Company in Minnesota and Anaconda American Brass Co. Ltd. in Noranda, area geologist for H.K. Porter Co. (Canada) Ltd. in Thetford Mines and regional geologist for the Mining Division of Amoco Canada Petroleum Co. Ltd. in Toronto.

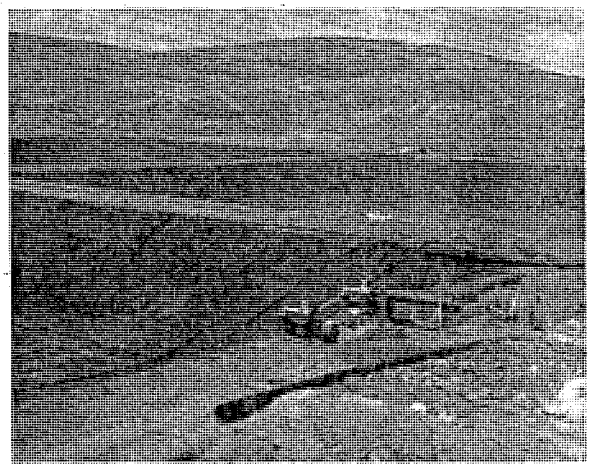
His recent activities and interests with the Ontario Ministry of Natural Resources, where he is mineral resources coordinator, Northwestern Region, include projects and research dealing with small mine development and custom milling in northwestern Ontario, the use of peat as an energy source, a gold heap leaching demonstration project, the establishment of an historical mining site in the Kenora-Dryden area, and the establishment of a building and ornamental stone industry in northwestern Ontario.

He is a member of the CIM and the Association of Professional Engineers of Ontario.

Keywords: Leaching, Heap leaching, Gold deposits, Mineral processing, Porosity, Column leaching, Sodium cyanide, Cyanidation.



Impervious pad under construction—Western Windfall Ltd., Eureka, Nevada.



Solution ponds under construction—Western States Minerals Corp., Elko, Nevada.

heap for further leaching, in an entirely closed circuit. The loaded carbon can be shipped away for stripping or can be stripped on site in a tank using hot caustic soda solution. The gold and silver are recovered from the hot caustic solution electrolytically, followed by smelting, producing doré metal.

Purpose

The purpose of this paper is, firstly, to provide a general overview of the heap leaching process. More importantly, it will attempt to assess the feasibility of using this method in northwestern Ontario, on Archean gold ores.

History

The heap leaching process is certainly nothing new. The concept dates back to about 1750 when acid solutions were percolated through large heaps of copper oxide ore on the banks of the Rio Tinto in Spain. Since then, the process has been used extensively throughout the world, largely to leach copper and uranium ores. Cyanide heap leaching of *gold* ores, however, is comparatively new. This concept was first developed by the U.S. Bureau of Mines in the mid 1960s. Since that time, the Bureau has carried out extensive test work on various types of ore, mostly from the western United States, and it has been largely through their experimental work and enthusiasm that the process has caught on in the western United States in the last decade.

Heap leaching has not been used to any extent in Canada to date. There have been some attempts at heap leaching of copper and uranium ores, but none of these attempts had apparently been very successful. The only cyanide heap leaching known to have been carried out on gold ores took place on a small scale in southeastern Ontario in 1980, but the results of this work have not been published.

This general lack of earlier success with heap leaching in Canada has resulted in an obvious lack of enthusiasm by the Canadian mining industry for further investigation of the process. Only three or four Canadian companies are known by the author to be carrying out heap serious leach tests on Archean ores.

Description of the Heap Leach Process

The following description of the heap leaching process is largely based on a fact-finding trip to Nevada, where six mining properties employing heap leaching were visited. These were the properties of Candaleria Partners, Smoky Valley Mining, Western State Minerals Corp., Carlin Gold Mining Company, Cortez Gold Mines and Western Windfall Limited. These operations range in size from very large (10,000 tpd) to relatively small (1,000 tpd). All are basically gold producers except Candelaria, which is a silver producer with only minor gold. At the Carlin Mine, approximately half of the production is treated in a conventional mill, with the remaining low-grade ore (2,500 tpd) recovered by heap leaching.

The mines described are scattered throughout central Nevada, mostly situated along the flanks of low mountain ranges of the Basin and Range Province. Significantly, the elevations of these operations range from 5700 to 7600 feet.

The gold ores currently being heap leached in Nevada are generally very low grade, by any standards, being in the range of 0.02 to 0.05 oz per ton.

These low-grade gold ores are typically highly altered and oxidized metasediments, including such rock types as siltstone, limestone and dolomite, bedded shales and cherts, volcanic ash and skarn. Ore at Smoky Valley is, however, significantly, different in character, consisting of hard rhyolitic ignimbrite, which is quite similar to many of the fresh "quartz porphyries" of the Ontario Archean. Mineralization is of hydrothermal origin, typically crosscutting lithologies and stratigraphy. The gold is finely disseminated in all cases, running in the 2- to 5-micron size range.

At all properties visited, mining is from open pits. As the grades are very low, they simply cannot stand up to high underground mining costs.

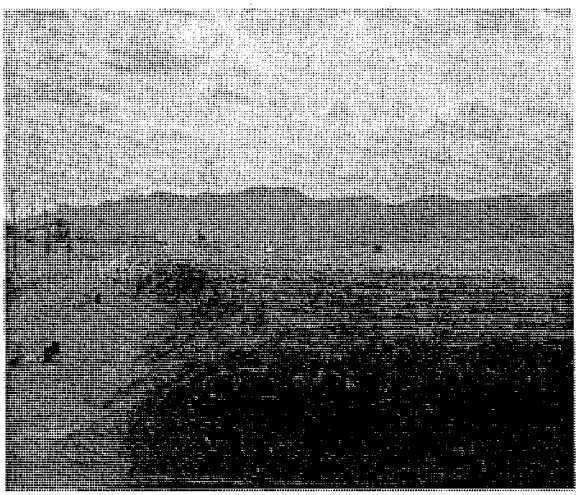
At three of the properties, pit-run material is leached directly, without crushing. These ores tend to break easily, producing a high proportion of sand-size fines. In some cases, a very wide size range was noted on the heaps, with some blocks running in excess of 3 feet. At three of the properties, ore is crushed before leaching. One property crushed to 90%-⅜ inch; another crushed to -1 inch.

Only one property uses agglomeration. Here, a caustic soda solution is sprayed onto the crushed ore as it comes off the conveyer belt, causing the fines to adhere to the coarser ore.

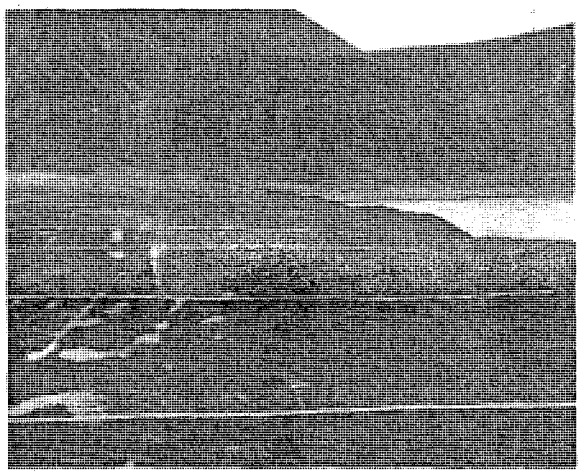
Basically, three types of pad construction are utilized. The first type, which is reuseable, employs a layer of asphalt layed down on a graded and compacted soil base. Asphalt thickness ranges from 4 inches to 1 foot. A second and perhaps the simplest type of pad consists of compacted clay. These impervious clay pads range from 5 to 18 inches in thickness. A third type consists of compacted clay or soil covered by an impervious plastic membrane. Whatever type of base is used, a 4- to 18-in. layer of coarse gravel or crushed rock is usually laid down on the base of each heap to protect the pad and also to act as an aquifer.

Pads are sloped at 2 to 4 degrees in two directions to provide for drainage of pregnant solution into plastic-lined ditches along two sides of the heaps.

Individual leach pads vary considerably in size. The Candaleria pad was one of the largest seen, and is probably one of the largest in the world. It is about 4200 ft long by 1000 ft wide



Gold ore heaps being leached—Smoky Valley Mining Division, Copper Range Company, Round Mountain, Nevada.



Sprinkling system distributing dilute cyanide leaching solution to top of silver ore heaps—Candelaria Partners, Hawthorne, Nevada.

and will contain 12 individual heaps, each approximately 1000 ft long by 300 ft wide. At most properties, however, a number of smaller pads are used. Carlin uses 5 pads, each being about 300 ft by 150 ft and containing 16,000 tons of ore.

Ore is typically dumped from trucks directly onto the pads and then carefully piled using front-end loaders or bulldozers. Formation of the heap is perhaps the most critical point in the whole heap leach operation. Care must be taken not only to prevent damage to the pad, but also to form the heap in such a manner as to provide maximum permeability and solution percolation, with minimum channelling. Most heap leach failures to date have reportedly been due to poor materials handling rather than to any chemical or metallurgical problems.

Individual heaps vary in height from 10 feet to 30 or 35 feet.

Once the leach cycle is completed, ore may be left on the pad and a new pad started elsewhere, or it may be removed from the pad and replaced by a new heap.

An alternative system, such as the one used at Western Windfall, is to stack heaps, one on top of the other. In some instances, new material is placed directly on top of the old heap, with the leaching solutions allowed to percolate through the full thickness of both stacked heaps. A related technique involves preparation of an entirely new heap on top of the previous one, with an impervious layer laid down between the two heaps. Stacking is useful where pad space is limited or there is a need to maximize recovery.

Solutions are delivered to the pads using centrifugal pumps and plastic pipes. At most the properties, solution is applied to the heaps using various types of sprinklers, either the rotary type or Bagdad Wigglers. Application rates vary from about 100 gallons per minute at some of the smaller operations to 16,000 gallons per minute.

At Western Windfall, a system of ponding and slow percolation is used in place of sprinklers. With this method, a number of small ponds are hand constructed on the top of each heap. A 3- to 4-ft hydraulic head of solution is constantly maintained in each pond.

Pregnant and barren solution ponds are usually located near the base of the pads and range in capacity from 20 million gallons for some of the larger operations to slightly over 1 million for some of the smaller ones.

Total leach cycles generally vary from 30 to 90 days. At Carlin, a 30-day cycle is maintained, including a 14-day leach period. Smoky Valley, which utilizes a slot advance technique, has a 48-day cycle which includes 40 days of leaching, 2 days of washing, 2 days of draining, and 4 days set aside for loading and unloading. At two of the other properties there seemed to be no fixed leach cycle and the stacked heaps were leached for as long as gold values could be economically recovered.

Theoretically, gold can be recovered off a heap for as long as it is leached. A plot of recovery vs time would usually show maximum recovery occurring several days to a week after start-up, with a gradual decrease thereafter. Thus, it is an economic decision as to what point it is more profitable to get rid of the old heap and start leaching a new and fresh heap.

Pregnant solutions collected off the base of the heap are passed through multiple-stage carbon column systems, where the gold is adsorbed onto the carbon. Four- or five-stage systems are usually employed, using gravity flow between stages. Individual stages are connected in series, and are either semi-detached or contained all in one unit. Each tank or stage contains anywhere from 150 to 3,000 lb of carbon. At Western State, the carbon adsorption unit consists of two vertical steel columns approximately 40 ft high, each containing 4 stacked stages.

The carbon generally loads anywhere from 100 to 150 oz Au per ton.

Carbon from the first stage is removed every 1 to 3 days, and the carbon from the other tanks is moved forward in the system by eduction. Following stripping, the carbon is regenerated in a kiln and/or treated with acid if necessary, then reintroduced back into the last stage.

Gold is removed from the loaded carbon in stripping tanks. The simplest stripping technique utilizes a solution of hot caustic soda (1-5% NaOH) and sodium cyanide (0.5-1.5%) in a non-pressurized tank. Temperatures are maintained at about 190°F. This simple and relatively inexpensive method requires a stripping period of 3 days, and is somewhat slower than other more sophisticated techniques. These other techniques use a pressurized hot caustic system maintained at 220°F, or a combined hot caustic, cyanide and alcohol system, which strips the carbon in less than 12 hours.

The highly loaded solutions from the stripping tanks are then passed through electrolytic cells having steel wool as cathodes. Gold plates out on the steel wool cathode, which is smelted in crucible furnaces, producing doré metal.

Recoveries at the Nevada operations reportedly range from 50% to 95%, with most falling in the 50% to 70% range.

The operations described above are mega projects compared to the type of projects envisioned for the typically small, high-grade deposits of northwestern Ontario. It is necessary, therefore, to scale the various processes down in one's mind.

Potential Use in Northwestern Ontario

Looking now at the potential for using the heap leach process in northern Ontario, two factors are usually cited as evidence

TABLE 1. Gold and silver assays of test samples

Sample No.	Au oz/ton	Ag oz/ton	Sulphides
1	0.040	0.21	2-4%
2	0.57	1.54	±50%
3	0.022	0.28	<1%
4	0.094	tr.	5-10%
5	0.567	0.47	10-15%
6	0.042	0.26	<5%
7	0.078	tr.	<1%
8	0.605	0.31	<1%
9	0.035	0.32	±15%
10	0.575	tr.	<2%
11	0.600	tr.	10-15%
12	0.140	0.16	<1%

Range—0.002 to 0.600 Au oz/ton

that the process will have little application here. Firstly, it has been suggested that, because of the physical characteristics of typical Archean ores, they will not be amenable to leaching. (Archean ores are supposedly hard and dense, with poor penetration, compared to the more oxidized and permeable hydrothermal ores of southwestern United States.) Secondly, it is suggested that the cold temperatures and harsh climatic conditions in northern Canada will so severely restrict leaching time as to preclude any economical operations on a production scale. The following will attempt to demonstrate that, although both of these factors are of obvious concern, neither will be insurmountable, and that heap leaching does indeed have potential application in northern Ontario for some ores.

The primary concern is whether typical Archean gold ores are heap leachable. This question had to be addressed before proceeding with any further research in this direction. A test program, carried out jointly by the Ontario Ministries of Natural Resources and Northern Affairs, was initiated in late 1980 to establish this fact. Twelve 50-lb samples of gold mineralization, representative of a variety of gold ores from northwestern Ontario, were collected and submitted to the Ontario Research Foundation for bench test work. Samples varied from hard quartz vein material to highly sheared and schistose material containing lesser amounts of quartz. Sulphide content varied from negligible amounts to about 15%.

The test work carried out by the Ontario Research Foundation included a full evaluation of the leachability of the ores by heap leach cyanidation. This test work included assaying, porosity studies, mineralogical studies, column leaching tests and conventional cyanidation studies.

The details of this test work will not be described here. These data are contained in Ontario Geological Survey Open File Report 5358, titled "Heap Leaching Metallurgical Studies on Several Ontario Gold Ores", by M.K. Witte (1981). A very brief summary of the test work carried out, and the more important results, will, however, be given.

As shown in Table 1, assay values for the 12 samples tested ranged in value from about 0.022 to 0.60 oz Au/ton.

Following assaying, representative 3-inch pieces from each sample were subjected to porosity tests. Of the 12 ores tested, five showed good to excellent penetration, and were subjected to further studies. These were largely the schistose ores. Of the 7 samples rejected, because of poor penetration results, 6 consisted largely of relatively unfractured and unaltered quartz vein material. The seventh rejected sample was somewhat schistose, but highly silicified, felsic tuff. Its strong silicification probably accounted for the poor penetration results.

Before column testing was carried out, the mineralogy of the 5 selected samples was studied. No discrete gold or silver minerals were observed in the samples, except for number 5. As might be expected, the amount of gold reported in the assays seemed to correlate with the quantity of pyrite occurring in the samples.

Next, the material from the 5 selected samples was crushed to -½ inch and placed in 4-inch glass columns for the column leach tests. The column tests lasted for 75 days or 1800 hours,

TABLE 2. Results of column leach tests

Sample No.	Head Assay (Au oz/t)	Au Recovery (from residue analysis)	Au Recovery (from solution analysis)
1	0.04	75 %	80.6%
4	0.094	22.3%	10.4%
5	0.567 (0.81)	good	good
11	0.60	91.5%	48.1%
12	0.14	97.1%	80.5%

with the cyanide solutions (0.1% NaCN and 0.1% $Ca(OH)_2$) actually pumped through the tubes for 18½ days of this total time.

No carbon adsorption technology was utilized in the test work, and the solutions were simply recycled back to the column.

The results of the column leaching, in terms of recoveries, are shown in Table 2. Gold recoveries calculated on both residue and solution assays are shown.

Samples 1 and 12 consisted largely of highly sheared mafic to intermediate volcanic rocks containing 5 to 10% quartz veinlets and only minor sulphides. Both of these samples showed good recovery rates, and relatively good agreement between the two calculation methods.

Samples 4 and 11 were also highly schistose or sheared, consisting of varying amounts of quartz vein material and carbonate, in more felsic tuffs or volcanics. Both contained moderate amounts of carbonate as well as significant amounts of pyrite. Sample 4 also contained arsenopyrite, and the very poor recovery rates indicated for this sample appear to stem from the presence of gold locked up with the pyrite and arsenopyrite.

Gold recovery from sample 11 is good. As will be noted, the solids and solutions do not balance well. The solution was full of slimes and it is felt that the dirty solutions contributed to poor solution analytical results. Thus, the 91.5% recovery from the residue analysis is probably the more reliable of the two figures.

Sample 5 consisted of 90% white to smokey quartz material in a granite host rock. The sample contained approximately 15% sulphides, including pyrite, galena and chalcopyrite. The sample also contained a significant quantity of free gold, which made it difficult to obtain a representative head assay and prevented an adequate balancing of the test work. Despite the difficulty in evaluating the test work, the report suggests that a gold extraction of 85 to 90% might be obtained by heap leaching techniques on this particular ore.

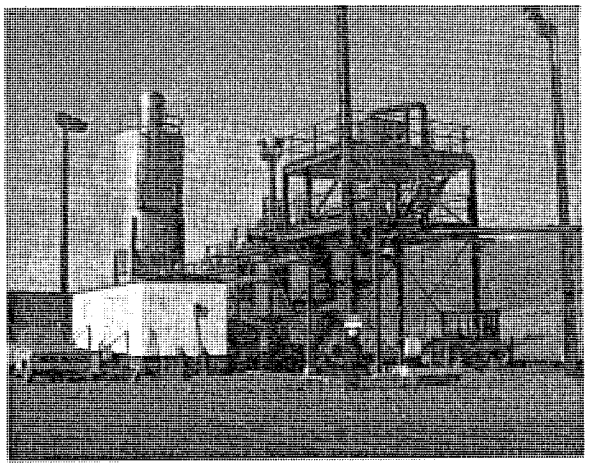
The over-all results of the test work are summarized in the Ontario Research Foundation report (Witte, 1981) as follows: "The heap leaching test work to date on the 5 ore samples selected produced results which were very encouraging. All ore samples, except number 4, produced recoveries which were acceptable for potential heap leaching projects. Based on these results, it is recommended that further work to study the feasibility of heap leaching by undertaken."

Thus, it would appear from the test work described that indeed at least some Archean gold ores from northwestern Ontario are amenable to heap leaching.

Looking now at some of the other factors that might tend to restrict or prevent the use of heap leaching in northern Ontario, it is helpful to make comparisons between northern Ontario conditions and experiences in Nevada.

The cold temperatures and harsh climate in northwestern Ontario are obviously a major concern. Cold temperatures not only cause freeze-up problems with plumbing systems, but also tend to decrease the kinetics of cyanide extraction. The question is, just how serious are these factors.

Although winter temperatures in Nevada are obviously not as severe as in northwestern Ontario, it is significant to note that most of the heap leach operations in that state are located at relatively high altitudes. Prolonged cold periods are not uncommon at these elevations, -23°C for 10 days at a time not being unusual.



Five-stage carbon adsorption plant—Carlin Gold Mining Co., Carlin, Nevada.

Large leaching pad, 4200 ft long by 1000 ft wide, able to produce 10,000 tonnes of silver ore per day—Candelaria Partners, Hawthorne, Nevada.

The usual procedure in Nevada is to simply turn off the plumbing whenever the temperatures drop below freezing. Several of the operations at lower elevations are able to operate straight through the winter months in this manner.

At Smoky Valley Mining, the mine has installed a solution heating system which is used, on and off, for about 3 months a year.

At Western Windfall, where temperatures are especially cold, the mine has gone to the ponding system described earlier. This system avoids many of the plumbing freeze-up problems. Solutions can be applied to the heap even though the outside temperatures are below freezing—at least until such time as the ponded solutions on the top of the heaps freeze up completely. This technique would certainly warrant investigation in any application in Ontario.

As to the effect of cold temperatures on the leaching rate of the cyanide solutions, it has been reported that this is not a significant factor, and even at near freezing temperatures leaching can be effectively carried out, although at a somewhat reduced rate.

Thus, it appears that despite the cold temperatures and shutdowns, even for prolonged periods, these very low-grade leaching operations in Nevada can operate at a profit. Two of the operations reportedly shut down operations completely for 3 or more months during the winter.

In northwestern Ontario, it might be reasonable to expect that any heap leaching operation might have to shut down completely for at least 4 months of the year, with additional intermittent shut-downs during the freeze-up and break-up periods. Mining could continue during the winter months, with the ore either stockpiled or placed directly on pads ready to start up in the spring. Careful scheduling of the total operation would obviously be required.

One of the main problems in the southwestern United States, and in some cases the limiting factor on production, is a limited water supply. Due to high evaporation rates in these dry climates, make-up water has to be continually added to the system.

This will obviously not be the case in northwestern Ontario, where one of the major problems could well turn out to be too much water. Heavy rainfall could, over short periods of time, overload the plumbing system, resulting in overflow from the solution ponds. Careful planning and special safety features will be required to avoid these problems. Solution ponds and pads will have to be constructed in such a manner as to minimize run-off. Solution ponds can be overdesigned to handle most overloading, and/or overflow ponds can be constructed down-slope from the heaps and the ponds, to provide a safety factor for emergency situations. If the economics are favourable enough, it might even be feasible to construct some sort of low-cost cover over the pads and ponds.

Serious dilution would obviously require additional cyanide and lime, which might, under extreme circumstances, lead to excessive chemical costs. The use of cyanide may, in itself, be a problem, especially if used in more populated areas. This may be especially true for the first one or two projects carried out in any given area. Other than the fact that it forms a deadly gas under certain circumstances, the general public knows little about the chemical characteristics of cyanide. Few people know that it breaks down quickly in sunlight, or that there are simple means available to break it down even more quickly in emergency situations.

These are all very valid environmental concerns, and they will have to be addressed and explained to the public in any contemplated heap leach operation.

At the Nevada properties visited, environmental concerns were closely monitored, including having one or more test wells located down-slope from the operations. It is encouraging to note that no serious environmental problems were reported at any of the properties visited.

Costs

The question of the costs of heap leaching methods is, of course, a critical one. It was noted at the start that estimates from the literature suggest capital requirements for a heap leaching operation to be only about 20 to 25% of the cost of a conventional cyanidation plant, and operating costs to be about 40%. While actual *operating* costs are hard to come by, some *capital* cost information was available at the Nevada operations visited. Without going into names, one 7,000-tpd operation was reportedly built at a total cost of $18 million. Another 2,500-tpd operation was constructed at a cost of $10 million.

D.W. Kappes (1980), in his excellent paper entitled "Leaching of Small Gold and Silver Deposits", has estimated that the total capital cost (excluding mining and crushing) to begin a 200-tpd operation, with ore stacked in heaps containing 10,000 tons each, would be less than $200,000 (U.S.) in 1978 dollars. In the same paper, Kappes also describes a very small "one-man" operation processing about 1 tpd on active heaps of about 60 tons. For an operation of this size, Kappes estimated that it could be started up for about (U.S.) $2,000, again probably in 1978 dollars. Although this figure does not include technical assistance, it still appears to be very low, even when converted to 1983 dollars.

It is worth noting that, for any operation being contemplated in Ontario, one of the largest costs will be for sheet plastic used to line solution ponds and pads. At one of the Nevada properties, two 1.3-million-gallon plastic-lined ponds were constructed at a total cost of about $125,000. This

averages out to about 75¢ per square foot of pond area. Heavy plastic sheeting is reportedly more expensive in Canada, and one should probably assume a cost of over $1.00 per square foot for plastic-lined ponds and pads.

The bottom line is, of course: can heap leaching operations be carried out *profitably* in northern Ontario? As with any new mining venture, many economic factors will have to be considered, and each project will have to be evaluated on its own merit. However, some generalizations can be made as to the three most important factors, i.e. tonnage, grade and recovery rates.

The scale of the operations in Nevada is considerably larger than anything being contemplated for northern Ontario at the present time. Volume treatment will obviously result in lower operating costs.

On the other hand, the grade of ores being considered for development in northern Ontario is significantly higher than that of the ores being mined and processed in Nevada, and this fact should tend to offset the scale advantage somewhat.

As noted earlier, recovery rates at the Nevada operations generally ranged from 50 to 70%. The bench test work on Ontario ores suggests that equivalent, if not better, recovery rates can be obtained, at least from some Ontario ores.

The length of time that leaching can be carried out during the year will be significantly shorter in northern Ontario than it is in Nevada, and it remains to be determined just how long this leaching period will be, and how significant the difference in leaching time will be. This could well turn out to be *the* controlling factor as to the economic viability of the process in Canada.

Summary and Conclusions

This paper has tried to demonstrate that heap leaching does have potential for use in northwestern Ontario. It has shown that, based on the results of preliminary bench test work, adequate recoveries can be obtained from at least *some* Archean gold ores. Further bench test work is obviously required on a larger variety of ore types from the area to assess this factor more fully.

Although here are a number of climatic and environmental problems that will tend to restrict its use in northern Ontario, these factors will hopefully not be insurmountable—and the process does, if nothing else, at least warrant further investigation by the Canadian mining industry.

Recommendations

The next step in the assessment of the heap leach process, in the northern Ontario context, should be a test or demonstration program, carried out by the private sector, under field conditions. A test heap of 1,000 tons or more is recommended.

The entire operation at this scale would consist, very simply, of a plastic- or asphalt-lined pad, 55 by 65 ft in area, with ore stacked 12 ft high, two plastic-lined solution ponds or tanks, three small carbon tanks approximately 3 ft in diameter, and various pumps, pipings and sprinklers. For a small demonstration operation of this size, it is suggested that, rather than trying to strip the carbon at the site, it be shipped out for stripping.

It is estimated that the total cost to carry out a test program as described, including surface mining, crushing, technical design and management, would be about $125,000, depending on the actual scale of the project and the amount of environmental test work and over-designing for safety purposes that is required.

The Ontario Ministry of Natural Resources has recently announced an incentive program to encourage and assist such a test demonstration project. With funding provided through the joint federal/provincial Northern Ontario Rural Development Agreement (NORDA), the Ministry will provide technical management and design, through a consultant, for one demonstration project. The successful proponent, to be selected by the Ministry, will be responsible for providing the ore, a suitable site, and the construction and operation of the project. The successful conclusion of the assisted demonstration project could lead to production leaching operations.

Four-stage gravity flow. Carbon adsorption plant—Cortez Gold Mine, Cortez, Nevada.

In conclusion, it is hoped that the readers will find the ideas and descriptions presented in this paper interesting and stimulating, and that they will be encouraged to investigate further the application of heap leaching of northern Ontario gold ores.

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First conference on the use of computers in the coal industry

The First Conference on Use of Computers in the Coal Industry, co-sponsored by West Virginia University and The University of Alabama, will be held at West Virginia University, Morgantown, West Virginia, **August 1-3, 1983.** This three-day conference will cover a wide range of computer applications in the coal industry, including topics on practical aspects of computer hardware, software and personnel. The conference will emphasize the practical "how to" approach, and highly technical and theoretical papers are discouraged.

Exhibitor space will be available. Organizations interested in displaying their products and services should contact the organizing committee for reservations before **March 15, 1983.**

Send abstracts and inquiries to:
Dr. Y.J. Wang, Chairman,
Organizing Committee,
First conference on Use of Computers in the Coal Industry,
213 White Hall,
West Virginia University,
Morgantown, WV 26506.

Montreal designated for international conference on the strength of metals and alloys in 1985

The seventh conference of the ICSMA series, in 1985, has been awarded to Canada by the International Committee meeting at ICSMA 6 in Melbourne, Australia, last August. The decision to designate Montreal as the site is the result of a year's planning by a committee of Canadian metallurgists who have developed an organizational plan, a theme and a budget, and have enlisted the cooperation of related technical societies and government agencies.

The ICSMA series of conferences has been devoted to advances in the understanding of the mechanical properties of metals, and the proceedings of the conferences have served as valuable reference sources.

In Montreal, ICSMA 7 will approach the traditional theme for the viewpoints of both property-microstructure and microstructure-fabrication relationships:
(1) the first includes the dependence of strength, hardness, toughness, and resistance to creep, fatigue and wear on the internal structure of the material.
(2) the second embraces the development of the desired microstructure by suitable processing, including casting, powder consolidation, hot working, cold working, thermo-mechanical and thermal treatments, welding and machining.

The sessions and keynote presentations will be organized to emphasize the link between research and industrial application. Some sessions on the final day may be devoted to discussing the impact on strength-related research of national science policy and research funding, of energy and mineral resource strategies and of product liability laws.

The Conference will be held at Concordia University, in Montreal, **August 19-23, 1983.** It will be jointly organized by Concordia University, Ecole Polytechnique and McGill University. The Conference languages will be English and French. The facilities under one roof include auditoria with capacities for 700, 380 and 190, a main entrance hall and mezzanine for receptions and poster sessions, cafeteria and faculty lounge. a budget, based on the conservative attendance estimate of 300, is expected to balance with registration fee of $350 in advance or $425 at the Conference.

The following societies have agreed to give their support to the request to ICSMA for the international conference: Materials Engineering Committee of The Metallurgical Society of CIM, Canadian Council ASM, CSIRA, ASM-Montreal, EIC, CSME, OEQ, WIC and CAP. The Natural Science and Engineering Research Council has expressed interest in seeing such a Conference come to Canada and recommend that a grant application be submitted as soon as the site is firmly designated. Support is also offered by PMRL of EMR, the Industrial Materials Research Center of NRCC and the National Research Council of Canada.

The Conference committee is now calling for volunteers to assist in making the Conference a truly Canada-wide endeavour. It wishes to have corresponding members from every major industry, institute and university who will encourage the submission of papers and assist in the review of manuscripts.

For information, contact the Chairman, H.J. McQueen, Professor of Mechanical Engineering, Concordia University, Montreal, Que., H3G 1M8.

Xerox for → JDS, Xerox to DHS. 4/6/83

ASARCO

Great Basin Exploration Division

WLK file Gold, Deposit Description

March 11, 1983

TO: Peter G. Vikre

FROM: B. J. Maher

Geology of the Jerritt Canyon District, Elko County, Nevada

Location: The Bell Mine, the only producing mine within the Jerritt Canyon district, is located approximately 45 miles north-northwest of Elko, Nevada, in the southern Independence Mountains (Sec. 33,34 35, T41N, R53E). Access is via Nevada 225 and graded dirt road. Milling facilities are located at the eastern margin of the range, approximately 8 miles from the site of active mining. Mining takes place near the crest of the Independence Mountains (approx. 9000 feet), ore being hauled by truck (required by the National Forest Service) to the mill, which in the winter and spring, can be slow and hazardous. This report is based on observations and data gathered during a mine tour on 4-30-82.

Ownership: The Bell Mine is a 70/30 joint venture between Freeport Gold Company (Freeport-McMoRan subsidiary) and FMC Gold Inc. (a FMC, Inc. subsidiary). Freeport controls pre-mining exploration and development as well as production.

Reserves: Published reserves are 9.1 m.t. with 0.22 oz Au/ton of "recoverable" Au. Mine personnel report reserves of 12 m.t. of 0.23 oz Au/ton. The reserves are distributed through 5 ore bodies, two of which are active. The two active ore bodies contain 6.0 m.t. (Marlboro Canyon) and 1.5 (North Generator Hill (NGH)). Present (5-82) cutoff grade appears to be 0.5 oz Au/ton based on ore control stakes observed in the NGH pit. The NGH pit has a higher oxide/carbonaceous ore content (oxide ore is cheaper to mill; $24/ton vs. $32/ton); this combined with the high cutoff suggests that Freeport is trying to recover capitalization costs early in the mine life. Figure 1 shows activity in the NGH pit. wow!

Geologic Setting:

The Jerritt Canyon district is hosted by highly faulted lower plate Paleozoic rocks of the Hanson Creek and Roberts Mountain Formations. Upper plate rocks are exposed at the district margins, but do not host gold mineralization. The Roberts Mountain thrust fault is not exposed within the main mineralized area, but the mine staff believe that low-angle imbricate structures, which locally increase permiability, were formed during the compresive

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MAR 16 1983

EXPLORATION DEPARTMENT

event. Mafic intrusive dikes are reported in the area, however the dikes as well as spotty erosional remnants of Tertiary volcanics are pre-mineral. Since mineralization is in part controlled by litho-stratigraphic variation, Freeport has conducted detailed stratigraphic studies within the district. The Hanson Creek Formation (Ordivician) hosts a majority of the mineralization, with the remainder in the basal portion of the Roberts Mountain Formation.

The exposed base of the Paleozoic section within the district is the Eureka Quartzite (Ordivician) which consists of 165 m of orthoquartzite. Unconformably above the quartzite is the Hanson Creek Formation which is subdivided into 5 members within the district. The basal members (Ohc_5 and Ohc_4) consist of 90 m of interbedded carbonaceous and shaley limestone with minor chert. The middle unit (Ohc_3) is dominated by 33 m of banded limestone which hosts most of the Au mineralization. The bands are repeatative laminations of carbonaceous limestone and micritic limestone which have appreciable primary bedding-plane porosity. This unit is overlain by 30 m of medium-grained massive dolostone (Ohc_2) and 13 m of cherty dolostone (Ohc_1). Both Ohc_1 and Ohc_2 show evidence of mineralization, e.g. pervasive silicification, calcite, realgar, and orpiment veinlets, but they don't constitute ore. The Hanson Creek is conformably overlain by the Silurian Roberts Mountain Formation. The Roberts Mountain is considerably more clastic in the Independence Mountains than at its type locality, containing less than 30% limestone. The basal portion of this unit locally hosts ore.

Although activity along the Roberts Mountain thrust did generate secondary permiability in several units, three sets of high-angle faults are more significant in localizing ore-forming fluids. An east-west set of high angle faults are the oldest normal faults in the district and partially influenced late hydrothermal events. A northeast trending set of faults appear to be the primary fluid conduits as Au values are distributed about this set. Northwest trending faults are post-mineral and slice up the ore bodies. The late faults produce ore bodies with sharp margins, e.g. the NGH ore body was missed by 10 feet due to a post-mineral fault in early drilling.

Associated with the northeast fault set are pod-like breccias which frequently contain the largest quantities of Au. The breccias also localized late hydrothermal fluids as they host barite, stibnite, and argillic alteration phases associated with the late fluids.

Deposit Description:

Ore within the mine has been subdivided into 2 catagories based on C content. Carbonaceous ores are grey-black, pyritic silty calcarenites or Ca-siltstones. Alteration consists of addition of minor C, aragonite, jarosite, and variscite (Al $(PO_4)\cdot 2H_2O$). Rock porosity decreases slightly. The carbonaceous ores contain up to several percent realgar and orpiment with minor cinnabar and arsenopyrite. Gold is submicroscopic.

"Oxide" ore is formed by two processes within the system: (1) Surficial oxidation by supergene fluids; (2) late stage evolution of a vapor phase and oxidation of H_2S in the hydrothermal fluid. The two types of oxide ore may be differentiated on the basis of clay composition and assemblages; primary (hypogene) oxide ore is characterized by the assemblage jarosite + alunite + kaolinite, secondary (supergene) oxide ore is dominated by smectite group clays.

Carbonaceous ore lies below the hypogene oxide ores which are in turn overlain by several jasperoid horizons in the dolomitic portion of the Hanson Creek (Figures 2 and 3). Although there is a close spatial relation between the jasperoids and gold ore in the NGH pit, jasperoids elsewhere in the district have been drilled with negative results. The jasperoids are brecciated and cut by multiple quartz veins suggesting several pressure release (boiling) events. Since the boiling event brecciates prexisting silicified rocks (the jasperoid), the C-ore event must also mobilize silica to the margins of the system forming jasperoid. Clearly, silicification is the dominant alteration product.

Widespread boiling is hypothesized based on the hypogene oxide mineral assemblage and analogy to other similar systems (e.g. Carlin). Boiling remobilizes large amounts of C into fault zones (Figure 2) or other low pressure areas (Figure 3). The acidic vapors and condensate leach Ca and introduce argillic, sulfate, oxide and silica phases into the rock (Figure 4). The Ca is transported to the margins of the system forming calcite veins and stockworks which are anomalous in trace metals. Au is also remobilized during this stage, forming microscopic particles up to 4 microns in size.

Stream sediment sampling along the west flank of the Independence Mountains for Sb originally led Freeport into the area. Rock chip sampling on the stream sediment defined target areas revealed protions of the presently known Au inventory. Additional targets were developed by geologic interpretation into areas covered by gravels and colluvium. During exploration and pit development, Au, Ag, As and Hg are corelatable. Au values are highly erratic until in the ore body, where they are consistantly above 0.1x oz Au/ton although they report

grades up to 2.9 Au/ton. Barium and Sb do not corelate with Au within the mineralized areas because they are late paragenetically and not always spatially associated with Au mineralization. The high level jasperoids contain anomalous As, Sb, Hg and Ba but Au values are low. Selenium and Tl analyses have probably been done but mine personnel refuse to discuss their distribution or significance.

Interpretation:

Freeport believes that the mineralization formed in a manner analagous to Carlin, Getchell, Alligator Ridge, et. al., in a near-surface, "hot-spring" environment. The close corelation between hydrothermal stratigraphy and metal distribution to the present surface support this concept. Mineralization does cut across stratigraphy although the apparent relationship between gold mineralization and the present erosion surface may be fortuitous.

This writer believes that the primary difference between Au-mineralized hot spring deposits (e.g. Florida Canyon) and "Carlin" type Au mineralization is the interaction of the hydrothermal fluid with a calcareous and carbonaceous host at moderate (1000-3000 feet) depths. The host composition and depth of formation place physio-chemical constraints on fluid composition as does the extent of fluid evolution at those depths. While the overall hydrothermal processes within hot spring and "Carlin" type systems may be similar, the differences are very significant and result in distinctly different ore bodies. Thus, the analogy to Carlin or other "Carlin" type deposits is more accurate than a comparison to true hot spring environments.

It is clear that the early hydrothermal fluids were able to transport and deposit Au while mobilizing silica to form the first of several jasperoid events. As these fluids evolved and boiled, perhaps repeatedly, the early formed carbonaceous ores were leached of their Ca and C content while mobilizing Au short distances where it nucleated into micron-sized particles.

Structural ground preparation is a major factor in localizing mineralization, first serving as a pathway for early Au-bearing solutions and latter as a low pressure site for C migration during boiling and leaching of the carbonaceous protore. Stratigraphy played an important role as host for the early C-rich Au mineralization. The favored horizon differed from adjacent units mainly in increased bedding plane porosity. As the relatively reduced early Au-bearing fluids migrated away from feeder structures along bedding planes, Au is deposited, possibly by oxidation of the Au bearing complex ($Au(HS)_2^-$?). Thus, the early stage Au mineralization may be deposited along a redox interface within the

favored horizon, perhaps similar to roll-front U deposits. As the hydrothermal fluid evolves through several episodes of silica sealing, breaching, and boiling, fluid compositions change such that oxide phases are formed, generating the upper level oxide ores.

B. J. Maher

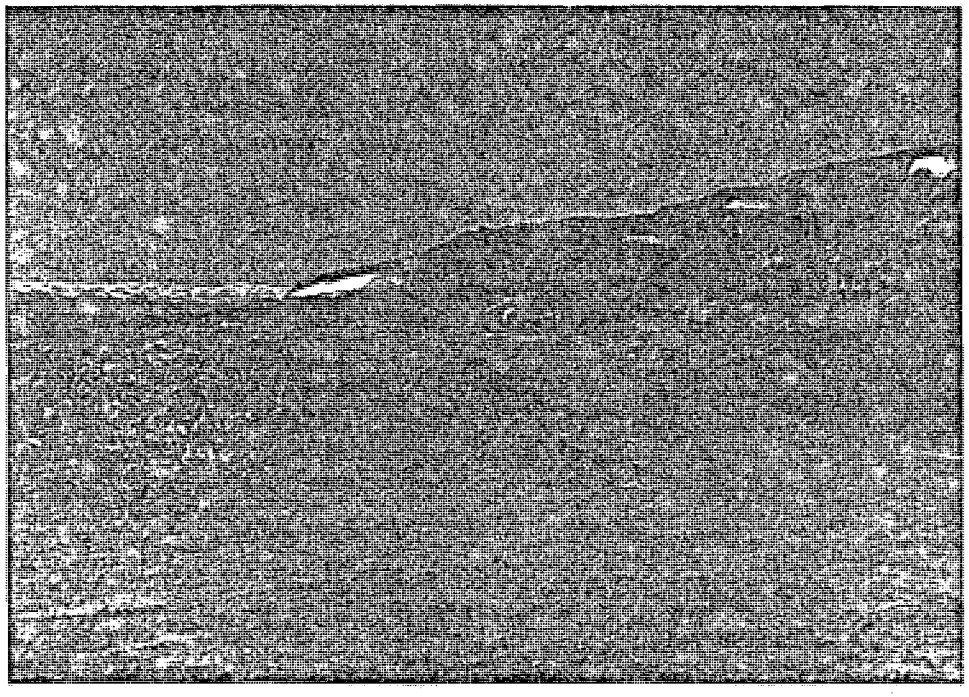
Brian J. Maher

BJM/jn

cc: Mr. F. T. Graybeal
Mr. W. L. Kurtz
Mr. G. J. Stathis

Figure 1. Mining activity in the North Generator Hill pit, Bell Mine. Photo taken looking west. Note tilted truck (Wabco 85 ton cap.) in right center of photo. Vehicle has sunk into mud after being loaded with ore.

Figure 2. Northern Face of NGH pit. Carbon has been remobilized into pods adjacent to fault from lower (right hand side) carbonaceous ore zone. Left side of fault is hypogene oxide ore. Ohc_1 and Ohc_2 are strongly silicified.



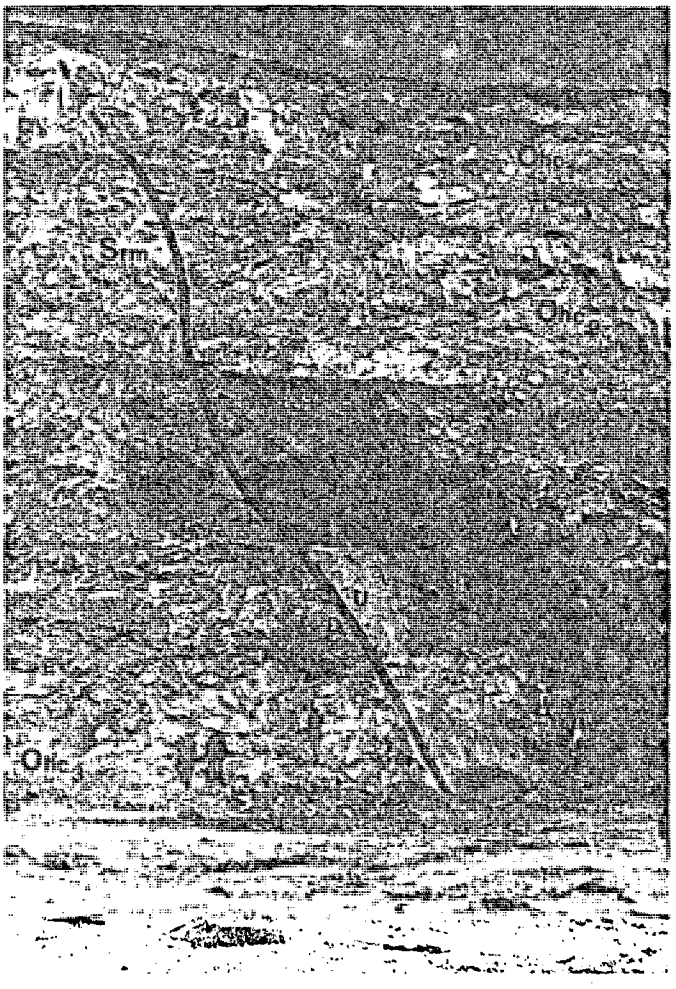

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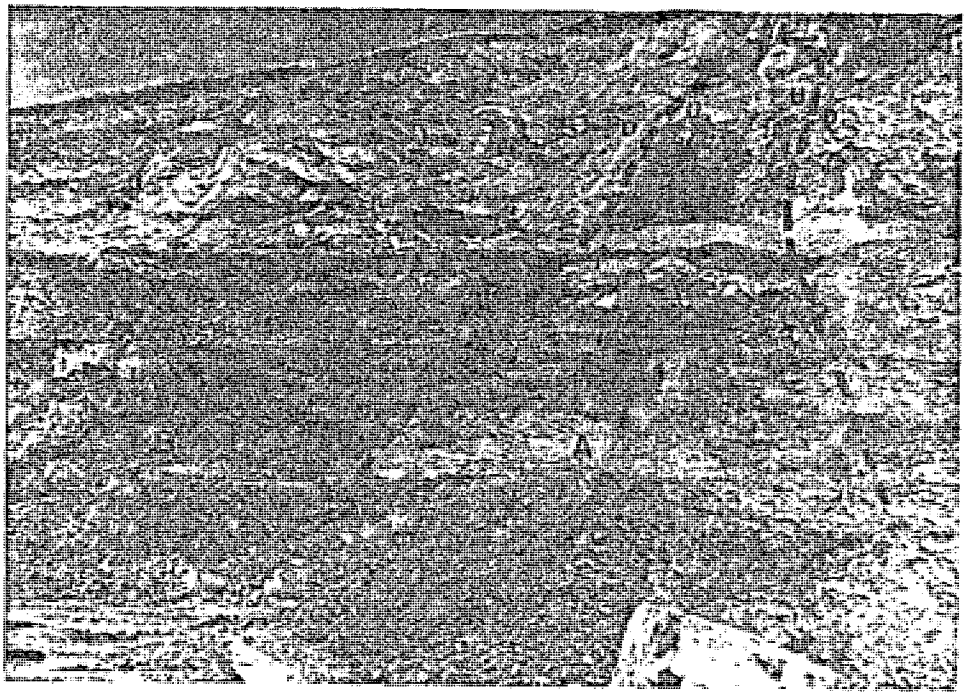
Figure 3. NGH pit. Large pod of carbonaceous ore that has been partly oxidized (a) by late hydrothermal fluids. Carbon was mobilized from margins near faults to central portion of pod (b) where abundent low angle imbricate jointly is developed.

Figure 4. Barite, quartz and iron oxide veinlet cutting jasperoid in NGH pit.



1982 Annual Report

Skilled equipment maintenance is critical to maximum operational efficiency and recoveries. For every three production people at the mine, and every two at the mill, there must be one involved with maintenance. As at other company locations, Freeport's training program develops men and women with newly acquired multiple skills such as carpenters who can weld, and machinists who can replace electrical equipment.

We are proud of the record output and design recoveries achieved in the first full year of operations by the some 265 people of Freeport Gold, particularly in view of the severe winter weather conditions which the project must endure.

Nevada Gold Exceeds Production Goals

Freeport Gold Company's 70-percent-owned Jerritt Canyon, Nevada, gold mining and milling operation recorded an excellent performance—producing at virtually design capacity of an average of 200,000 ounces of gold annually. Despite severe winter conditions, ore throughput achieved design capacity, average recoveries approached the design 87.5 percent level, and unit costs were within budget.

Freeport Gold's share of 1982 production and sales amounted to 137,000 ounces and 136,300 ounces of gold, respectively. Sales realization, including earlier hedging transactions, averaged $390 per ounce.

A total of 90,000 ounces or about half of Freeport Gold's share of expected output remains sold forward for 1983 delivery at an average price of $383 per ounce.

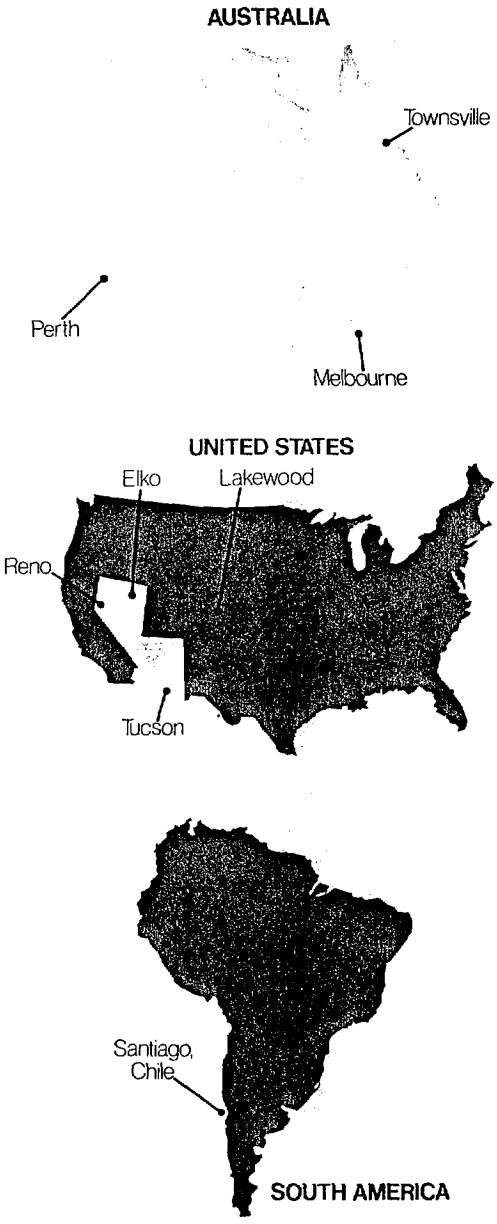
Earnings amounted to $14,406,000 in 1982 and, if current output plans succeed and market prices of near $500 per ounce can be obtained, profits will rise in 1983.

In 1982, step-out drilling around the Jerritt Canyon operating mine added reserves of over 1,500,000 tons of mineable gold ore at an average grade of about 0.15 ounce per ton. These newly found reserves contain approximately 222,000 ounces which about replace the 228,525 ounces contained in the 968,370 tons of 0.236-ounce-per-ton ore milled during the year. Mining of one area in 1982 indicated that the lateral extent of the higher-grade ore zones was less than expected in our original reserve computation, and therefore conservative downward adjustments were made to all remaining reserves. Proven and probable mineable reserves as carried at the beginning of 1983 are 11,614,000 tons containing 0.233 ounce per ton and, if mined and processed at the mill design rate of 2,750 tons per day, will sustain operations for over 11 years. Drilling in four other widely separated areas within the joint-venture lands encountered significant gold mineralization. We are encouraged by these still preliminary results, but much additional drilling will be required to complete the evaluation of any of these areas.

Gold Exploration Encouraging

Freeport Exploration Company, headquartered in Reno, Nevada, is concentrating on disseminated gold prospects in the western United States, Australia and, most recently, in Chile in an effort to apply the knowledge gained at the Jerritt Canyon orebodies to other locations.

Exploration offices on three continents



Ranchers Completes $5.5 Million Purchase of Colo./N.M. Volcanic Rock Producer

Purchase made by newly formed subsidiary

Ranchers Exploration and Development Corp. has purchased all of the assets of Colorado Aggregate Co., a division of Petroleum Corporation of America, for $5.5 million.

Colorado Aggregate produces volcanic rock, principally scoria, for landscaping, roofing and use as briquettes in gas barbeque grills. Sales are expected to average about $4.5 million annually. Principal mining properties are situated at Mesita, Colo., and San Antonio Mountain, N.M. The company has about 50 employees.

Petroleum Corporation of America received $1.19 million in cash at closing and will receive an additional cash payment of about $324,000 on June 1, 1983. The balance of the purchase price involved restructuring a $4 million loan from Texas Commerce Bank that had been made against Colorado Aggregate's properties. The loan is now payable by Colorado Aggregate over the next five years and will become non-recourse financing to Ranchers on Nov. 14, 1983, if certain performance objectives are met.

The acquisition, which was closed on March 29, 1983, was made as of Jan. 1, 1983, and will give Ranchers the benefit of Colorado Aggregate's earnings for the quarter ended March 31, 1983. Ranchers said that it expects Colorado Aggregate's earnings, before taxes and interest expense, to be in the range of $1.4 million annually, about half of which should be reflected in the quarter ended March 31 as a result of the seasonal nature of the company's operations.

Colorado Aggregate was purchased through a newly formed subsidiary of Ranchers, Colorado Aggregate Co. of New Mexico.

Felmont Oil/Case, Pomeroy Consider Inquiries For Round Mountain Gold Mine Interest

Felmont Oil Corp. and Case, Pomeroy & Co. Inc. are considering inquiries from numerous parties expressing strong interest in purchasing their combined 50% interest in the Round Mountain Gold mine, situated in Nye county, Nev.

The Louisiana Land and Exploration Co., through its Copper Range Co. subsidiary, is the operator and owner of the remaining 50% interest in the mine. The operator currently estimates that proven and probable reserves at Round Mountain are 228 million tons of ore, with an expected ultimate recovery of close to 8 million ounces of gold. For the year 1982, total production at the mine was 72,561 ounces, from which Felmont and Case, Pomeroy each had revenues of $6.5 million from their 25% shares.

Felmont and Case, Pomeroy & Co. Inc. have retained the New York-based private banking firm of Brown Brothers Harriman & Co. to act as their financial advisor in reviewing expressions of interest in the Round Mountain gold project.

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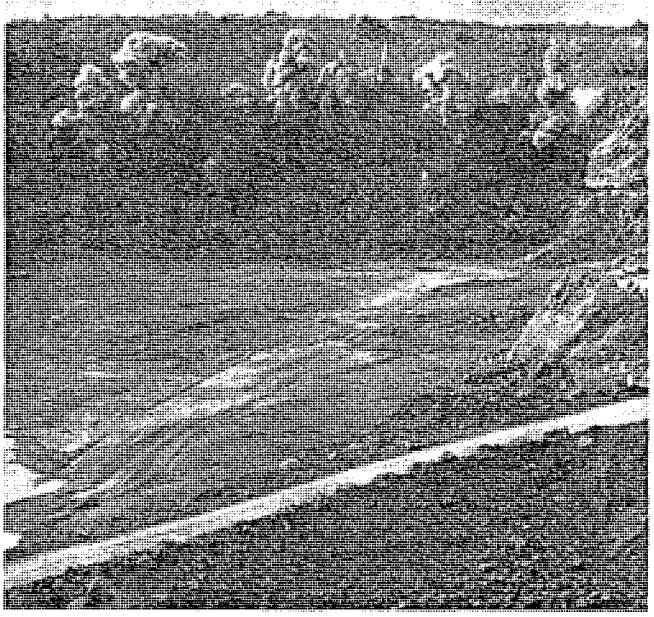

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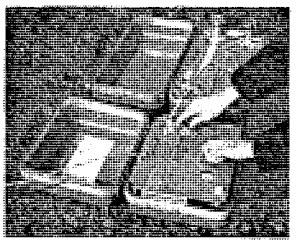

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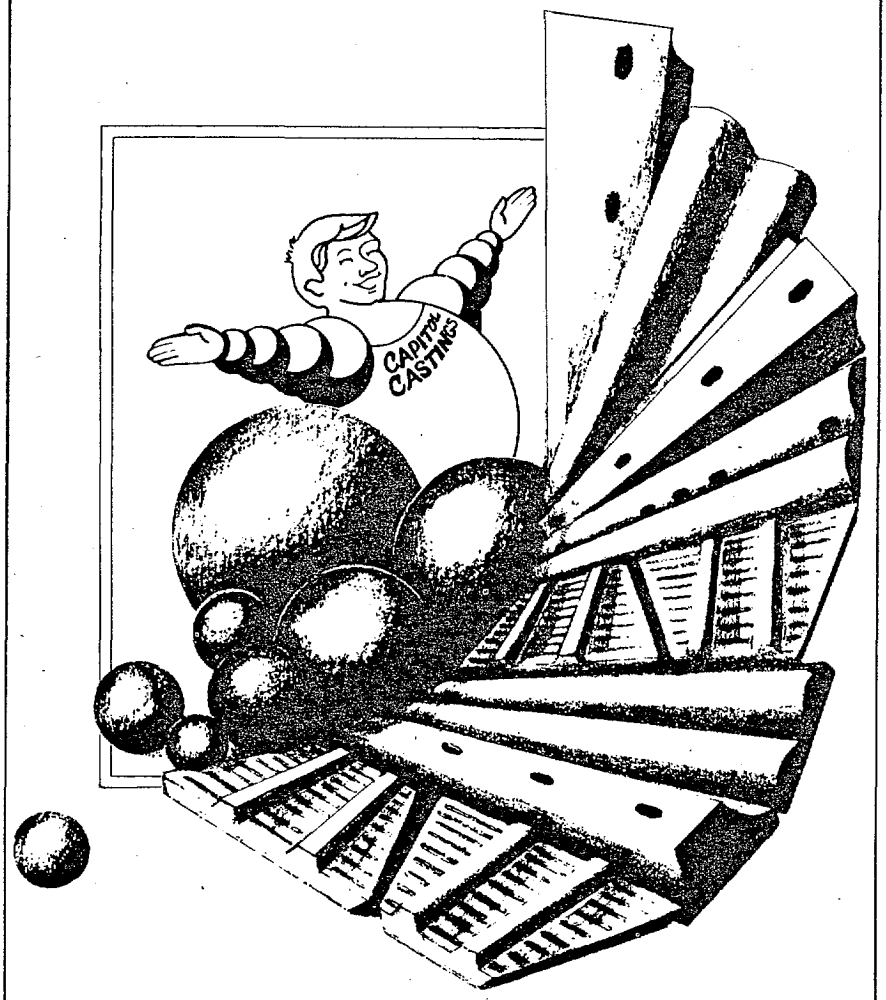

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MIDLAND ROSS

Bull Run Gold Receives Favorable Results on Mesona Exploration

Exploration being conducted in Nevada by Freeport Minerals Co.

Bull Run Gold Mines Ltd., Vancouver, B.C., has received the results of the 1982 exploratory work on its Mesona property, situated in Elko county, Nev., from Freeport Minerals Co., a wholly-owned subsidiary of Freeport-McMoRan Inc.

The exploration is being conducted by Freeport under an exploration agreement that gives it the exclusive right to explore and prospect Bull Run's Elko county, Nev., mining claims. Freeport will earn a 60% interest in Bull Run's claims upon completion of a $2.5 million exploration program. Freeport has so far expended about one-tenth of that amount.

Most exploration work completed on the Bull Run claims consist of exploratory drilling, geological mapping, surface sampling and the construction of access roads. A total of 21,705 ft. was drilled in 58 vertical rotary drill holes in two target areas known as the North and South Sammy Creek zones. Both target areas were identified by geological mapping and surface sampling.

The North Sammy Creek zone was tested with 38 holes, the majority of which were located on a 200 ft. grid. Near surface gold mineralization was encountered in about 50% of the drill holes. The South Sammy Creek zone was tested with 20 widely spaced holes, with significant gold mineralization intersected in 9 of the holes.

Drilling was suspended on Nov. 18, 1982, because of heavy snow, but is currently scheduled to resume in the spring of 1983 as weather conditions allow.

Associated Manganese Mines' 1982 Manganese/Iron Ore Shipments

In 1982, The Associated Manganese Mines of South Africa Ltd. shipped a total of 1,977,000 metric tons of ore from various mines operated by the company in the Beeshoek-Blackrock area near Postmasburg in Cape Province, South Africa. The principal quantity consisted of 1,132,000 tons of manganese ore, and the remaining 845,000 tons comprised iron ore.

U.S. Steel Mill Imports Increase to 1,173,000 N.T. in February

U.S. imports of steel mill products continued to increase in February, both in tonnage and in percentage penetration of the depressed domestic market, according to U.S. Commerce Department figures compiled by the American Iron and Steel Institute.

Imports in February were 1,173,000 net tons, compared with 1,098,000 tons in January and 1,600,000 tons in February 1982. Imports in the first two months of 1983 were 2,271,000 tons, compared with 3,569,000 in the same 1982 period. While the 1983 figures were below the record-high levels of early 1982, in both import tonnage and market penetration, the 1983 imports thus far have still taken about 20% of the depressed American market.

Imports in February represented about 21.4% of the apparent supply in the American market. This preliminary figure is based on an estimated 4.4-million tons of domestic mill shipments. Market penetration in January was 19.7%, and for the first two months of this year, about 20.5%. In February 1982 market penetration by imports had been 23.3%; and for the first two months of last year, 25.0%.

Kennecott Sells Tintic Division to HMC Mining Inc.

Kennecott announced the sale of the assets of its Tintic division near Eureka, Utah, to HMC Mining Inc., a Utah corporation headed by Paul L. Hunter. Specific terms of the agreement were not disclosed.

Kennecott's Tintic division, which provided flux materials for the company's smelting operations, had been shut down since Nov. 29.

Pioneer Nuclear Inc. Exploring Silver Property Near Creede

As part of its program to extend its mineral activities beyond uranium, Pioneer Nuclear Inc. signed a joint venture agreement last year to explore a 2060-acre silver property near Creede, Colo. Pioneer Nuclear, as operator, began exploration immediately and may commence mine development in 1983.

Iron Ore Exports from Goa Total 1,208,083 M.T. in February

Cumulative amount to March 1 shows major decline at 2,227,562 M.T.

In February, iron ore shipments from the port of Mormugao in the territory of Goa, India, amounted to 1,208,083 metric tons for an improvement from 1,019,479 tons in the previous month of January but a major decline from 1,632,466 tons in Feb. 1982. The total quantity consisted of 1,169,473 tons of fines and 38,610 tons of sinter fines.

During the first two months of 1983, Goan iron ore exports aggregate 2,227,562 metric tons, a reduction of 1,209,415 tons from 3,436,977 tons in the corresponding period of 1982.

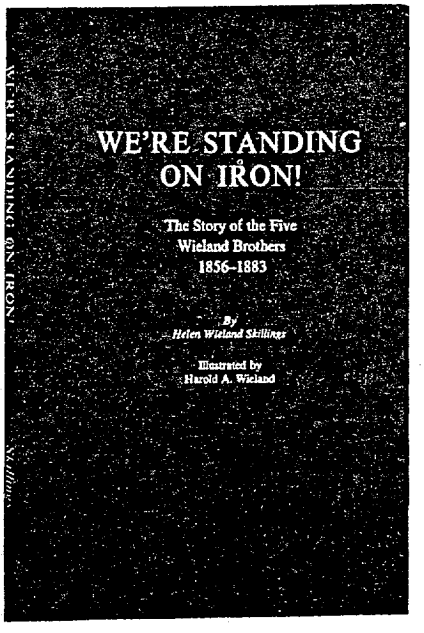
The principal receiver in February was Japan taking 1,002,883 metric tons, with the remainder consisting of 106,200 tons for South Korea and 99,000 tons for Italy. The top exporter was Chowgule & Co. shipping 324,263 tons followed by V.M. Salgaocar with 261,200 tons, Sesa Goa with 197,868 tons, V.S. Dempo with 137,501 tons, A.V. Sarmalkar with 114,100 tons, and Orient Goa with 102,621 tons. The other two organizations that were active handled minor tonnages.

The accompanying tabulation compiled by Fomento Industrial contains figures for Goan iron ore exports by shippers and destinations for February.

IRON ORE EXPORTS FROM MORMUGAO IN FEBRUARY—METRIC TONS

SHIPPER	JAPAN Fines	JAPAN S. Fines	S. Korea	Italy	Total February	GRAND TOTAL
CHOWGULE & CO.	324,263	—	—	—	324,263	544,452
SESA GOA LTD.	398,868	—	—	99,000	197,868	419,084
V.M. SALGAOCAR	155,000	—	106,200	—	261,200	407,800
V.S. DEMPO	137,501	—	—	—	137,501	278,986
FOMENTO	37,000	—	—	—	37,000	232,289
A.V. SARMALKAR	114,100	—	—	—	114,100	114,100
ORIENT GOA PVT. LTD.	64,011	38,610	—	—	102,621	102,621
DAMODAR MANGALJI	—	—	—	—	—	37,800
P. TIMBLO INDUSTRIAS	33,530	—	—	—	33,530	33,530
TIMBLO PVT. LTD.	—	—	—	—	—	—
-do-(MMTC)	—	—	—	—	—	30,400
V. METHA & CO.	—	—	—	—	—	26,500
TOTAL	964,273	38,610	106,200	99,000	1,208,083	2,227,562

Early Development of Lake Superior Region Lumber — Shipping — Exploration



This is the story of five brothers who came from Germany to begin a new life in the U. S. and founded the townsite of Beaver Bay on the north shore of Lake Superior in 1856. Sawmills were built at Beaver Bay, and the lumber was shipped in the Schooner Charley to ports in Michigan, Wisconsin and Ontario as well as Minnesota. In 1865, the low grade iron formation near the present site of Babbitt, Minn., was discovered by one of the brothers. The original exploration of this property by the Ontonagon Syndicate was unsuccessful. Today the taconite at Babbitt is mined by Reserve Mining Co. for the production of high grade iron ore pellets at its Silver Bay plant, four miles east of Beaver Bay.

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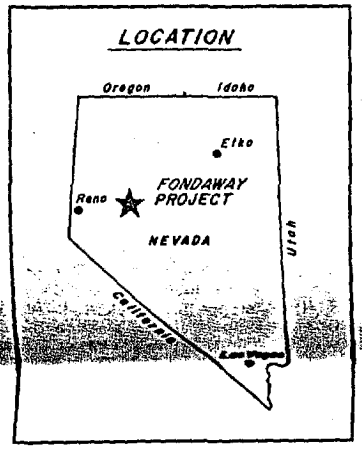


Qal

TRrS

TRrS

ADDITIONAL CLAIMS TO THE WEST

x .07/50'

x .11/15' GOLD-BEARING ZONE

x .09/15'

Qal

TRrS

x .17/20'

x .12/20'

SECTION 2 | SECTION 1

SECTION 11 | SECTION 12

x .07/40'

TRs

TRs

TUNDRA GOLD MINES LIMITED

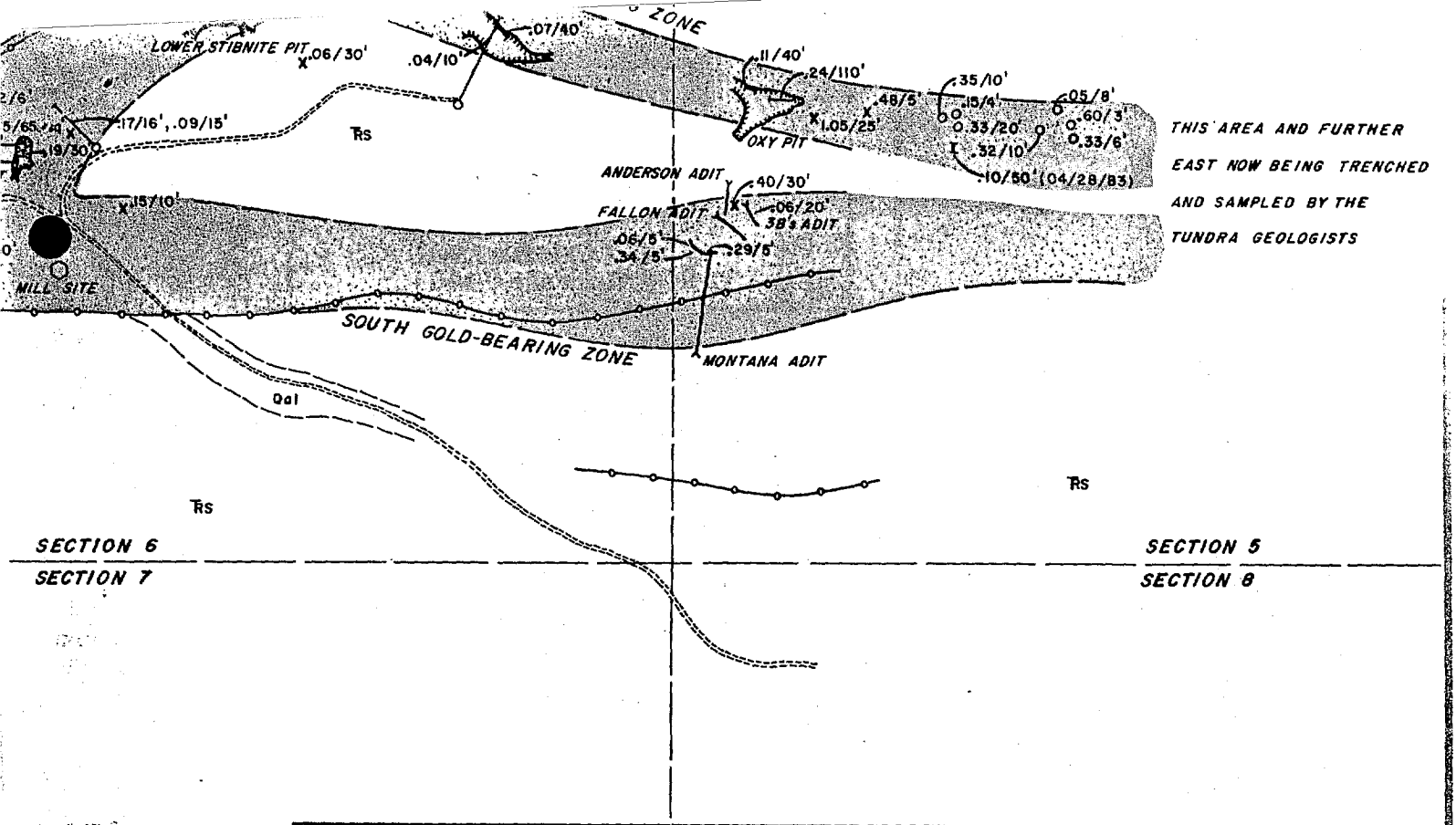
P.O. BOX 30107, ALBUQUERQUE, NM 87190
FONDAWAY PROJECT
ARSENICAL GOLD BELT, NEVADA, U.S.A.

Bert Applegath, President
Tony Miller, Vice-President
Bevan Anderson, Assistant to the President

Project Manager: Robert Akright, Geologist
Project Consultants: Dr. Jean Descarreaux, Geologist
Dr. Geza Kisvarsanyi, Geologist
Fenton Scott, P.Eng., Geologist
Project Geophysicists: Peter Ferderber
Harry Ferderber

Northern Mine
May 5 1983

Compiled by JEAN DESCARREAUX & ASSOCIATES LTD., April 1983



IN 1982 THE MINE PRODUCTION OF RECOVERABLE GOLD IN THE U.S. SHOWED 1,226,530 TROY OUNCES. THE STATE OF NEVADA IN 1982 PRODUCED 592,564 TROY OUNCES, OR ALMOST ONE-HALF THE U.S. PRODUCTION.

LARGEST GOLD DEPOSITS OF THE ARSENICAL GOLD BELT OF NEVADA

			PAST PRODUCTION PLUS PRESENT RESERVE		
DEPOSIT	OPERATOR	DISTANCE FROM FONDAWAY (air miles)	Tonnage (short tons)	Average grade (ounce of gold/ton)	Ounces of gold (total)
ROUND MOUNTAIN (open pit)	Louisiana L. & Expl.	98 S.E.	12,700,000	0.66	8,400,000
GOLD QUARRY (open pit)	Newmont	117 N.E.			
Higher grade zones			25,100,000	0.11	2,660,000
Lower grade zones			150,000,000	0.03	5,400,000
CARLIN (open pit)	Newmont	123 N.E.	30,700,000	0.21	6,500,000
GOLDFIELD (underground) (mined out)	Newmont	158 S.E.	24,960,000	0.19	4,854,000
JERRIT CANYON (open pit)	Freeport-FMC	153 N.E.	13,200,000	0.21	2,900,000
GETCHEL		93 E.	13,526,000	0.18	2,447,000
BATTLE MOUNTAIN	Duval	72 N.E.	6,000,000	0.40	2,400,000

Compiled by Dr. Geza Kisvarsanyi, consulting geologist, and Robert Akright, geologist.

"FROM "QUOTE" Rhythm Edition
MAY/JUNE 1993

CMI: RLB, PGW 5/24/93
Circulate JDS, JRS → WLK
JDS

...are basically prospectors.

Quote: *What is U.S. Minerals Exploration Company?*

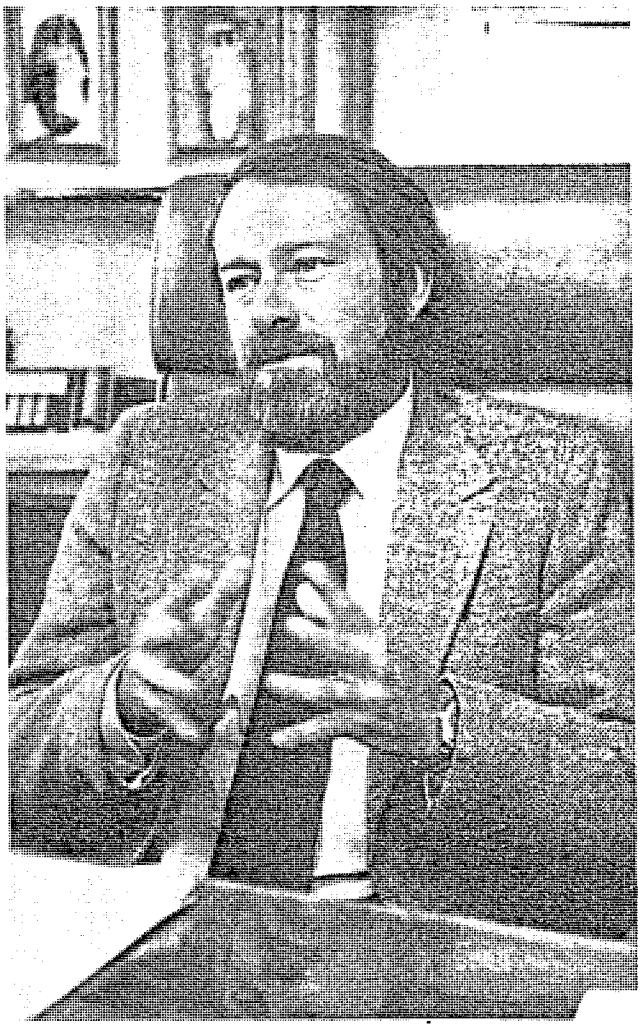
Grauberger: U.S. Minerals Exploration Company (USMX) is an exploration company looking for precious metal deposits that are low-grade and that can be mined by open pit methods. We are looking for very large tonnage situations that can last for a number of years. We basically do this by acquiring properties we believe have merit, and then farming them out, or joint venturing them to other major mining companies. They then spend the initial money exploring these tracts to decide whether they have the potential to meet that kind of a large tonnage situation. We have executed joint venture agreements on 45 properties with over 15 major companies during the last three years.

Are there any major differences between USMX and the average mineral exploration company?

Yes, ours is a small publicly held company, and I can only compare our company to other small publicly held companies. Most of the other companies usually only have one or two properties that they raise money on through public stock offerings. Then, they spend the shareholders' money on these properties trying to develop them and usually go under.

We do not do that. We are working essentially a "numbers" game. We have a large number of properties (currently 30) that we are looking at. The odds in this business are at *least* 200 to one against finding an economically viable situation that you can mine for a long period of time. We are essentially trying to survive until we find what we believe is a very large deposit. Once you *find* the deposit, it's not too hard to finance it. Most people don't really realize the money it takes to put them into operation.

It all boils down to this: The major differences are 1.) We're still here, 2.) We have almost as much money in the bank now as we did when we first started, 3.) We are usually at least marginally profitable, and 4.) We have accomplished our main goal, which is to discover and initiate development of a significant open pit precious metal mine.



Story Photos by K.J. Fleming

Gary Grauberger

President, U.S. Minerals Exploration Company (USMX)

Interview by Liz Walker

How are you able to finance your operations if it is so expensive?

Our operations are financed by joint venture partners. The major mining companies that we make each deal with essentially refund us the money we have invested in the property. Then, they are required to

We have accomplished our main goal, which is to discover and initiate development of a significant open pit precious metal mine.

spend anywhere from a half million dollars to as much as $20 million on the property to completely explore and develop the property before we ever have to put a dime into it. In other words, before *we* have to put money in the property, we know what's really there (from their own data and what they have spent). In exchange for that, USMX gives up a part of our interest in the property; usually 50 to as much as 75 percent. If we don't want to put *any* money in the property, we usually go to a straight royalty interest of about five or ten percent.

So, who is primarily taking the risk?

The major company, not our shareholders.

Could you give our readers some information into your own background and how you got involved in all of this?

In 1968 I first got involved in a small silver mine in Nevada and was promoted out of every dime I had. I was a broker at the time, but the sour experience prompted my interest in the minerals and the oil and gas industry. I went back to school and received a degree in geology from the University of

...I have received, to date, over ten times my original investment.

Colorado. I graduated in 1971, and started working for mining companies looking for base metals, uranium and precious metals. I got involved exclusively with precious metals in about 1973 and have been working in that field ever since. The silver mine I originally lost my money in is now in production and I have received, to date, over ten times my original investment.

What gave you the idea to start this company using the "joint venture" method of financing?

I was working for Earth Resources Company until 1979. They were very successful and discovered and operated the DeLamar silver mine. They then decided to get out of the mining business, and the company was eventually merged into Mapco in 1980. Because of this, I was going to be out of a job, and decided to form USMX. The Penny Stock Market at that time was beginning to come back, and a good friend of mine, Greg Pusey, had been involved with several Denver underwritings. We talked about this idea for some time, so I quit Earth Resources, just as the two companies began

serious merger negotiations, and we formed USMX. I was aware of a number of properties because I had been doing reconnaissance for Earth Resources for several years. I picked up one property that I thought had a great deal of merit. Greg and I founded the company with $35,000 of our own capital. We then raised an additional $150,000 from interested friends and investors. We went public through J. Daniel Bell in August of 1980 at $0.10 per share and raised a net of $1.7 million. After founding the company and putting several property groups into it, we immediately started making joint ventures. The company has been marginally profitable ever since, just using the joint venture technique.

Do you use a specific method for your gold and silver property acquisition and exploration that is different from the average method?

We don't really do anything that is different from the major companies, but we do have several people that are very good at what they do. We are basically prospectors. We take all of the information that we are able to get from literature, from land stat photos, and from known geologic maps. Then we use a small aircraft to fly these areas that we believe (on a map) have potential. Once we spot something from the air that looks like it may have some potential, basically by coloration of the rocks that we see below us,

It is more or less what we call "grass roots" exploration.

we send in our prospectors, or myself, and sample the area extensively. If the samples turn out (if they show anomalous gold, silver or some of the other signature elements of these types of deposits), then we go in and either stake it (if it's open ground), or we acquire it from landowners. It is more or less what we call "grass roots" exploration. We can move very quickly, whereas the major companies need four or five different memoranda and reports filled out by their managers and geologists before a decision can be made to acquire an area of interest. Whereas, in about 20 seconds after we get all of the information, we move.

So that time savings is also a savings in money to the stockholder?

Right, plus there are actually very few landowners that we end up dealing with because we are looking at areas that most companies haven't done much work on. We are able to acquire the land through the old 1872 mining laws, where we can actually just put posts on the ground and stake mining claims. When we do deal with landowners, they know we are not a very big company. We tell them exactly what we are trying to do. We can't afford big payments or big royalty interests. And most of the time, they will go along with us. We usually acquire the properties very cheaply compared to what a major company could acquire the property for. The we put it all together in a clean package. We have been very successful with this approach. Since our inception in 1979, we've joint ventured about 45 separate properties with mining companies and have had all of the properties drilled. Today, we have eight joint ventures that are still drilling and we know that we have one definite success.

Why is gold and silver exploration such a precarious business and what are the variables involved?

In order to just get the 45 properties we have acquired, we have weeded through at least 500. The ones that we acquired have a definite geological and geochemical signa-

ture of a low-grade gold or silver deposit. In other words, how the rocks have been changed by hydro-thermal solutions. Once you are able to put together all these different geological aspects, the results do not

Even with all of the sophisticated techniques we have today, no one can see under the ground.

necessarily mean there is going to be an ore deposit there. What you have is a very large area that has all the right indicatiums at the surface. Next you try to figure out where the better areas are located within this large zone. Then you drill it. Even with all of the sophisticated techniques we have today, no one can see under the ground. So we drill the holes and usually discover inconsistencies or just blank holes. After the first round of drilling, you usually wipe out about 90 percent of the properties that you think have any merit. Then, the ones that you have gotten some halfway decent results on, you drill again. This again eliminates several more prospects. It is a disciplined step by step process of trying to take an area that looks good and destroy it. That's exactly what you go through. After you feel you might have something, that's when exploration begins to get really expensive. For example, just to walk away from a property and know that you haven't overlooked something significant, you are going to spend at least $300,000 to $500,000. If the prospect survives the initial expenditures, another $1 to $3 million will be required to determine if it's an ore deposit.

Could you give us an average number of years it takes to go into production from the time that you first decide you are going to investigate a property?

On a significant low-grade precious metal deposit that is mineable by open-pit methods, the average time from investigation to production is usually about six to seven years. The process of a prospect becoming a major mine usually requires large expenditures through numerous exploration programs by many different companies. You can't become a millionaire overnight in this business. For a deposit of this type, the quickest one I have ever seen put into production has taken about four years from the acquisition date. More often than not, the initial locator of the prospect (whether a company or an individual) gives up on the property long before it reaches production and profits are forthcoming.

What are the odds of finding a big discovery?

Two hundred to one, at least.

Could you tell us about the Grayhill Exploration Company and how you are involved with that?

USMX owns about 24 percent of Grayhill Exploration Company, which went public in January of 1981. Grayhill is doing the same thing we are except they are doing it in Mexico, other foreign countries and through a joint venture with USMX in the United States. Grayhill's experienced management consists of top geological talent

What is considered mineable ore at one point in time can be considered waste the next.

and they have been putting together properties and exploring them. Grayhill is trying to put one property into production in Mexico with their Mexican partners. The difference in working in foreign countries is that you must have very, very good partners that have been mining in that foreign country for a long time, otherwise there is no way that you will ever make a dime out of those ventures. The laws are very different, and it is just incredibly difficult to work in foreign countries.

What are your hopes for your activities in Mexico?

From the results we have received so far, we are very pleased with what Grayhill has been able to accomplish. We think they will have an operating mine there within the year which will be profitable. They have a number of exciting properties that actually look much better than what you would ordinarily be able to find in this country. You have to balance that against the unstable government and all the problems, such as high taxes and logistics, you find in foreign countries.

Could you tell us something about the oil and gas part of USMX?

We have been investing in oil and gas since shortly after going public. We have been spreading our risk through a large number of wells. We have participated in nine wells and, to date, six of those wells have been successful. We just take a small interest (between three and 15 percent) per well.

In mining terms, that is extremely profitable.

That way we don't have to put too much money in it, and, if it does become successful, we will profit. We have actually invested more money in oil and gas since we started than we have in mineral properties because we are the promoted joint venture partner in the oil and gas properties. We have been successful at finding reserves, but the last year has really hammered the oil and gas industry. We have four wells on stream and two shut in, but the buyers are only taking a third to a half of the capable production of these wells because of the current prices and glut situation. We plan to stay involved in the oil and gas business using the same technique of promotion that has been effective in the minerals business. We now feel we have the expertise, and have formed a joint venture with Grayhill to develop our own oil and gas properties and it is currently paying for itself.

What do you need to know before buying stock in a mining company?

One should check into the people who are running the company. These people should have been involved in the mining business working for other companies for at least a period of ten years. Everyone involved should be experienced, because it is a very, very tough business. Unless you've got the experience, you probably won't understand the real economics involved. This is crucial to the success of the company. I was familiar with economics from my own past experiences and mistakes; that is why we *always* share the financial risks as well as the rewards. For example, to date there has been about $7.5 million spent on our properties at our partners' expense. We would have gone broke long ago had we not known how to properly finance with joint venture partners. With our method of operation we still have almost as much money in the bank now as we did when we first started and our assets have increased dramatically.

Would you like to comment on some of the "fly by night" mining operations we have been hearing about recently?

There were a number of other companies that went public about the same time we did. As far as I know, there are only two of them that are still active today. One of them is doing fairly well because they have found a small deposit and are mining it. The other one has been very conservative, as we have, and still has most of its cash left. Most of the rest are either broke, gone, inactive, or being investigated by the Securities Exchange Commission. Many millions of dollars of investors' money has been lost. The key to financial survival in a small mining company is to insure that there is limited risk to the shareholder's investment. We have been able to accomplish

It doesn't happen over night.

this. There is no way that I know of to go out and immediately find a big ore deposit. I have been working at it for twelve and a half years and I have been involved in only *one* significant discovery that is presently in production. The name of the game is exposing the company to the maximum number of potential mineral discoveries and surviving until a significant deposit reaches production.

You also have to be cognizant of the fluctuations of the price of gold and silver. As you see, it has changed dramatically in just the last three or four years. Gold has been everywhere from $200 per ounce to $800 per ounce. Silver has been everywhere from $5 an ounce, to as high as almost $50 an ounce. What is considered mineable ore at one point in time can be considered waste the next. This can alternate dramatically simply because of the price. All of the low-grade deposits are very sensitive to the price of gold and silver. If you don't take that into account, you can spend an excessive amount of money on a marginal property when the price of the metal is high and by the time the mine goes into production, the price could have dropped dramatically, changing a potential profit into a loss. Due to this, we calculate all of our inferred reserves on about $7 per ounce silver and about $300 an ounce gold. We don't really believe that we will see it go much below that again. But no one can ever really be certain.

You are about to announce a very profitable discovery. Could you give us some information about that?

We have had four joint venture partners that have drilled this particular property since we started. We have already publicly released reserves of 25 million tons of 0.05 ounces per ton gold and about 0.50 ounces per ton silver and significant credits of lead and zinc. We know that the reserves are still in the probable category. We don't have enough holes drilled to put it in the proven category, which is what we are doing now. We feel very confident of our reserves because of the number of holes we have drilled, but it could still go 20 percent either way. That is what "probable" means. Besides finding the ore, the next problem was what percentage of the gold and silver can be recovered profitably. We have completed extensive metallurgical testing and we have come up with a very cost-effective method for recovery of gold and silver. This method will make this particular deposit, which normally would not be that profitable *very* profitable. The technique was developed by the Colorado School of Mines Research Institute. From all the information that we have at this point, including the recent bulk testing, indications are that the ore is worth about $35 per ton at present prices in the ground. We can recover about $27 worth of that at a total cost projected to be around $7 to $8 a ton. We feel that we are going to make, after all operating costs, expenses, and severance taxes, $19 to $20 per ton on this property. We plan on mining 10 thousand tons per day. In mining terms, that is extremely profitable. Operating costs on most producing open pit precious metal deposits range between $12 and $25 per ton. The technique planned will result in significantly lower costs compared to other open pit mining operations.

Could you explain the process that USMX will use to recover the gold and silver on this property?

Yes, we are going to use a gravity preconcentration process using devices that are called Reichert cones and Reichert spirals. Our recent pilot test results on bulk samples of the ore deposit indicate that we can recover over 90 percent of the gold, about 80 percent of the silver and around 75 percent of the lead and zinc in the deposit at a very low operating cost. In addition, it will significantly reduce the capital investment required to put the property into production. The conventional cyanide or flotation mill that would normally be required for this property would amount to an investment of $100 million. We think with the Reichert devices, we can get by with $20 to $25 million capital investment on the same property. Operating costs on the conventional cyanide or flotation mill would be between $12 and $15 a ton. Here, we think we are going to get by with $5 a ton.

What does all of this mean to USMX?

It means that at present prices of $400 an ounce gold, $10 an ounce silver and about 20 cents for lead and 40 cents for zinc, we have discovered a billion dollar ore deposit. At the planned production rate of 10,000 tons per day, the property should generate, after all operating expenses, at least $190,000 a day for the USMX-CM joint venture. Production is estimated to continue for 10 years. We still don't know what the total reserves are, as the deposit has still not been completely defined. Significant potential still exists for further expansion. Based on 100 drill holes on approximate 100 foot centers, drilled by eight different companies, we are confident of our announced reserves.

What are your plans for the future then?

We are hoping the price of gold and silver stay up! We are going to make *absolutely* sure the ore is there by large scale underground bulk sampling. We have a continuing drill program on the property right now to hopefully expand the reserves. Our joint venture partner, Centennial Minerals Ltd., has agreed to invest another $2 million in the next year to make certain that everything is right. This money will provide complete engineering, pre-feasibility and metallurgical studies so that a final production decision can be made. Then, the *real* grief starts: construction, engineering, permitting and the usual start up problems and an investment of $20 — $25 million. We believe that we can still make a profit with gold, as low in price as $200 an ounce. There are very few companies that have been able to meet those kind of costs. There are only three of them in the U.S. that I know of.

Any question we have not covered?

The only thing I would like to mention is that the Denver OTC Market has taken a lot of abuse over the last two years. I think a lot of it is justified, but there still are little companies around that do deserve financing and actually become successful. USMX is still not through the woods yet, but I really, *sincerely* believe that we have found the "pot of gold" at the end of the rainbow. It doesn't happen overnight. It takes patience, perserverance, commitment and faith.

JD.5

June 2 — 1903

FROM: W. L. KURTZ

TO: Exploration Mgrs + Staffs
Sell, Vikre, Bella, Smith, Harper, Gray, Gee

FTG requested this be sent to division offices — if you want to see the photos I will send you the originals — but please return

W. L. Kurtz

New York, June 16, 1983

RECEIVED
JUN 20 1983
EXPLORATION DEPARTMENT

FILE MEMORANDUM

Telfer Mine
Western Australia

In October 1982 I visited the Telfer gold mine which is located in a remote portion of northeastern Western Australia. The area is extremely flat with the largest mountain rising no more than 100 ft. above the surrounding plains. The rocks which host the mineralization are exposed on two structural domes which partly overlap at one end forming part of the larger so-called Telfer dome. The mineralization occurs in the Proterozoic Yeneena group which is about 19,000 meters thick. Minor Proterozoic granites have been mapped roughly 20 miles west of the mine site. An earlier description of the deposit by H. C. Williamson is attached.

The Telfer Formation, which is host to the mineralization, represents a transition from underlying quartzitic rocks to a siltstone turbidite series and consists primarily of silty shales which average about 800 meters thick. This shaley sequence is overlain by arenites and dolomites.

The gold mineralization is confined to the bottom half of the Telfer Formation in three separate horizons. The low grade footwall ore occurs at the base of the Telfer formation in a fine grained sandstone. A second ore horizon called the Middle Vale Reef occurs in a siltstone bed about 20 meters above the base of the Telfer Formation. This is overlain by another fine grained sandstone and then 500 meters of mixed fine grained sandstone and siltstone which contain numerous low grade beds of mineralization. All three horizons of gold mineralization are very strongly strataform and stratabound.

The footwall mineralization, which has not been well studied, consists of bedded gold and pyritic stockwork quartz veining which appears to be related to conjugate and unloading joints. The gold appears to become more stratabound at the top of the basal sandstone. The veins are not considered to be hydrothermal feeders from an underlying source. Grades are generally lower and reserves are not extensive.

The Middle Vale Reef ore is the highest grade and was the original mineralization developed at the mine. When unfolded, the Middle Vale Reef measures about 1.5 x 2.5 kilometers and is ore grade throughout, although there is fair discontinuity in the stratigraphy and thickness. The mineralization varies from about 2-12 meters in thickness and varies up and down through a 20 meter stratiggraphic thickness. Pyrite is localized near the top of the Middle Vale Reef

and is locally massive within it. The pyrite contains traces of chalcopyrite within the pyrite grains and also has a high arsenic content. Gold is locked in the pyrite. The gold is very fine in terms of its silver content (which was not given). The highest arsenic values in the ore were said to be around 2000 ppm. The pyrite content can vary from a few percent to 80% over 20 centimeters or so. The central part of the Middle Vale Reef has the aspects of a bedded quartz vein containing 10-50% white quartz with pervasive iron staining and this rock commonly contains gold grades in excess of 1 oz. Gold mineralization tends to die out quickly to the west in the dome, but extends more than 200 meters downdip into the east limb where the grades taper off more gradually. There is said to be a rather sharp break at the top of the Middle Vale Reef ore zone to a siderite-rich bed which extends for about 6 meters above the ore and contains very little gold. Several photographs of the mineralization are attached.

The E reefs which lie about 500 meters above the Middle Vale Reef are generally lower grade and more erratic in their distribution. The siltstones which enclose the E reefs are also sideritic with 5-20% carbonate which is confined to the bedding. The siderite extends an unknown distance beyond the area of known gold mineralization. In the West dome the E reefs occur in 6-10 different overlapping en echelon beds. The E reefs in the West dome are also characterized by surface leaching of gold to depths of 10 to 20 meters with no recognizable gold enrichment. Newmont geologists first flew in and landed between sand dunes to confirm the assays submitted by a prospector. They have said if they had sampled the leached E reefs, they would have rejected the prospect. Instead they sampled the Middle Vale Reef.

Jim Stewart, the Chief Mine Geologist, said that the current data favors introduction of gold mineralization into a shallow marine environment by hot springs. There is a student working on a PhD thesis covering the entire deposit which should be completed by 1985. The only deposit that might be similar to Telfer which Stewart knew of was Pilgrims Rest in South Africa. Stewart said that there is no real change in grade in the mine area between the supergene oxidized zone and the hypogene sulfide-bearing zone. In addition, he said that there is no obvious correlation between the percent of iron oxide or pyrite and the gold assay. Stewart estimates that the uneroded Telfer system originally contained more than 32 million oz. of gold. The nature of the sedimentary environment is still unclear because several visitors have identified what they think are casts of various evaporite minerals in the three gold-bearing horizons.

My impression of the deposit is that it is undoubtedly sygenetic or at least the gold-bearing fluids were introduced into a lacustrine or shallow marine environment, with gold being precipitated at some later time. There were no obvious hydrothermal vent areas in the areas we saw, but neither was there any type of pervasive rock alteration. I see no reason that a similar deposit couldn't form anywhere gold-bearing water is introduced into a shallow aqueous environment such as a playa lake in Nevada. Telfer is completely unlike Carlin-type deposits to which it has been compared in the past and is not

similar to any other gold deposit I have seen or read about. It may be the sulfide facies equivalent to the hematitic strataform gold ores such as Morro Velho, which I have not seen, and sounds vaguely similar to gold deposits in the Pritchard Fm. described recently by John Balla (memo attached).

The mine came on stream in 1977. There are 4 mine geologists in addition to Stewart and 200 employees overall. The average length of stay of an employee at the mine is 20 months. All stripping is done by the contractor. The ore is mined by Newmont using cats equipped with rippers. The current mining rate is 500,000 tons of ore/year within overall stripping ratio of 33:1.

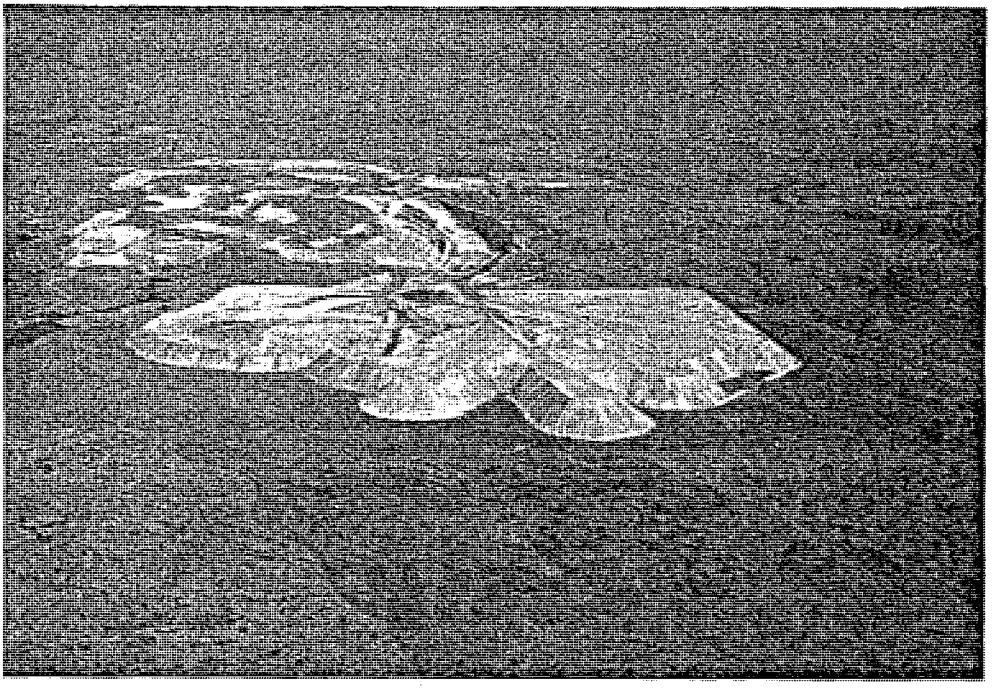
The reserves require drilling at a 25 meter grid. Scheduling for the purposes of mining requires drilling on a 12-meter grid before stripping begins. The main dome required roughly 1500 holes to define the reserve. All drilling was done by reverse circulation rotary with an air vacuum and samples were collected on half-meter intervals. Newmont is currently doing some heap leach research at the mine and an initial test on a thousand ton pad is complete.

Original signed by
F. T. Graybeal

F. T. Graybeal

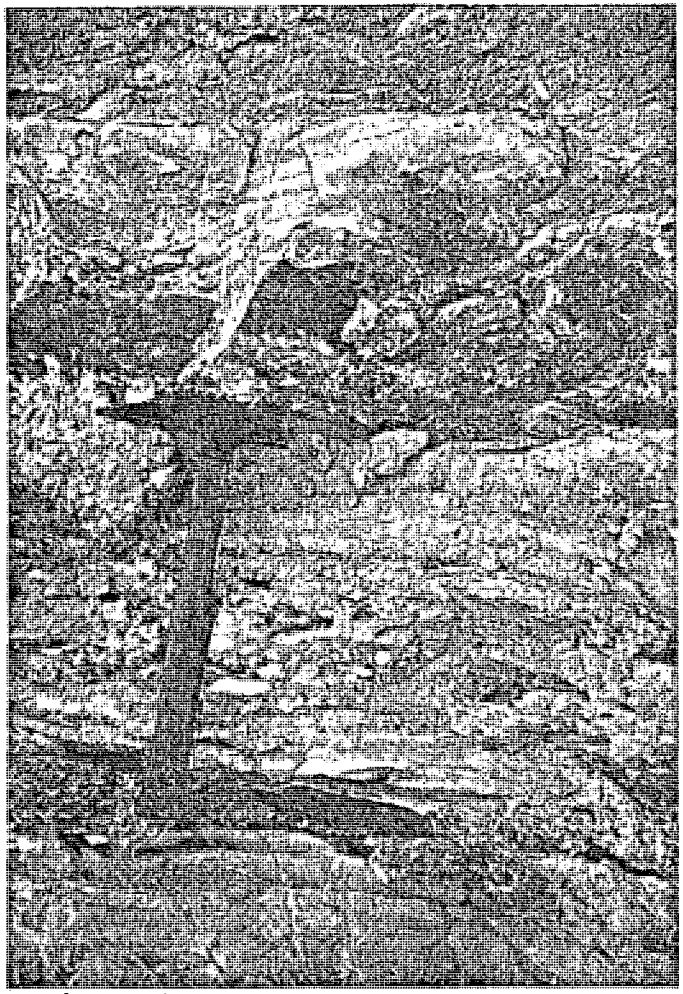
Attachments

cc: WLKurtz
Route to
JDSell (and staffs)
PGVikre
JCBalla
DMSmith, Jr.
DDHarper
RSGray
REGale
return to WLK



Telfer Mine looking northwest. The arcuate shape of the pit follows the shape of the Telfer dome. Footwall mineralization on the left side of the pit; E Reefs on the right side.

Middle Vale Reef outcrop. Former pyrite content in this area was 10-20%. Free gold is easy to find. Outcrop forms a dip slope which dips into the pit seen behind vehicle. Assays of the rock commonly exceed 0.5 oz. Au.



Typical Middle Vale Reef. The highest grades in this zone are usually confined to an obvious 1-2 ft. thick zone of heavy quartz and iron staining. In this picture the high grade (about 1.5 oz. Au) extends from iron staining at base of hammer to top of iron staining 6 in. above hammer. Mining commonly takes lower grade material above the high grade bed in the picture.

Typical E Reef mineralization. Zone extending 2 ft. above base of pen averages about 0.4 oz Au. E Reef mineralization has very little massive white quartz and only minor iron stain as compared to the Middle Vale Reef. E Reef in picture was, however, very quartzy (granular) and poros (could be due to weathering). The E Reef bed may have contained more quartz than beds above and below but otherwise was not greatly different from barren rock.

July 12, 1978.

FILE NOTE:

Telfer Goldmine
Paterson Range Sheet
SF51-6-2346-I, W.A.
J.V. Newmont (70%) & BHP (30%)

1. INTRODUCTION

This note summarizes a talk given on the 4th July, 1978, to the Joint Meeting of the Geological Society of Australia and the Australian Institute of Mining & Metallurgy at the University of Western Australia by Dr David Tyrwhitt, Exploration Manager W.A. for Newmont Pty. Ltd., on the Telfer goldmine situated in the Great Sandy Desert in the North-West region of W.A. at 21°40'S, 122°12'E.

2. SUMMARY & CONCLUSION

Newmont discovered and, with BHP, brought into production a relatively small high grade open cut goldmine containing 3.8 Mt, grading 9.6 g Au/t, in the remote desert of W.A. The total cost was $27 million. Intense activity by Newmont and others has failed to find a similar mechanically enriched syngenetic ore body in the stratigraphic extensions away from the two principal gold occurrences at Telfer.

3. LOCATION & ACCESS

The goldmine can be reached by air or by 300 km long graded dirt road leading from the abandoned mining camp of Mt Sydney. Mt Sydney is situated 230 km by dirt road from the small town of Nullagine on the Great Northern Highway; by road Nullagine is 320 km from the major town of Port Hedland and 1364 km from Perth.

The goldmine is situated on the eastern flank of a discontinuous low range of bare rocky hills known as the Paterson Range. The hills rarely rise more than 100 metres above the plateau covered with west north west striking fixed sand dunes of the Great Sandy Desert. The dunes make initial access into the Paterson Range extremely difficult but oil explorers and prospectors had made tracks across the Desert before the gold discovery.

4. HISTORY

In August 1971, Ronnie Thomson, a geologist employed by Day Dawn Minerals, sampled the gossans and quartz veins of the Paterson Range and obtained samples which contained up to 3 ozs gold per ton with minor Cu. Day Dawn failed to act on Thomson's recommendation to acquire the ground. Three months later, Day Dawn ceased business and Thomson was released. Thomson joined Newmont as a contract geologist in April 1972 and focused Newmont's attention onto the gold samples from Paterson Range. Field inspection by Newmont led to the staking of 21 mineral claims over the prospect in May 1972. Dr Tyrwhitt states that Thomson and his assistant geologist were the first to sample the gossans and quartz veins.

Our file record indicates that the prospect was first recognized,

Turcaud induced several companies to examine the area for Cu, including Anglo, Day Dawn and WMC, but only Day Dawn assayed for gold even though the gossans and quartz contained visible free gold. The prospect was offered to Asarco by Turcaud in May 1972; however, the records show that by this time Newmont was pegging out the main deposit.

5. GEOLOGY

5.1 Regional Geology

The Paterson Range consists of a folded metamorphosed middle Proterozoic sequence of quartzite siltstones, shales and dolomites lying unconformably on the Rudall Metamorphics and overlain by horizontal Permian clastic sediments. Granites, dated at 600 Ma, intrude all the Precambrian sediment. The stratigraphic column as recently defined by Geological Survey of W.A. is set out below:

	Permian Clastic Sediments	
		Unconformity
Yeneena Group	Wilki Quartzites	
	? Dolomite	
	Telfer Sandstone	
	Malu Quartzites	
	Isdell Dolomite	
		Unconformity
	Rudall Metamorphics	

The Yeneena Group is folded into distinctive domal structures.

5.2 Mine Geology

The Telfer Sandstone consists of a thick sequence of sandstones separated by thin siltstone bands and has been divided into the following units by Newmont:

Camp sandstone
Outer siltstone with minor sst + dolomite
Rim sandstone
Upper Vale siltstone
Median sandstone
Vale siltstone 15 m thick.
Middle ~~Bell~~ Vale Reef gossanous siltstone 0.5 to 2 m thick
F.W. sandstone
Lower Vale siltstone
Core sandstone

The Middle ~~Bell~~ Vale Reef is a 4 km long continuous gossan horizon outcropping irregularly around the Telfer Dome. The beds dip uniformly outwards at 25° and are cut by local minor faults. The bulk of the ore is contained in the Middle Vale Reef as an oxidized free milling ore with the gold associated with limonite - quartz impregnation within a siltstone host rock mineable by open pit to a vertical depth of 60 metres. Auriferous stratiform quartz reefs occurring 100 metres above the Middle Vale Reef are known as E1 and E2 horizons. They contain a smaller part of the estimated ore reserves and are not being mined at present.

Gold mineralization, at about half the grade of the Telfer deposit,

km from the Telfer Dome.

The depth of the oxidation varies from 60 to 100 metres and the water table is about 60 metres below the surface. Below the water table the ore is a pyritic siltstone with grades about half the oxidized ore grades.

6. EXPLORATION AND ORE RESERVES

Initially, exploration was by percussion drilling, which indicated a large low grade ore body; however, later vertical reverse cycle drill holes on a 200 metre grid outlined a smaller, higher grade, ore body. Subsequently, the drilling pattern was closed into a 50 metre grid. The final grade and tonnage did not vary greatly from the initial estimate based on a 200 metre grid pattern.

The ore reserve calculations were made by computer and checked by conventional methods. The ore reserve calculations were based on the following features:

1) sampling interval 0.5 metre;
2) gold assays were uncut;
3) the maximum depth of ore was 60 metres vertically below the surface with a maximum waste to ore ratio of 10:1;
4) plans and sections at 1:250 scale;
5) sampling methods and assaying results for 3 drill holes compared favourably against results obtained from sinking shafts over the three holes and sampling the ore horizon.

The drilling programme of some 400 drill holes outlined 3.8 Mt of ore averaging 9.6 g Au/t. The mill head grade of ore treated to date correlates very well with the ore reserve grade, giving further confidence to the ore reserve estimates.

7. MINING AND MILLING

The cost of the facilities, including a townsite for 300 people, was A$24 million, with exploration and feasibility costing an additional A$3 million.

Currently, nine-tenths of the overburden is removed by contractors working 13 shifts/fortnight. The last two metres of the siltstone H.W. and the two metres of ore (average) are removed by Newmont. Newmont operates 6 days/week and removes 2500 tons of ore and waste/shift, including 1200 tons of ore. The siltstone can be ripped with a D9 bulldozer but the H.W. sandstone often has to be blasted by the contractor.

Gold is recovered by jigs and counter-current decantation cyanide leaching. Initially, the gold recovery averaged 93% but recovery now averages 96%. (The head grade of the ore treated so far indicates an 86% recovery in the first year of operation but some gold would be locked up in the mill circuit.)

[illegible] of ore at an average grade of 15.2 g/tonne

1977 stand at 3,517,000 tonnes, averaging 7.3 g/tonne. The new estimate contains an additional 363,000 tonnes of ore, grading 11.3 g/tonne, down dip from the current open cut.

Attached is a geological map of the Telfer goldmine and a road map showing the location of the deposit.

H.C. WILLIAMSON

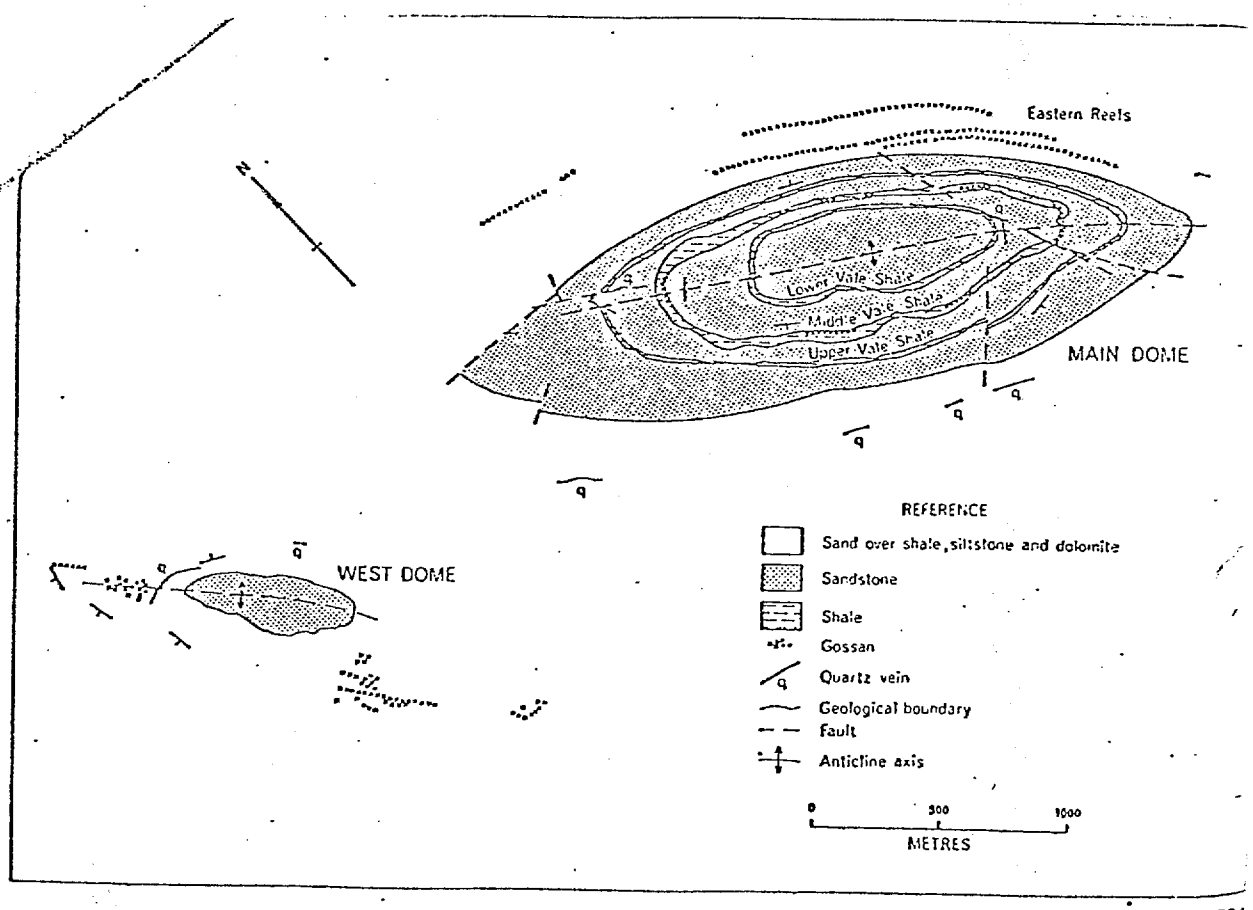


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Figure 46. Telfer Dome gold prospects—geological map (after Newmont Pty. Ltd.).

July 5, 1983

FROM: W. L. KURTZ

TO: JC Bella
DM Smith
P Gillikoe
JD Sell
DD Harper
RS Gray
RE Hale

This should have been included with the TEXPER data

ASARCO

Northwestern Exploration Division
E 920 Wolverton Court (N 2900 Nevada)
Spokane, WA 99207
(509) 489-7870

FTG

FILE

APR 26 1983

APRIL 21, 1983

RECEIVED
APR 25 1983
EXPLORATION DEPT.

MEMORANDUM TO: W.L. KURTZ

FROM: J.C. BALLA

SUBJECT: GOLD IN THE PRECAMBRIAN

Reference is made to your memorandum of April 8 concerning the abstract on placer gold in the Prichard formation in the Abstracts with Programs for the 1983 G.S.A. Rocky Mountain and Cordilleran Section meetings, (I had seen the abstract, as I am a member of G.S.A., and routinely get the abstracts volumes).

As I mentioned to you on the phone, I have been aware of gold mineralization in the Prichard for about 10 years, having looked at a number of gold prospects in northern Idaho and western Montana. One of the interesting aspects about the above general area is that whenever there is a significant exposure of upper Prichard formation, there are placer gold diggings downstream from it.

In outcrop, the gold occurs in two general classes. (There is probably a continuum between the two classes). The first are high angle (to the bedding) quartz veins. These quartz veins are along major fault zones and go for miles. These generally represent the major gold mines of the region. The other class are small quartz veins which are sub-parallel to parallel to the bedding. These veins are generally discontinuous along strike. They pinch and swell, forming pods several feet or tens of feet along strike, and then will pinch down to nothing very quickly. None of these veins have ever amounted to anything, production-wise.

Having looked at a number of these prospects, I get the impression that the amount of gold veins is proportional to the amount of deformation of the Prichard. The more deformation, the more gold-bearing quartz veins.

I also have the impression that the gold may have been originally concentrated in chert beds in the Prichard during non-deposition of the Prichard. That is, during periods of non-deposition of the shales, chert was chemically precipitated, and accumulated gold. Subsequent deformation remobilized the chert beds into gold-bearing quartz veins.

I have also never seen an area where there was a great concentration of gold-bearing quartz veinlets.

In the Seward Peninsula of Alaska, Pete Sainsbury worked for the U.S.G.S. for several years, and presented a paper entitled "Geologic Factors Controlling the Distribution of Placer Gold Deposits of the Seward Peninsula, Alaska," copy attached.

Sainsbury came to a similar conclusion concerning the placer gold on the Seward Peninsula: It was derived from the Precambrian York State, and subsequent deformation remobilized it. Whether the York State and the Prichard formation are correlatable is unknown.

Finally, with respect to looking into this environment, I think it would be a worthwhile program. However, our commitments for this year effectively preclude such a program.

John C. Balla

JCB/dt
Enclosure

cc: R.L. Brown with enclosure

Geologic Factors Controlling the Distribution of Placer Gold Deposits of the Seward Peninsula, Alaska

by

C. L. (Pete) Sainsbury

1976

Paper for oral presentation at the Northwest Mining Association Annual Meeting, December 3-5, Spokane, Washington, 1976

Copy for reproduction

ABSTRACT

Ranking as the second largest producer of placer gold in Alaska, the Seward Peninsula had produced more than 6,260,000 fine ounces of gold by 1961, and is currently producing from 2 dredges and several smaller operations. Placer gold deposits are widespread over the Peninsula, and their areal distribution is practically coincident with the outcrop areas of the York Slate. This formation, of Precambrian age, is composed principally of fine-grained quartz siltite with variable amounts of carbonaceous and/or graphitic material, and subordinate amounts of limestone, graywacke, shale, slate, and calcareous graywacke and siltite. Although the York Slate governs the distribution of placers, the size and richness of individual placers is dependent upon several other factors, the most important of which are: 1. The degree of deformation of the York Slate 2. The size and composition of intrusive bodies in the slate and other rocks 3. The nearness to major overthrust faults involving the York Slate 4. The nearness of gold-bearing veins in the slate containing native gold along with such minerals as stibnite, arsenopyrite, galena and other sulfide minerals 5. the geomorphic history of the region, especially the distribution of Pleistocene glaciers, and sealevel changes by both eustatic and tectonic changes of shoreline. Many placers created during Tertiary and Pleistocene time undoubtedly were buried beneath lava flows of Late Tertiary and Pleistocene age, and were both destroyed by glaciation and preserved beneath glacial deposits.

The relationships are so clear-cut that one is tempted to conclude that most of the gold must have been present as an original, syngenetic constituent of the slate. However, the richer placers are associated with mineralized structures of probable Cretaceous or early Tertiary age; in such places, gold was probably deposited by the carbonaceous material in the slate. Recognition of the various factors outlined above can lead to the prediction and possible location of new placers.

INTRODUCTION

Between the years 1961-1972, the writer and his assistants completed the geologic mapping of almost the entire Seward Peninsula during the course of mineral investigations for the U. S. Geological Survey. During the early years, detailed mapping at scales 1:63,360 in the Lost River area led to the discovery of important lodes containing fluorite and beryllium (Sainsbury, 1962, 1969a). Beginning in 1967, mapping was re-oriented toward the reconnaissance mapping of the entire Peninsula at scales of 1:63,360 and 1:250,000. During this mapping, the writer had the good fortune to map in the vicinity of practically every known placer deposit of the Seward Peninsula and nearby areas. These placer deposits included those of tin, gold, mixed tin and gold, and mixed gold and platinum.

As a result of the mapping many new interpretations of structure, stratigraphy and localization of ore deposits were elucidated and published (see references cited list). After leaving the Geological Survey in 1972, the writer continued work in nearby regions, and in 1974 returned to the Peninsula in search of primary uranium deposits, several of which were found in the Darby Mountains. These deposits were "rediscovered" by the Geological Survey in 1976 (Miller, et al, 1976). Because large amounts of new information has been published in two summary papers published in 1974 and 1975 (Sainsbury, 1974, 1975), no attempt is made in this paper to present all the background data needed to support conclusions made herein. For all the supporting data, the reader is referred to papers bySainsbury and his many co-workers, including Donald Grybeck, Travis Hudson, Rueben Kachadoorian, Thomas E. Smith, Thomas Richards, William R. Marsh, and Rodney Ewing. These papers are cited in the References Cited list; in addition, papers by Gordon Herreid, John J. Mulligan, Thomas P. Miller and co-workers, and C. L. Hummel give information more recent that that collected by the oldtime workers in the area, notably A. J. Collier, A. H. Brooks, G. B. Richardson, P. S. Smith, and a host of lessor-known workers. Omission of any worker from the References Cited list is not a purposeful slight. In particular, papers by D. M. Hopkins

on the Pleistocene history of the Nome area have been of great value.

For a good reference publication that gives many facts about placer gold on the Seward Peninsula and elsewhere in Alaska, the reader is referred to Cobb(1973). For very good descriptions of placer mining on the Seward Peninsula in its heyday, see Moffit(1905) and Collier, Hess, Smith and Brooks (1908) amoung others.

In some respects the conclusions of the writer are in good agreement with earlier writers; in others, especially with respect to the origin of the Nome placers from reworking of glacial deposits, the writer has some sharp disagreements with published literature. It is perhaps especially pertinent to note here that many of the great placer gold areas of the world are associated with belts of black slates, and that lode deposits of the typical "banded quartz" type are often very rich, the gold being associated with carbonaceous films in the quartz. In this respect, the lode and placer deposits of the Seward Peninsula conform to the norm.

The method of presentation of this paper will be largely pictorial and illustrated by kodachromes of the actual rocks, mines and geologic relations depicted by good published maps. The written material that follows will conform to that method of presentation, and must of necessity be without the maps and slides.

LOCATION

The Seward Peninsula adjoins the Bering Straits at its narrowest part, and is most probably a continuation of the Chukotsky Peninsula of eastern Siberia (figure 1, index map). If sealevel were lowered by a few hundred feet, land would be continuously exposed across the Bering Straits in a band more than 100 miles wide, and the geologic units exposed on the Seward Peninsula would undoubtedly be found extending into Siberia beneath a thin veneer of marine sediments that would include glacial moraine deposits from a major ice sheet that

moved south through the Bering Strait area and left moraine now covered by seawater as far south as St. Lawrence Island in the Bering Sea west of Nome.

GENERAL GEOLOGY

The Seward Peninsula is made up of rocks of Precambrian, Paleozoic, Mesozoic and Tertiary age, all of which were intensely deformed. Eastward thrusting of all the pre-Tertiary rocks is recorded throughout the Peninsula, and the criteria of superposition of units by descending age is applicable only locally. The Precambrian rocks consist of gneisses exposed in the Kigluaik Mountains (figures 2, 3), and of slates, schists and thin carbonate rocks exposed elsewhere. Two main types of schists are recognized- the deformed and schistose equivalents of the York Slate, and diverse, generally green-colored schists of mafic composition, characterized by chlorite, albite, amphibole, quartz and sphene or rutile. Many of these rocks contain glaucophane, and many of the other schists are believed by me to be retrograded blueschists. As such, these rocks have an unmistakeable relationship to widespread tectonic deformation under relatively shallow conditions. All the Paleozoic rocks dated by fossils consist of carbonate rocks, as does the latest Precambrian unit, the Kanauguk formation. The Kanauguk formation is transitional above the York Slate, and is important in excluding placer gold deposits. The Paleozoic carbonates include thick sections assigned to the Ordovician, Silurian, Devonian and Mississippian? Systems.

The Mesozoic rocks, confined almost entirely to the eastern part of the Peninsula, consist of mafic volcanics of Jura-Cretaceous age, and of sandstones, shales, graywackes and silstones of Cretaceous age. Some of these rocks are coal-bearing. During the Tertiary, coal-bearing beds were deposited in local basins at several places on the Peninsula; these sandstones and shales are mostly covered by muskeg and tundra, for the Tertiary basins are still the lowest-lying areas of the Peninsula. In latest Tertiary and earliest Pleistocene time, volcanic flows and volcaniclastic rocks were deposited over much of the central

Seward Peninsula and possibly elsewhere. Remnants of these andesitic flows cover some of the placer gold deposits in the Inmachuk River area, and to the south, dark basaltic flows from Lost Jim cone mantle the older volcanics, and are probably only a few hundred years old. In the Devil Mountain area, Recent tuff, ash and local flows cover hundreds of square miles.

In Cretaceous time, after the culmination of the intense eastward thrusting, great quantities of igneous rocks ranging in composition from syenite to quartz monzonite were intruded into areas that are now the axis of mountain ranges. These rocks range in age from about 105 M.Y. to about 90 M.Y. In latest Cretaceous time, several stocks of biotite and biotite-muscovite granite were intruded on the western Seward Peninsula-these are the "tin granites" which have guided all the main sulfide-cassiterite ore deposits, as well as the fluorite-beryllium lodes of the Lost River area. These granites range between 68 and 74 M.Y. in age. Because only the Precambrian rocks are of interest to the interpretation of the placer gold deposits, only these are further discussed here.

York Slate

This formation extends throughout the Seward Peninsula, and also into adjacent areas, such as the Candle quadrangle, where it was assigned to the Paleozoic by workers in that area. Although many slate areas are separated widely, and hence are not easily correlated, it is indisputable that at many of these separated areas, the distinctive orange-weathering argillaceous and dolomitic limestone of the Kanauguk formation lies transitionally above the slates, giving confidence to the correlation.

The York Slate displays many variants, the most common rock type being a black, slaty-to-blocky rock with a very high silica content-locally reaching 92 percent. Such rock most likely would be classed as quartzite elsewhere, but because of its dark color and carbonaceous content it is here called siliceous slate or quartz siltite. In other variants, calcite and argilleceous material increases, but the dark color persists even in the carbonate-rich facies. In the least deformed and

least metamorphosed areas, particularly in the western Peninsula and around the headwaters of the Kougarok River, original bedding, cross-bedding and original carbonaceous laminations are present. Thin sections show that such rock is composed of silt-size rounded quartz grains with small flakes of white mica and carbonaceous clots. With moderate folding, silica begins to migrate, and small white vitreous quartz veinlets become conspicuous. With continued deformation, the quartz veinlets become larger, more conspicuous, and are folded and fractured. In some, native gold can be observed as minute specks and flakes, especially where veinlets are fractured. Many of these larger veinlets contain pyrite, ankeritic carbonate, and other sulfide minerals in minute amounts. Because such veinlets lie "blind" throughout large volumes of York Slate, the gold and other sulfide minerals are considered to be derived from elements deposited syngenetically with the slate. It is these early, mineralized veinlets that account for the small amounts of placer gold found everywhere in streams in large areas of slate.

With continued deformation, the carbonaceous material clots, and a semi-gneiss develops that is characterized by thin laminations of white quartz. Near major overthrust faults, intense shearing and recrystallization of quartz and total expulsion of carbonaceous material leads to a white quartz gneiss. Wherever streams flow along such rock, placer gold deposits have been worked at least locally. It took many years of meticulous mapping on the Seward Peninsula before I and my associates were able to recognize that all these units were merely tectonic equivalents of the York Slate, and some workers to this day are not willing to assign them to the York Slate.

Nome Group

This designation was given by the oldtimers to rocks near Nome that consist of chloritic schists, graphitic schist, limestones, marble and marble schists. I now consider the Nome Group rocks to be intercalated in the York Slate, and they are not everywhere correlative. For a full discussion of the problem, see Sainsbury (1975, p. 15-16). Of almost equivalent

chemical composition but of certain volcanic origin is the Casadepaga schists of Smith(1910), which forms a wide belt of somber green schists on the west side of the Casadepaga River.

The most significant factor of the chloritic schists is that they contain ESSENTIALLY NO PLACER GOLD. In fact, where the mineralized fault structures so important in the Nome area pass into the chloritic schists, placer gold deposits of economic importance ESSENTIALLY CEASE. This important relation has no single exception on the Seward Peninsula, and it will become apparent in subsequent slides of maps.

STRUCTURE

The widespread thrusting of the Seward Peninsula has been described by the writer in many published papers, especially in 1975, and in a paper that describes the blueschist of the Peninsula (Sainsbury, Coleman, and Kaachadoorian, 1969). Throughout the Peninsula, rocks are made schistose, several S-planes are developed, and dragfolds locally isoclinal with limbs and cleavage horizontal are everywhere seen (photos with this paper). There can be no doubt that there was an immense amount of deformation with tectonic transport to the east. Even in the deeper gneisses, where recrystallization and tectonic smearing has largely obliterated all original bedding, local exposures show intense folding of preserved carbonate beds(photo with the oral presentation). Because A. J. Collier, an astute geologist never adequately honored by contemporary workers, was the first to recognize that thrust relations probably existed on the Seward Peninsula (Collier, 1902), I named this great thrust belt after him (Sainsbury, 1969b). P. S. Smith recognized the thrusting near Nome, but showed a photograph of the thrust sheet on Mt. Dixon, calling it an unconformity(Smith, 1910), otherwise I might have named it after him! The thrusting largely culminat prior to the injection of the main intrusives in late Cretaceous time, and the compressive effects extended beyond the Peninsula, for the Cretaceous sedimentary rocks east of the Peninsula were strongly folded into folds trending northeast (see Patton and Miller, 1969).

Because the thrust plates of Ordovician limestone in the York Mountains, on the western Seward Peninsula, were thrust northward over older thrusts that display ubiquitous signs of tectonic transport to the east, similar to the rest of the Peninsula, I must conclude that a later period of northward thrusting was superposed upon the general eastward thrusting in the York Mountains. This second event of thrusting culminated before the intrusion of the tin granites of latest Cretaceous age. Contemporary workers on the Peninsula only gradually are realizing the extent of the thrusting, but me and my co-workers have mapped thrusting throughout the Peninsula- it is widespread and real! Major ore deposits of the York Mountains are related to the Rapid River thrust, and some seeking ore deposits on the Peninsula today are searching along thrust faults elsewhere on the Peninsula.

PLACER GOLD DEPOSITS

The remainder of this paper will follow rather closely the oral presentation, largely supported by numerous slides which cannot be reproduced for the printed paper. In order to illustrate the close correlation between placer gold deposits and the York Slate, slides 5 and 6 show, respectively, the known major placer gold and tin deposits, and the main outcrop areas of the York Slate. As can be seen from the slides, EVERY SINGLE PLACER OF ECONOMIC IMPORTANCE lies within areas where the York Slate forms most of the bedrock! This is more than coincidence.

In slide 7 , the larger intrusive rocks of granitic compositionare added. Note again that all main placer gold deposits have either outcropping granitic intrusives, or abundant evidence, such as tactites, hornfels, etc, that granite lies beneath the area at shallow depth. Of especial interest is the fact that very large batholiths have invaded the Darby Mountains, yet there are no important placer gold deposits in or near the Darbies! The single small deposit known lies within the York Slate near the north contact of a large intrusive of quartz monzonitic composition.

In slide 8 , the the main mappable mineralized fault zones are added to the map. We immediately see that the most important

placer gold district of the Seward Peninsula, the Nome area, is marked by a system of mineralized fault zones trending NE through the York Slate and into the overlying limestone of the Kanauguk formation, and the chloritic schists. Note too that not a single placer deposit of importance occurs within the chloritic schists- all are restricted to the York Slate or marine strand-lines on the slate! Again, the relationship is not coincidental! Note too the mineralized faults SW of the Hannum Creek and other placers on the Inmachuk River. The richer placer mined by Willie Foster on Hannum Creek stops abruptly at the mineralized fault zone. Samples taken by me from this zone contain gold, silver, tin, antimony, lead, zinc, arsenic and copper. Note too that the richer placers in the Solomon area lie immediately downstream from the lode gold mine at Big Hurrah, and the gold-bearing stibnite veins nearby, known as the Stepovich mine. To my knowledge, the Big Hurrah lode was the onlygold-quartz mine of importance in this part of the Seward Peninsula, although several small lodes were mined in the Nome area. Stibnite from the Stepovich mine contains coarse gold embedded in stibnite, a relationship which is noted at many other places in Alaska.

If we look west to the Lost River area, we see an apparent exception to the rule that mineralized faults and lode deposits insure productive placer gold deposits. Certainly the Lost River area contains by far the largest and diverse deposits known on the Seward Peninsula. These consist of sulfide-cassiterite deposits in altered dikes and the underlying granite, as well a very large deposits of fluorite containing beryllium minerals (Sainsbury,1969). Tom Smith and I discovered all these deposits between 1960-1964.

However, in spite of the large and diverse deposits, only a small amount, about 50 tons, of placer tin with essentially no gold was mined downstream from the deposits. Not too far west, the placers at Potato Mtn produced over 2000 tons of placer tin, with substantial gold-an apparently anomalous situation, which however, can be explained on the basis of glacial scouring. Lost River valley was heavily glaciated during

during the York Glaciation of Early Wisconson age, some 20,000-40,000 years ago, whereas the Potato Mtn area was not. Glacial scouring of the Potato Mountain area was much older, and related to a probable polar ice cap that I think covered this part of the western Seward Peninsula, as well as the adjoining Chukotsky Peninsula. I proposed this glaciation in 1967 (Sainsbury, 1967b), based upon the fact that I thought the topography was ice-scoured, and upon the fact that glaciers more than 2000 feet thick in the York Mountains were prevented from flowing out onto the lowlands to the north and depositing moraine. The only possible explanation was that another glacier had blocked the valley glaciers, and as this glacier would have occupied the lowlands now the Chukchi Sea, it must have been a polar glaciation. Subsequently, Dave Hopkins and other marine geologists recovered striated cobbles and moraine beneath the Bering Sea as far south as St. Lawrence, thus proving that an ice sheet had indeed flowed south through the Bering Straits. The tin placers at Potato Mountain and Cape Creek on the west tip of the Peninsula formed after the retreat of this ice sheet, but time since the York Glaciation was too short to allow the accumulation of important placers at Lost River. It should also be noted that the bedrock at Lost River is limestone, whereas that at Potato Mountain is the York Slate. Small amounts of placer gold were mined from the Anikovik River and Baituk Creek in addition to that in the rich tin placers at Buck Creek. All these creeks flow on the York Slate.

In the next slide, I have added the larger glacial deposits in solid orange, and the areas known to have been glaciated sometime during the Pleistocene in lined orange. We note immediately that not a single placer of importance lies within any of the extensive glacial deposits, in spite of the fact that such streams as the Sinuk River and Canyon Creek have reworked and removed large volumes of ground moraine and outwash gravels. However, we do have auriferous beach deposits in the subsurface at Nome, and according to Hopkins, Tagg and Nelson of the USGS, the richer gravels and sands offshore at Nome lie in part on glacial moraine. I consider the Nome area to be a special case that I will discuss later.

In the next slide I have added the larger areas of volcanic rocks as well as the extensive muskeg-covered lowlands (yellow), and our interpretative picture is now complete. We see that large areas of the central Seward Peninsula that might be expected to have placers were covered by the flows and volcaniclastic rocks of the Imuruk Lava Fields, originally described and studied by Dave Hopkins. As proof that undiscovered placers should exist here we note that one gold placer beneath lava flows is being being mined by Hagendorn on the Inmachuk River. If it were not for the fact that large and rich placer tin deposits are being mined beneath lava flows in Nigeria, I might be tempted to discount the potential importance of such placers under the Imuruk lava fields. But exploration geologists and geophysicists someday will locate and mine other placer gold deposits south of the Inmachuk River.

One final comment on this map, and then we will look at some of the individual areas in a more interesting way. Note that I have shown that platinum placers are restricted to the easternmost Seward Peninsula at Haycock and mixed with gold near Granite Mountain. The Haycock deposits contain gold as well as platinum-group metals, and are quite large, though much smaller than those well-known deposits at Goodnews Bay. All the platinum-gold placers lie upon bedrock consisting of Jura-Cretaceous metavolcanic rocks of mafic composition. These mafic volcanics are unusual in two respects- they contain an unusual amount of chrome, and they are loaded with fragments of eclogite (garnet-pyroxene rock) probably derived from mantle material. The platinum is derived from the mafic volcanics and the eclogite fragments.

INDIVIDUAL AREAS

In the following slides we will look at some of the individual areas by reference to Kodachrome slides, which are much more expressive than words, starting at Nome. I will then attempt to illustrate areas that are typical of each of the main geologic controls that I have outlined.

Nome Area

It is commonly said that if a circle with a radius of 5 miles is drawn around the mouth of Anvil Creek near Nome, then almost 90% of the gold produced from the Nome area will lie within this circle. Obviously, special conditions must have led to this unusual enrichment of placer gold around Nome!

If we look at the general Nome area (slides 12, 13,14), we can see immediately that gold has been produced from many sources from the present beach landward to and beyond the Third Beach, which forms the general escarpment along the front of Anvil Mtn. As seen in slide 14 ,which because of the snow cover admirably shows the general trend of Second Beach from Cape Nome across the wide coastal plain behind Nome, both the modern and fossil beaches have been very important in localizing workable ground. In addition, other beaches not discernible from surface topography have been mined, such as Monroeville and Submarine "beaches", which may in part be reworked older beaches left on the abrasion platform of an enchroaching sea.

As we all know, the initial history at Nome involved the mining of the extremely rich stream placers of Anvil, Osborne and other short streams north of Nome. Anvil Creek alone produced more than $6 million in gold at $20/oz. That these rich stream placers once led out across an old surface subsequently removed by marine erosion is proved by the fact that richer placers on the abrasion platform in general lead across the coastal plain to the mouths of these rich streams. In addition, I would attribute part of the gold of these richer paystreaks to marine erosion of the zone of primary deposits that lies along and near to the Anvil Creek fault zone.

Following the exhaustion of the stream placers, and luckily for the thousands of miners stranded at Nome with no stream ground to stake, gold was found in the modern beach, which belonged to noone. In a critical part of Nome history, the present beach sustained thousands of miners who worked with rockers, pans, and a multitude of other devices to recover almost a million dollars from the Nome beaches. Later discoveries of the submarine and fossil beaches sustained Nome during its second cycle of

mining, and lastly, the discovery and development of cold water thawing enabled the dredges to mine large areas of frozen ground that never could have been mined if steam thawing had been required! Lastly, the increase in price of gold from $20./oz to $35/oz, and thence upward from $35./oz to the wildly-fluctuating current price provided shots in the arm at precisely times when mining otherwise was about economically dead.

By reference to figure 15, the geologic map of the Nome area, we see that all the right conditions were at Nome to lead to a very great enrichment of placer gold: the York Slate is everywhere present beneath the richer placers; the granitic intrusives at Cape Nome, at Sledge Island, at Cape Rodney and in small exposures elsewhere have mobilized gold from the slates into veinlets; a strong system of faults with known gold-bearing sulfide veins and veinlets in and nearby is unmatched anywhere else on the Peninsula (note that the 5-mile radius circle reaches almost to the northwesterly Penny River fault zone and southeasterly to the last major fault at the junction of the Nome River and the coastal plain). Lastly, a complex geomorphic history involving major fluctuations and stillstands of the sea has led to a marine enrichment superposed upon older stream enrichment.

Finally, glaciation has played an important part in the removal of gold at Nome, rather than being the source of the gold through moraine. If we look to the northwest of Nome, we see large expanses of moraine originating in the Kigluaik Mountains. This moraine floors the Sinuk River, and has been reworked by the modern and an older strandline. Yet not a single placer gold deposit of economic importance occurs within the moraine or along the beaches in the moraine. The moraine of the oldest glaciation probably contained considerable gold derived from Tertiary stream placers of great richness, and possibly contributed some gold to the younger marine deposits, but the glaciations at Nome, as elsewhere on the Seward Peninsula, effectively removed or dispersed the economic placer deposits. I think that this is an important point of economic significance.

Solomon Area

Moving now to the east of Nome (photo 16), we see that an important placer area around Solomon occurs almost entirely within the outcrop area of the York Slate. The richer placers lie downstream from gold-bearing lodes in the slates, such as at Big Hurrah and the Stepovich mine. Some placer gold continued for many miles up the Solomon River and into the drainage of the Casadepaga River, but only in areas floored by the York Slate. As soon as auriferous streambeds leave the slate and enter onto chloritic schists or limestone, the mined ground ceases abruptly. There is no exception to this rule known to me.

Council Area

East of the Solomon placers, the next area of importance is the Council district (photo 17). Here we have a complex intermingling by thrust faulting of the York Slate, the limestones of the Kanauguk formation, and the Paleozoic carbonate rocks. As seen in photo 18 , the York Slate is intensely crinkled , lineated, and cut by quartz veinlets. The contact between the York Slate and the Kanauguk formation is a thrust fault, as seen in photo 19. Although no intrusive rocks are present in any number in the Council area, the richer placers lead upstream along Ophir Creek (which was quite rich) until they end at the glacial moraine mantling a large area at the south side of the Bendeleben Mountains! It is probable that rich placers once extended much further up Ophir Creek, where they may remain buried by moraine. Note the large granitic intrusive some 5 miles north of the last placer on Ophir Creek.

Darby Mountains

Moving east of Council to the Darby Mountains, we see a very great expansion of the granitic intrusives, yet we have essentially no placer deposits, the only thing of any interest being the marine placers in Golovin Lagoon, which have not yet produced. The only placer gold deposit worthy of mention lies in a small expanse of York Slate at the north margin of the Darby pluton. Some lode uranium-thorium deposits may have

contributed chemically resistant allanite and an unidentified uranium-titanium niobate into streams within the Darby Mountains, and possibly some workable placers may exist. However, none are known to date. Note that the bedrock in the Darbies where not intrusive, is largely limestone, chloritic schist or metamorphic rocks, with little York Slate. Too, the major valleys may have been glaciated prior to the Wisconsin.

Candle Area

Substantial placer gold deposits in the Candle area have pruduced large amounts of gold, and smaller dragline operations are still producing. All the deposits lie within the York Slate, which contains several small and altered granitic intrusive rocks within the drainage basins of several of the auriferous streams. Across the Kiwalik River, the large granitic plutons are intruded into mafic volcanic rocks, and associated placers are small and lean, even though there are many large areas of mineralized bedrock, such as shown in slide 19 . Nevertheless, York Slate must extend out under the alluvial fill of the Kiwalik River, and it is entirely possible that buried placer gold deposits will be found. Certainly there must be older marine strandlines beneath the alluvium, and future prospecting may be directed toward the finding and testing of such deposits.

Inmachuk River

Placer gold has been mined throughout much of the drainage of the Inmachuk River, and one placer beneath basalt is still being mined (Hagendorn property). Most of the mining ceased about 1970, however. Note that the bedrock is largely York Slate (photo 20), which to the south is intruded by both large and small granitic intrusives. At Hannum Creek, a rich placer ended abruptly where a mineralized fault zone crosses the creek, and other parallel fault zones to the southwest are altered and stained. Limestone of uncertain age forms the bedrock north of Hannum Creek, and there are no known placers. Tactite developed in the limestone suggest that a granitic

intrusive underlies that area at shallow depth. To the east of the Inmachuk River, the argillaceous limestones of the Kanauguk formation underlie much of the country, and there are no important placer deposits. Remnants of basalt in the order of 3 M.Y. old cover somm valleys south of the Inmachuk, and the Imuruk Lava Fields have covered any old placer deposits to the south. However, it is probable that placers containing gold and possibly cassiterite are buried beneath lava flows in that area.

Kougarok Area

The Kougarok area was developed after much of the Nome district was worked out, and mining has extended to the present by a few diehards such as the Tweet brothers (photo 21), who are second generation goldminers from Teller. Photo 22 shows a typical operation of the Tweets on Washington Creek, and photo23 shows some of the typically coarse gold they mine. During the 1930's, placer miners worked most of the Kougarok River and all its tributaries that ARE UNDERLAIN BY YORK SLATE. In all cases, as soon as streams leave the York Slate, placer mining ceased. The north part of the Kougarok district included the tin granite at Serpentine Hot Springs. To the east, on Humboldt Creek, important placer gold deposits contain commercial amounts of cassiterite, and dredgeable ground remains to be mined along the lower course of the creek. Galena-bearing lodes were found by the writer and his coworkers in the headwaters of Humboldt Creek (Sainsbury, Hudson, Kaachadoorian and Richards, 1968), but they are not of commercial size. Nevertheless, they contain gold, and have probably accounted for the richness of the Humboldt Creek placers. I would like to mention that a detailed study of this area proved that geochemical prospecting using stream sediments alone would not find either the lodes or the cassiterite-bearing placers, but that panned concentrates would lead the exploration man directly to the lodes, as well as demonstrating that important amounts of tin occur in Humboldt (Sainsbury, Hudson, Kaachadoorian and Richards, 1970). This has important ramifications for geo-chemical

exploration elsewhere in the Arctic.

Dahl Creek

Photo 24A shows the placer operations at Dahl Creek, on the south end of the Kougarok, and photos 24 to 26 show a typical placer operation. The gold is fine, lies in part on a decayed and bleached, clayey bedrock and partly in the Kougarok Gravels. These gravels are frozen, and contain old peat zones as well as fossil ice-wedge casts (photo 27). I consider them to be of early Pleistocene age, and the gold to represent old slip-off slope concentrations of the ancient streams, including the Kougarok River, that was eroding the York Slate. Mining is hampered by a general lack of water, and much gold probably could be recovered if more water was available. Work by the writer and his colleagues demonstrated that fine gold is distributed throughout the Kougarok Gravels in amounts approximating 10-15 cents a cubic yard at $35.00 gold, and many more paystreaks representing concentrations on old slipoff slopes unquestionably remain hidden.

To the south, the placers at Coffee Creek were mined for almost 40 years. Large amounts of stained vein quartz in Coffee Creek and the headwaters of Dahl Creek suggest that veins contributed the coarse gold fond in Coffee Creek, and nuggets with quartz attached are found commonly. Coffee Dome shows thermally metamorphosed York Slate with abundant white quartz veinlets, and a granitic intrusive is assumed to lie beneath. Note that all the bedrock in the auriferous parts of the Kougarok and Dahl Creek areas is composed of the York Slate.

Teller Area

Going southwest toward our point of origin at Nome, I wish to discuss the placer gold deposits here, for they are somewhat unusual for two reasons- much of the gold is very coarse, and the gold may be related to mafic bodies that have intruded the York Slate and created wide belts of semi-chloritic schists around them. Photo 28 shows an operation

known locally as Bond Davis' Camp, and photos 29 to 30 show some of the typical gold being mined in 1970. Some cinnabar occurs with the gold, and many nuggets are surrounded by amalgam. In days past, many placer miners worked the valleys of the Bluestone River, Gold Run , and Deese and Coyote Creeks near Teller. The gravels are shallow and thin, and I personally believe that in early Pleistocene time, the area was glaciated. Today, however, the main glacial deposits lie to the southeast near the mouth of Canyon Creek, where they probably bury a paystreak in an old channel of the Bluestone River.

In photo 31 the geologic relations are depicted, based upon the writer's mapping in 1967-68. Note that all the bedrock in the vicinity of the placers consists of the York Slate, and placers cease abruptly when the chloritic schists form the bedrock. There may be covered veins in many places in the Teller area, but none are exposed at the surface.

CONCLUSIONS

I hope that the above brief descriptions of the placer gold deposits of the Seward Peninsula will convince you that without the York Slate, there are no large and rich placer deposits developed on the Seward Peninsula. Without question, all the richer placers are associated with mineralized veins in bedrock, and generally near to outcropping intrusive bodies that may include granite or gabbro, but principally granite. No deposits were concentrated from glacial till, and glaciation destroyed or covered many significant placers that otherwise would have been mined.

Exploration potential for hidden placers exists largely in the regions covered by lava flows, in the valley of the Kiwalik River east of Candle as far as the York Slate extends, in the Kougarok Gravels, and probably around the granitic intrusive east of Riney Berg's camp on Independence Creek. Buried creek placers may exist offshore at Nome, between Nome and Teller, but at considerable depth, and on hillslopes in many areas where coarse gold may form lag depositssimilar to those mined by Bond Davis at Teller. In general, however, we may be sure that the heyday of mining of placer gold on the Seward Peninsula

is long gone, and likely will never return. After all, there are few people today with the fortitude and training of the oldtimers such as the Tweets, and new methods will have to find placers even harder to mine than those mined in the past. I have little hope for any large-scale placer mining in the future on the Seward Peninsula.

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ASARCO

Southwestern Exploration Division

June 8, 1983

To: JHCourtright
FRKoutz
HGKreis
WLKurtz
GJStathis
JRStringham

From: J. D. Sell

Heap Leaching Gold

Attached is a copy of the cover letter by R. L. Brown and the article by F. T. Graybeal on the heap leaching of gold.

Please note the final paragraph on page 2 of the cover letter.

Although field work and a vast number of drill holes with ore is the prerequisite for an operating mine, the ASARCO Library also has a number of publications which have pertinent thoughts on gold. These include:

a) Principal Gold-Producing Districts of the United States, by A. H. Koschmann and M. N. Bergendahl: USGS Prof. Paper 610 (1968).

b) The Geochemistry of Gold and Its Deposits, by R. W. Boyles: Geol. Survey of Canada Bulletin 280 (1979).

c) Precious Metals in the Northern Cordilleran, edited by A. A. Levinson: The Assoc. of Exploration Geochemists (1982).

d) The Geology of Gold in Ontario, edited by A. C. Colvine: Ontario Geological Survey, Miscellaneous Paper 110 (1983).

James D. Sell

James D. Sell

JDS/cg

Attachments

J. D. Sell

New York, May 31, 1983

MEMORANDUM FOR: Exploration Managers

Heap Leaching - Gold

Five years ago most of us considered that heap leaching operations in gold camps were mainly conducted by underfinanced, fly-by-night groups attempting to salvage remnants of surface oxidized surface material and old dumps around some of the various abandoned gold mining localities in the west. The view was that there was no real money to be made from this activity, but that these junior companies might make something promoting their operations on various over-the-counter and regional stock exchanges. Since that time, recognition that such organizations as Selection Trust, Goldfields, Canadian Superior, and Placer Development among the experienced mining companies; and Louisiana Land, Amoco Minerals, and Occidental as examples of very-well financed groups had made investments in heap leach operations, and were deriving from them handsome profits and cash flows, obviously necessitate a re-evaluation of that view.

Mr. Graybeal has put together a compilation of data concerning the heap leach operations and it, together with his observations concerning the state of the "heap leaching" art, is in my opinion a valuable piece of background information which I am herewith distributing to you.

The advent of heap leaching as a viable metallurgical recovery technique quite obviously has added a new class of mineral deposit - the class of deposits containing pervasively distributed free gold grading say from .04 gold to those deposits grading say .15 Au (which might be more economic under conventional milling techniques - to our list of legitimate exploration targets. There are of course, limitations - not all .04 gold deposits are heap leachable, one reason being that in some of these deposits the gold is caught up in pyrite or in some other sulfide, and the development of this technique obviously does not constitute a basis for the assumption that all areas in which there is presumptive very low grade mineralization of these grades should be drilled. Nevertheless, new technology has opened up a new group of mineral deposits which can earn attractive rates of return, and mining operations on these deposits are in operation and are now continuing to be constructed in the Western United States and in Australia, and operations of this type are either under construction or being contemplated in the Philippines, and in South Africa.

RECEIVED

JUN - 6 1983

S. W. U. S. EXPL. DIV.

In summary I think you will find the information Mr. Graybeal has compiled very useful to you, and I ask that you circulate it among your staff and make sure they understand both the possibilities and the limitations inherent in heap leaching.

R. L. Brown

Attachment

cc w/attachment

RLHennebach
RdeJOsborne
TCOsborne

New York, May 25, 1983

MEMORANDUM FOR: Mr. R. L. Brown

Review of Gold and Silver
Heap Leach Operations

Heap leaching is a simple process in which a cyanide solution is percolated through heaps of gold-bearing rock dissolving gold which is usually recovered on activated carbon. Geological factors of importance include distribution and grain size of gold in the ore, supergene oxidation which liberates sulfide-bound gold and reduces consumption of cyanide, and physical nature of the ore which governs permeability in the leach heaps.

The first significant operation to recover gold by heap leaching began at Cortez in 1971 as an addition to a conventional milling operation. The first significant independent heap leach operation began at Windfall in 1975 and there are presently 12 independent heap leach gold mines. I estimate that 320,000 oz. Au, or roughly 20% of the total USA gold production, was recovered by heap leaching in 1982 equivalent to about half of the primary open pit gold produced in the USA.

The attached fold-out chart summarizes information on 15 significant precious metal heap leach operations. Data are incomplete, change almost daily, and come from many sources of variable quality. Twelve of the 15 deposits are gold mines of which 6 are in Nevada and only 1 is outside the USA. Seven of the 15 are run by junior mining companies. Five additional operations for which only incomplete data are available are described briefly below. The following discussion will focus on the gold operations.

Reserve data in Table 1 suggest that at a gold price of $400/oz. a minimum exploration target size would be about 1 million tons averaging 0.08 oz. Au. Minimum ore grade for a large and low cost operation with recoverable gold and silver might be about 0.040 oz. Au equivalent. The mineralization in heap leach ores is usually disseminated with assay boundaries so reserves may change substantially with the price of gold. Exceptional care is required to measure the gold content of heap leach material which is near the lower limit of acceptable fire assay precision. Increased confidence in reserve estimates is obtained by running duplicate assays, assaying large samples (10-20 lbs.), drilling large diameter holes, and drilling a large number of holes.

TABLE 1, Summary of Heap Leach Reserves

Mine	Tons (millions)	oz. Au	oz. Ag
Haveluck (1)	2	0.054	-
Borealis	2	0.08	0.5
West End	3	0.085	-
Windfall	5	0.035	-
Alligator (2)	5	0.12	-
Annie Creek	7	0.043	0.16
Ortiz	7-16	0.053	-
Northumberland	10-16	0.052	-
Round Mountain	12-230	0.043	-
Zortman-Landusky	40-190	0.030	0.7

(1) Additional carbon-in-pulp ore was included in overall reserve which justified development.

(2) Highest grade heap leach operation; considering that carbonaceous ore must be stockpiled and additional reserves are present, this might have been better suited to conventional milling.

Major capital cost variables are the need for a crushing plant and whether mining is done by contractor. Capital costs in 1983 dollars for heap leach operations including Candelaria (Ag) ranged from $2300-6600/daily ton of ore for 6 of the operations where good data are available (see Table 2). Alligator Ridge was $15,000/daily ton and may be in error. Capital cost for the Zortman-Landusky complex are not known accurately, but if estimates of $1 million (1983 dollars) are close the cost would be $200/daily ton. This remarkably low cost resulted from contract mining of ore which did not require crushing. Table 2 shows that heap leach capital costs are substantially less than those for more conventional milling operations due to a relatively simple flowsheet. Capital costs per ounce of recoverable gold for heap leaching are equivalent to or less than costs for conventional milling as shown on Table 3.

TABLE 2. Capital Cost/Daily Ton of Ore Mined (1983 dollars)

Heap Leach		Conventional Milling	
Haveluck	- $2,300	Golden Sunlight	-$10,000
Santa Clara[1]	- 2,700	Pinson	- 22,000
Annie Creek	- 2,900	Mercur	- 27,000
Round Mountain	- 3,500	Jerritt Canyon	- 44,000
Candelaria	- 4,300	McLaughlin	- 66,000
Ortiz	- 6,300		
Northumberland	- 6,600	Average	-$34,000
Alligator Ridge	- 15,000		
Average	$ 5,400		

(1) Not on fold out table; see text

TABLE 3. Capital Cost/oz. of Recoverable Gold in Reserve (1983 dollars)

Heap Leach		Conventional Milling	
Haveluck[1]	- $16/oz.	Golden Sunlight	- $43/oz.
Annie Creek	- 18	Jerritt Canyon	- 49
Santa Clara	- 23	Pinson	- 63
Northumberland	- 58	McLaughlin	- 69
Round Mountain	- 59	Mercur	- 99
Ortiz	- 67		
Alligator Ridge	- 83		

(1) Heap leach and CIP reserve and plant.

Capital cost per ounce of recoverable gold in the original reserve for the Zortman-Landusky complex would be less than $1.00.

Construction intervals vary from 8-15 months. Six of the operations began production in the interval 1979-1981. Mining rates (ore) for 8 operations cluster between 1000-3000 tpd. The two largest operations are Round Mountain at 8000 tpd and Zortman-Landusky at 25,000 tpd. Round Mountain produces more gold than Zortman due to a longer operating season and higher grades. Although cold weather is a problem for plumbing systems and reduces the rate of dissolution of gold Zortman-Landusky, which is located in northern Montana, resumed mining on February 21, 1983, indicating cold weather is not a complete deterrent. The Victor Mine leaches heaps in enclosed vats at Cripple Creek and Round Mountain recently installed a heater to keep circulating fluids warmer. Because of the low ore grades stripping ratios are low and pit slopes are kept steep where possible. Cutoff grades are about 0.02 oz. Au.

Gold recoveries are in the range 60-70% for a 30-day interval. The most important and unpredictable variable in heap leaching is the movement of solution through the heaps. In order to obtain any recovery a cyanide solution must be brought in contact with the gold. Where the gold occurs on natural fractures crushing may not be required. If crushing is required or if the ore is unusually soft, excessive fines may reduce permeability and cut the flow of solution below that required to maintain production. Fines have been a problem at Alligator Ridge, Haveluck, and Candelaria. Agglomeration techniques have been developed by the U.S. Bureau of Mines to handle fines, but they increase capital and operating costs. Proper bedding of material on the leach heaps has been extensively researched to avoid compacting the ore with haulage equipment and to reduce sorting of fine and coarse material as individual loads of ore are added to the heaps. Careful selection of samples for metallurgical work is necessary as a single orebody may have several distinct ore types. Recognition of different ore types is difficult because most low grade deposits are explored using air hammer drilling methods which pulverize the sample, but careful logging can provide useful data. Most companies construct one or several 1000-5000 ton test heaps before final operating procedures are selected. The importance of accurate measurements of ore grades and recoveries cannot be overstated; for a 3000 tpd operation with $270/oz Au operating costs at a gold price of $430/oz., a change in recovered gold of six thousandths of an ounce per ton of ore (0.006 oz. Au) will change the annual net profit before taxes by $1 million.

Operating costs per ounce of gold produced seem to cluster at $250-270/oz. The Round Mountain estimate of $190/oz. Au is the 1983 GNP adjusted equivalent of estimates made in 1977 by Copper Range and Mountain States using a head grade of 0.045 oz. Au and is probably too low. Costs at Northumberland are unusually high at $402/oz., even when $68/oz. haulage costs are deducted.

Heap leach operations for which less data are available include:

1) Carlin Mining Co. Heap leaching is now in progress at Carlin. Major reserves at Maggie Creek and Gold Quarry of at least 150 million tons averaging 0.036 oz. Au will be agglomerated and heap leached. The Bootstrap dumps were previously leached as they lay after mining without prior sealing of the base. By 1984 it is estimated that 30% of the gold production at the Carlin complex (Carlin, Maggie, Gold Quarry, Rain, Blue Star) will be by heap leaching.

2) Picacho District, Calif. Chemgold, Inc. (Glamis Gold, Inc.) is leaching 300,000 tons and producing 600 oz. Au/mo. Agglomeration techniques are being studied. Recent drilling expanded reserve to 10 million tons averaging 0.05 oz. Au. Mineralization rumored to occur in mid-Tertiary detachment fault zones.

3. Santa Clara, Costa Rica. Located 40 miles from San Jose and owned 60% by United Hearne and 40% by Canadian Barranca. Reserves are 4.1 million tons averaging 0.063 oz. Au and recent drilling indicates substantial increases are probable. Came on stream September 1981 at 1500 tpd for $4 million. Operating costs are estimated at $6.00/ton.

4) Darwin Mine, Calif. Anaconda is drum agglomerating and heap leaching flotation tails averaging 1.6 oz. Ag and 0.01 oz. Au. Leach pads are compacted tailings.

5) Grubstake Mine, Montana. Resources America, Inc. is drum agglomerating and heap leaching 20,000 tons of dump material averaging 0.07-0.15 oz. Au. The Merrill-Crowe recovery plant is highly automated

Serious feasibility studies or pilot heap leach test work is ongoing at:

1) Pinson mine, Nev. A test pad is being built to study heap leach characteristics of 5 million tons of low grade material involving two ore types.

2) Bell Mtn. mine, Fairview Dist., Nev. American Pyramid is developing a combined milling and heap leach operation.

3) Dayton mill tailings, Comstock Lode, Nev. Tailings average 2.5 oz. Ag and are 65% minus 200 mesh. A 40-ton experimental heap was agglomerated and yielded 78% recovery in Ag in 5 days.

4) Packard mine, Nev. This property is located 3 miles southwest of the Rochester deposit. Reserve is estimated at 2 million tons averaging 1 oz. Ag. An initial unagglomerated 30,000 ton test was unsuccessful, but subsequent 300 ton pilot heaps with ore crushed to 80% minus 9/16 in. and agglomerated gave 65% Ag recovery in 6 days. Follow-up work on a 1000 ton heap gave similar results and work is continuing.

5) Goldfield, Nev. Initial 8000 ton test heap failed due to low permeability. Column leach tests on agglomerated rock crushed to minus 1 in. gave 80% recovery of gold in 14 days. Existing heaps will be recrushed and agglomerated as a pilot test.

6) El Plomo, Colo. E&B Explorations (German money) is evaluating 1 million tons of 0.057 oz. Au. Large test heap program was completed in 1982.

7) Gold Stripe, Plumas County, Calif. Test production by NCA Minerals will begin June 1983 from 10,000 ton test pad. Deposit has reserves 1.2 million tons averaging 0.063 oz. Au.

8) Zaca Mine, Alpine County, Calif. California Silver has operating permits for a 3000 tpd/heap leach mine. Reserves are 4.1 million tons of 0.036 oz. Au and 0.61 oz. Ag at 0.01 oz. Au cutoff. Column leach test of combined oxide and sulfide ore crushed to minus 3/8" recovered 76% Au and 38% Ag in 19 days.

9) German Gulch, Montana. Reserve is 11 million tons of 0.048 oz. Au. Heap leach recoveries on two 4500 ton heaps are reportedly 85%.

10) La Ronge Dist., Saskatchewan. Waddy Lake Resources has conducted tests said to be successful on low grade rock containing coarse gold.

11) Hedley, British Columbia. Banbury Gold Mines has applied for permit to begin heap leach production on gold-silver property.

12) Telfer mine, W. Australia. Newmont has completed a 1000 ton test heap leach pad and will crush ore to minus 1" and agglomerate. They say there is considerable heap leach ore, my guess is at least 5 million tons.

13) Masbate mine, Philippines Is. Atlas is building an asphalt heap leach test pad to evaluate characteristics of an eventual 4 million ton stockpile grading 0.030-0.045 oz. Au.

14) Gabiet Mine, Sudan. Robertson Research (U.K.) in complex ownership package is conducting pilot heap leach tests on mine tailings. Ultimate production might be 3000/oz. Au/yr.

15) Witwatersrand S. Africa. Feasibilities are reportedly underway to evaluate heap leaching on old dumps and tailings.

Heap leach operations are attractive because capital costs are low, technology is simple, and development intervals are rapid. These features permit the development of deposits otherwise too small or too low grade to justify a conventional milling operation. A substantial body of literature is available on the technology, but less information is available on existing operations, particularly operating costs. Operating costs per ounce of gold recovered may exceed costs at open pit milling operations, but are less than costs at the Homestake mine (1982 costs about $400/oz.). Although heap leach operating costs/oz. Au may be somewhat more than conventional milling operations, the capital investment may be somewhat less, and heap leaching therefore represents a valid and competitive mining investment. A 3000 tpd heap leach plant mining 0.07 oz. Au with 65% recovery at a gold price of $430/oz. and operating cost of $270/oz. would net $8 million before tax, clearly of sufficient size to be of interest to Asarco.

The many low grade gold prospects in various stages of exploration now known in the western US substantially outnumber known prospects of higher grade mineralization which might become conventional milling operations. This ratio may also apply to future discoveries and a significant effort is currently underway in low grade environments where the odds for discovery would appear to be more attractive.

F. T. Graybeal

F. T. Graybeal

CHARACTERISTICS OF GOLD AND SILVER HEAP LEACH OPERATIONS

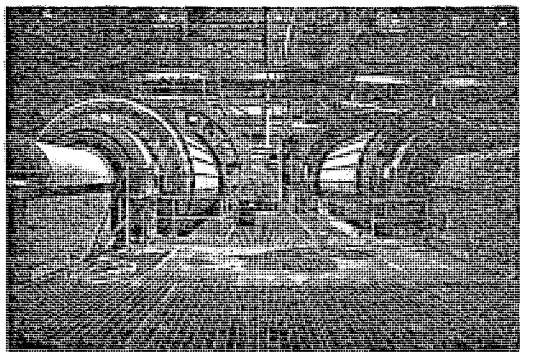
MINE	OWNER	RESERVE (M=million)	CAPITAL COST STARTUP	PRODUCTION RATE	WASTE/ORE, CUTOFF	RECOVERY	OPERATING COST (1983)	GEOLOGY	COMMENTS
GOLD MINING OPERATIONS									
ALLIGATOR RIDGE Nevada	50% Amselco 50% Occidental	14M tons - 0.12 oz Au	$30M; permitting and construction took 15 months; startup Oct. 1980.	2200 tpd 60,000 oz Au/yr	W/O=3/1; 40-45° pit slope	70-75%	NA	Disseminated gold in zone of strong jasperoid alteration of silty limestone; deeper ore is carbonacous.	Crushing and agglomeration required; carbonaceous ore is stockpiled; more ore in 2 other zones; pad made from compacted clay mined locally; 140 employees.
ANNIE CREEK S. Dakota	Wharf Resources	6.5M tons - 0.043 oz Au 0.16 oz Ag	$4M; startup Summer 1983.	750,000 tpy	Cutoff = 0.02 oz; 45° pit slope	74%	$225-275/oz	Similar to Gilt Edge (?); disseminated gold in pyritic (oxidized) latite and trachyte porphyry in breccia and shear zones.	Wharf holds 50% interest in adjacent Bald Mtn. property (Homestake 50%) with reserves of 9.9 million tons averaging 0.041 oz Au, 0.17 oz Ag.
BOREALIS Nevada	Houston Int. Minerals (Tenneco)	2.1M tons - 0.08 oz Au 0.50 oz Ag	Startup Nov. 1981.	2300 tpd 30,000 oz Au/yr 30,000 oz Ag/yr	NA	NA	NA	Disseminated gold and silver halides in oxidized, silicified and argillized tuff and volcanic breccia.	40 employees.
CORTEZ Nevada	Placer Development	2.2M tons - 0.041 oz Au at Gold Acres 2.8M tons at Cortez.	Startup 1971; part of a milling operation.	NA (not available)	Cutoff = 0.015 oz	50% from clayey Gold Acres ore; 65% from blocky Cortez ore.	NA	Largely Carlin-type with fine-grained disseminated gold in silty limestone, variably carbonaceous.	No crushing, 170,000 ton heaps.
HAVELUCK W. Australia	Whim Creek	1.4M tons - 0.054 oz Au heap leach.	$2M; startup June 1980; 10 mo. for construction.	1000 tpd 14,500 oz Au/yr	W/O=2/1; cutoff = 0.018 oz	70%; 30% in 30 days	$250/oz HL $270/oz CIP	Disseminated gold in strongly weathered and talc-altered mafic volcanic rock.	Began as 100% heap leach; excessive fines required conversion to 50% heap leach, 50% CIP; mining by contractor; total reserve is 4.5M tons at 0.066 oz Au.
NORTHUMBERLAND Nevada	Amoco Minerals	10M tons - 0.052 oz Au	$17M; startup April 1981; 8 mo. for construction.	3000 tpd 30,000 oz Au/yr 30,000 oz Ag/yr	W/O=1.8/1 cutoff = 0.020 oz	60% from jasperoid, 70% from shaley limestone	$402/oz excluding deprec./depl.; haulage is $68/oz.	Carlin-type with disseminated gold in jasperoidized limestone and in decalcified silty limestone.	Crushing required; 10 mile haul pit to heaps; plastic liner under heaps; additional 6M tons averaging 0.034 oz Au at W/O=6.9/1; 110 employees.
ORTIZ New Mexico	Goldfields	6.8M tons - 0.053 oz Au	$14.9M; startup Feb. 1980; 12 mo. for construction.	3000 tpd 60,000 oz Au/yr	W/O=1.2/1; cutoff = 0.02 oz.	78% in 56 days; 60% in 23 days.	$261/oz excluding deprec./depl.	Disseminated gold in fractures and vugs in oxidized breccia pipe cutting quartzite and associated with latite dikes.	Additional 10M tons are below water table; mining done by contractor; crushing required; asphalt-lined heaps are cycled; 48 employees.
ROUND MOUNTAIN Nevada	Smokey Valley Mining Co. (50% Louisiana Land)	Started at 11.6M tons - 0.061 oz Au; now 195M tons - 0.043 oz Au, 0.08 oz Ag	$18M; startup Jan. 1977; 15 mo. for construction.	8000 tpd 75,000 oz Au/yr 30,000 oz Ag/yr	W/O=1/1; cutoff = 0.02 oz; 56° pit slope	67% in 26 days.	$190/oz (est. by FTG based on current ore grade of 0.045 oz Au).	Disseminated gold in iron-stained joints cutting dense weakly altared ash flow tuffs.	First large heap leach gold operation; crushing required; asphalt-lined heaps; heaps are cycled every 28 days; 240 employees.
VICTOR Colorado	Silver State Mining Corp.	3.5M tons - 0.06 oz. Au; potential for 8M tons.	$3M; startup March 1982; construction took 9 mo.	1000 tpd 20,000 oz Au/yr	NA	Close to 90% in 4 days.	$160/oz Au; mining cost is $.20/ton	400 ft. diameter breccia pipe.	Cripple Creek District; 75% of ore is mined by ripping; ore crushed to minus 1/2 in. and agglomerated with 90% of Au in solution by time cement cures; leaching in concrete "vats" covered for year-round leaching; 4-day cycle.
WEST END Idaho	Canadian Superior	3M tons - 0.085 oz Au.	Estimate $10M; startup July 1982; 11 mo. for construction.	3000 tpd	W/O=2/1	75%	NA	Oxidized gold-bearing arsenopyrite in shear zones in argillized schist, quartzite, and cargonate cut by trachyte dikes.	5-month operating season; 2 years to do EIS: 60 employees.
WINDFALL Nevada	Idaho Mining Corp.	5M tons - 0.035 oz Au 0.01 oz Ag	Startup July 1975.	1500 tpd	W/O - low; cutoff = 0.01 oz Au	NA	NA. Profitable in 1975 when gold was $160/oz (equal to $264/oz. Au in 1983).	Disseminated gold in jasperoidized and sanded dolomite.	No crushing; mining reduces sanded dolomite to 35% minus 200 mesh, ponds instead of sprinklers.
ZORTMAN-LANDUSKY Montana	Wharf Resources Ltd. Pegasus Explorations, Argo Gold Mining Corp., and others.	Zortman is 20M tons - 0.032 oz Au, 0.4 oz Ag; Landusky is 19M tons - 0.027 oz Au; 0.9 oz Ag.	Startup July 1979; construction took less than 12 mo.	12,000 tpd each; 67,000 oz Au and 146,000 oz Ag is total for both.	W/O=0.7/1	70% Au 35% Ag	$250/oz Au exclusive of Ag; equivalent to $4.50/ton ore including adminis., excluding deprec., depl., tax.	Disseminated gold and silver in weakly pyritic (now oxidized) shear zones cutting syenite. Zortman is old gold district, Landusky is old silver district; 8 miles apart.	Mining by contractor; heaps sealed with 2 ft. compacted bentonite; no crushing; 500K-1M ton heaps are permanent; early Zortman grades were 0.06 oz Au; additional Zortman reserves 30-40M tons, possibly 190M tons at Landusky; 65 employees; 1983 production began Feb. 21, 1983.
SILVER MINING OPERATIONS									
CANDELARIA Nevada	60% Occidental; will sell to NERCO.	13M tons - 3.15 oz Ag 0.002 oz Au.	$30M; startup September 1980.	8000 tpd 1,700,000 oz Ag/yr. 9,200 oz Au/yr.	NA	40% in 60 days	NA	Argentiferous Fe and Mn oxides in formerly pyritic chert, sandstone, and shale cut by quartz veins and chalcedonic zones.	Only full-scale heap leach silver mine in world; crushing and agglomerating required; closed Spring 1982 due to low Ag prices; 140 employees.
TOMBSTONE Arizona	Tombstone Exploration Company	Dumps contain 1M tons - 2.26 oz Ag and 0.037 oz Au. Lode averages 1 oz Ag.	Startup of dump material January 1981.	1500 tpd from dumps; 2000 tpd tpd from Contention zone.	NA	Lab tests indicate 75% for Au, 60% Ag both in 16 days.	Lode is profitable at Ag price of $10/oz.	Silver halides, sulfates in quartz-veined and silicecus limestone conglomerates, variable manganese.	Lode ore is mined by ripping; dump material crushed to minus 1/2" and agglomerated; leach pad is compacted mill tailings; agglomeration increased recoveries by 5x.
TUSCARORA Nevada	Tuscarora Assoc.	500,000 in dumps 2.5 oz Ag, 0.02 oz Au.	$700,000; startup October 1979; cost includes moving first 200,000 tons.	NA	NA	NA	$6.90/ton of ore.	Silver halides and gold in dumps in an old epithermal quartz-adularia vein camp.	Plastic-lined leach pads; plant treats 1400 tons solution/day; mining by contractor; 6 employees.

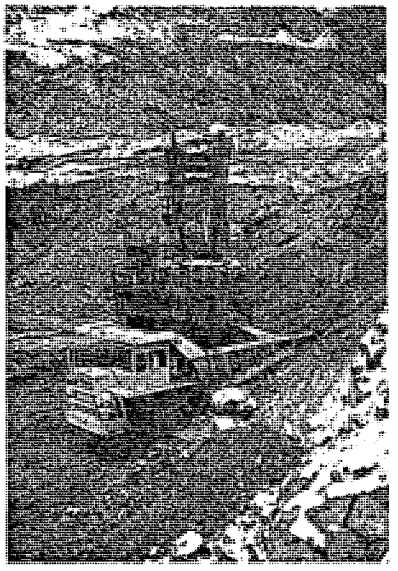
FTG - 5/26/83

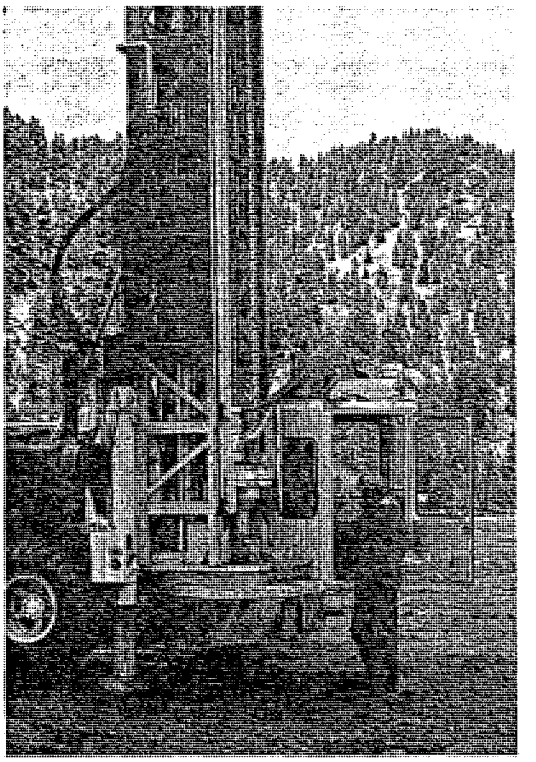
Placer Amex Inc. Operating Golden Sunlight Mine at

Project completed ahead of schedule and below in Montana to extract average of 72,000 oz. of

by DAVID N. SKILLINGS Jr.







(Top Left) Close up of loading blast hole at Mineral Hill with blasting agent consisting of ANFO initiated with boosters and primacord. (Top Right) Interior of Golden Sunlight plant showing grinding section comprising 13½ by 20½-ft. rod mill in open circuit and 15 by 20-ft. ball mill in closed circuit. (Bottom Left) View of electric shovel equipped with 8-cubic yard bucket loading ore into 60-ton mechanical drive truck for haulage one mile to the crusher. (Bottom Right) Scene of rubber-tire mounted D40K-II unit drilling 6¾-in. diameter blast hole on 15 by 15-ft. pattern to 30-ft. depth in Mineral Hill pit.

EARLY IN 1983, PLACER Amex Inc., a wholly-owned U.S. subsidiary of Placer Development Ltd., Vancouver, B.C., placed in operation its Golden Sunlight gold property near Whitehall in Jefferson county, Mont., 25 miles east of Butte. The Golden Sunlight project was marked with its completion five months ahead of schedule at a capital investment totaling U.S.$50 million, 40% below the original estimate of U.S.$88 million as a result of lower interest rates, reduced inflation, extremely competitive construction bidding and on-time delivery of mining and proccessing equipment.

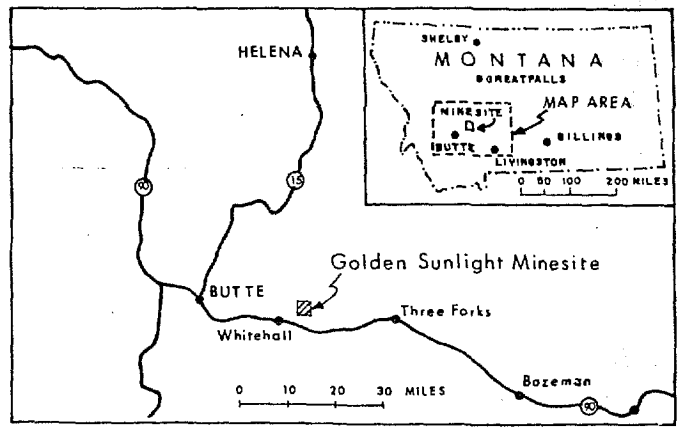
Following completion of the Golden Sunlight project and its successful start up, the first dore was poured in the plant on Feb. 7, with production of the initial bar of bullion on Feb. 17. The project consisted of development of the Mineral Hill orebody by the open pit method and construction of a processing complex designed to extract bullion at an average annual rate of about 72,000 oz.

The Golden Sunlight mine, having a life estimated at 13 years based on the initial ore reserves, is situated on the northeastern side of the small community of Whitehall. The mine and plant are operated by Golden Sunlight Mines Inc., a California corporation organized on March 3, 1982, as a wholly-owned subsidiary of Placer Amex.

Presently an ownership of 39.6% is held by Placer Amex in Cortez Gold Mines, a joint venture that will produce an estimated 40,000 oz. of gold annually commencing with development of the Horse Canyon mine in Nevada during 1983 at the rate of 2200 tons per day. Extrac-

Full Capacity

cost estimate gold annually



tion of gold from heap leaching of old mine dumps also is conducted by Cortez Gold Mines. In northern Nevada, an interest of 51% is held by Placer Amex in the McDermitt Mine producing about 25,000 flasks of mercury per year.

Main Primary Gold Mine in Montana

Although Anaconda Minerals Co. extracts gold as a by-product of its copper mining operations at Butte, Golden Sunlight becomes the principal primary source of gold in Montana. In Phillips county of northern Montana, Zortman-Landusky Mining Co., owned jointly by Pegasus Gold Ltd. and Gold Reserve Mining Co., is mining ore by the open pit method for recovery of gold through heap leaching. In 1982, about 10,000 tons of ore were mined per day for a total of 3,873,000 tons from which 67,000 oz. of gold and 145,000 oz. of silver were extracted.

The Golden Sunlight mine consists of a group of patented and unpatented mining claims as well as fee lands totaling 4200 acres. The mining claims have been under the same ownership for many years, with ore mined from several different deposits. The initial orebody to be mined at Mineral Hill is situated in a breccia pipe ore zone that has been somewhat faulted but is fairly well defined.

In 1958, a 10-year lease and option to purchase the Golden Sunlight property were acquired from A.O. Smith Corp., Milwaukee, Wis., by American Exploration & Mining Co., which was renamed Placer Amex in 1973. During 1968, the option was exercised by the company, and it took fee simple title to the property.

Over a 22-year period, an exploration program amounting to more than 50,000 ft. of rotary and diamond drilling was completed by Placer, which also mined 100,000 tons of ore by both open pit and underground methods and produced 2500 oz. of gold and 300 oz. of silver through test heap leaching.

First Mining Claim Staked in 1890

On April 1, 1890, the first claim of the Golden Sunlight group was staked by Anthony A. Hedley, who previously had been working as a miner in the St. Paul's gulch area to the northwest across the Bull Mountain range. Mr. Hedley later staked the Telluride, Sunlight and Last Chance claims.

In June 1890, American Development & Mining Co. entered the district and acquired Mr. Hedley's claims and also staked the first claims on Mineral Hill. This company continued to extend its holdings until July 1894. A stamp mill with amalgamation plates was erected by American Development in 1895, but this plant was unsuccessful, and milling operations were abandoned in 1905.

From 1890 to 1910, considerable high grade ore was mined from the Golden Sunlight property for shipment to various smelters. Measurement of known stoped out areas using grades mined during that period indicates that more than $1.5 milion probably were mined from the old Sunlight workings on the south end of the property. These workings are known as Sunlight, Buffalo, Moonlight, Lamplight and Excelsior veins.

In 1910, the Golden Sunlight property was acquired by H.C. Bacorn and his associates, who commenced a systematic program of development. Three long adits were driven to reach the Sunlight vein, and two long adits and several short workings were driven in the Mineral Hill deposit. Only about 5000 tons of ore having a value of $200,000 were mined until work ceased in 1917.

From 1917 to 1935, the Golden Sunlight property was operated by numerous lessees. Dan Zink and Lot Borden of Whitehall mined the Mineral Hill deposit and are reported to have shipped ore with a gross value of $90,000, and later C.L. Tidball of Whitehall with several partners scavenged high grade ore from Mineral Hill and the Ohio claim with shipments having a gross value of about $50,000.

During the 1920s, John B. Wellcome, an attorney of Butte, and Alex McKay, a grocer of Whitehall, secured the entire Golden Sunlight property for legal fees and grocery bills, obtaining interests of 5/8ths and 3/8ths, respectively. Late in the 1920s and early in the 1930s, Messrs. Hutchins and McGilvary shipped old material from the Sunlight stopes and grossed about $20 per ton for a total of $30,000. At about the same time, H.W. Carver shipped old tailings from former milling operations up to 1905, with shipments grossing about $20,000. Later Mr. Carver, working with Messrs. Hutchins and McGilvary, mined out a stope on the Bacorn vein near the portal of the Bacorn adit from which 3000 tons of ore grossing $12 per ton were removed for a total of $36,000.

(Continued on page 6)

Placer Amex Operating Golden Sunlight Mine at Full Capacity

(Continued from page 5)

Exploration by Anaconda in 1930s

During 1935, Anaconda Copper Mining Co. acquired an option to purchase the Golden Sunlight property. Existing openings were sampled, and five diamond drill holes totaling about 1000 ft. were completed. During Jan. 1936, activities were discontinued by Anaconda, which failed to exercise its option to purchase.

In Sept. 1936, A.O. Smith Corp. acquired a lease and option to purchase the Golden Sunlight property from the heirs of Messrs. Wellcome and McKay. This company conducted an extensive program of development and diamond drilling as well as metallurgical testing until 1938, with some direct-shipping ore produced in that year. In June 1939, the property was subleased to M.F. Riebhoff. On Feb. 26, 1946, the option was exercised by A.O. Smith, which took fee title to the property.

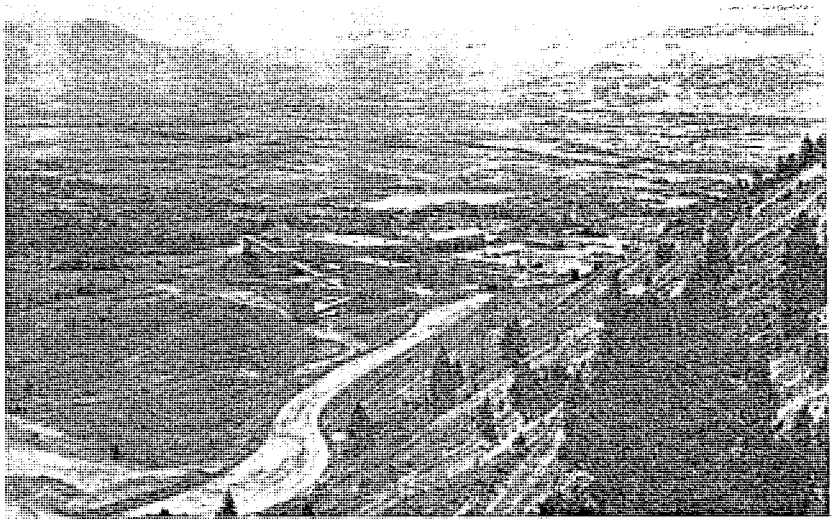
During that year, Martin Bros., Butte, obtained an option to acquire the Golden Sunlight property from A.O. Smith, and in association with Winston Bros. Co., Minneapolis, Minn., a pilot mill was erected and operated for four months to check metallurgy based on laboratory testing. Six diamond drill holes totaling 1831 ft. were completed in the Mineral Hill deposit, but the lease option was terminated.

In 1955 and 1956, an attempt was made by White Development Co. to operate the Golden Sunlight mine as a moderate-sized open pit. A glory hole and a connecting conveyor belt haulage adit were developed in Mineral Hill to mine a low grade orebody situated around what was thought at that time to be a breccia pipe. The ore was processed by screening, with the screened material treated by cyanidation. This operation experienced difficulties and produced only small quantities of gold.

From 1901 to 1980, production of ore from mines on the Golden Sunlight property is estimated at 253,618 tons from which were extracted 59,271 oz. of gold, 80,061 oz. of silver, and 54,598 lb. of copper.

Engineering Contract to Kilborn

In Aug. 1981, the contract for detailed engineering for the Golden Sunlight project was awarded by Placer to Kilborn Engineering (B.C.) Ltd., Vancouver. This firm provided engineering design, drafting services and field coordination



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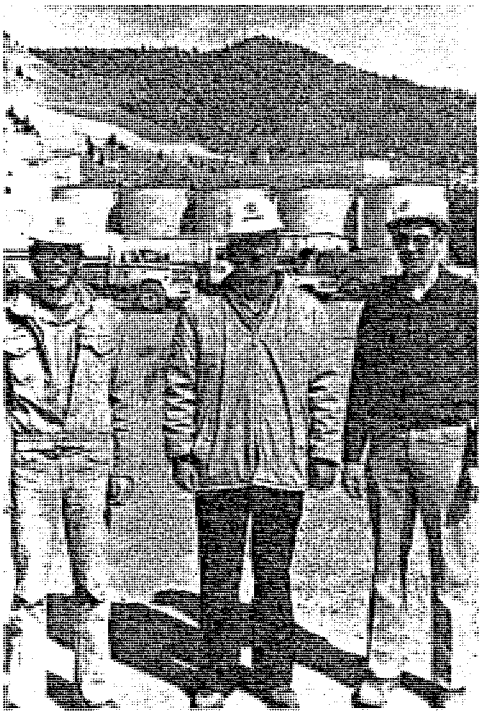

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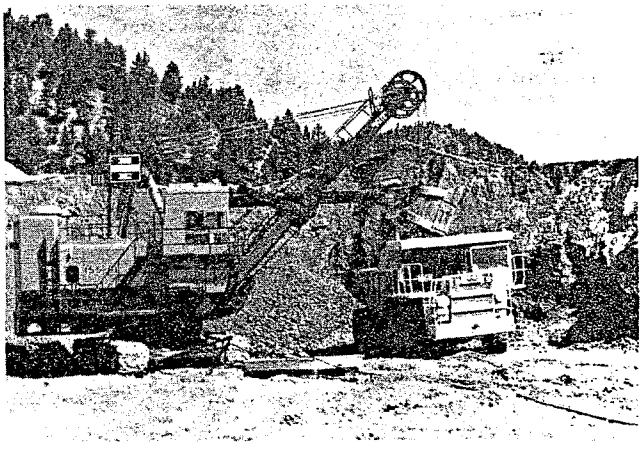
(1) Looking southeast toward the plant at the Golden Sunlight mine in which initially about 2,000,000 tons of ore will be processed annually and bullion will be extracted at the average rate of 72,000 oz. per year. (2) View on 5800-ft. level of Mineral Hill pit showing waste being loaded by an electric shovel equipped with an 8-cubic yard bucket into a 60-ton for haulage one-half mile to dumps situated on the north side of the pit. (3) Jim Smolik, right, mine manager at the Golden Sunlight mine, pictured recently at the plant site with Donald E. Jenkins, administrative superintendent, and Michael W. Roper, chief mine engineer.



2

of surface facilities from primary crushing to bullion production.

During Oct. 1981, work on building a 2-mile access road from Interstate 90 was commenced for completion in Dec. 1981. Site preparation in the plant area also began in Oct. 1981. During the winter of 1981-82, engineering and equipment procurement were conducted in preparation for starting the Golden Sunlight project with full scale construction in the spring of 1982. On April 26, 1982, the first concrete was poured for the processing complex.

By Jan. 1983, the Golden Sunlight project had been completed with Placer acting as the general contractor and Commonwealth Construction as construction advisor. Between 25 and 30 subcontractors were engaged in the project, with a total of 350 personnel involved in the field at the peak of construction activity in Nov. 1982.

The Golden Sunlight operation is based on ore reserves totaling 25,800,000 short tons averaging 0.05 oz. of gold per ton that are suitable for mining by the open pit method from the Mineral Hill deposit. Geological studies indicate additional reserves at depth that may be mineable depending on mining costs and gold price by underground methods or open pit method with a high stripping ratio. Later drilling will be conducted to delineate deeper ore reserves. Presently surface drilling is under way to determine sites for waste dumps.

The Mineral Hill orebody will be mined over 13 years in a pit developed on 25-ft. high benches, with an overall stripping ratio of 2 tons of waste and low grade material to 1 ton of one. The present mineable reserves will be mined in three stages, with the stripping ratio increasing with depth.

Although the stages will overlap, the waste-to-ore ratio is 1-to-1 in the first stage, with 0.6-to-1 in the start-up year of 1983. The first stage will continue for the initial two or three years, and in the second stage the milling rate may be increased to about 6000 tons per day with softer ore. Following the second stage, covering seven or eight years, stage three will comprise the final five or six years of Mineral Hill production. Additional mining equipment will be required for stages two and three including one more drill, another two shovels or mechanical excavators, and an additional eight trucks.

Pre-Mine Stripping Begins Dec. 1982

In Dec. 1982, training of crews began in preparation for the pre-mine stripping of the Mineral Hill deposit with removal of 150,000 to 200,000 tons of waste from the top benches. The highest waste bench was at 5950-ft. elevation. Also in December, shakedown of the crushing circuit commenced with processing of aggregate for road-bed material.

During Jan. 1983, ore mining was started from the top bench at 5850 ft. for stockpiling at the mill and subsequent feeding to the crusher. In February, mining began on the 5825-ft. level, followed by the 5800 and 5775-ft. benches, all of which are currently providing ore to the mill. During 1983, two additional benches will be mined at 5750 and 5725 ft. Based on the feasibility study, the bottom bench of the Mineral Hill pit will be reached ultimately at 5300 ft., which is the crusher elevation.

On a schedule of two shifts, five days per week, total material mined daily from the Mineral Hill pit amounts to 12,000 to 14,000 tons, including 7000 to 8000 tons of ore for the mill. On an annual basis about 2,000,000 tons of ore will be mined initially. At Golden Sunlight, the ore is different geologically from other gold deposits in the western U.S. The ore is harder with a 20 to 22 Bond work index, which is kilowatts per ton, as compared with 12 to 15 Bond work index for most orebodies.

Presently the cutoff for ore is 0.02 to 0.03 oz. of gold per ton, with the ore cutoff adjusted with changes in the gold price. The material averaging less than the cutoff is segregated in stockpiles for possible future blending with mill feed.

In the Mineral Hill pit, the ore control program is based on a geostatistical ore control estimation system known as Kriging, in which blast hole assays are used. Blocks containing milling ore, low grade material for stockpiles, and waste for dumps are flagged for the production crews.

Drilling with Two Driltech Units

Blast hole drills consist of two D40K-II Driltech units, one of which is rubber-tired and the other track-mounted, and drilling patterns vary according to rock competence and rock type. Generally 6¾-in. diameter blast holes are drilled on 15 by 15-ft. patterns to 30 ft. providing 5 ft. of subdepth. Blasts range from 100 to 200 holes and break from 50,000 to 100,000 tons of ore for about one week of mining. The blasting agent is ANFO initiated with boosters and primacord.

Broken material is loaded with a 155B Bucyrus-Erie electric shovel equipped with an 8-cubic yard bucket backed up by a 988 Caterpillar front-end loader. The haulage fleet consists of five 60-ton Wabco mechanical drive trucks with the ore hauled one mile to the crusher and waste moved one-half mile to dumps situated on the north side of the pit.

The remaining mining equipment includes two D8 and one D6 Caterpillar dozers, one model 14G Caterpillar motor grader, and water trucks and sand trucks.

On Jan. 19, 1983, the mill was started up at the Golden Sunlilght mine with processing of ore and filling of leaching tanks. Currently the plant is operating with an average capacity of about 5000 tons of ore per day. Initially, hard silicified ore from the top of Mineral Hill is being treated, and gradually this will become softer with depth.

From trucks, ore is dumped into a 42 by 65-in. Allis-Chalmers primary gyratory crusher driven by a 400-hp. General Electric motor. The crusher building contains a system to collect dust from which gold is recovered as well as to improve the environment of the plant area. Minus 6-in. ore is fed with a Stephens-Adamson apron feeder to the primary belt for conveying to the primary screen. Ore is screened on an 8 by 20-ft. primary Hewitt-Robins double-deck screen having 2 by 5-in. openings on the top deck and ¾ by 3-in. slotted openings on the bottom deck. The plus ¾-in. ore is conveyed to the coarse ore stockpile having a live capacity of 12,000 tons, whereas minus ¾-in. ore goes to the fine ore stockpile.

(Continued on page 8)

Placer Amex Operating Golden Sunlight Mine at Full Capacity

(Continued from page 7)

Through four draw points, coarse ore is withdrawn from stockpile and fed onto a belt for conveying to the fine crushing facility. Ore goes to a secondary 7-ft. Standard cone crusher in open circuit for reduction to 1¼ in.

The Standard cone crusher product combines with the 7-ft. Short Head cone crusher product and is conveyed to a secondary 8 by 20-ft. Hewitt-Robins single-deck slotted screen having ¾ by 3-in. slotted openings. All minus ¾-in. product is conveyed to the fine ore storage pile having a total capacity of 35,000 tons, 7000 tons of which are live.

Rod/Ball Mill Grinding in Plant

Through eight draw points, fine ore is withdrawn and fed onto a belt for conveying at rates ranging from 220 to 238 tons per hour to the primary grinding unit, which is a 13½ by 20½-ft. Rexnord rod mill operating in open circuit. This unit is driven by a synchronous motor of 2000 hp. and is charged with 4-in diameter by 19½-ft. long rods.

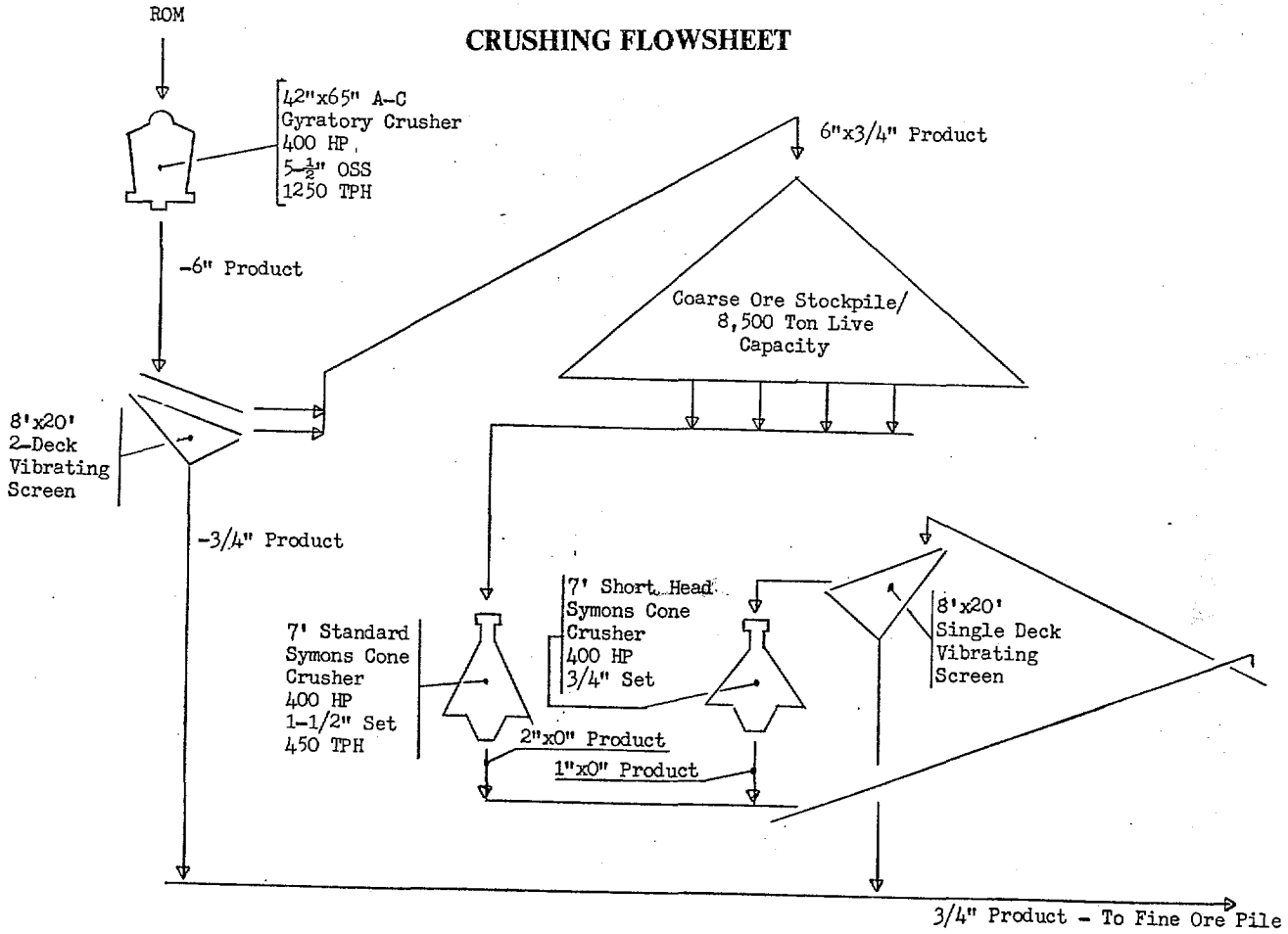
For fine grinding, pulp is fed to a 15 by 20-ft. Rexnord ball mill driven through an air clutch with 3000 hp. and charged with 2½-in. diameter balls. This unit is in closed circuit with four 26-in. cyclones from which underflow is recycled to the ball mill and overflow reports to the #1 Enviroclear thickener of 35-ft. diameter.

From the thickener, underflow is pumped to the leaching circuit consisting of ten 40 by 44-ft. agitated tanks having a capacity of 400,000 gal. each. In the tanks, cyanide is added for leaching along with lime and air, with a retention time of 48 hours. The pulp containing gold is directed to a 5-stage hydrocyclone washing circuit to remove the sand. Underflow progresses through the first, second, third, fourth and fifth stages while the overflow moves countercurrent through the fifth, fourth, third, second and first stages. About 1700 gal. per minute of wash solution are added to the washing circuit.

From the first stage of the washing circuit, the gold-bearing solution at 15% to 20% solids goes to the #2 35-ft. diameter high capacity thickener, in which solids are settled and the clean overflow solution flows to the carbon column upflow absorption circuit comprising five tanks for recovery of gold.

Six Agitated Tanks in CIP Unit

The carbon-in-pulp circuit comprises six agitated tanks in which the #2 thickener underflow is contacted with suspended carbon to recover



soluble gold. Loaded carbon containing the gold goes to the #1 and #2 pressure stripping vessels, each holding up to 4 tons of carbon that is heated to 250° F., with the solution recirculated. This is a batch process in which the solution goes up the vessel and gold is electrolytically stripped from the solution.

The loaded carbon from the carbon-in-pulp circuit and the carbon columns is taken in 4-ton batches to the pressure vessel, in which 1% to 2% sodium hydroxide and .1% sodium cyanide are added. From the pressure vessel, the solution goes to electrowinning cells followed by electroplating on steel wool cathodes.

Steel wool cathodes are heated to 1950° F. in a rotary gas-fired furnace, and the bullion is remelted and poured into 800 to 1000-oz. dore bars containing about 75% gold with 15% to 20% silver. Production averages 200 oz. of gold per day with one bar poured every three or four days and sold to refineries. In addition to 72,000 oz. of gold per year, from 10,000 to 20,000 oz. of silver will be extracted annually at Golden Sunlight.

From the mill, tailing flows through a 12-in. diameter pressurized polyethylene slurry pipeline with orifice plates rather than drop tanks, a distance of 10,000 ft. to the disposal area. Tailing is directed to cyclones situated on the dam to produce sand to build up the dam and slimes are discharged into the pond, from which clean water is reclaimed and recycled to the mill.

From 1000 to 1200 gal. of water including 500 to 700 gal. of fresh water pumped from the Jefferson slough storage basin are used per minute in the processing facilities. The plant is highly automated with controls provided by Gould, Modicon. Electric power for the Golden Sunlight mine is supplied by Montana Power Co., which built a substation and constructed a line two miles to a substation at the property.

In the plant area, an assay laboratory is equipped with a furnace room, sample makeup room, wet laboratory, weighing room, and atomic absorption room.

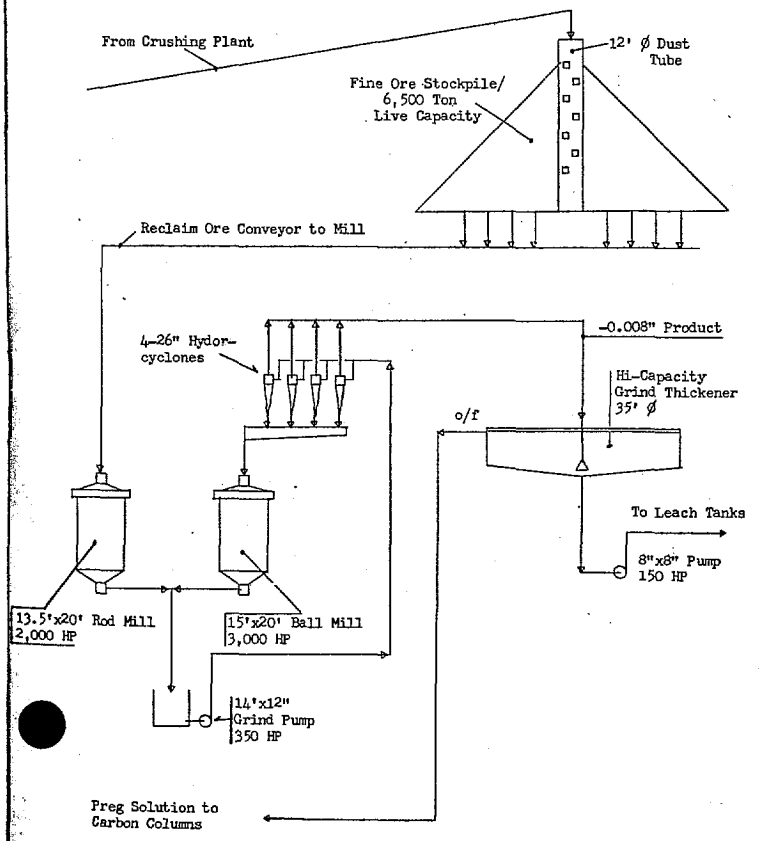
The administration building, which was completed in Sept. 1982 as the first major facility in the Golden Sunlight project to be occupied, includes offices of the manager and his staff as well as accounting and engineering departments.

A maintenance shop building completed in Dec. 1982 contains five bays consisting of two truck bays, two heavy equipment bays, machine shop and welding shop as well as the warehouse. The dry building contains a conference room, first aid room, oxygen room and ambulance.

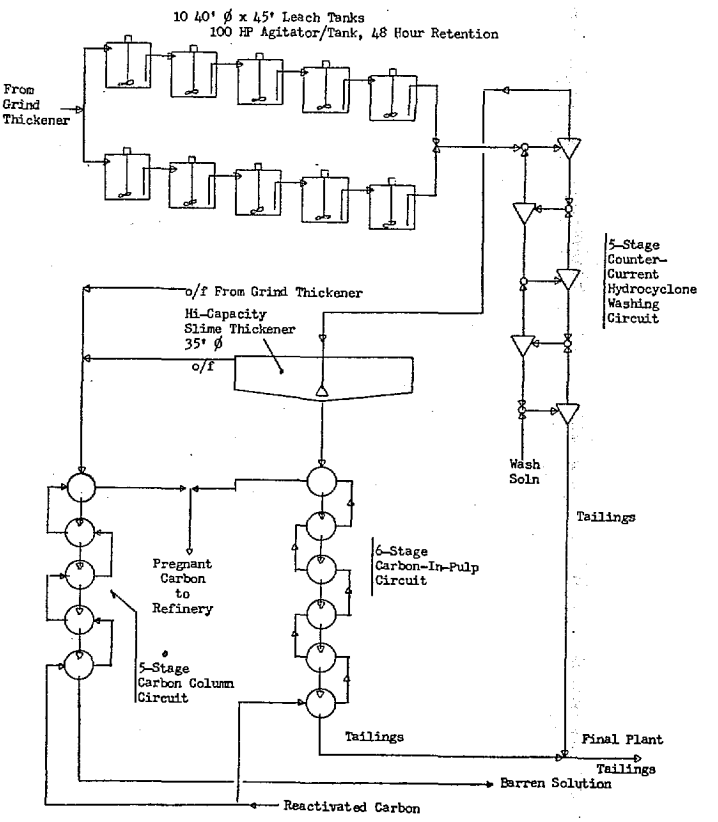
Jim Smolik is mine manager at the Golden Sunlight mine, with Donald E. Jenkins, administrative superintendent; Michael W. Roper, chief mine engineer; Quayle Lusty, mine superintendent; Gordon Ziesing, mill superintendent; Gerald Spear, maintenance superintendent; and Gerald Gouge, chief accountant.

About 120 personnel are employed at the Golden Sunlight mine, and employment will continue near this level until increasing during the second stage of pit development.

GRINDING AREA FLOWSHEET



GOLD RECOVERY AREA FLOWSHEET



JDS

ASARCO

Southwestern Exploration Division

June 21, 1983

Mr. R. L. Brown
New York Office

Gold Exploration

A small, say 300 tpd, lode gold mine recovering 0.25 oz/T produces over 25,000 ounces of gold annually equivalent to a 1700 tpd heap leach that recovers 0.042 oz/T. At reasonable operating costs and $400 gold, a 300 tpd operation should provide $3-5 million net per year. Any lode gold mine having this grade and a potential to provide a minimum of 700,000 tons should be of interest.

The mines closed by either War Production Order L-208 or labor shortages caused by WWII might be of interest since they should contain proven and probable reserves. Some of these, as indicated by the 1942 Minerals Year Book, are listed below:

STATE	MINE	DISTRICT, COUNTY	1942 PRODUCTION GOLD IN OUNCES
Arizona	Gold Road	Oatman, Mohave	28,570
	Tom Reed	" "	
	Others nearby	" "	
	Alvarado	Weaver, Yavapai	13,245
	Octave	" "	
	Yarnell	" "	
	Martinez	Congress, Yavapai	3,107
California	Idaho-Maryland	Grass Valley, Nevada	153,887
	Brunswick	" " "	
	Empire Star	" " "	
	Lava Cap	" " "	
	Golden Queen(Tropico)	Mohave, Kern	47,000
	Cactus Queen	" "	
	Kern Mines, Inc.	Cove, "	2,325
	Surcease	Yankee Hill, Butte	7,343
	Various	East Belt, parts of Amador, Calaveras, Eldorado, Mariposa, Tuolumne	+9,000 grading +.
	Alabama	Ophir, Placer	14,716
	Oro Fino		

STATE	MINE	DISTRICT, COUNTY	1942 PRODUCTION GOLD IN OUNCES
California	Original 16 to 1	Alleghany, Sierra	10,613
		Brown Valley, Yuba	5,994
Colorado		Gold Hill, Boulder	8,156
		Leadville, Lake	27,114
		Empire, Clear Creek	7,893
		Idaho Springs, Clear Creek	6,171
Idaho	Talache Mines	Middle Boise, Elmore	23,106
	Lincoln Group	West View, Gem	2,909
Montana	Ermont Mines	Argenta, Beaverhead	3,300
	Ohio Keating	Cedar Plains, Broadwater	8,775
		Norris & Norwegian, Madison	7,323
		Renova, Madison	9,818, Anaconda
		New World, Park	4,847
Oregon		Cracker, Baker	2,206
Washington	Knob Hill	Republic, Ferry	24,981

Each manager ought to review his files and literature to determine if any are worth pursuing (we already know Gold Road is!).

W. L. Kurtz

W. L. Kurtz

WLK:mek

cc: J.D. Sell
D.M. Smith
J.C. Balla
P.G. Vikre

New York, July 7, 1983

PERSONAL/CONFIDENTIAL

MEMORANDUM FOR: Mr. W. L. Kurtz

Your idea (your memo June 21, 1983) to look at small gold properties closed by government edict in 1942 is a good one and I endorse it. However, all concerned should bear in mind that the economics of your model are very skinny.

Total on-property cash costs at Galena (800 tpd) for 1983 year-to-date are $73/ton, for Coeur (535 tpd) $66/ton. These are cash costs without overhead, depreciation, or depletion, and at established properties with all production facilities including shafts, haulage levels, etc., already in place. Costs at small underground mines will vary widely, of course, but in their class (cut-and-fill in narrow veins) the Galena and Coeur are low-cost mines by comparison with other similar properties.

If we apply $70/ton costs to your model, then the maximum net per year is $3 million. If we assume it would cost a minimum of $12 million spread over 3 years to develop a small underground mine and get it into production with a mill (bear in mind we have already spent $6.4 million at Aquarius for underground development and sampling), then the return on investment for a 10-year operation (1 million tons) would be less than 17% before tax. This is without any provision for buying a property or profit royalties, etc.

Although there are different perceptions as to what is an acceptable return, certainly with a small mine we would want to see a substantially higher return than for a large investment, simply to justify management time and overhead. My own view is that 25% before income taxes would be a minimum, 30% would be more comfortable. Of course the distinctions between exploration, development, and exploitation get blurred with small underground mines as contrasted with open pits, and in special situations relatively modest returns could be accepted if the mine had real potential for continued development of new ore and possible expansion on a bootstrap basis after production starts, e.g., Aquarius.

The above is not meant to discourage you, but please sharpen your pencil before flooding us with recommendations for expensive underground development based on a model such as you suggest.

I've marked this memorandum and the copies "confidential" because we don't like to advertise our costs at Galena and Coeur: therefore, please don't circulate it; however, please encourage your people to use cash cost figures of $70/ton or more for small cut-and-fill underground mines when evaluating prospects.

T. C. Osborne

cc: RLBrown
FTGraybeal
JDSell
DMSmith
JCBalla
PGVikre

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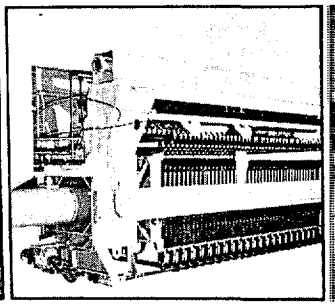

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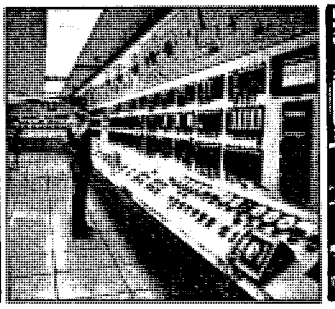

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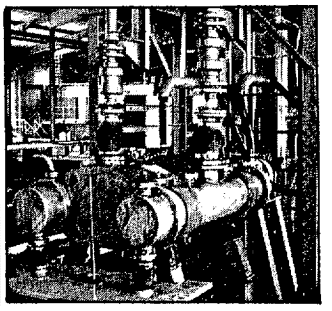

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On the cover: Carbon adsorption columns at the Welkom gold mine, now part of Anglo American's Western Holdings in South Africa's Orange Free State.

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Leaching and precipitation for gold and silver ores 48
Higher prices and better process technology have sparked new interest in low-grade deposits. This review highlights some of the important developments in precious metals leaching and precipitation technology now being used to exploit deposits once considered uneconomic.

How pressure filtration dewaters Butte concentrates 60
Ingersoll-Rands automatic "Lasta" pressure filters are the heart of Anaconda's new copper products dewatering plant in Montana.

Copper flash smelting 76
Phelps Dodge Corp. is using clean, energy-efficient technology to produce anode copper at its Hidalgo smelter in Playas, N. Mex. The operation's flash smelter, designed by Outokumpu, combines conventional roasting and smelting in a single, autogenous process.

Purification of zinc solutions for the tankhouse 83
A new solution purification process, using a continuous vibrating reactor and a novel combination of solid-liquid equipment, may cut costs at the Nordenham zinc plant in West Germany.

NaOH-STPP thinner improves wet grinding of phosphate rock 90
Research on mixtures of NaOH, Na_2CO_3, and sodium tripolyphosphate (STPP) has shown that a marked synergism occurs in the NaOH-STPP binary system—one specific mixture gave a strong performance in improving grinding efficiency.

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LEACHING AND PRECIPITATION TECHNOLOGY

FOR GOLD AND SILVER ORES

Alvin Lewis, associate editor

Many gold and silver deposits that were once considered uneconomic are now being exploited, thanks to higher prices and important improvements in the technology of leaching and precipitating precious metals from ore. Significant advances have been made in systems that use activated carbons to adsorb and desorb precious metal cyanides, and these advances are responsible for the development of carbon-in-pulp (CIP) plants worldwide, as well as for the increasing usage of carbon columns in heap and vat leaching operations, bleed streams, and tailings return waters.

The following is a summary of some of the more important developments in precious metal leaching and precipitation technology. Covered are the processes of cyanidation and alternative solvents for leaching gold and silver ores, heap and vat leaching, carbon-in-pulp, the technology of carbon adsorption and desorption, electrowinning, Merrill-Crowe zinc dust precipitation, comparisons between CIP and CCD (countercurrent decantation), and tables outlining how some important gold and silver producers process their ores.

DISSOLVING GOLD FROM ORES

For the many cases where the gold and silver particles in an ore deposit are too fine-grained (less than 50 microns) or too low-grade to be concentrated by gravity and/or flotation techniques, operators have been able to leach precious metals from ores via cyanidation. For a predominantly gold ore, a dilute solution of about 1 lb of sodium cyanide per ton of water (about 250 gal) is typically prepared to dissolve out the gold. For an ore containing significant amounts of silver, the cyanide strength of the solution is usually doubled. The exact solvent strength for any given ore must be determined during a pilot test program before going into production.

The chemical equations that govern the solubility of gold and silver during cyanidation are as follows:

$$4Au^0 + 8NaCN + O_2 + 2H_2O \rightarrow 4Na[Au(CN)_2] + 4NaOH$$

and

$$4Ag^0 + 8NaCN + O_2 + 2H_2O \rightarrow 4Na[Ag(CN)_2] + 4NaOH$$

These equations indicate that *oxidizing* conditions must be maintained in order for cyanidation to proceed. About 1-2 lb/st of lime or sodium hydroxide (caustic) is added to keep the system at an alkaline pH of 10-11. This "protective alkalinity" is required to counteract the generation of acid during cyanidation, thereby preventing cyanide degradation and the formation of deadly HCN gas.

Before opting for a cyanidation program, a mineralogical examination should be conducted to identify any interfering substances. A gold-silver orebody can contain cyanicides (partially oxidized sulphides of As, Sb, Zn, Fe, and Cu that consume cyanide), carbonaceous material (which adsorbs precious metal cyanides and causes premature gold and silver precipitation), and organic substances (which consume the vital dissolved oxygen from the leach solution). Also, cyanidation rates decrease with altitude and decreasing temperatures because of lower available oxygen at these conditions.

The cyanide leaching process is extensively used because of its economy and technological simplicity. Nevertheless, serious R&D efforts are underway to find leachants that are less toxic and dissolve precious metals at faster rates than cyanide. One potential candidate that has received considerable attention lately is acidothiourea.[20, 28] Preliminary studies in South Africa and Taiwan have shown that acidothiourea leaches gold 10 times faster than cyanide, is less affected by interfering ions, is not affected by cyanide-soluble minerals, and is less consumed by metal reactions on a per-weight basis. The three key problems that are preventing thiourea from commercial use are its high expense (about 25% more than NaCN), its degradation by oxidation, and the recovery problems encountered when using

10 largest Free World gold producing mines
(all are in South Africa)

Company	Location	Mine type	Ore Reserves '000 st	Ore Reserves Au (oz/t)	Ore Reserves '000 oz	Leaching method	Recovery method	Annual product ('000 oz)
Driefontein Cons.	Transvaal	U	20,011	0.543	10,866	Cn	Zn ppt	2,444
Vaal Reefs	Transvaal	U	20,717	0.347	7,189	Cn autoc	Zn carbo	2,363
Western Holdings	Orange Free State	U	16,026	0.334	5,353	Cn	Zn carbo	1,340
Western Deep Lev.	Transvaal	U	5,971	0.582	3,475	Cn	Zn ppt	1,254
Harmony	Orange Free State	U	23,443	0.222	5,204	Cn	CIP & Zn	1,027
Hartebeestfontein	Transvaal	U	14,649	0.395	5,786	Cn	Zn ppt	982
Kloof	Transvaal	U	3,681	0.643	2,367	Cn	Zn ppt	957
Buffelsfontein	Transvaal	U	10,081	0.341	3,438	Cn	Zn ppt	905
Free State Geduld	Orange Free State	U	7,430	0.400	3,418	Cn	Zn ppt	884
President Brand	Orange Free State	U	7,872	0.408	3,212	Cn	Zn & CIP	859
Total			**129,881**	**0.3873342**	**50,307**			**13,015**

10 Largest United States gold producing mines

Property	Company	Location	Mine type	Ore Reserves '000 st	Ore Reserves Au (oz/t)	Ore Reserves '000 oz	Leaching method	Recovery method	Annual product ('000 oz)
Homestake	Homestake	SD	U	17,518	0.220	3,854	Cn vat	CIP	278
Jerritt Canyon	Freeport	NV	O	11,614	0.233	2,706	Cn Cl	CIP	200
Carlin	Newmont	NV	O	8,718	0.127	1,107	Cn Cl	Zn ppt	145
Golden Sunlight	Placer Amex	MT	O	50,800	0.060	3,048	Cn	Zn ppt	72
Battle Mountain	Duval	NV	O	15,400	0.148	2,279	Cn	CIP	71
Alligator Ridge	Amselco/Nerco	NV	O	5,000	0.120	600	Heap	Carbon	68
Round Mountain	Louisiana Land	NV	O	195,400	0.043	6,402	Heap	Zn ppt	59
Pinson	Rayrock/Lacana	NV	O	5,853	0.071	416	Cn	CIP	56
Ortiz	Amcon (Cons. Goldfields)	NM	O	6,000	0.065	390	Heap	Carbon	49
Sunnyside	Standard Metals	CO	U	2,500	0.160	400	Fl, Gr	Amalgam	40
Total				**318,803**	**0.073**	**23,202**			**1,038**

Abbreviations: autoc—autoclave; Carbo—carbon columns; CIP—carbon-in-pulp; Cl—chlorine oxidation; Cn—cyanide; Zn ppt—zinc precipitation; Fl—flotation; Gr—gravity; O—open pit; U—underground. Data prepared for E&MJ by Homestake Mining Co. Figures are for 1982, or latest known data.

activated carbon.

Other potential leachant candidates include ammoniacal thiosulphate for recovering gold and silver from leach residues in the presence of copper,[3] and malononitrile for leaching gold from carbonaceous ores.[10]

LEACHING IN HEAPS AND VATS

Since the early 1970s, when Cortez Gold Mines proved that gold and silver in low-grade deposits could be economically recovered on a large scale by leaching heaps, operators worldwide have been able to successfully recover precious metals from 5,000- to 2 million-ton heaps grading as low as 0.03 oz/st. The flexibility, low capital costs, and reasonable operating costs of heap leaching have become attractive to the small operator, especially at today's high precious metal prices.

Heap leaching typically recovers only about 60-70% of the precious metals in the ore, significantly less than the 90-99% frequently reported for ores treated by carbon-in-pulp or countercurrent decantation leach circuits. Nevertheless, heap leaching has enabled certain small, low-grade deposits that were once considered waste to be exploited at a profit.

Laboratory and pilot tests are recommended before deciding to go ahead with a full-scale operation. The ore should contain fine gold particles, be free of cyanicides and carbonaceous material, and have only a minimum of acid-forming constituents. Ores can be run-of-mine or crushed to 10 mm, and are mixed with 3-5 lb of sodium hydroxide per ton of ore before being piled on a sloped, impervious pad. Operators have found that sodium hydroxide is preferred to lime, because it does not scale up in the sprinkler system on top of the heap.

If the ore has a high clay content, agglomeration with Portland cement, water, and cyanide is often recommended to produce uniform feed and enhance permeability throughout the heap. State-of-the-art agglomeration procedures were researched and published by the USBM and have proved to be invaluable and inexpensive procedures for leaching ores with fines and/or clays. Preliminary testing, pad design, ore preparation, loading, cyanidation, and recovery reviews have also been published (see reference numbers 1,5,6,11,22,25,26, and 29).

Once the heap has been prepared, the sprinkler system applies 4-75 gal/ft^2/day of dilute alkaline cyanide solution from a "barren reservoir." The cyanide solution is prepared by adding 1-4 lb of cyanide per ton of water, and is applied preferably by large-drop sprayers. The cyanide solution dissolves gold and silver as it percolates through the oxygenated heap, and the "pregnant" solution drains from the bottom of the heap to plastic-lined channels and finally to a "pregnant solution storage basin." Pregnant solutions are processed through a precious metal recovery system, the most popular being either carbon adsorption or Merrill-Crowe zinc dust precipitation. Alternative recovery systems include ion exchange, direct electrowinning, soluble sulphide precipitation, and aluminum dust precipitation.

Once the pregnant solution has been stripped, cyanide and sodium hydroxide are added to the barren solution to bring it back up to a pH of 10-11 and the required cyanide concentration. The revitalized cyanide solution is then recycled to the heap.

Carbon adsorption recovery. Adsorption onto activated coconut shell carbon has become a popular method for recovering gold and silver from large volumes of low-

Some carbon-in-pulp operations

Name, location	Ore: Type	Ore: Grade (oz/st)	Ore: Reserves (million st)	Ore: Treatment	st/24 hr processed recovery (%)	oz/24 hr produced	Direct cost/st processed	Number of employees
Battle Mountain (Duval) Lander, NV	Sulph; oxide exposures	0.080 Au 0.10 Ag	4	48 hr cyanid in pachucas; CIP for 6 hr	3,000 st (92.5% Au) (40.0% Ag)	220 Au 120 Ag	$13.21 (1979)	270
Bulldog Mtn. (Homestake) Creede, CO	Cyanid slimes from flot mill	2.00 Ag Au (?)	N/A	36 hr cyanid in pachucas; CIP for 6 hr	400 st (70.0% Ag)	(?) Au 560 Ag	$ 9.16 (1979)	11
Jerritt Canyon (Freeport Gold) Elko, NV	Calcareous slts	0.220 Au Ag (?)	12	24 hr cyanid with carbon in agitators; zn ppt of strip soln.	2,750 st (87.5% Au)	600 Au (?) Ag	N/A	N/A
Pinson Golconda, NV	Metaseds	0.120 Au Ag (?)	3.2	24 hr cyanid in agitators; pump through carbon columns	1,500 st (86.2% Au)	160 Au (?) Ag	$14.25 (1981)	83

Some Nevada heap leaching operations

Name, location	Ore: Type	Ore: Grade (oz/st)	Ore: Reserves (million st)	Ore: Treatment	Leach pad	Recovery (%)	Ore processed (st/yr)	Production (oz/yr)	Direct cost/st ore processed	No. employees
Alligator Ridge (Amselco) White Pine, NV	Silicic slts	0.120 Au Ag (?)	5	Crush to -3/4 in.; agglom.	12 in. compact clay	Carb ads	750,000	70,000 Au	N/A	140
*Bootstrap (Carlin) Elko, NV	Oxid. slts	0.044 Au Ag (?)	<1	-1/2 in. mine run	Compact clay lake bed	Carb ads (54% Au)	220,000	6,000 Au	$1.70 (1979)	N/A
Borealis (Houston Oil & Min) Mineral, NV	Min volc	0.090 Au Ag (?)	2.5	Crush to -2 in.; agglom.	5 in. compact asphalt	Merrill-Crowe (70% Au)	450,000	28,000 Au	N/A	60
Candelaria Mineral, NV	Dissem Ag in oxid silicic sh	Minor Au 3.15 Ag	N/A	Crush to 1½ in.; agglom.	18 in. compact clay	Merrill-Crowe (50% Au) (42% Ag)	2 million	2.6 million Ag	N/A	160
*Gold Acres (Cortez) Lander, NV	Oxid ls, chert, breccia	0.036 Au Ag (?)	5	Un-crushed mine run	15 in. compact mill tails	Carb ads (55% Au)	1 million	20,000 Au	$0.66 (1976)	N/A
Round Mountain (Smoky Valley) Central NV	Welded rhy tuff	0.060 Au 0.07 Ag	>6	Crush to -1/2 in.	2 in. rubberized asphalt, 5 in. asphalt	Carb ads (67% Au) (30% Ag)	2 million	60,000 Au 30,000 Ag	$5,32 (1980)	129
Tuscarora Elko Co., NV	Andesite intrusive	0.020 Au 1.75 Ag	0.5 (mine waste)	Un-crushed mine run (fines to 6 in.)	Wet, compacted, lake bed clays	Merrill-Crowe (50% Au) (40% Ag)	100,000	1,000 Au 70,000 Ag	$6.50 (1980)	9
Windfall Eureka, NV	Dol-omite	0.028 Au Ag (?)	<1	Un-crushed mine run	2 in. compact asphalt	Carb ads (80% Au)	220,000	5,000 Au	$3,93 (1980	39
Ortiz (Gold Fields) New Mexico	Qtz breccia	0.050 Au Ag (?)	>5	Crush to -3/4 in.	2 in. rubberized asphalt, 5 in. asphalt	Carb ads (90% Au)	750,000	35,000 Au	$3.57 (1980)	46 plus con-tractors

*not operating

Abbreviations: and—andesite; calc—calcareous; carb ads—carbon adsorption; conc—concentrate; cy—cyanide; dol—dolomite; int—intrusive; min volc—mineralized volcanics; N/A—not available; ppt—precipitate; rhy—rhyolite; sh—shale; slts—siltstone; py—pyrite; po—pyrrhotite.

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grade pregnant solutions that contain mainly gold. Activated carbons have surface areas of about 1,000 m^2/gm and can adsorb up to 30,000 ppm of gold in a cyanide complex, leaving a barren solution with only 0.005 ppm of gold. Ores with very high Ag and Au values can be treated with sodium sulphide to selectively precipitate the silver values prior to reaching the carbon adsorption circuit.

Carbon adsorption from heap leached pregnant solutions commonly occurs in a series of four or five columns or tanks, which are usually arranged in an open cascade design with overflow launders on each tank leading to a feed pipe at the bottom of the following tank. Where floor space is at a premium, the columns can be vertically superimposed. Solution velocity and volume are controlled to maintain a suspended bed of carbon in the stream without carrying the carbon away from the system. Carbon samples from the lead column are fire assayed and when fully loaded, the lead column is removed for desorption. The second column in the series now becomes the lead column, the third becomes the second, and so forth. A last column is returned with regenerated carbon plus any new make-up carbon that may be required due to attrition.

Precious metals desorption. The precious metals loaded onto activated carbon are usually stripped on site, but some smaller operators may prefer to sell their loaded carbon to a smelter. Various stripping technologies are available, including a hot (90°C) strip with 1% NaOH in a 0.1-0.2% NaCN solution; a hot (about 60°C) strip with 10% ethanol or propanol in a 1% NaOH plus 0.1-0.2% NaCN solution; a hot (120-130°C) pressure strip with NaOH plus NaCN; a strip with various aqueous organic mixtures at room temperature; and stripping with zinc dust to yield a filterable precious metal product.

High-temperature and high-pressure stripping techniques are more expensive but much faster than room-temperature techniques. Care must be taken when stripping with alcohol because of its flammable, explosive, and toxic nature.

Merrill-Crowe precipitation. If the pregnant solutions are to be treated by Merrill-Crowe zinc dust precipitation instead of by carbon adsorption, the pregnant solution must first be clarified and de-aerated before introducing the zinc dust. This technique is preferred for very rich pregnant solutions, small volumes of solution, or solutions containing large amounts of silver. The zinc dust precipitates gold according to the following reaction:

$$2NaAu(CN)_2 + Zn^0 \rightarrow Na_2Zn(CN)_4 + 2Au^0$$

A similar equation holds for precipitating silver.

Leaching in vats. Gold does not readily dissolve in cyanide at low temperature or at very high elevations because of reduced oxygen activities in such environments. Therefore, some operators, especially those who encounter cold winter months, leach their ores year-round in indoor vats. Silver State Mining Co. operates four separate leaching vats on four-day cycles in Cripple Creek, Colo., despite high altitudes and sub-freezing winters.[19] At Silver State, the pregnant solutions are drained from the vats and moved to a carbon adsorption circuit.

CARBON-IN-PULP

Carbon-in-pulp (CIP) has recently become a standard practice for recovering gold and silver from low-grade, leached ore. The main advantages of CIP are that it can process slimy ores without a liquid-solid separation, and that precious metals recovery percentages are often very high. Significant savings in capital costs are realized, because a filtration or countercurrent decantation system is not needed. Several excellent papers dealing with details of the basic unit operations, new developments, costs, etc., are available (see reference numbers 1, 2, 9, 13, 14, 15, 16, 17, 18, 23, 26, and 29).

Unit operations. The unit operations for CIP include grinding, pre-screening of fines, adsorption onto activated carbon, acid washing, elution, carbon reactivation, and electrowinning. Ore is typically ground to minus 35 mesh, prescreened to remove grit, classified, thickened, and conditioned with air and lime before entering the leach circuit. The leach circuit consists of a series of mechanical or air agitators (i.e., pachuca tanks), where precious metals are dissolved from the pulp in an oxygenated solution of cyanide plus caustic. The pulp then flows to the six to eight tanks of the CIP circuit, where it is further contacted with NaCN, lime slurry, and activated carbon that is coarser than the pulp, and onto which the precious metals are adsorbed (6x16 mesh carbons are common with minus 35-mesh pulps). Various types of adsorption vessels are available, including mechanical or air agitators, simple propeller tanks, pachuca tanks, and draft tube agitator tanks. The draft tube agitator is the most recent development, requiring only about ⅓ the input horsepower of a conventional agitator and having features to control pulp velocity and therefore avoid undue shearing of the activated carbon.

In the adsorption vessels, the leach pulp is moved countercurrent to the flow of the activated carbon. In one of the earlier CIP designs, the carbon and the pulp are both air-lifted to external screens, which allowed the fine-ground pulp to move on while the coarser carbon grains were recovered and then returned to the same tank. A newer technology now being pilot tested in South Africa involves transferring the carbon and pulp to stationary peripheral and launder screens at the top of each adsorption vessel, again retaining the carbon and advancing it countercurrent to the flow of the pulp. The carbon continuously loads precious metal cyanides and, when fully loaded, is air-lifted to screens and moved to stripping vessels.[29] The barren pulp is screened as it leaves the circuit and is disposed as tails.

In the adsorption vessels, pulp densities are usually maintained at 40-45% solids. During a CIP discussion at the 1981 AIME annual meeting,[2] panelist Kenneth B. Hall of Homestake indicated that at higher pulp densities, carbon in the adsorption vessels will begin to float rather than intimately mix with the pulp, while at lower densities, the contact time is decreased and the carbon may begin to settle out of the pulp. Furthermore, the pH should be maintained at 10.0-10.5. "At too low pH levels, HCN gas will evolve [and cause] high cyanide consumption [but at] too high pH levels, dissolution is slowed," Hall said. At Homestake's Lead, S.Dak., operation, cyanide levels are kept above 0.015% for optimum recovery. Independent consultant George Potter cautioned that unpublished data from the US Bureau of Mines Center in Salt Lake City indicated that if sodium hydroxide is used (instead of lime) and if the pH rises above 10.2, gold and silver loading onto activated carbon in the adsorption circuit will be inhibited.

Hard carbons from coconut shells are preferred

Some operations employing other treatments

(includes counter current decantation, batch cyanidation of gravity and flotation concentrates, and leaching flotation mill tailings)

Name, location	Ore			st/24 hr processed recovery (%)	oz/24 hr produced	Direct cost/st processed	Number of employees
	Type	Grade (oz/st)	Treatment				
Agnico-Eagle, Joutel, Que., Canada	Sulphide ore, py and minor po	0.190 Au Ag (?)	Grind; flotation; cyanidation; Merrill-Crowe ppt.	1,200 st (91.0% Au)	210 Au (?) Ag	$ 12.32 (1980)	190
Delamar Owyhee, ID	Ag & Au in rhyolite	0.020 Au 4.70 Ag	Grind; 72 hr cy leach; CCD thickened; Merrill-Crowe ppt.	2,200 st (92.0% Au) (85.0% Ag)	40 Au 8,800 Ag	$ 6.37 (1979)	135
Elandsrand (Anglo American Corp.) Carletonville, Transvaal, S.Af.	Ventersdorp contact reef	0.170 Au Ag (?)	Grind; gravity concn; flotation; smelt. Cyanid of flotation tailings in pachucas; Merrill-Crowe ppt.	5,000 st (95.2% Au)	800 Au (?) Ag	$ 41.70 (1980)	7,759
Gooseberry (Westcoast Oil & Gas) Sparks, NV	Qtz veins	0.190 Av 7.50 Ag	Grind; flotation; cyanid; filter; Merrill-Crowe ppt.	350 st (83.0% Au) (85.0% Ag)	55 Au 2,230 Ag	$115.18 (1980)	98
Manhattan (Houston Intl. Min.) Manhattan, NV	schist	0.080 Au Ag (?)	Ground; gravity concn; flot tailings; batch cy. gravity and flot concn; Merrill-Crowe ppt.	650 st (85.0% Au)	45 Au (?) Ag	$ 7.66 (1980)	49
Pueblo Viejo (Rosario Dominicana) Dominican Republic	Ag/Au laterite	0.130 Au 0.60 Ag	Grind; cy leach in agitators; CCD thicken; Merrill-Crowe ppt.	8000 st (91.0% Au) (77.0% Ag)	950 Au 3,700 Ag	$ 3.06 (1977)	750
Segovia (Frontino) Segovia, Antioquia, Colombia	Qtz veins in grndior	0.190 Au 0.52 Ag	Grind; flotation; cyanid; Merrill-Crowe ppt.	660 st (85.0% Au) (75.0% Ag)	110 Au 260 Ag	$ 12.12 (1975)	N/A
Telfer (Newmont Prop.) N.W. Australia	Qtz limonite	0.230 Au Ag (?)	Grind; gravity concn; cyanid tailings; Merrill-Crowe ppt.	1,600 st (98.0% Au)	360 Au (?) Ag	N/A	36

Abbreviations: and—andesite; calc—calcareous; carb ads—carbon adsorption; conc—concentrate; cy—cyanide; dol—dolomite; int—intrusive; min volc—mineralized volcanics; N/A—not available; ppt—precipitate; rhy—rhyolite; sh—shale; slts—siltstone; py—pyrite; po—pyrrhotite.

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because of their resistance to breakage and abrasion. Abrasion-resistant carbons are required to minimize precious metal loss from fine but loaded carbons passing through screens with the barren pulp. Designs are implemented to reduce carbon abrasion. For example, carbons are sometimes pre-abraded to wear off any sharp edges before being put into the CIP circuit. Static screens and plastic piping are beginning to replace vibrating screens and iron piping, and researchers are looking into lubricating and suspending ore pulps with clay and chemicals.

Early CIP designs incorporated separate processes for leaching and adsorption, but studies are now underway to perfect the combining of these two processes simultaneously in what is known as the *carbon-in-leach* setup. The first tanks are used solely for leaching, and subsequent leaching plus adsorption goes on simultaneously in the remaining tanks. A separate adsorption circuit is not necessary.

Desorption and electrowinning. As the loaded carbons leave the CIP circuit, they are carefully washed with water sprays on a vibrating screen before entering stripping vessels. A variety of desorption techniques are available, including atmospheric, pressure, or alcohol strip. The atmospheric strip requires less equipment, lower capital expenditures, lower operating costs, and is optimum for many small plants. However, atmospheric desorption takes the longest amount of time—about 40 hr at Homestake's Lead operation and up to 72 hr at other plants.[2] The other strip methods can cut treatment time to 4-10 hr and are recommended for large-tonnage plants with high loadings. Potter indicated that in a pressure strip, a O.1-O.2% NaCN solution with 1% NaOH is heated to 120-130°C under 75 lb/in.^{2} in a stainless steel vessel. The chemistry of the alcohol system is "among the best," he said, but caution is necessary because of the toxic, flammable, and explosive nature of alcohol.[2] It may take only 6 hr to strip carbon from 10,000 to 100 ppm.

According to Donald Duncan of Pinson Mining Co., the South African's are developing an acid-conditioning

procedure prior to stripping that cuts the stripping time to about 8 hr. Most of the work has been done with HCl, but HNO_3 is recommended, because it is a much safer acid to use around cyanides.[2]

After desorption, provisions are made to reactivate the carbon for reuse with a dilute nitric acid wash, water wash, 1% NaOH wash, drain cycle, and heating cycle to about 630°C for 30 min in an indirectly fired rotary kiln. The carbons are screened to remove fines and presoaked at least 12 hr before reuse. This procedure removes various contaminants, such as organic oils and carbonates that coat or adsorb onto the carbons, thereby decreasing the number of available sorption sites.

The strip solutions are electrowon onto steel wool cathodes in Zadra cells or rectangular cells. The rectangular cells are often used when floor space is at a premium, and they have a further advantage in that the cathodes can be removed from the bottom of the cells without shutting down the circuit.[2]

Electrowinning consists of passing the strip solutions through a series of at least three cathodes at a negative potential of less than 3 v. Gold and silver are electroplated as loosely adhering metals, along with some copper, lead, and mercury.[26]

Instead of electrowinning, strip solutions can be precipitated by zinc dust, and the precipitate removed by filtration. Steel wool cathodes from electrowinning or precious metal precipitates from zinc dusting are smelted to doré.

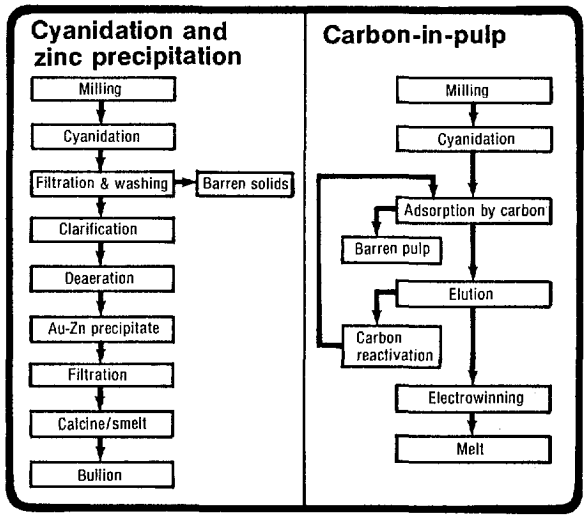
COMPARING TECHNOLOGIES

The leach CIP process was compared with the leach countercurrent decantation (CCD) circuit by Acton and Charles at the 14th International Mineral Processing Congress in October, 1982.[1] CCD is a well-established practice for washing leached pulp in thickeners, where the barren solution moves countercurrent to the pulp. The subsequent pregnant solution is clarified with flocculants, filtered, de-aerated, and precipitated with zinc dust and lead salts. A filter press forms a precious metal cake, which is sometimes digested to remove zinc, refiltered, and smelted. The barren cyanide is then recycled to the last thickener.

Recoveries by CCD and CIP are comparable on fast-settling ores, but CIP is faster with clayey ores. CIP requires less equipment for lower capital costs, occupies minimal space, and offers greater recovery efficiencies with dilute mill solutions. Also, adsorption is not inhibited in the presence of soluble sulphides, copper, or iron—common interferents with the zinc precipitation process. The zinc precipitation process requires careful control of alkalinity and free cyanide, and is more labor-intensive than CIP.

CIP technology does have some drawbacks. These include the need to screen all of the pulp, the use of more cyanide (because it is not regenerated), high gold inventories in process, and lost gold to tailings with the fine, abraded carbon.

An innovation that could prevent the loss of loaded fine carbon to tailings is "Magchar" (magnetic activated carbon).[12] Magchar carbons are combined with magnetite so that carbon recovery can be accomplished with magnetic separators instead of screens. The technology is not new—a patent was awarded to American Cyanimid in 1946, and pilot tests were conducted in 1947 at the Getchell mine in Nevada and at Eagle-Picher's Sahaurita pilot plant in Arizona. But, with the resurging interest in



low-grade gold and silver ores and higher precious metals prices, Magchar carbons may merit reconsideration, especially for slow-settling, slimy ores. The carbons can be finer-grained for faster adsorption, and the need to grind the pulp finer than the carbons is eliminated. One drawback to the Magchar process is that magnetic separators have higher capital costs than the screens currently used in CIP.

A critical advantage of CIP as compared to a CCD leach circuit, pointed out by Potter at the AIME panel discussion, is that it "gets virtually all of the soluble values that would otherwise go out in your filter cakes or CCD final underflow."[2] Hall added that as the tonnage of ore and precious metal values increase, the amount of carbon required increases substantially and "at some point, it will become less economical to operate a CIP plant." Nevertheless, Potter reiterated that "every gold plant and silver plant, [even] if you are using Merrill-Crowe, should be using a carbon column somewhere in the system" to scavenge precious metals from bleed streams and tailings return water. Homestake, for example, treats its tailings return water to recover 0.005-0.01 oz/st gold by passing the water through carbon columns.

For refractory and carbonaceous ores where gold cannot be extracted by direct cyanidation, a 30-yr-old pressure leaching technology from Sherritt may be commercially justified at high gold prices.[21] The concept involves aqueous oxidation in autoclaves at high temperatures and oxygen over-pressures. Once the gold-bearing sulphides are oxidized, the gold is liberated, and extraction is possible via cyanidation.■

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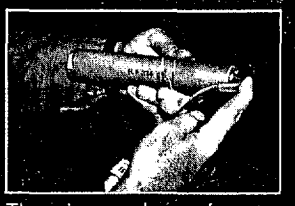
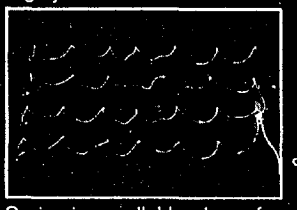
HERCUDET*


The primer makeup of Hercudet is uncomplicated.

Miners use conventional loading systems with Hercudet.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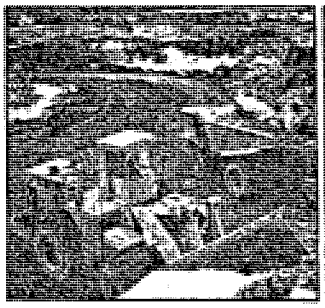
CARLIN GOLD
A RENEWABLE ASSET FOR NEWMONT MINING



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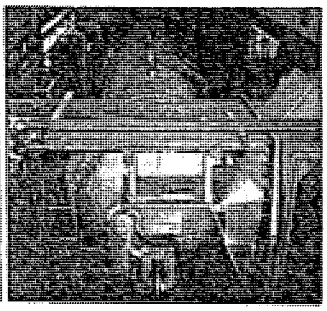

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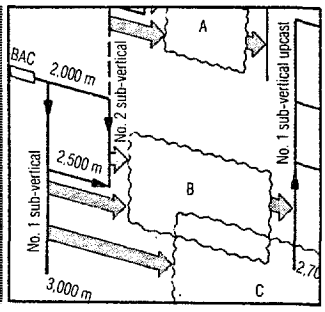

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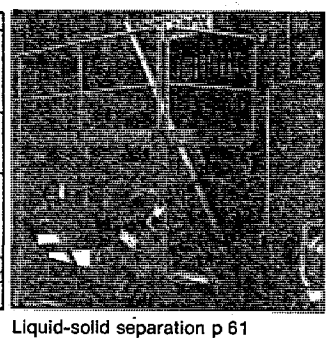

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CARLIN GOLD

A NEWMONT MONEY GENERATOR KEEPS ON RENEWING ITSELF AFTER SPARKING THE REBIRTH OF GOLD MINING IN NEVADA

Dan Jackson, western editor, COAL AGE

With newly identified deposits, Carlin Gold Mining Co., a 100%-owned subsidiary of Newmont Mining Corp., is now studying significant growth potential after generally being credited with launching Nevada's most recent gold rush in 1965. Since then, Carlin and a number of other companies have pushed Nevada to the lead as the largest source of gold in the US.

Newmont took a chance on the district 40 mi west of Elko in the early 1960s, after publication of a now classic study by the US Geological Survey. To the discerning mind, that report suggested some interesting exploration possibilities in eroded windows of the Roberts Mountain thrust fault. The company's exploration efforts soon identified 11 million st of 0.32-tr-oz/st, microscopically sized gold at a time when the yellow metal was worth only $35/tr oz. But Newmont had a concept and a plan for gold mining based on open-pit mining of low-grade ore. With rare vision, then all but unknown among large US mining firms, who had little inclination to get involved with modest-sized ventures, Newmont developed the mine and built an 1,800-st/d cyanidation plant. The Carlin subsidiary has served the parent well ever since.

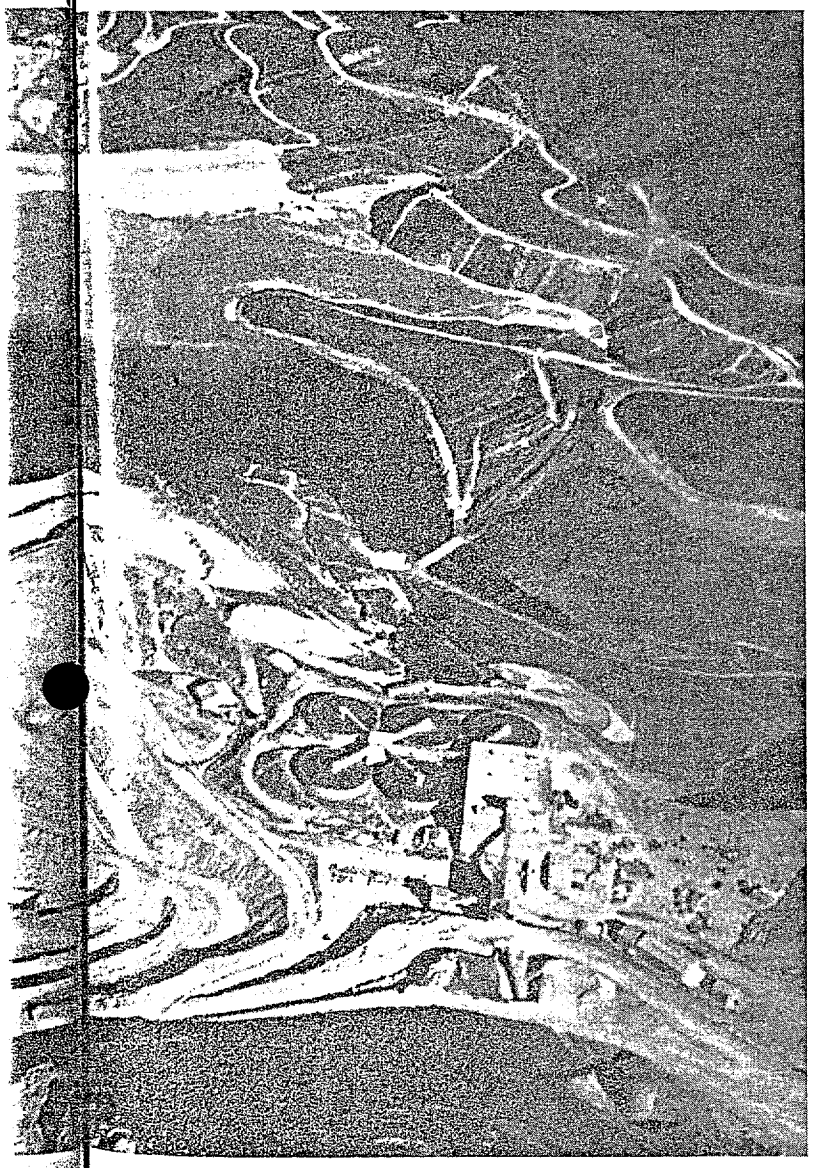
Today, Carlin Gold now operates three open pits within a 20-mi radius of the plant—the Carlin (the main orebody), Maggie Creek, and the Blue Star. Carlin and Blue Star are producing mill feed ore. Maggie Creek excavates mine-run ore for the mill as well as low-grade ore for on-site heap leaching and carbon adsorption columns. The Bootstrap mine is now depleted of ore but supports ongoing leaching activities.

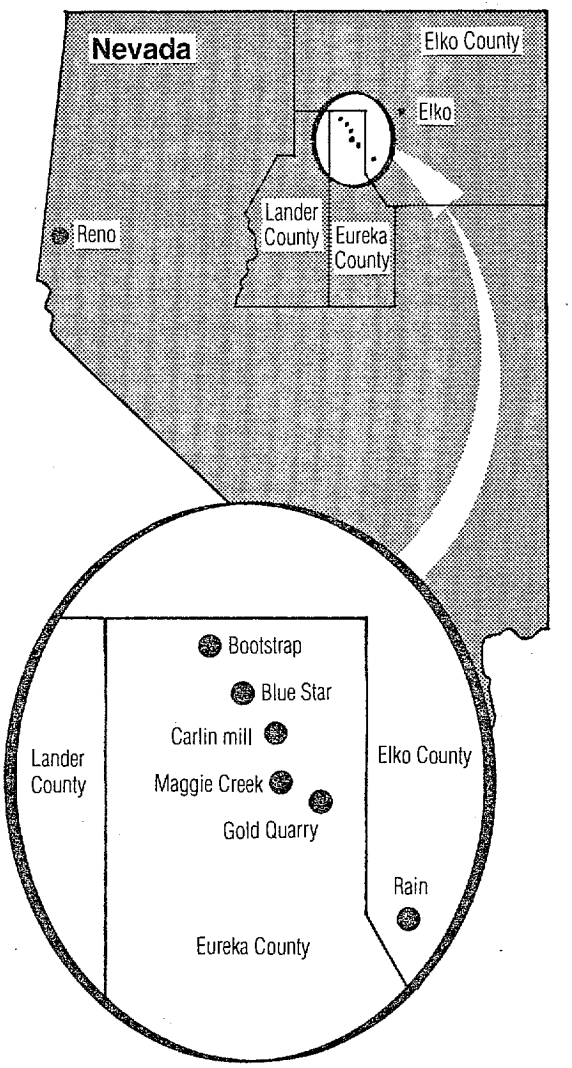
In the past two years, Carlin has announced new nearby gold finds, including the Gold Quarry and Rain deposits. According to Ralph Mattson, executive vice president and general manager, the company has completed geological studies and has established *in situ* reserves in these deposits. Carlin is now engaged in the extensive geological and metallurgical research necessary to develop a mining plan for the Gold Quarry deposit.

A traditional agitation-leach and CCD (countercurrent decantation) cyanide process was developed for the Carlin plant at Newmont's Danbury, Conn., Research Center. This process worked efficiently on oxide ore. Later, the company, in cooperation with the US Bureau of Mines Metallurgy Research Center at Reno, Nev., developed a chlorination pretreatment for more troublesome carbonaceous ores and added a processing module to the Carlin plant in 1971. Since then, the pretreatment circuit for the more refractory ore had been modified to accept chlorine at higher rates. Additional equipment was also installed to partially oxidize carbonaceous ore prior to chlorination. This alteration permitted the processing of ores having higher carbon and pyrite contents than was previously possible. The carbonaceous pretreatment circuit accepts mine-run feed at a rate of 500 st/d and the oxide circuit treats 1,800 st/d of mine-run feed. Slurry from the oxidation/chlorination circuit travels through the leach and CCD circuits of the mill concurrently with the oxide pulp.

When Gold Quarry goes into production, Carlin will nearly double its gold output with process tonnages that approach 12,500 st/d, up from the current 2,500-st/d rate. The company began an 8,000-st, heap-leach, test

When it went into production in 1965, Carlin Gold Mining Co.'s operation ranked as the first US gold discovery of major importance in more than 35 years. The original mine and plant are still the central focus of a number of satellite orebodies that furnish heap-leach and mine-run milling ore. The new Gold Quarry and Rain deposits are now under study for development and production.





facility last summer to try new ideas aimed at improving the state of the art. To date, leaching of ores from Maggie Creek has followed standard practice, including agglomeration of the low-grade ore placed on the heaps with Portland cement.

The company currently excavates about 15,000 st/d of material from Maggie Creek and 30,000 st/d from Carlin. Gold production in 1982 totaled 145,000 tr oz, and recovery in 1983 is projected to reach approximately 160,000 tr oz. Since the plant started in 1965, Carlin has mined 118 million st of waste to extract 16 million st of ore. Bullion output through 1982 amounted to 3.4 million tr oz. from the company's mines.

THE NEVADA CONNECTION

Newmont became interested in exploring for gold in Nevada in 1960. At that time, the US Geological Survey published Professional Paper 400-B, "Alignment of Mining Districts in North-Central Nevada," by Ralph J. Roberts, who investigated a major fault system now referred to as the Roberts Mountain thrust fault.

After digesting the report, Newmont exploration geologists pinpointed an area deserving further studies. It was located in the old Lynn Creek mining district, where placer prospecting had been carried out sporadically for years, with little success. The area is located 18 mi north of Carlin and 40 mi west of Elko in the Tuscarora Mountains. A 6-mi^2 area was staked in 1961 and 1962, followed by a drilling program that showed gold values in the third borehole. More than 800 4¾-in.-dia rotary drill holes were sunk on a 100-ft grid and sampled at 5-ft intervals (250,000 linear ft total) to block out the Carlin orebody.

GEOLOGIC WINDOWS OF TIME

Carlin's ore deposits occur in windows within the Roberts Mountain thrust. According to Roberts, these windows are a result of doming, part of which may have occurred during or shortly after thrusting and part of which may have occurred much later. Alignment of the windows, notes Roberts, indicates that doming occurred in zones of structural weakness in which movement has taken place intermittently. The nature of the deformation is not fully understood, and the precise date of its beginning is not known. However, the zones of weakness and the windows along them are penetrated by conduits, along which igneous rocks and related ore-bearing fluid rose. The zones probably penetrate to great depths within the crust and possibly date back to Precambrian time.

This crusting is related to the Pre-Carboniferous Antler Orogeny and occurred along a regional thrust plane that forced western, eugeosynclinal, siliceous, clastic rock into contact with eastern, miogeosynclinal, carbonate rocks of similar age.

The Carlin mine is located on the northeastern edge of the Lynn window near the Roberts Mountain thrust fault. Other windows within the area, including the Carlin and Bootstrap, are typical of the Lynn.

Host rocks for the Carlin deposit are exposed in the central section of the Tuscarora Range uplift. They strike easterly and dip about 45° north. Upper plate rocks adjacent to the thrust are designated the Ordovician Vinini formation. They consist of bedded cherts and shales with sandstone lenses. Higher in this section, limestones, siltstones, and quartzites are exposed.

Immediately below the thrust is about 300 ft of massive to locally thin-bedded, silty to sandy, carbonaceous Devonian limestone, locally referred to as Popovich limestone. Below the Popovich, the Silurian-Early Devonian Roberts Mountain formation consists of silty to sandy, carbonaceous, dolomitic limestone in fresh outcrops, totaling about 1,500 ft in thickness.

Most of the gold orebodies lie within the upper 300 ft of the Roberts Mountain formation, which contains a higher sand-silt content. The Ordovician section below the Roberts Mountain formation include the Hansen Creek dolomite, Eureka quartzite, and Pogonip group.

Gold mineralization occurs in upper- and lower-plate sedimentary rocks ranging in age from Ordovician to Upper Devonian. Dikes of Tertiary age also are mineralized. The most commonly occurring metals are barium, arsenic, strontium, manganese, antimony, and gold.

Barite, pyrite, and iron oxide are the most commonly occurring minerals. Calcite is present in moderate amounts. Galena, cinnabar, and stibnite can be recognized by sight.

EVOLVING MINE DESIGN

Since discovery and development of the original orebody by Newmont, Carlin Gold Mining Co. geologists have carried on in the traditional manner of finding and developing additional orebodies, says Jay McBeth, vice president and manager—operations. McBeth details the procedure as follows: After a discovery is made, mineralized zones are drilled on a 100-ft grid spacing. Each 5 ft of hole is then assayed. Rock-chip boards are constructed for each hole and logged to determine rock type and alteration features. All data are then transmitted to Newmont's Danbury Computer Center for storage.

Generalized ore outlines are plotted on computer-generated, vertical assay sections and then adjusted to conform to known geologic data. These refined projections are transferred to plan, along with known and inferred geological information. By combining the vertical and plan (bench) sections, a trial economic pit bottom is established, and the first trial pit is designed. As new drill-hole and geological data become available, the sections are updated, and new reserve figures are calculated. Final pit design is determined upon completion of the drilling program.

At this point, final pit slope has already been established from a study of rock types, bedding angles, and major structural features. The haul road, containing the necessary mining requirements (such as 8% maximum grade, 80 ft wide, and 100-ft-radius curves), is also planned to minimize, where possible, adverse effects that bedding and structure would have on the final pit wall.

Mine and mill production figures are projected annually on a five-year schedule, with the first year projected on a monthly basis and the remaining years, quarterly. Mine production, says McBeth, can come from a number of locations in any given year, and this requires tight sequence planning of waste and ore removal from each area to assure an uninterrupted supply of ore to the mill. This schedule is used by the six operating departments to develop cost estimates, which are integrated into the total budget, as well as to forecast income to be derived from sale of the product. The five-year plan also assists in anticipating where additional mining capacity may be needed to maintain existing production or bring in new orebodies.

MINING PROFILE

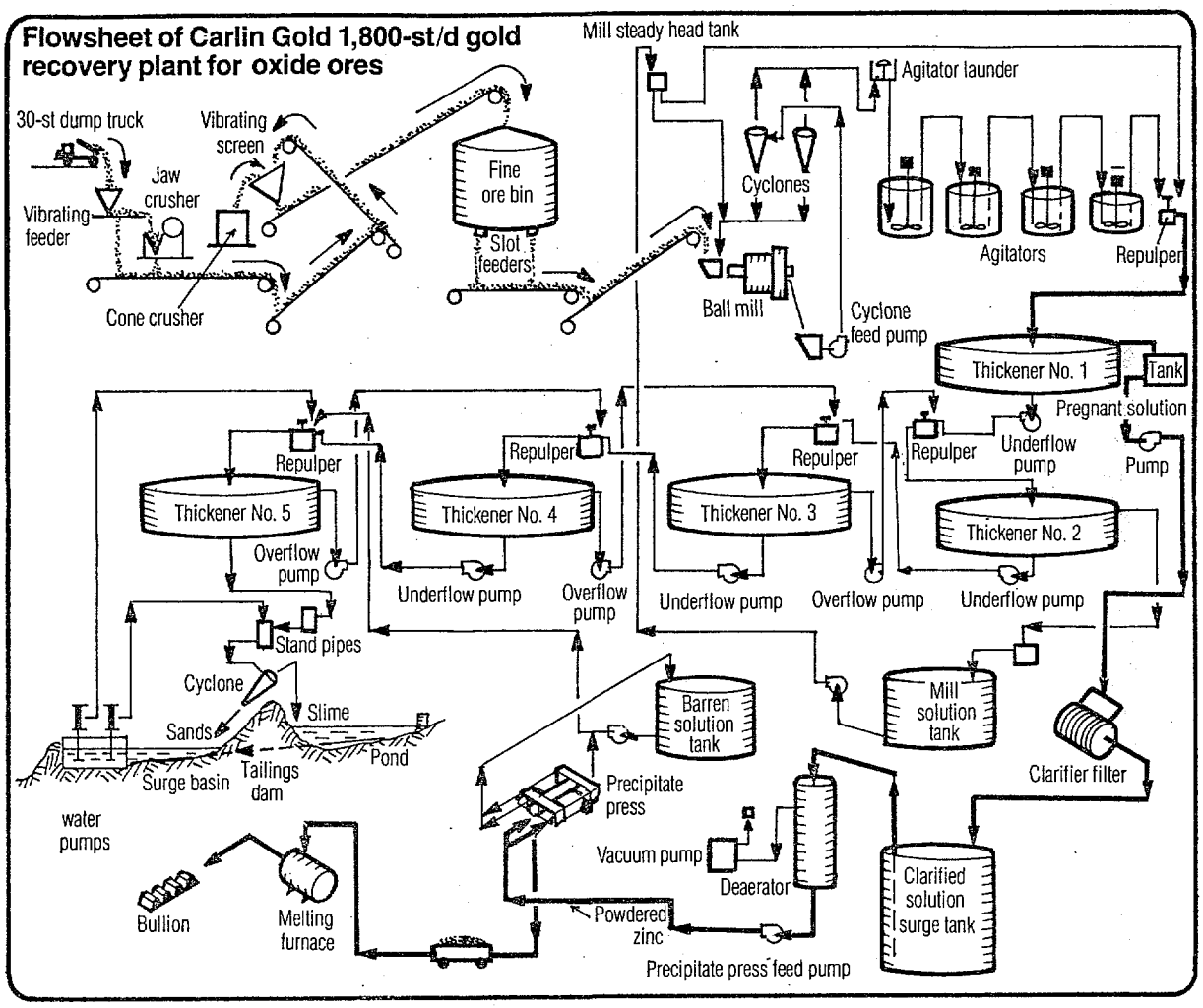
An average stripping ratio of about 9:1 is currently maintained at Carlin's open-pit mines, but the original reserve had a stripping ratio of 3.2:1. During periods of very high gold prices, the ratio, due to expansion of the Carlin pit on down-dip ore, has been as high as 18:1.

Major mining equipment required at the Carlin mine for drilling and blasting includes a D40K Driltech blasthole drill, four T-750 Chicago Pneumatic drills, and two Amerind-McKissick prill trucks. For loading, a 5-yd^3 P&H diesel-electric shovel, two 12-yd^3 Caterpillar 992C loaders, 11- and 6-yd^3 International Harvester loaders, and 6- and 7-yd^3 Caterpillar 988B loaders are available. For hauling, 16 Wabco haul trucks—15 75-ton units and one 50-ton unit—and three 85-ton Euclids are on hand. Auxiliary equipment includes two Caterpillar 14E graders, three Caterpillar D8 dozers, a Caterpillar rubber-tired dozer, two water trucks, and tire, fuel, and lube trucks. Maggie Creek and Blue Star pits are mined with part of the above equipment.

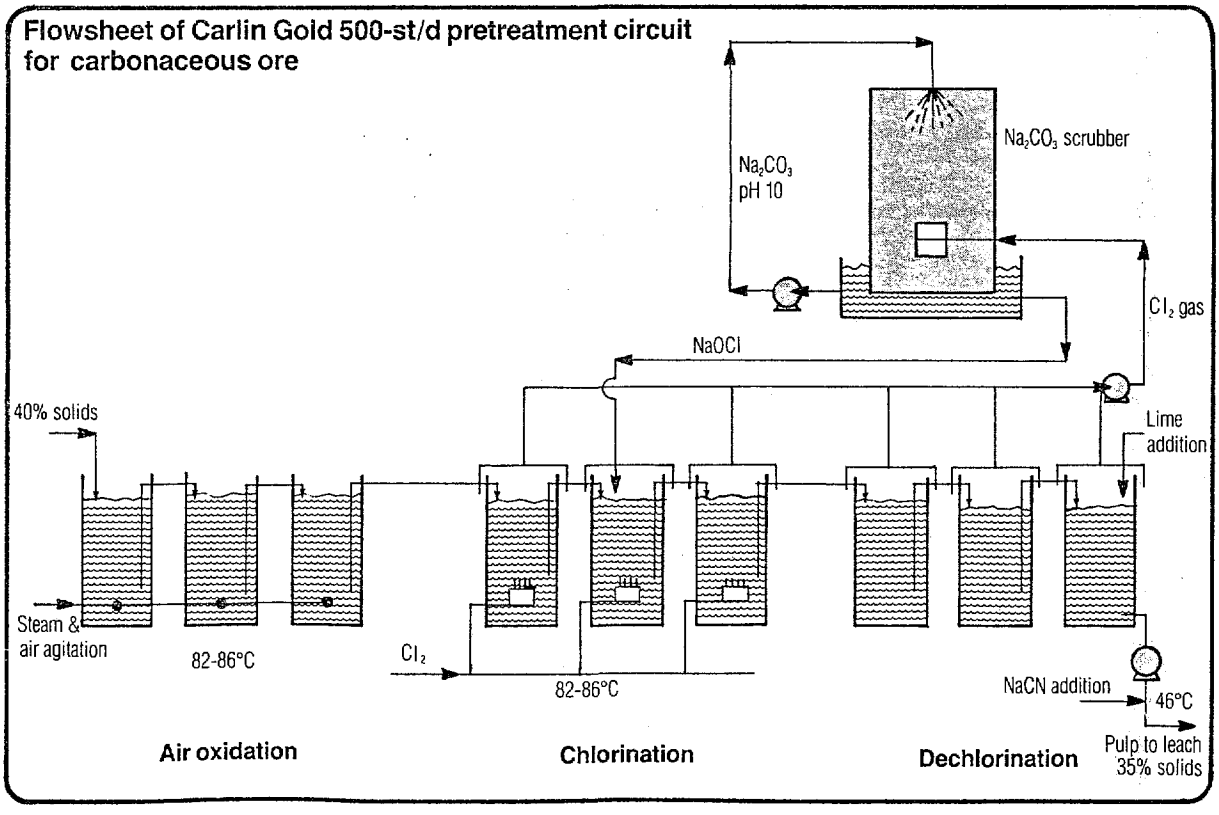
For the most part, mining is carried out by drilling and blasting 20-ft benches, but 10-ft benches are not uncommon at Blue Star. The main reasons for using 20-ft benches, where possible, are to control ore grades, to minimize dilution, and to separate carbonaceous and oxide ores on the benches.

All drills are truck-mounted for mobility, operated by one man, and capable of drilling the required 25-ft depth—20 ft of bench height plus 5 ft of subgrade—without a steel change. Composite samples of each 20-ft drill hole are taken for assay purposes. Holes are drilled on 14-ft centers, but this varies according to rock hardness, which ranges from hard limestone and cherts to soft shales and siltstones. The average footage drilled per machine-shift is 500 ft.

Flowsheet of Carlin Gold 1,800-st/d gold recovery plant for oxide ores

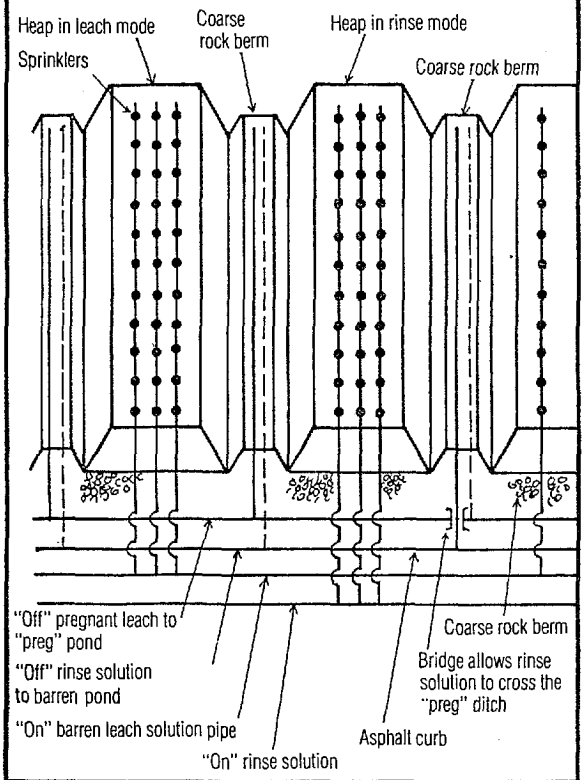


Flowsheet of Carlin Gold 500-st/d pretreatment circuit for carbonaceous ore

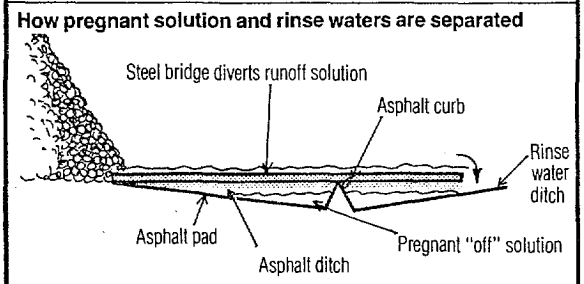


Details of Maggie Creek heap leach arrangement

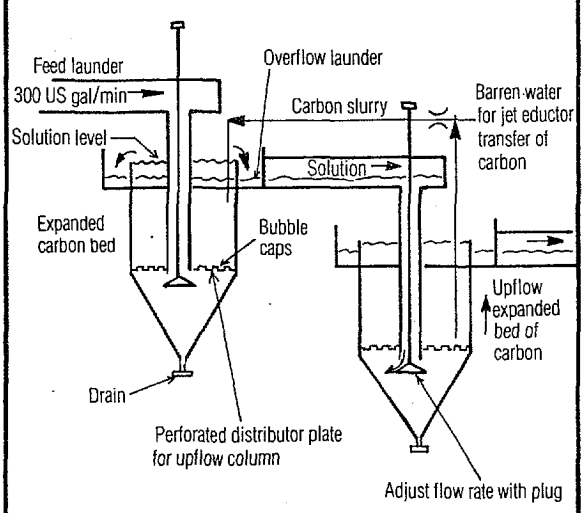
Idealized solution distribution and recovery



How pregnant solution and rinse waters are separated



Schematic diagram of carbon adsorption columns



Tri-cone carbide-button and milltooth bits are used to drill blastholes. Bit sizes vary from 7⅞ to 6½ in. "Most bits," says McBeth, "are purchased as re-run or re-tip bits, which are cheaper than new bits." McBeth says that re-run bits cost one-tenth that of a new one and will drill an average of 1,100 ft of hole. A new bit would have to average 11,000 ft to be equal to the cost of a re-run bit, which McBeth says has not been attained.

Blasthole patterns are loaded with ANFO by a two-man crew using a prill truck. Initiation in each hole is achieved by using one 2 x 8-in., 65%-gelatin dynamite stick tied in with E-Cord trunk liner and cross ties. Detonation is by two caps and a fuse connected just prior to blasting. To minimize displacement of blasted material, 17-millisecond surface delays are used between rows. All holes are bottom-primed. The powder factor averages 0.31 lb of explosive per ton of blasted material. Wet holes are loaded with ANFO placed in 5-in.-dia, waterproof bags weighing 30 lb. Reinforced Primacord is used down the hole with flat, 1-lb cast primers placed on the cord to make contact with each bag of ANFO.

After blasting, the original hole locations are restaked by the pit survey crew for ore control purposes. Each hole in ore zones is tagged with its assay value and flagged according to a color code. Ore loading and hauling is carried out during daylight hours only to improve sorting the various ore grades and types. The front-end loader operator, using the color coding system and operating under the direction of the mine foreman, sorts ores and directs haulage units to the various stockpiles using a horn code.

Atomic absorption assays of blasthole samples are the norm but areas known or suspected to contain carbon ore undergo both atomic absorption and fire assays. If the two types of assay do not correlate reasonably well, the ores in these areas are considered carbonaceous. The carbonaceous ores do not respond well to cyanidation without pretreatment by means of oxidation and chlorination. Thus great care is required to segregate the two ore types during loading since carbonaceous pretreatment capacity is limited to 500 st/d.

It is critical to overall gold recovery that carbonaceous ores remain undiluted and unbulked in quantity by oxide ores. It is just as important to avoid mixing carbonaceous ore into the oxide plant feed which by-passes pretreatment because gold recovery from the carbonaceous fraction will suffer.

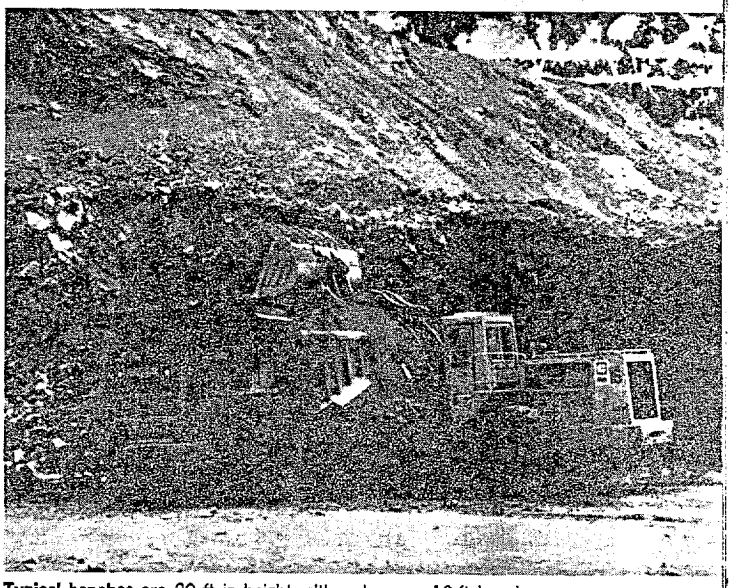
MILLING THE ORES

Since mill start-up in 1965, few process changes have been necessary, primarily because the gold is finely divided—ranging in size from 3 to 5 microns—and is void of cyanicides. Such characteristics provide for good cyanidation of oxidized ore, and gold recoveries from the 1,800-st/d oxide circuit have been good at 90%, says McBeth. Carbonaceous ores were not encountered in quantity until about 1972. These ores required pretreatment in order to gain acceptable gold recovery.

As noted, Carlin installed a new oxidation process ahead of the initial chlorination pretreatment process. This process now converts the amenable pyrite to iron oxide and oxidizes some of the carbonaceous material by first agitating the slurry with air at about 90°C. This is followed by chlorination to complete oxidation of the two mineral species of carbon and iron.

Carbonaceous ore, reduced to ¾ in., reports to an 8 x 7-ft, grate-discharge ball mill and is ground in fresh water

Ore grade control is important; thus, survey crews relocate and flag drill hole locations with appropriate data after bench blasts.



Typical benches are 20 ft in height, although some 10-ft benches are used. Excavators are sized at 5-11 yd^3 and trucks at 50-85 st.

to 65% minus 200 mesh in a closed circuit with spiral and cyclone classification. Pulp at about 45% density and 90°F is then pumped to the agitation circuits, consisting of four insulated, rubber-lined tanks equipped with mechanical agitators.

Vaporized chlorine, purchased in 17-ton tanks, is fed to the first set of agitators through a system of iron and PVC pipes. Each of the tanks is equipped with four 1-in., open-ended PVC pipes extending almost to the bottom to distribute chlorine gas to the agitated pulp. The remaining tanks are used to digest hypochlorite before the pulp enters the main circuit for cyanidation.

Feed rate to the chlorination circuit is controlled by monitoring the amount of excess hypochlorite in the pulp. An excess of 0.03-0.5%, determined by titration, indicates that all carbon that will react has been oxidized, and the ore is ready for cyanidation. Carbonaceous ore is oxidized at a pH of 6 and then raised to 11.5 pH with lime at the end of the chlorination process, prior to entering the oxide ore circuit. Without treatment, the carbonaceous ore would yield nil to 20% of the contained gold. Once oxidized, the recovery rate averages about 83%.

From the beginning, Carlin selected a simple, straightforward cyanidation process, including (in flow sequence): coarse grinding, ground-pulp leaching, liquid-solid separation by countercurrent decantation, clarification and deaeration of pregnant solutions, precipitation of gold values by zinc dust, pressure filtration for recovery of precipitated values, and furnacing precipitate for bullion production.

The mill has 169 control, alarm, and recording stations. Such instrumentation permits a favorable ratio of tonnage treated to mill operators. About 145 st of ore per manshift are crushed, milled, and sent to tails.

Reagent consumption in pounds per short ton is as follows: sodium cyanide, 0.33; zinc dust, 0.15; lime, 8.50; filter aid 0.66; flocculant 0.03; lime deposit inhibitor, 0.07; and soda ash, 0.50.

LEACHING OPERATIONS

Maggie Creek produces about 500,000 st/yr (2,500 st/d) of heap-leach ore along with 1,150 st/d of mill-grade ore, which is trucked to and processed at the Carlin mill. In round figures, capital cost of this operation was about $10 million, including crushing plant, ponds, carbon column recovery plant, trucks, drills, loaders, leach pad, and equipment repair shop. Bechtel Corp. was responsible for design and construction of the project.

A portable crushing plant, designed by E. F. March Engineering, St. Louis, Mo., was assembled by Morgan Equipment Co., San Francisco, Calif. It is complete with a Cedar Rapids jaw crusher, Simplicity double-deck vibrating screen, and Symons Standard cone crusher.

In the beginning, leaching was carried out without agglomerating the fine ore. Shortly after start-up, however, the necessary equipment was added to agglomerate fines by adding Portland cement and water.

The asphalt leaching pad is 7 in. thick and slopes at about 4% grade. The pad is designed to accommodate five heaps side by side. Each heap measures about 280 x 160 ft and contains about 16,000 st of ore at a height of 10 ft. Three of the heaps are leached simultaneously, while the others are being built or removed. A berm of crushed coarse rock is constructed along the downhill edge of the heap. Separate drain curbs are built into the pad, so that rinse solutions can be separated from leach solutions along the main drainage ditch, which extends along the ends of all the heaps.

Leaching solution, at 45-60 $lb/in.^2$, is applied to three rows of sprinklers at a rate of 0.24 $gal/hr/ft^2$ to each 16,000-st heap. Each row is equipped with five equally spaced sprinklers. Heaps also contain rows of sprinklers (five sprinklers per row) on each heap edge. Solution is applied at 10.5 pH with 0.75 lb of NaCN/st of ore. Off solution contains 0.3 lb of NaCN/st and a pH of 9.

Pregnant solution flows by gravity to the pregnant solution pond and on to the 300-400-gal/min carbon-column gold recovery plant, consisting of five stages of upflow fluid-bed carbon contactors in series. Adsorption vessels are 6 x 8 ft and contain overflow launders.

Each tank requires 1,800 lb of carbon. Carbon is transferred as a slurry by a water eductor located above the level of the adsorption tanks. Loaded carbon is shipped to the Carlin mill, where the gold is recovered by electrowinning on steel wool cathodes.

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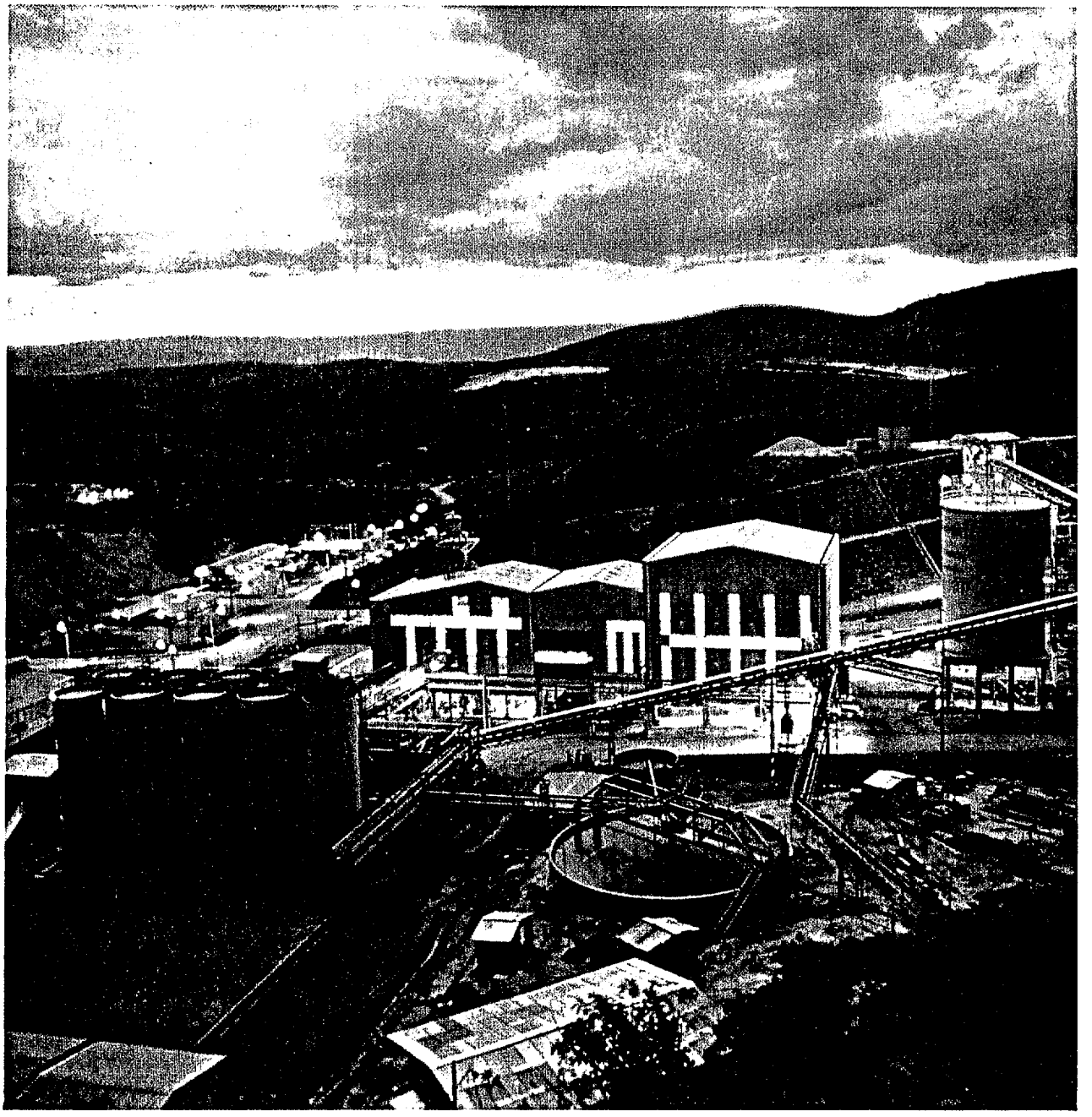
SKILLINGS' Mining Review

PUBLISHED EVERY WEEK



View of reduction plant at Mineracao Morro Velho S.A.'s new Jacobina gold mine in state of Bahia, Brazil.

5th WORLD GOLD CONFERENCE

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5th WORLD GOLD CONFERENCE

Total of 270 participants attend program held on June 22 and 23 in Lugano, Switzerland

by DAVID N. SKILLINGS Jr.

ON WEDNESDAY AND Thursday, June 22 and 23, about 270 participants from 37 countries convened at the Lugano Convention Center in Lugano, Switzerland, to attend the World Gold Conference sponsored by The Financial Times in association with The Banker. This was the fifth in the series of these meetings, which previously were held in Montreux, Switzerland, in 1982 and 1979 and prior to that in London. General theme of the conference was "The Outlook for Gold and Silver," marking the first time that silver was included in the program.

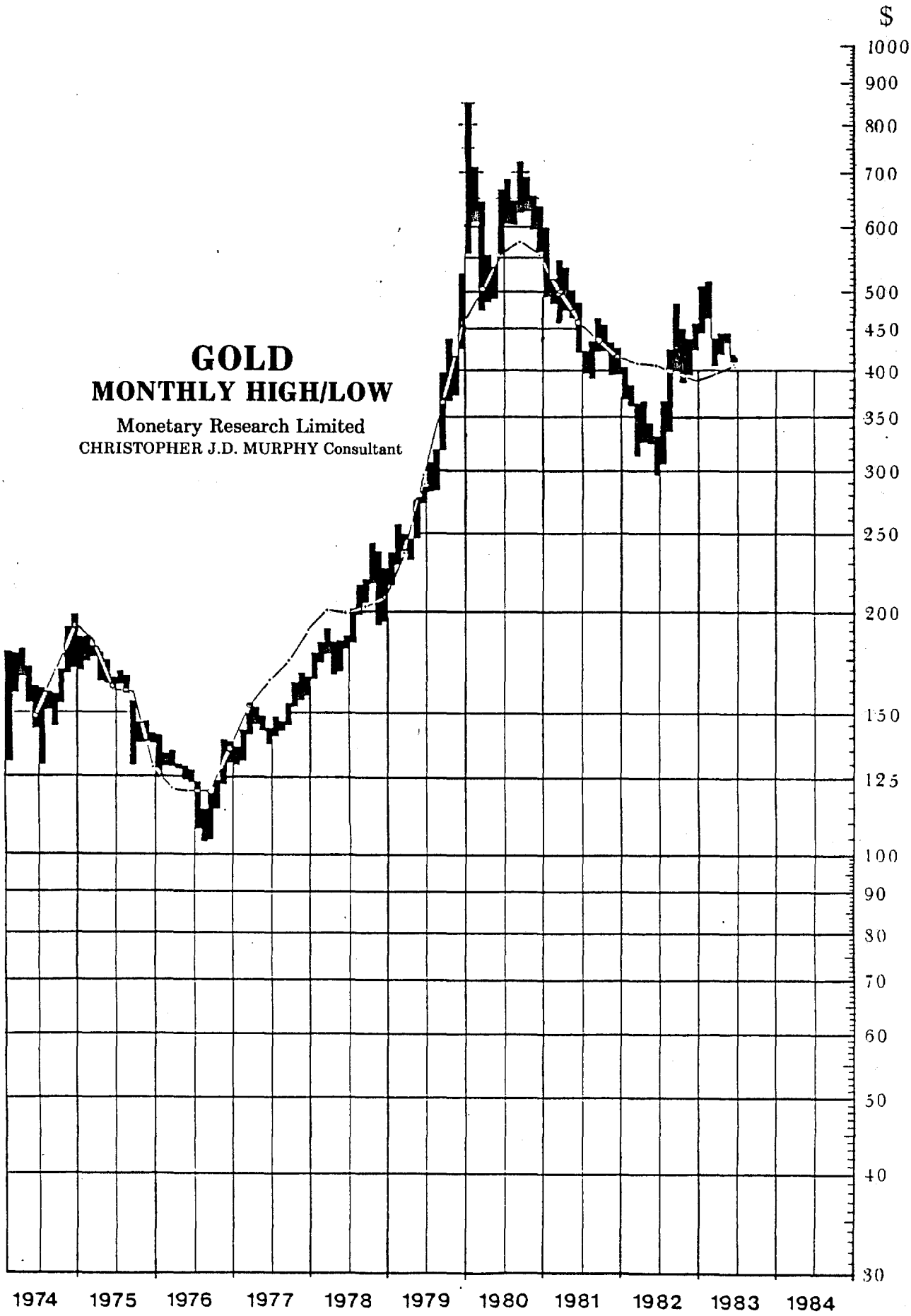
On June 22, the World Gold Conference was opened by Robert Guy, director of N.M. Rothschild & Sons Ltd., who served as chairman of the initial session. In his introductory remarks, Mr. Guy set the scene in the marketplace for gold and silver and reviewed recent price movements as well as discussed the future role of gold as a reserve asset. Improvement in price is anticipated to be the result of a general economic recovery rather than revival of inflationary expectations. It is too early to say whether continuation of the high U.S. budget deficit will be accompanied by a pre-election reflationary economic program and will arrest declining expectations, but reappointment of the chairman of the Federal Reserve gives encouragement to those looking for a controlled recovery. Signs of recovery lead some industrial consumers of silver to anticipate an increase in their consumption this year of 5%, but they demonstrate no willingness to chase the market in upward flurries that recently saw the price close to the $15 level before its reaction to $11. Justification for this approach was underlined by the higher price levels attracting significant dishoarding and encouraged forward sales by several mining companies.

Mr. Guy emphasized that the gold swap technique has become accepted as a means of creating short term liquidity for individual countries. During the 1970s, swaps as a means of mobilizing gold reserves were used by Italy, Portugal and South Africa. The South African Reserve Bank has renewed its activity, and a loan of $200 million guaranteed by Yugoslavia's gold reserves was reported in March, with a gold swap of $400 million with Portugal reported in April. Other countries within the Comecon group and from Latin America also have undertaken such transactions although not necessarily via central banking institutions but with commercial banks involved. In themselves, swaps are a sign of weakness. In addition, central banks prefer to swap rather than to sell because they do not like the idea of a permanent reduction in their gold reserves, but if enough of them did sell a significant quantity the effect on the price would be dramatic. Gold remains a sensitive issue for central banks, and statistics produced on official transactions can be confusing. Some transactions are never reported, and others may fail to distinguish between sales and swaps.

The Outlook for Gold

In his address titled "The Outlook for Gold," Dr. Chris Stals, senior deputy governor of the South African Reserve Bank, stated that the supply/demand analysis of the gold market indicates a fairly delicate balance between supply and demand at the existing price level. Looking at prospects for gold and possible changes in the supply and demand components of the formula, the present price becomes suspiciously low, especially with a longer term view. Gold supply is fairly inelastic and stable, but demand is changeable and sensitive, susceptible to many volatile and uncertain macro-economic variables. Since the price of gold was released from the shackles of Bretton Woods late in the 1960s and gold markets were liberalized around the globe, the gold price moved through two distinct typical business cycle-like phases. The important question that must now be answered is whether the upturn in the price that commenced in July/August of last year is the beginning of a new cyclical upswing or whether it results from temporary influences only and might, therefore, be reversed again soon.

(Continued on page 8)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5th WORLD GOLD CONFERENCE

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Dr. Stals remarked that gold is not just another commodity and its price behavior is affected by many other considerations that normally would not have the same effect on the prices of other commodities. Prospects for economic developments in the major industrial countries over the next year enhance the bullish outlook for gold. Should present recoveries in the U.S. and United Kingdom gain momentum and also pull economies of other industrial countries of the west in their wake, demand for gold naturally would also rise. What is even more important, such recovery would almost unavoidably be accompanied by various monetary developments that would all be conducive to a rise in the gold price. Increases in the world money supply, declines in interest rates, a turnaround in inflationary trends, and a decline in the value of the U.S. dollar could each on its own and all moving together in a consistent economic recovery program, exert upward pressure on the gold price. The astute investor may follow the conservative approach to buy now in a relatively quiet market so as not to miss the opportunity of acquiring gold at a fairly low price.

The Official Use of Gold

Rene Larre, adviser to Schneider S.A. and former general manager of the Bank for International Settlements, reviewed "The Official Use of Gold—An Historical Assessment and Forecast for the Future," in which he noted that central banks hold in their vaults nearly one-half of the total world inventories and influence gold prices through their own transactions. During certain periods, they are net buyers and at other times they are net sellers, but in both cases the balance of their operations tips the gold price one way or the other. In 1982, for example, central banks reduced their gold holdings by 72 tons after an increase of 65 tons in the previous year. The turnaround was 137 tons, representing 10% of gold utilization for last year. The role of monetary gold has been reduced to a considerable degree by the Jamaica compromise in Jan. 1975. This results, on the one hand, from suspension of convertibility of the dollar, and on the other hand from abolition of any fixed price for gold. The abolition of dollar convertibility into gold has been the most important feature of the new system, as it prompted the absorption of all western countries into the dollar area.

DATE: June 20, 1983 Profile: $400 per ounce
James Capel & Co.—GOLDVAL

Mine Name	Present Value: 15% 350	400	450	Volatility factor	'right' R of R	'right' price	latest price	% change
East Daggas	5.27	5.99	6.76	1.13	6.8	13.81	5.50	151.1
GOLD						400.00	409.00	-2.2
Deelkraal	2.46	2.95	3.23	1.15	7.0	4.47	4.60	-2.8
E.R.G.O.	4.01	4.89	5.64	1.19	7.5	8.32	8.88	-6.3
Randfontein	60.03	76.12	90.22	1.23	8.0	123.79	146.00	-15.2
F.S. Geduld	18.84	24.72	27.99	1.23	8.0	38.68	48.75	-20.7
Western Deep	24.30	28.99	33.27	1.17	7.3	47.63	60.75	-21.6
Western Holdings	20.78	26.66	28.91	1.19	7.5	41.26	54.00	-23.6
Harmony	9.91	12.99	15.80	1.26	8.5	18.34	24.00	-23.6
Stilfontein	8.95	10.61	12.47	1.18	7.4	15.30	20.25	-24.5
Welkom	5.41	6.94	7.53	1.19	7.5	10.74	14.50	-26.0
Durban Deep	15.97	19.69	27.07	1.31	9.1	26.49	36.00	26.4
St. Helena	17.15	20.77	24.67	1.20	7.7	31.80	43.50	-26.9
Venterspost	8.44	11.46	13.34	1.26	8.5	14.24	19.75	-27.9
Driefontein	11.95	13.72	15.44	1.14	6.9	24.39	35.25	-30.8
Southvaal	19.66	23.29	25.06	1.13	6.8	42.85	63.25	-32.3
Pres. Brand	17.76	21.64	25.55	1.20	7.7	32.54	49.00	-33.6
Blyvoor	7.49	8.90	10.11	1.16	7.2	11.74	18.00	-34.8
Doornfontein	11.91	14.28	16.63	1.18	7.5	20.82	32.00	-34.9
Elsburg	1.72	1.96	2.24	1.14	7.0	2.83	4.40	-35.6
$ Ft Gold Mines						644.3	1003.6	-35.8
E.R.P.M.	7.28	9.13	11.45	1.25	8.4	12.19	19.00	-35.8
Vaal Reefs	39.62	48.48	59.40	1.22	8.0	75.09	118.00	-36.4
Western Areas	2.65	3.02	3.45	1.14	7.0	4.37	7.20	-39.3
Hartebeest	29.00	34.87	40.40	1.18	7.4	51.96	86.00	-39.6
Bracken	1.34	1.85	2.04	1.25	8.3	2.20	3.80	-42.0
Libanon	13.77	17.05	19.95	1.20	7.7	24.09	42.25	-43.0
Zandpan	4.92	5.91	6.85	1.18	7.4	8.81	15.50	-43.1
Winkelhaak	12.65	15.96	18.27	1.20	7.7	24.31	43.25	-43.8
Kloof	14.45	18.75	21.50	1.22	8.0	27.80	50.50	-45.0
Elandsrand	3.71	4.46	5.13	1.18	7.4	7.50	13.75	-45.4
Pres. Steyn	16.97	20.71	23.41	1.18	7.4	30.60	56.50	-45.8
Loraine	2.14	2.95	4.18	1.40	10.4	3.86	7.30	-47.2
Buffels	16.68	22.46	26.10	1.26	8.4	32.03	64.50	-50.3
Unisel	4.35	5.45	6.27	1.20	7.7	8.52	17.25	-50.6
Kinross	5.91	7.87	9.58	1.28	8.7	12.86	26.75	-51.9
Grootvlei	5.05	6.27	7.30	1.20	7.7	8.50	19.75	-56.9
Leslie	0.94	1.32	1.70	1.35	9.7	1.60	4.20	-61.9

Mr. Larre pointed out that abolition of the fixed price for gold is more technical. The new rule is that monetary authorities are free to deal in gold, but they cannot establish a fixed price or define the value of their currencies as a weight of gold. Purchases and sales of gold have to be made at market related prices. In practice, central banks are severely impaired by the absence of any fixed price for gold. Given the market volatility and size of the price fluctuations, it has become hazardous for central banks to deal in gold, as the wisdom of any decision can be denied the day after by a new move of the market. These considerations have discouraged outright operations of monetary gold, such as sales or purchases, so that national gold reserves are standing still, showing only small changes from one year to the next. The only exceptions occur in time of crisis when governments have to mobilize their gold in a hurry to finance a balance of payments emergency. This seems to have been the case in 1982 when Latin American countries sold a total of 82 tons and an eastern country 32 tons.

Gold Trading

Paul Zubler, senior vice president of Union Bank of Switzerland, discussed "Gold Trading—A View from the Swiss Market," in which he commented that Swiss banks were the first to transact gold swaps. Granting of credits on the basis of swaps can also represent an alternative to the collateral loan. During times of increased supply pressure, this instrument can be of valuable support to the market. Gold lending and borrowing, which have flourished in recent years, offer lenders, generally central banks, the possibility of obtaining a small return on an otherwise non-income bearing physical gold holding. It serves the borrower as a vehicle for interest arbitrage operations and/or as a source of liquidity. Physical gold has not only managed to avoid losing its significance but has even grown in importance. A key reason is that gold, in the wake of the mobilization of last reserves by several central banks, despite its official banishment from the monetary system, has again moved into the spotlight. The impressive role played by gold, especially in times of crisis, has again been evident.

Mr. Zubler explained that the question marks increasingly arising with the international financial system, the worldwide political and military tension as well as the underlying doubts of an ultimate victory over inflation, are several important reasons why physical gold will continue to assume primary significance even in the future. This will not only apply to traditional gold areas of the world such as the Middle East, India and Southeast Asia but also to an increasing extent to Japan and the U.S. The explosive growth in demand for bullion coins and gold jewelry in all forms are solid evidence of this. This trend is expected to continue in coming years. Among other things, the supply of physical gold will hardly keep pace with development of purchasing power in nominal terms so that the degree of scarcity of physical gold is likely to increase still further in the future. As a result, Switzerland as a gold trading center will display an even higher commitment in the physical gold trading sector. The final talk of the opening session was "Gold and the Banking System," by John Forsyth, director of Morgan Grenfell & Co. Ltd.

U.S. After Great Gold Debate

On June 22, the afternoon session of the World Gold Conference was commenced with an address "The U.S. After the Great Gold Debate," delivered by Thomas W. Wolfe, president of Wolfe/Wire Inc., and he reported that the fixed exchange rate-quasi gold standard cause still simmers on the back burner but has no high level advocacy in Congress. The U.S. is now in general accord with the other powers on a passive policy toward gold, with no significant change foreseen over the next year or two. A development that bears watching is the probable issue of a U.S. gold bullion coin by 1985 after the medallions have run their course. The key point in the proposal being discussed is that gold for a new coin is not intended to come from official reserves, at least in major part. Legislation drafted for the coin puts a permanent cap on the quantity of gold that may be drawn from existing official stocks. By far the most significant change in American gold policy is a vastly different attitude toward the gold price that has occurred in the past year. Although the U.S. was the largest official holder of gold, Treasury officials felt better when the price was going down than when it was going up.

Mr. Wolfe said that all this has changed. Early in June, Undersecretary of the Treasury Beryl W. Sprinkel in opposing future IMF gold sales expressed concern over a possible depressing effect on the gold price, which he noted could cause large losses in the value of gold reserves of the U.S. and other countries. No Treasury official in memory has ever hinted at such concern. If the implication of a drop in the gold price means an official loss, then a rise must mean an official profit. U.S. policy has not gone this far, but there is now a conscious awareness of a relationship between the price of gold and the official reserve position. In the U.S. view, a gold price rise is not without some benefit, and that is a new factor in the international monetary scene. The prospect for stable economic growth at least for the next year or two looks better than it has in a decade. In this setting, precious metals will do moderately well, with real prices fairly steady. Vigorous economic expansion would mean substantially higher gold prices in 1983-84.

Mr. Wolfe's talk was followed by a paper titled "Speculation and the Silver Market," by Dr. Henry G. Jarecki, chairman of Mocatta Metals Corp., who noted that the GSA holds 120 million ounces of silver and also said that options, which are a right to buy but not an obligation, are growing substantially in importance. Next speaker was P.J.K. Smith, director of Johnson Matthey Bankers Ltd., whose subject was "The Outlook for the Silver Market."

Industrial Market for Silver

The afternoon session on June 22 was completed with a discussion of "The Industrial Market for Silver,"

(Continued on page 10)

(Continued from page 9)

presented by Ulrich Kunze, director of Degussa AG, and he pointed out that for 10 years the industrial use of silver has reflected a downward tendency. Although basic statistics show a slight increase in 1981 and 1982, silver consumption since 1973 has declined by 28% from 517 million troy ounces to 371 million troy ounces. Speculative price development from 1979 to 1981 contributed significantly to decreasing consumption. Research programs conducted to reduce consumption of silver became an absolute necessity because of the prices at that time. The process of substitution set in action as a result of these research programs is irreversible for cost reasons. The worldwide recession also has been reflected in consumption of silver. The photography industry continues as quantitatively the principal area for silver application. Although in the U.S. and Britain, demand decreased considerably between 1979 and 1981, the photography sector has proven to be extremely resistant to external forces.

Mr. Kunze commented that precious metals because of speculative influences are subject to wide price fluctuations. In the past, therefore, research and development were concentrated to an increasing degree on significantly reducing contents of precious metals in switching devices and plugs. For example, new precious metal alloys have a lower precious metal content with a higher resistance to abrasion through better processing methods that limit the use of the precious metal to the technically necessary volume, or through new switching equipment and plug designs. Better methods to recover precious metals from punching wastes or from contact pieces that have served their use contribute to saving precious metals. During recent years, the jewelry sector has characterized the downward quantitative development the most. Unmistakable is the trend toward innovative, technological development in all sectors, consumer goods, machine building, transportation and construction industries, with silver being saved through more efficient production processes.

Structural Changes in Gold Market

On the morning of June 23, Hubert Baschnagel, central manager and member of the executive board of Swiss Bank Corp., served as chairman of the World Gold Conference. The opening speaker was Meinhard Carstensen, general manager for Dresdner Bank AG, whose topic was "Structural Changes in the Gold Market." He said that a real revolution, which is not yet terminated, was introduction and development of the gold futures market, with Comex and IMM still setting the pace. The futures market reaches deeply into the banking system. Through the mechanism of futures, switches of funds are being made from the banking market into commodity markets and vice versa. Comex and IMM brought an entirely new dimension into gold trading. The latest report of the Bank for International Settlements quantifies worldwide gold mining production excluding U.S.S.R. and other Comecon countries as well as China and North Korea at just over 1000 tons in 1982. The attraction of the futures market is that it offers a cheap and highly efficient way of getting into gold. On margins of a few thousand dollars, futures contracts offer the chance to get a claim for little outlay.

Mr. Carstensen noted that only less than 1% of the total volume is tendered and accepted for delivery, with this figure amounting in 1982 to about 350 tons representing one-third of the entire world production of gold or one-quarter of annual supply. Currently the knowledge of chart technique, together with the market incentives, which are influenced by daily political events and the short term trading strategy of many professionals dictate largely the pricing of gold. Chart technique, computer dealing, a profound knowledge of the shorts and longs in the market and the ability to foresee their next reactions are the tools to develop a trading strategy, which must be flexible enough to be overthrown within the next few minutes. It seems to be advisable to return to more conservative behavior in trading gold. This could give the market more price stability together with confidence. If market makers could again develop medium to long term trading strategies as well as for their investment advisory, the market would have a fairly good chance to eliminate short dated erratic price fluctuations in the future.

In "Markets in the Far East," Timothy S. Green, consultant to Consolidated Gold Fields on the Middle East, Far East and South American markets, said that Sumitomo's recently discovered mine in Japan has reserves containing an estimated 120 tons of gold. He also remarked that the base price of gold is presently between $400 and $500 per ounce, being cheap below that level and expensive above. Speaking next was T.M. Othman, deputy general manager for Al-Saudi Banque, who mentioned that the gold at Fort Knox included 400-oz. bars of 995 fineness as well as melted down gold. Gold still serves as a reserve for money supplies and is a reserve against money losing value through devaluation or inflation. Generally the gold price tends to be higher in unstable parts of the world. He agreed with the previous speaker that gold is dishoarded above $500 and in demand below $400, with a current range between $420 and $430. The private market for gold has grown widely from demand for jewelry, coins and industrial uses, with central banks absorbing a loss in reserves.

Strategy of Major U.S. Bank

Following a talk "Gold's Future and the Retail Product Revolution," by Dennis Suskind, partner in charge of precious metals of J. Aron & Co./Goldman Sachs & Co., the morning session on June 23 was completed with remarks by Ronald G. Layard-Liesching, vice president-foreign exchange analysis of The Chase Manhattan Bank N.A., on "Gold—The Strategy of a Major American Bank." He suggested that a return to a fixed exchange rate system, as outlined in the Bretton Woods agreements, is totally impossible, except temporarily if a world financial collapse were experienced. The reason is that the private sector can marshal more resources than governments in

foreign exchange. In practical terms, the dollar is totally dominant in the world monetary system. Not only is 80% of world trade and finance denominated in dollars but, more importantly, there are 10 markets for dealing in quantities of $20 million or higher. Only the yen and gold could provide a real alternative to the dollar as the basis for the world monetary system.

Mr. Layard indicated that international banks have many particular characteristics that are advantageous in gold dealing. Major features include size and diversity of a bank's customer base that enables them to borrow currencies and bullion at advantageous rates; their breadth of customer relationships that enable them to deal with all participants in the gold market; treasury operations enabling them to integrate the financial cover for any gold transaction in any major area worldwide; and unmatchable international clearing facilities. Gold is developing as an international monetary medium, and if this is the case ultimately all major banks will be forced to deal in gold to some extent. The question is: How should one enter this market as an international bank? Chase Manhattan is approaching it as an adjunct to its treasury operations and via its commodities division. As with currencies, it borrows and lends, buys and sells, arbitrages and gaps, the gold cash, forward and futures markets. At this stage, Chase Manhattan has not emphasized retail gold, which is expensive for customers and expensive to run.

On the afternoon of June 23, the concluding session of the World Gold Conference was commenced with an address "Gold Shares in the Medium Term," delivered by Charles Smedley, mining partner of James Capel & Co. He explained that the South African gold mining companies, which pay out revenue in dividends rather than engage in exploration and other developments, rank as the world's largest as well as lowest cost producers. He also suggested that $400 per ounce is the new base price for gold and also that it will keep up with inflation. Following this talk was a paper "Inter-Relationships of Bullion and Futures Markets," by Robert M. Rubin, managing director of Drexel Burnham Lambert.

Multiple Reserve Currency System

Paul Bareau, economic adviser to the Mirror group of newspapers, discussed "Gold in a Multiple Reserve Currency System." He stressed that return to a gold standard is off, at least for the foreseeable future, but gold will have an important role to play as an adjunct and major element in the monetary system toward which the world is moving. At the latest count, world reserves expressed in U.S. dollar equivalent amounted to $796 billion, of which gold at the price of around $440 per ounce represented 53.5%, foreign currencies made up 40.1%, reserve positions in the IMF 4%, and 2.4% for special drawing rights. The amounts held in reserve currencies of $319 billion included 71% in U.S. dollars, 13% in German marks, 4% in Japanese yen, 3% in Swiss francs, and only 2% in sterling. During recent years, the composition of the total of reserves kept in foreign currencies has diversified. The mark and yen have come up in the list. The European monetary system unit, ECU, itself made up of member currencies, has become more important. What we are seeing is a gradual emergence of a multiple reserve currency system, each reserve currency grouping around it a bloc of national currencies, within which a measure of exchange stability will prevail.

Mr. Bareau stated that in envisaging the evolution and extension of a multiple reserve currency system he foresees a comparable extension of the role of gold. The reentry of gold in the international monetary system would imply some official intervention in the market for the metal. This would be intervention designed to prevent such volatility as occurred in Jan. 1980 when the gold price rose and then fell by $100 on successive working days. It would not be intervention comparable with the expensive attempt by the international gold pool of major central banks in the 1960s to keep down the market price to the then official $35 figure. The case for giving gold a greater role to play in the monetary system of the future is that it is there, the principal element in international reserves; that there is too much of it to get rid of it; that is has served the world as a monetary metal for some millennia; and that it has the intrinsic value and independence on which no paper has ever been able or will be able to claim unless it be freely convertible.

David Marsh of the Paris bureau of The Financial Times presented a summing up of the addresses presented at the World Gold Conference to complete the program, which was then formally closed.

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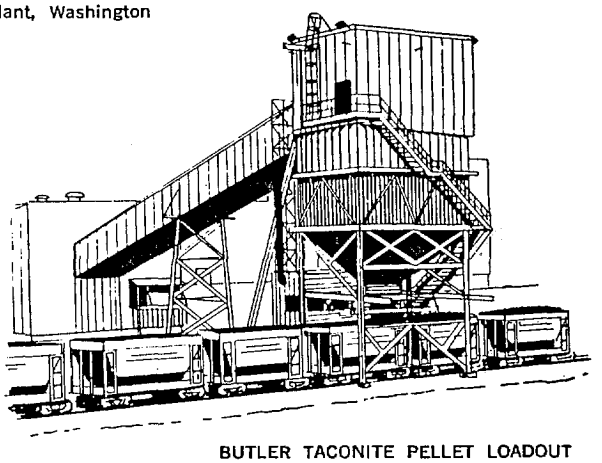

BUTLER TACONITE PELLET LOADOUT

Water-Rock Interaction Conference/Paper #4 1983 ?

WRI-4

REGIONAL METAMORPHIC ORE DEPOSITS - STRUCTURAL AND CHEMICAL CONTROL

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Regional metamorphism and deformation are fundamental processes in crustal evolution, and several important classes of ore deposits are formed as a result of these processes. This paper examines the relationship between deformation, metamorphism, fluid behaviour and the general characteristics of these classes of deposits. In particular, we will concentrate on the mobility of the fluid, the "structural control" of ore distribution and the chemistry of metamorphic fluids as it relates to the scavenging, transport and deposition of valuable metals. To illustrate these principles we shall draw mainly on examples of metamorphic gold deposits and also unconformity-related uranium deposits.

Fluid mobility and metal sources: There is widespread evidence of tensile fracturing (veining) during regional deformation and metamorphism (also in retrograde environments), indicating that pore fluid pressure (P_f) commonly exceeds the minimum principal compressive stress (σ_3) (e.g. Etheridge et al., 1983a). At these high fluid pressures, active microcracking is to be expected, with resultant significantly enhanced permeability. Permeabilities of the order of 10^{-15} to $10^{-18}m^2$ (10^{-3} to 10^{-6} darcy) are not unreasonable under these conditions and the metamorphic fluid will therefore be highly mobile. It may even be convectively unstable and the resultant large scale fluid circulation would provide a very effective means of scavenging elements which are present in low concentrations. The overpressured nature of metamorphic fluid systems requires that fluids are derived from devolatilisation of the metamorphic pile and fluid chemistry would then reflect the bulk composition of the pile. Typical metamorphic fluids are of low salinity, CO_2-bearing and somewhat reduced (except where evaporitic rocks are involved), these properties controlling the scavenging ability of the fluid and influencing deposition from it.

Structural control of transport and deposition: A regionally metamorphising and deforming terrain is expected to have widely varying permeability, and fluid flow may be strongly focussed into high permeability lithological and structural settings. Several mechanisms for enhancing permeability and therefore focussing fluid flow will be discussed. Enhanced permeability is most readily achieved by penetrative fracture, which is initiated at high P_f where σ_3 is minimal. In rock masses undergoing heterogeneous deformation (folding, faulting) stress varies in a systematic manner. Hence, the geometry of plumbing systems may be predictable and represent a useful exploration guide. Structural control is essentially the systematic relationship between rock structures, stress field, and zones of dilation. These points are well illustrated by the geometry of gold and uranium deposits discussed below.

Chemical controls on deposition: Ore deposition within the fluid channelways can take place due to: 1) changes in temperature; 2) changes in pressure; 3) fluid/wall rock interaction; or 4) mixing of migrating fluids with local rock fluids. Temperature gradients in regional metamorphic terrains are too small to provide an effective means of concentrating deposition within a small rock volume, but each of the other three processes may be important. Again, specific examples, both prograde and retrograde are given below.

EXAMPLES

Metamorphic Au-mineralisation: Low grade (e.g. greenschist facies) regional metamorphic belts constitute a major source of gold on a world-wide basis. Australian Archean greenst[illegible] and Palaeozoic slate belts have been particularly productive. Here we focus on an example of the latter: the Castlemaine-Chewton region of Victoria.

Gold-quartz mineralization in this area is hosted by a folded lower Ordovician quartz wacke/slate sequence and is restricted to narrow, north-south trending belts

parallel to major fold axial traces. Auriferous quartz-vein systems have developed in fault- and fold-related dilatant sites generated at low or negative effective pressures ($P_f \geqslant P_{total}$) during regional deformation. Faults with related gold mineralisation are dominantly west-dipping reverse structures. Among the several types of quartz-vein systems recognised, within-fault veins developed by oblique opening along faults predominate. The largest veins have formed where refraction of faults across folded structures has caused the local fault orientation to be inclined to the bulk displacement direction. Adjacent to fault zones quartz-vein systems have been generated in extension fractures related to the modification of the local stress field during fault movement. Saddle structures (more common in thick bedded, arenite rich sequences) and some bedding parallel vein systems have developed during flexural slip and fold growth.

Vein microstructures indicate that vein growth has occurred by repeated hydraulic fracturing and the operation of crack-seal mechanisms over an extended period, some veins involving up to 2×10^3 crack-seal increments per centimetre thickness. Quartz deposition is best explained by transient fluid pressure decreases attending repeated hydraulic fracture in the dilatant zones. On the basis of experimental data for quartz solubility and fluid pressure decreases of less than 100 bars, deposition of one gram of quartz (at 300°C, 2Kb) requires a minimum of 10^4 grams of H_2O. The development of major quartz vein systems must thus involve the throughput of tens of cubic kilometres of fluid and emphasises the significant structural control and focussing of fluid transport.

Vein assemblages involve quartz (≥95% by volume) + ankerite + chlorite + phengitic mica ± albite, with pyrite and arsenopyrite being the major sulphides. Sphalerite, galena, chalcopyrite and pyrrhotite also occur especially associated with high gold values which are commonly developed where vein systems abut carbonaceous, pyritic slates. Wall rock alteration is minor, and restricted to zones at few centimetres wide which exhibit similar assemblages to the veins' lower greenschist facies hosts. Arsenopyrite enriched haloes occur in arenites up to several tens of metres from major vein systems. Sphalerite-galena sulphur isotope fractionation indicates temperatures around 300°C. Calculated $\delta^{18}O$ values for aqueous fluids in equilibrium with vein quartz at 300°C range from +8 to +10‰, whereas $\delta^{13}C$ for fluids (as CO_2) were -1 to -5‰. Both aqueous (3-5wt% NaCl equivalent) and CO_2-rich primary inclusions are present in auriferous quartz and exhibit homogenisation temperatures consistent with trapping in a two-phase fluid regime and 300°C and 2Kb pressure. Mica-albite equilibria and fluid salinities imply weakly acid fluids under these conditions.

The evolution of the auriferous quartz-vein systems during the regional deformation history (predominantly late in fold growth and cleavage development), the lack of extensive wall-rock alteration, the depositional conditions, as well as the isotopic and chemical composition of the fluids all attest to the formation of mineralization from regional metamorphic fluids. Gold precipitation in the vicinity of pyritic and carbonaceous slates may be ascribed to mixing of locally derived methane-bearing fluids with more oxidised fluids traversing faults and other dilational zones. Associated base-metal sulphide and arsenopyrite deposition reflect increased aH_2S from the mixing of these fluids. Consideration of the total gold production from the region and gold content of the Ordovician sequence and probable underlying rocks (Glasson and Keays, 1978) indicates source regions of the order of $100km^3$. The large volumes of both the fluids and source regions imply deep circulation of the fluids which were ultimately derived from devolatilisation of the metamorphic pile.

The formation of auriferous quartz-vein systems of widely varying geometry in the Castlemaine region reflects deformation under conditions of very high fluid pressure within a stress field substantially locally modified by developing folds and faults. The restriction of auriferous veins to narrow belts within the lower Ordovician succession reflects the temporal and spatial sequence of reverse fault development, the ability of these fluids to tap auriferous fluids deeper in the pile and the mixing of these fluids with more reduced fluids within the Ordovician succes-

sion. Evolution of slate belt gold mineralisation contrasts significantly in several ways with Archean greenstone- and BIF-hosted metamorphic Au deposits.

Unconformity-related Uranium deposits constitute a significant portion of the world's economic uranium mineralisation. Such deposits occur at or near unconformities between a metamorphosed, strongly deformed basement and a gently dipping sandstone cover. In this paper we emphasise the role of basement-cover contrasts in controlling the plumbing system responsible for mineralisation and associated alteration utilising examples from the East Alligator River (E.A.R.) uranium field in northern Australia. Significant (perhaps essential) elements of the geological environment of the deposits include: (i) A basement comprising nighly evolved supracrustals (including U-enriched granitoids and carbonaceous schists) which have undergone medium to high grade metamorphism and polyphase deformation producing strongly foliated rocks. (ii) A thick cover rock sequence containing basal terrestrial quartz-rich clastics of very low metamorphic grade. (iii) Localised, post-cover sequence brittle-ductile deformation.

High grade uranium mineralisation in the E.A.R. (Ranger, Jabiluka, Koongarra, Narbalek deposits) in reverse fault zones and associated kinked, chevron folded and brecciated regions in the basement. The reverse faults have non-planar surfaces controlled by mechanical anisotropy of the basement schists and their contrast with the overlying sandstone : the faults preferentially follow shistose units and transform along the unconformity into steep narrow zones in the cover. Breccia zones and mineralisation are best developed where the faults are subparallel to basement foliation. The breccias are initiated by hydraulic fracture, commonly in kink zones, and may evolve by extensive dissolution into stylobreccias. Chloritic and other styles of magnesian alteration are most strongly developed in the brittle-ductile deformation zones related to the faults. The vast mass transfer required to effect alteration evidences the large volumes of fluid focussed through these zones and emphasises the role of structurally-enhanced permeability in their development. We present simple geometric/mechanical models to illustrate these relations.

Fluid inclusion data (e.g. Ypma and Fuzikawa, 1980), stable isotope (e.g. Binns et al., 1980) and mineralogical studies suggest that the major phases mineralisation and alteration took place at 150-200°C and may be dated between 1300 and 1600 m.y. B.P. from radiogenic isotope studies (e.g. Riley, et al., 1980; Gulson and Mizon, 1980). Fission track dating (Koul, Wall and Johnston, in prep.) indicates 150-200°C regional temperatures during this period, resulting from the very low grade regional metamorphism of the cover rocks under ≥4-5 kms of burial. This regional system provided the fluids responsible for mineralisation and alteration. The hypersaline character and isotopic signature of the fluids is consistent with their derivation from evaporitic facies in the cover rock sequence. Fault controlled circulation of these relatively oxidised fluids through U-enriched basement rocks resulted in uraniferous solutions which on further migration locally mixed with CO_2/CH_4 bearing fluids (from interaction with graphitic schist) to effect uraninite precipitation. The inferred chemistry of cover rock derived fluids would also produce the observed magnesian alteration.

The essential contributions of deformation and metamorphism to the evolution of the E.A.R. uranium field involve: (i) Pre-cover rock regional metamorphism producing foliated (mechanically anisotropic) and locally U-enriched basement rocks. (ii) Very low grade regional metamorphism of the cover rock sequence leading to the development of saline, relatively oxidised fluids and low permeabilities of the basal sandstone. (iii) Circulation of these fluids through the U-enriched basement in fault zones. (iv) Strong focussing of these heated, through uraniferous fluids deformation zones in the basement, the geometry of these zones being controlled by the basement/cover mechanical contrasts and their enhanced permeability resulting from hydrofracture and dissolution. (v) Magnesian alteration resulting from the passage of the fluids and also their mixing with fluids containing locally derived reductants effecting mineralisation in and around the deformation zones.

Similar models may be applied to unconformity-related deposits of the Athabasca region, Saskatchewa. Contrasts in the size, geometry and distribution of deposits between the Athabasca and E.A.R. probably reflect differences in the structural geometry, structural- and thermal-history and related permeability distribution of the two regions.

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The Role of Heat in the Development of Energy and Mineral Resources in the Northern Basin and Range Province

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EPITHERMAL GOLD/SILVER DEPOSITS: THE GEOTHERMAL CONNECTION

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ABSTRACT

Epithermal precious metal deposits in the western United States can be related to two main plate tectonic regimes: (1) active subduction zones of Mesozoic to Quaternary age with associated subduction-related magmatism and geothermal phenomena and (2) extensional tectonics of Tertiary-Quaternary age with associated rifting, magmatism and geothermal activity.

Three main deposit types and numerous subtypes of epithermal gold-silver deposits can be recognized in the western U.S.: (1) volcanic- or volcaniclastic-hosted vein, disseminated and stockwork deposits in or adjacent to volcanic centers; (2) sediment-hosted, stratigraphically-controlled replacement and stockwork deposits, typically in thin-bedded carbonaceous limestones or calcareous, carbonaceous shales (Carlin type), or in solution breccias, paleokarst horizons and channels (Sherman type), (3) hot spring-related stockwork and fissure systems, in which permeable rock horizons including volcanicalastics and hydrothermal explosion breccias are common host rocks for massive silicification and mineralization. Volcanic and subvolcanic rocks are common in the vicinity due to the association of volcanism with geothermal systems.

Mineralized fossil goethermal systems are directly analogous to a number of modern geothermal systems and the processes involved are essentially identical. Modern geothermal systems that contain significant amounts of base and precious metals (e.g. Taupo zone, New Zealand; Salton Sea, California; Steamboat Springs, Nevada; O-Yunuma and Sakurajima, Japan) are related to volcanic centers active within the Quaternary. Geothermal systems unrelated to volcanic centers, the deep circulation type related to high regional heat flow, typically contain very minor amounts of metals (e.g. Beowawe, Buffalo Valley and Darrough in central Nevada; Rio Grande rift in New Mexico).

EPITHERMAL GOLD/SILVER DEPOSITS: THE GEOTHERMAL CONNECTION

INTRODUCTION

Epithermal precious metal deposits in the western United States can be related to two main plate tectonic regimes: (1) subduction zones of Mesozoic to Quaternary age with associated subduction-related magmatism and geothermal phenomena, and (2) extentsional tectonics of Tertiary-Quaternary age with associated rifting, magmatism and geothermal activity. Deposits related to subduction-generated magmatism are associated with volcano-plutonic belts that range in magma chemistry from calcic through calc-alkaline, high potassium calc-alkalic, alkali-calcic and alkalic (Westra and Keith, 1981; Keith, this volume). The subduction-related deposits occur in subparallel, time-dependent belts and their metallogeny reflects the magma chemistry of associated igneous rocks (c.f. Keith, this volume). Deposits related to extensional tectonism are associated with rhyolites of the bimodal, basalt-rhyolite assemblage (McKee, 1971; Christiansen and Lipman, 1972). The rhyolites of this suite are high-silica, subalkaline or peralkaline types. No significant precious metal deposits are associated with volcanic centers in which only basalt or peralkaline rhyolite is present.

Three main types and numerous subtypes of epithermal gold/silver deposits can be recognized in the western U.S.: (1) volcanic and/or volcaniclastic-hosted vein, disseminated and stockwork deposits in or adjacent to volcanic centers; (2) sediment-hosted, stratigraphically-controlled replacement and stockwork deposits, typically in thin-bedded carbonaceous limestones or calcareous, carbonaceous shales (Carlin-type), but also in solution breccias, paleokarst horizons and channels (Sherman-type); (3) hot spring-related stockwork and fissure systems, in which permeable rock horizons including volcaniclastics and hydrothermal explosion breccias are common host rock for massive silicification and mineralization. Volcanic and subvolcanic rocks are usually associated with hot spring precious metal deposits, reflecting the common relationship between volcanism and metal-bearing geothermal systems (White, 1981).

EPITHERMAL Au-Ag DEPOSIT TYPES

The term epithermal (Lindgren, 1933) denotes Au-Ag deposits with the following generalized characteristics: deposition from relatively low salinity fluids (12% equivalent NaCl or less) at temperatures in the 150°-300°C range; limited vertical range of

Au-Ag deposition, typically 400 m or less and within 1000 m of the premineral surface; and, presence of As, Sb, Hg, Tl, Se, Te minerals associated with electrum, silver sulfides and sulfosalts with variable base metals. Au-Ag tellurides or Ag selenides are important ore minerals in some districts.

The source of the metals in epithermal Au-Ag deposits is an unresolved problem. Stable isotope data from many deposits indicate that the ore fluid was dominantly composed of highly exchanged meteoric water and sulfur derived from a sedimentary source. This has led many workers to propose that the metals were derived by hydrothermal leaching of source rocks, principally sedimentary rocks. Conversely, the close spatial association with intrusive and extrusive igneous rocks at nearly all deposits suggests to some that igneous rocks may have been the source of at least part of the metals present.

Continuing detailed geologic studies at a large number of deposits, involving stable isotopes, geologic mapping, geochemistry of the mineralized zones and related unmineralized sedimentary rocks and associated igneous rocks, and detailed fluid inclusion studies will be required in order to more closely define sources of metals.

Volcanic-Hosted Epithermal Au-Ag Deposits

Volcanic-hosted Au-Ag deposits include many of the famous bonanza deposits of the Circum-Pacific rim. In the western U.S. and Mexico they include such major districts as the Comstock Lode, Tonopah and Goldfield in Nevada, Cripple Creek in Colorado, and Guanajuato, Pachuca-Real del Monte and Tayoltita in Mexico. Deposits are typically in the 1-10 million ton range but often occur in districts with as many as thirty discrete orebodies. Grades vary widely up to a hundred ounces per ton silver and ten ounces per ton gold, but are generally less.

The observed ages of mineralization vary from late Mesozoic through the Cenozoic, coinciding with the ages of the volcanotectonic complexes with which they are genetically associated. Conceptually, older volcanic-hosted Au-Ag deposits will occur insofar as their supracrustal host rocks are preserved. Conversely, these deposits - as well the other epithermal types - are in the process of formation today at various locales around the Circum-Pacific region.

Three main subtypes of volcanic-hosted Au-Ag deposits can be distinguished by differences in alteration mineralogy, major and trace metal content, gold/silver ratios, and associated igneous rock types.

I. Enargite-gold subtype

One major type is characterized by advanced argillic alteration, gold associated with enargite-group minerals, high total sulfur content in both sulfides and sulfates, and by associated volcanic and subvolcanic igneous rocks ranging in composition from andesite to rhyodacite. Goldfield, Nevada, El Indio, Chile, and Summitville, Colorado are major examples of this deposit type. Silver-bearing tetrahedrite, galena and sphalerite underlie gold-enargite mineralization in some districts but, are virtually absent in others. Bismuth and tellurium are important trace metals. The gold mineralization commonly occurs in silicified hydrothermal breccias. Grades are often erratic and can be spectacularly high. Alteration, both hypogene and supergene, is widespread and typically extends far beyond economic mineralization.

Sillitoe (1983) has recently described enargite-bearing massive sulfide bodies with associated gold mineralization which occur in volcanic host rocks within zones of advanced argillic alteration. His enargite-bearing massive sulfide deposits, several of which he shows as genetically related to porphyry copper systems, are a variant of our enargite-gold deposit type.

II. Silver-base metal subtype

A second major subtype is associated with quartz-adularia-K-mica alteration, a generally high silver to gold ratio, a low total sulfur environment, geochemically anomalous amounts of Mo, W, Mn, F, and Se, variable amounts of Pb and Zn, and minor Cu. Genetically related igneous rocks are high and low-silica rhyolites. Deposits related to high-silica rhyolite systems typically contain more Mo, W, Sn, F and Mn than those associated with low-silica rhyolite systems. Many of the major bonanza, epithermal silver districts of western North America such as Creede, Colorado; Comstock and Tonopah in Nevada; Delamar, Idaho; and Tayoltita and Pachuca-Real del Monte in Mexico are examples of this deposit type.

Propylitic alteration is pervasive in these deposits, with potassic and phyllic alteration typically restricted to veins and immediately adjacent wall rocks. Quartz, adularia, fluorite, manganese carbonates and silicates, calcite and barite are the common gangue minerals. Argillic and phyllic alteration can form an alteration halo adjacent to and above the silver-gold ore bodies (Buchanan, 1981; Randall, 1979; Giles and Nelson, in press). The major precious metal ore minerals are electrum with silver sulfides, sulfosalts and/or selenides. The chief base metal sulfides are galena, sphalerite, and chalcopyrite with variable pyrite. Molybdenite is present in trace to minor amounts in many deposits, and huebnerite or wolframite occurs in several districts and may be present, but unidentified, in many more.

Economic silver-gold mineralization is underlain in some deposits by a zone of high base metals, chiefly Pb and Zn with some Ag. The silver-gold ore zone is typically exposed at the present erosion surface, but blind orebodies are present in some districts, e.g. Pachuca-Real del Monte and Tonopah. Structures above buried orebodies are typically narrow zones of quartz-calcite stringers which are essentially devoid of precious metals and sulfides. Anomalous amounts of Hg, As and Sb are usually present with sulfidic silicification above the precious metal ore shoots. If the water table at the time of alteration and mineralization did not intersect the paleosurface, there will be no hot spring sinters deposited, and hypogene solfataric alteration will take place. A productive precious metal system can be overlain by rock which contains no anomalous silver-gold mineralization, but may contain alunite, cinnabar, native sulfur and various sulfate minerals.

III. Gold telluride subtype

The third major deposit subtype is characterized by quartz-fluorite-adularia-carbonate alteration, a low silver to gold ratio with gold and silver occurring chiefly as tellurides, and a genetic relationship to alkalic igneous rocks. Fluorine and tellurium are diagnostic geochemical indicators for this type of deposit. Small amounts of base metal sulfides, stibnite and cinnabar may also occur with the precious metal ores. Cripple Creek, Colorado and Zortman-Landusky, Montana, are examples.

Gold telluride mineralization at Cripple Creek is hosted by quartz-fluorite-carbonate-adularia stockworks, veins, breccia shoots (Cresson Pipe) and sheeted zones which are surrounded by an envelope of propylitized volcanic rock. It seems probable that the Cripple Creek type of deposit is the volcanic and sub-volcanic expression of alkalic porphyry Cu-Au-Ag-Pt mineralization, such as that recently described by Werle and others (1982) in the Allard stock, Colorado.

Carbonate-hosted Epithermal Au-Ag Deposits

Deposits in this group are frequently referred to as "Carlin-type" deposits after the Carlin gold mine in Eureka County, Nevada. Host rocks for this group of deposits are typically thin-bedded, calcareous, carbonaceous, clastic sediments.

The carbonate-hosted deposits are hydrothermal, disseminated-replacement gold deposits. They are characterized by a high gold/ silver ratio and a geochemical association of Au, As, Sb, Hg, Ba and Tl. Anomalous amounts of W, Mo, Sn and F are usually present. Gold grades are typically in the 0.1 to 0.4 ounce/ton range with initial reserves ranging from a few hundred thousand tons to over one hundred million tons. Typical ore bodies range from five to fifteen million tons in size.

Host rocks are decarbonated in the ore horizon and variably silicified and argillized. Hydrocarbons are dissolved in hydrothermal fluids and reprecipitated adjacent to zones of acid alteration and in haloes around the ore zone. Jasperoid replacements occur in and adjacent to high-angle fault conduits. Jasperoid usually shows evidence of multiple periods of brecciation and may occur above, within or below the main ore horizon. Jasperoid is ore grade in some deposits, and almost always contain geochemically anomalous amounts of gold. In addition to the ubiquitous silicification, alteration minerals typically present include illite, montmorillonite, kaolinite, chlorite and sericite.

Spatially associated with nearly all Carlin-type deposits are stocks, dikes, sills and plugs ranging from granodiorite to rhyodacite in composition. Intrusive rocks are usually altered and mineralized and, in some deposits, contain economic gold mineralization. Primary controls of ore deposition in Carlin-type deposits are high-angle faults which transect a favorable host rock type, typically a thin-bedded, silty to sandy carbonaceous siltstone or carbonate. The gold in Carlin-type deposits is typically submicroscopic and has particle sizes in the micron to submicron range (Radtke, 1981). Gold occurs as free gold, coatings on pyrite, as gold-organic complexes dispersed on amorphous carbon grains, as discrete grains in realgar, and in solid solution in realgar and native arsenic (Radtke, op. cit.).

Sulfide minerals present in Carlin-type deposits include pyrite (2%) and, typically, highly variable amounts of cinnabar, realgar, orpiment, and stibnite. Small amounts of base metal sulfides including sphalerite, galena, chalcopyrite and molybdenite are usually present. Several rare thallium-bearing minerals have been identified in a few deposits. The amount of arsenic present in Carlin-type deposits is highly variable; some deposits such as Getchell, Nevada and Mercur, Utah contain abundant arsenic sulfides; others such as Northumberland and Alligator Ridge, Nevada contain only small amounts of arsenic.

Unoxidized primary ore at a number of Carlin-type deposits is overlain by a zone of hypogene oxidation produced by a late stage of boiling of the hydrothermal solutions. Supergene oxidation may be superimposed on the zone of hypogene oxidation, making separation of hypogene and supergene alteration stages difficult (Radtke, 1981; Wells and others, 1969).

The age of Carlin-type deposits is a subject of some controversy. Many workers prefer a late Tertiary age for all deposits of this type. Their argument is based on apparent control by Basin and Range faults of late Tertiary age, and an inferred relationship to late Tertiary intrusion and volcanism as a heat source for the hydrothermal fluids (Radtke and Dickinson, 1974; Radtke,

1981). Indeed, a remnant volcanic surface of the same age as the younger felsic dikes and sills associated with mineralization is frequently located within several kilometers of the deposits (Stevens and Hawkins, in press). On the other hand, K-Ar ages of hydrothermal sericite from several of the Carlin-type deposits give ages that range from at least Cretaceous to late Tertiary (e.g. Silberman and others, 1974; Norton and others, 1977; Berger 1980; Bonham and Silberman, unpub. data).

In some deposits, as at Getchell and Gold Acres, the geologic relationships further suggest a genetic connection between mineralization and older granodioritic or granitic plutons near the ore zone (Silberman and McKee, 1971; Wrucke and Armbrustmacher, 1975; Berger, 1980). Clearly, more work is needed to establish whether the ages of formation of Carlin-type deposits range from late Mesozoic through the Teritiary, or are confined to a distinct time span within the Tertiary.

Several published models for the formation of Carlin-type deposits have placed a strong emphasis on hot-springs model. This is somewhat misleading, because there does exist a class of disseminated gold-silver deposits (discussed later in this paper) which clearly formed in a hot springs environment, within 100 meters or less of the paleosurface. The Carlin-type deposits are certainly related to fossil geothermal systems, and some or all of these systems must have vented to the surface as hot springs. However, none of the Carlin-type deposits have as yet been conclusively shown to have formed in an actual hot spring environment.

Current estimates on the depth vary from as shallow as 200 meters to as deep as 2000 meters. Such estimates usually reflect the inferred connection - or lack thereof- of mineralization with a deeper plutonic body, shallow volcanic surface, and the like as discussed above. Proponents to the shallower environment cite as further evidence the apparent open and short lived nature of the mineral systems, probable boiling of the hydrothermal fluids, and abrubt changes in fluid Eh and pH as reflected in the mineral and alteration assemblages.

Dissiminated, stratiform replacements in solution breccia (paleokarst) horizons developed on thick limestone units are another type of carbonate-hosted deposit. These gold-silver deposits have been informally termed "Sherman-type" after the Sherman mine in the Leadville district, Colorado. Paleokarst zones in upper lower Mississippian strata (e.g. Madison, Leadville) provide favorable ore locales throughout the Rocky Mountain region. Examples include the Warm Springs (Gilt Edge) and Kendall districts in Montana, the Aspen and Gilpin districts in Colorado, and possibly the Lake Valley district in New Mexico. In all cases there is evidence of associated igneous activity.

The host in these deposits is dominantly paleokarst fill material, usually a heterogeneous mixture of limestone solution-breccia fragments and blocks in a carbonaceous, calcite-cemented siltstone and clay matrix. The old solution channels, combined with the porous and reactive infill, focus hydrothermal fluids and localize ore. Gold/-silver ratios vary widely within and between deposits, as does the accompanying base and trace metal assemblage. As examples, Gilman and Aspen are Pb-Zn-Ag districts with little gold; Sherman and Lake Valley are oxidized silver deposits; Gilt Edge and Kendall contain micron-size gold with pyrite, fluorite, minor base metals, and an As-Sb-Hg trace metal signature (Giles, 1983). The ore shoots are very irregular in plan, reflecting the primary ore control. The various central Montana deposits contained 1-2 million tons averaging around 0.4 ounces Au per ton.

Hot Spring-related Au-Ag Deposits

Hot spring lode gold deposits are silicified breccias and vein stockworks of quartz-sulfide mineralization found at or very near the pre-mineral surface. They represent an important and relatively new source of gold production. Examples currently include Kasuga, Iwato and Akeshi in Japan; Waihi and other deposits of the Hauraki goldfield in New Zealand; Cinola in British Columbia; Hasbrouck, Borealis and McLaughlin in the United States. Hydrofracted vein stockworks and mineralized explosion breccias are the main ore hosts.

Ascending fluids produce a typical mushroom-shaped cap of near surface silica replacement and flooding. The cap is often overlain by hot spring sinter. Ore grades are typically 0.05 to 0.5 oz per ton with local grades in confined volcanic throat areas and vein clusters up to 1 ounce per ton. Supergene processes may lead to local silver enrichment. The Au-Ag ratio is overall about 1:1, but is widely variable within and between deposits. Individual orebodies known to date range from 1.5 million tons (e.g. Kasuga) to more than 20 million tons (e.g. McLaughlin).

Ore is typically composed of micron-sized native gold and electrum. Associated minerals include fine-grained pyrite-marcasite and a large suite of silver sulfosalts. The gangue mineralogy is microcrystalline quartz and chalcedony with lesser calcite and adularia. Cinnabar and stibnite are zoned toward the surface; minor base metals are enriched at depth. Host rocks are typically hardened by a pre-ore episode of K-Na metasomatism and near-surface silicification. Sulfide ore is often enveloped by alunite,

kaolinite and montmorillonite, and late acid supergene leaching often overprints the mineralization.

Hot spring environments contain the strongest enrichment in the epithermal geochemical suite, a response to shallow boiling and steep near-surface gradients in temperature. Mercury, antimony, thallium and arsenic are typically anomalous. The geochemical suite is identical to that identified in the bonanza and disseminated replacement environment, but concentrations are up to several orders of magnitude higher. Individual elements may be absent, e.g. mercury in some Japanese deposits, or additional elements may be present, e.g. tungsten at Cinola, British Columbia, depending on local source rock characteristics.

Hot spring lode gold deposits are associated with the sealed portions of fossil geothermal systems and may be considerably offset from currently active centers of geothermal activity. All deposits discovered to date are associated with explosive fracturing (hydrofracting) and hydrothermal brecciation. Explosive release of pressure and resultant high permeability contribute to shallow boiling of rapidly rising hydrothermal fluids and to repeated episodes of mineralization. Temperatures of precious metal deposition are under 200°C, and the fluids are meteoric and dilute.

SUMMARY AND CONCLUSIONS

Epithermal gold/silver deposits can be subdivided into a series of subtypes based upon mineralogy, alteration characteristics, host rock lithology, magma chemistry of associated igneous rocks, depth of formation and deposit form (vein, stockwork, breccia-hosted, etc.). They are essentially confined to arc and back-arc settings and are predominantly of Mesozoic or Cenozoic age. Depth of formation ranges from surface hot spring environments to at least 2000 meters. The fluids associated with ore deposition range from dilute alkali chloride waters to acid sulfate waters with temperatures ranging from 150°C to 300°C.

This diversity in host rock lithology, ore fluid chemistry, tectonic setting and depth of formation, leads to an equal diversity in deposit form, ore mineralogy and wallrock alteration. Recognition of this diversity of epithermal gold/silver deposit types is a key factor in exploration. For example, gold/silver deposits in carbonate host rocks typically have inconspicuous wallrock alteration haloes, but engargite-gold deposits have very extensive wallrock alteration haloes.

The unifying factor in epithermal gold/silver deposits is their clear relationship to geothermal systems. The study of modern geothermal systems in the Circum-Pacific belt, in particular, has led to a better understanding of this diverse group of deposits. Future research in active geothermal systems will clearly lead to a better understanding of epithermal gold/silver deposits.

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UNITED STATES DEPARTMENT OF THE INTERIOR

GEOLOGICAL SURVEY

A compilation of fluid inclusion and stable isotope data on selected precious- and base-metal epithermal deposits

By

Daniel O. Hayba[1]

Open-file Report 83-450

This report is preliminary and has not been reviewed for conformity with U.S. Geological Survey editorial standards.

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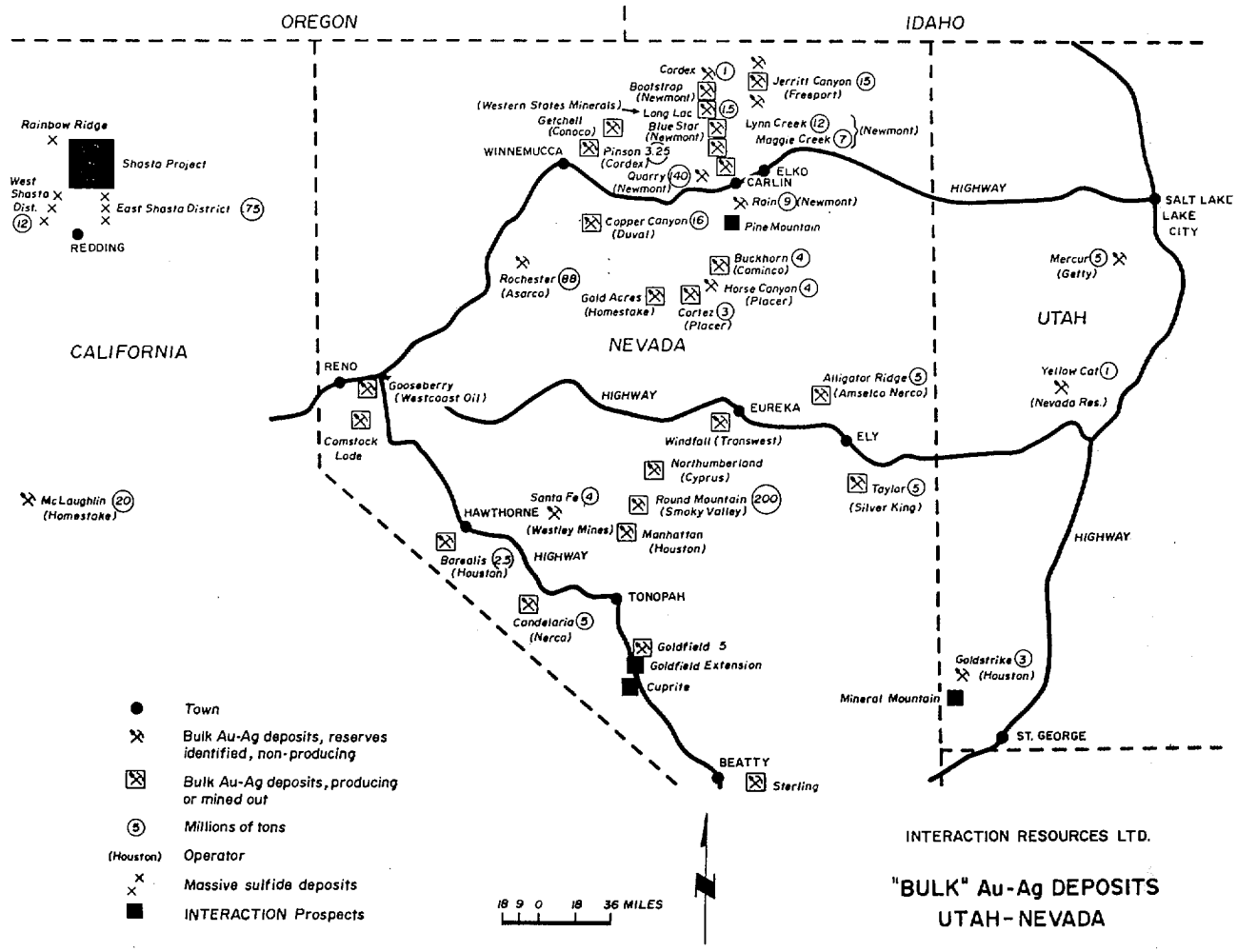
Canadians looking for epithermal gold deposits

The State of Nevada has been a source of mineral production for a long time. It has recently become the largest producer of gold in the United States, and many companies are involved in exploration and development. Among these are several Canadian groups and companies.

The experience of one Canadian group, Interaction Resources Ltd, is used here to illustrate the potential of epithermal, or 'Carlin-type' gold deposits.

Carlin Gold Mining Company (100% subsidiary of Newmont Mining Corporation) went into production at the Carlin mine in Nevada in 1965. This 'marked the development by Newmont of a new concept in gold mining based on large, relatively low-grade ore reserves mineable by open-pit methods' (Newmont annual report 1981). The ore grade milled was relatively high (about 0.21-0.30 oz/ton range during 1970-78), but reserves are now estimated at around 0.16 oz/ton.

In addition to the conventional milling carried out on this ore, lower grades were treated from about 1979 by heap leaching processes. Several of the newer projects are based on heap leaching of



low grade material. Carlin mine gold production since 1970 has ranged, roughly, between 107,000 and 215,000 oz a year from milling, while leached gold has risen from 2800 oz in 1979 to 24,200 oz in 1982.

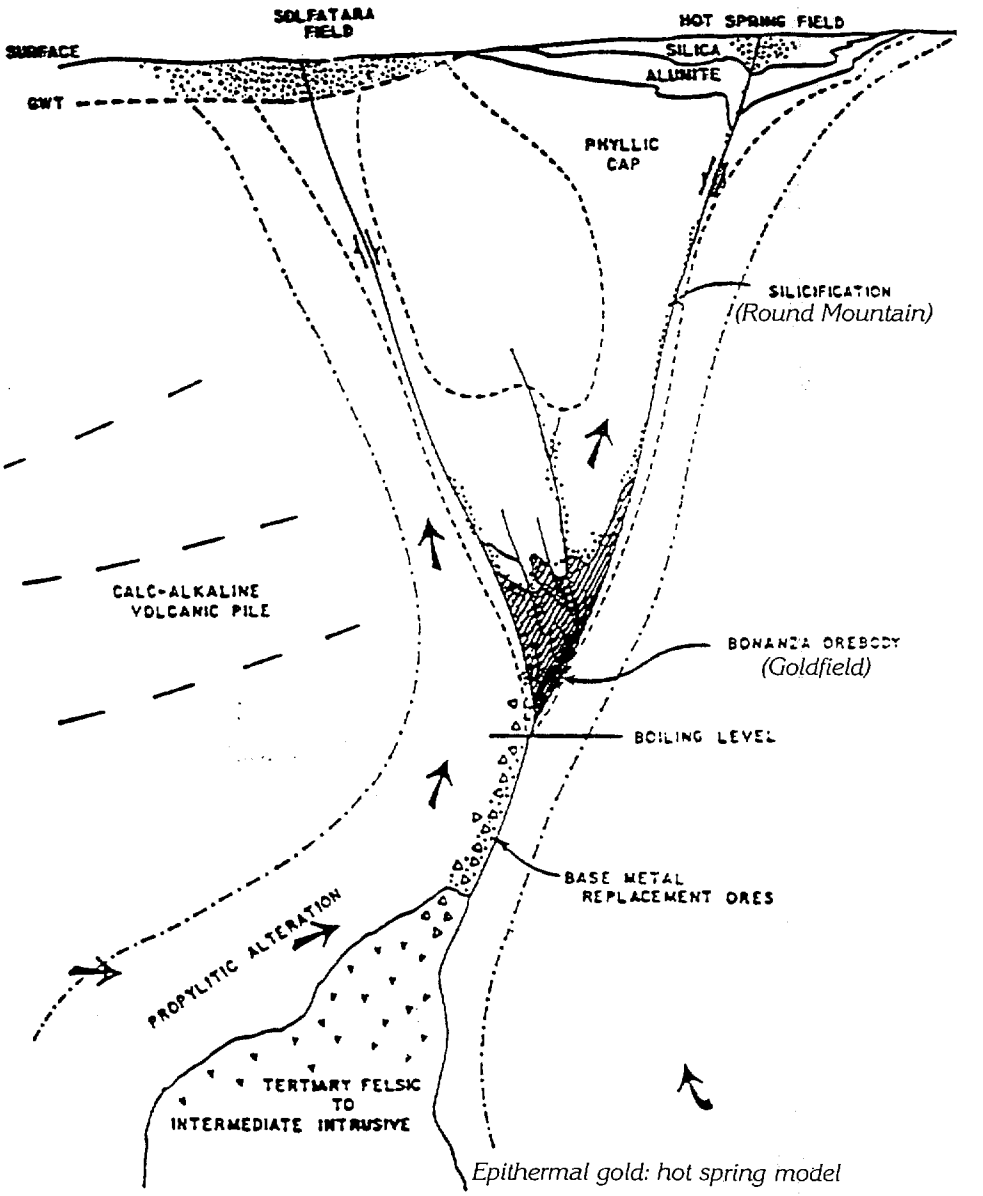
The concept of bulk gold deposits in a hot springs environment has only recently been widely accepted by the exploration industry. This concept, coupled with recent advances in geochemical and analytical methods, now permits exploration for precious metals by experienced geologists at reasonable cost and with a good chance for success.

The 'Carlin-type' bulk gold deposit occurs in a hot springs geologic environment and is usually associated with typical hot springs minerals such as mercury, sulphur, arsenic, antimony, barite, etc. The detection of these accessory elements is an integral part of the geochemical exploration program used to identify the 'Carlin-type' geologic environment. Geological and geochemical reconnaissance surveys are used to locate suitable precious metal environments followed by detailed geology, geochemistry, and geophysics to define the targets. Drilling will be required to delineate the quantity and quality of the precious metal deposit.

The precious metal deposits, originally identified in the Carlin District of North Central Nevada, are also being found in California, Arizona, throughout the State of Nevada, and Western Utah. The deposits are generally large, ranging in size from 500,000 tons up to as much as 200,000,000 tons. The gold content varies from 0.03oz to 0.35oz per ton and averages about 0.12oz. Mining is usually by economical open-pit methods. The ore is often amenable to low cost cyanide heap leaching to recover the contained precious metals.

The mining of Carlin-type deposits is characterized by low capital costs and low production costs. The total cost to produce an ounce of gold, including capital write-offs and depreciation ranges $US 121-205 depending on the complexity of the ore. This compares with an estimated average South African cost of $280 per oz. The underground Homestake Mine in South Dakota produces an ounce of gold at an estimated cost of $US365. The average cost of the major free world gold producers is estimated to be $US279 per ounce.

It is believed that the Carlin-type deposit will become increasingly important in the future and will represent a significant percentage of world gold production in the years ahead.



Epithermal gold: hot spring model

INTERACTION RESOURCES PROJECTS

Interaction Resources started a program in 1982 to explore for precious metal prospects in the southwestern USA. Because the group is managed by people with extensive experience in mineral exploration, Interaction carried out its own programs.

Subsequently, four properties have been acquired: Pine Mountain, Nevada; Mineral Mountain, Utah; Cuprite, Nevada; and the Shasta project, in northern California. The map shows these properties and others which are held by various interests, including Cominco, Cordex Syndicate, and others based in Canada.

The Mineral Mountain project in Utah is the most advanced. It is a joint venture in which Interaction has 25% interest. It consists of over 100 staked mining claims located in Washington County, Utah.

Rock geochemical anomalies high in gold, silver, arsenic, antimony, and mercury have been located along the favourable geologic contact separating Cretaceous Iron Springs formation from the underlying Permian limestones. This is the same geologic setting as the nearby Goldstrike deposit.

Additional geochemical surveying, geologic mapping and bulldozer trenching will be completed before selecting specific drill targets.

Also in Nevada is the **Pine Mountain project** (Interaction 25%). This consists of 27 claims staked in Elko County, Nevada, some 5 miles southwest of the Newmont Mining Corp Rain deposit and within the very productive north-trending Carlin District.

The area staked includes geochemical rock anomalies high in gold, silver, arsenic and antimony and within a geologic environment similar to the Rain deposit. Additional and more detailed geochemical work will be completed before selecting drill targets. Preliminary exploration drilling will consist of wide-spread rotary non-core drilling.

The Cuprite property project consists of 73 staked mining claims near the old Cuprite District, located 10 miles south of the Goldfield District, Esmerald County, Nevada.

The property overlies an area of intense hot spring type alteration. The geologic model used in exploring the Cuprite claims is based on both the Goldfield District and the Round Mountain District.

The Goldfield District, one of the bonanza districts of the western USA, produced 5,000,000oz of gold from 5,000,000 tons of ore in the period 1902-1938. Heap leaching operations are currently underway to recover gold from the lower grade wall rock.

Good possibilities also exist on the Cuprite claims for finding a gold-silver stockwork system similar to the very large Round Mountain type deposit which would occur above the Goldfield type in the hot springs model. Initially, four rotary drill holes each 1000ft deep are planned to test the prospect for both types of deposits.

The Shasta Project (Interaction 66.67%): On this 500-mile^2 area near Redding, California, the first year of a planned three-year program has been completed.

Well known former producing mines in the region are the East Shasta District that has produced 750,000 tons of 3.0% copper, 15% zinc, 0.03oz/ton gold, and 3.0oz/ton silver; and the West Shasta District that has produced 17,000,000 tons of 4.0% copper, 5.0% zinc, 0.04% oz/ton gold and 3.0oz/ton silver. The two favourable geologic horizons containing the above ore bodies extend into the Shasta project area.

In 1982, 2200 stream sediment geochemical samples and 60 rock chip samples were collected and analyzed for copper, lead, zinc, gold, silver, mercury, and barium. Four anomalous geochemical zones were located, and over 12,000 acres of land have been acquired to cover the discoveries.

Dighem Ltd, of Toronto, has completed a helicopter borne electromagnetic and magnetometer survey over one of the geochemical anomalies. This survey will be followed by detailed ground geophysics and grid geochemical surveys. It is expected that drilling will commence in early Fall.

CONCLUSIONS

From results obtained, Interaction considers that the four new gold prospects merit more definitive work, leading to drilling. It is also suggested that further discoveries will be made in 1983 in the project areas.

Economics. Studies of various possible programs suggest that 480,000 ounces of bulk 'Carlin' type gold in place at $500 per ounce would have a net value, before taxes, of $74,000,000. Because of the low capital costs required to bulk mine and leach a 'Carlin' type deposit, it is possible for a smaller exploration and development company, such as Interaction, to develop this type of deposit.

Conventional milling of the ore, as opposed to heap leaching under the case cited above, would result in an estimated net value before taxes of $85,000,000. This step would be taken if tonnage and grades were sufficient to warrant the higher capital costs of a conventional milling operation.

Reference. The Cordex Syndicate, mentioned above, includes Rayrock Resources, Dome Exploration, and Lacana Mining, which has been successful with the Pinson mine, Nevada (producing since Jan 81).

The group has a new mine in the making: the Dee, formerly known as the Boulder Creek property, which is due to produce in September 1984. Information about this and the Pinson operation appeared in *The Northern Miner*, May 26, 1983 (p1, p6).

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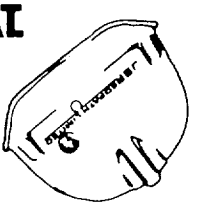

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What is the value of gold?

Market analysts mostly agree that the price of gold will vary under the influence of a complex set of factors, including political, economic, and speculative pressures. It is therefore interesting to see how this basic agreement is interpreted by the various analysts. In a recent issue of *Business Week*, predictions for the price of gold at January 1984 ranged up to US$650/oz and down to less than US$300/oz.

Whatever the official or unofficial attitude to gold as a currency or standard may be, there is no doubt that mankind has a deeply-ingrained respect for the value of the metal, for whatever reason. The industrial and scientific uses of gold are also being extended, to provide a further, and more tangible basis for the regard in which the metal is held.

In a recent talk to a *Financial Post* conference (Toronto 11 March 83), Richard L Davies, managing director of The Gold Institute/l'Institut de l'Or, spoke of gold as the 'bright light of industry'. Here are some of the points he made.

Gold: bright light of industry. After millennia of seeking, finding, mining, and refining gold, there are no more than 3-billion troy ounces (oz) of it. Of this, 1.2-billion oz form the basis of the world's monetary structure; the rest is held as jewelry, works of art, dental elements, industrial items, and people's savings and investments.

Canada has for some time been the third largest producer of primary gold (1982, 3.4% of the world total). Its production is projected to increase by 74% in 1985, to 2,458,000oz: expected to be 5.1% of the world total production in that year.

The two largest producing countries, South Africa and The Soviet Union, have their own gold programs. In the rest of the world, Canada has shown creative leadership. Canadian gold producing companies, with the encouragement of

WORLD GOLD MINE PRODUCTION PATTERN CHANGING

With cooperation of gold mining and related enterprises throughout the world, The Gold Institute/l'Institut de l'Or has issued *World Mine Production of Gold 1981-1985*. The annual amounts of gold produced from underground, surface, and alluvial sources are indicated for each of the 57 countries of the world known to produce at least one thousand troy ounces of gold per year, arranged according to the size of their expected 1983 production.

The report shows that, while the production rates of the two largest producers, South Africa and the Soviet Union, are showing little or no increase, other countries' production is growing.

The report gives production figures for each country in thousands of troy ounces, and relationships in tenths of a percent. The following summary indicates the magnitude of these interesting changes in rounded millions of ounces and in rounded percentages.

	million troy ounces			*percent of world totals*		
Country	*1981*	*1983*	*1985*	*1981*	*1983*	*1985*
South Africa	21	22	22	50%	48%	45%
Soviet Union	10	10	10	23%	22%	22%
Other Countries	11	13	16	27%	30%	33%
World Totals	42	45	48	100%	100%	100%

For the leading gold mining countries after South Africa and the Soviet Union, following are their updated 1983 and 1985 expected mine production figures.

	thousand troy ounces	
Country	*1983*	*1985*
Canada	1,960	2,458
China	1,930	2,160
United States	1,777	2,098
Brazil	1,477	1,707
Australia	831	805
Philippines	791	794
Papua New Guinea	584	1,187
Colombia	532	560
Chile	515	526
Ghana	450	490
Zimbabwe	422	426
Dominican Republic	340	339
Peru	223	230

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Henry C Armstrong, Energy, Mines and Resources Canada, organized The Gold Institute/l'Institut de l'Or. This is made up of furnishers of gold in 12 countries. This Institute is dedicated to increasing the beneficial uses of gold, to increasing and extending the awareness of the value of gold, and to providing precise and timely information on gold and gold products.

The United States Bureau of Mines reports an increase in use of gold for jewelry and arts, dental, and industrial applications, from 2,771,000oz in 1981 to 3,403,000oz in 1982.

Aircraft use reliable gold connectors in flight control equipment; some have 82:18 gold: nickel bonding alloy to assure jet engine performance of turbine blades and afterburners. An international symposium on gold industrial bonding alloys was held on 27 April 83 by the Gold Institute and the American Welding Society.

Many attractive modern buildings use gold-coated glass, which combines aesthetic and practical advantages. The coated glass admits light but reflects heat radiation both from outside (to help keep the interior cool in summer) and inside (to prevent heat loss in winter). The result can be a considerable saving in air conditioning and heating costs. Some of the advantages are discussed in a publication 'Gold in Architecture' published by The Gold Institute.

Gold in the form of ingots, coin, and artefacts has been widely used as an incentive or reward to people in business and industry. Gold prizes offered to distributors and customers have reportedly helped to produce sharp increases in market shares and total volume of gold during 1982.

Catalysts. DuPont has used gold to produce vinyl acetate plastics and adhesives; Exxon has been improving high octane gasoline catalysed by gold; and there are other notable industrial catalyst applications on record.

In recent years the Gold Institute has collected information about creative catalyst workers internationally. In May 83, in Philadelphia, an international symposium on gold for catalysis was held by the Gold Institute and the Catalysis Society.

Money structures. The nations of the world, directly and through the International Monetary Fund and Bank of International Settlements need and use about 1.2-billion troy ounces of gold as the primary component, representing well over half the value, of their international reserves. The Gold Institute report on World Mine Production of Gold shows that all the 57 known gold producing countries of the world produced 43,477,000 troy ounces of gold in 1982. Much of this was used for jewelry, dentistry, and industry as well as in corporate incentive programs. The remainder was purchased by individuals and private institutions in the form of coins, ingots, bars, and medallions, as savings and investment. Although there was considerable shifting of gold from one government to another, there was no significant change in the total gold held by the world's monetary authorities.

Although total gold holdings remained constant, governments were increasing their creation of units of currency at the rate of around 10% per year. Money supply (M1) which includes actual currency as well as checking accounts, and which is 3 to 4 times the size of actual currency, was also increasing at rates from 8% to 15% per year. *WM*

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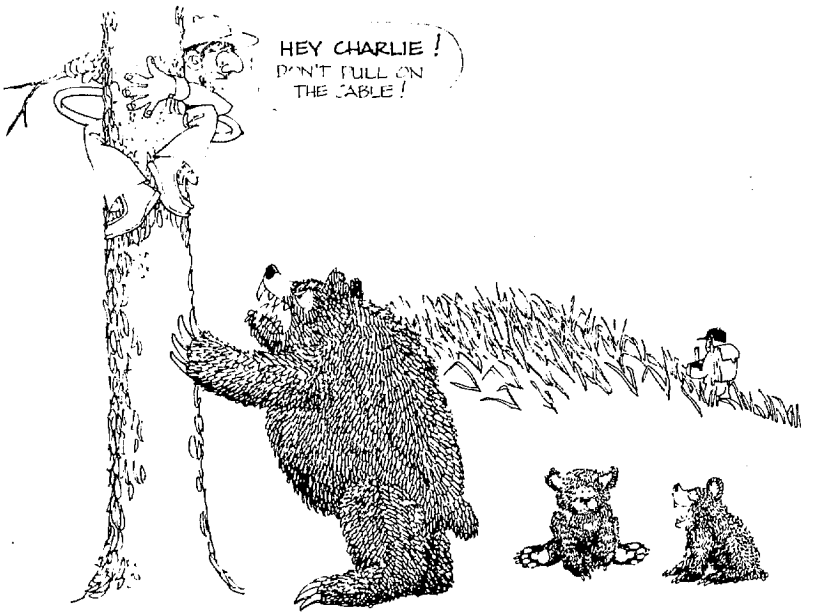

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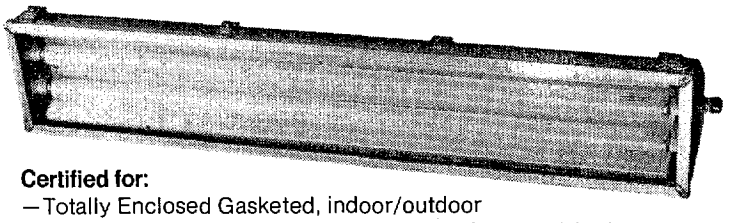

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SILVER: OVER-OPTIMISM?

Silver price projections may be too high

The world mining industry is in danger of having to suffer the consequences of inadvisable business decisions, in both precious and base metal mining, taken on the basis of an over-optimistic assessment of the future price of silver. Many mining companies at present seem to be calculating on the basis of long-term silver prices in the region of US$10-15 an ounce. In the view of Commodities Research Unit (CRU) Ltd, the price of silver, in constant money terms, will be substantially below this range. This is one of the main conclusions of a new study *Silver — The Next Five Years*.

Many mining companies that traditionally rely on base metals to provide the bulk of their revenues, have only been able to break even or show profits thanks to substantially improved silver prices since the middle of 1982.

It would be wrong, in CRU's view, for mining companies to expect this support to continue. According to CRU, better base metal prices should allow a majority of silver-producing mines to cover their operating costs in the forecast period, but those that rely on silver for most of their revenue will be vulnerable.

The study presents strong evidence that, in the final analysis, silver prices react to the level of above-ground stocks of bullion. After estimating the present and future levels of unreported silver stocks, CRU concludes that total stocks in private hands have reached, and are likely to remain at, higher levels than at any time in modern history. The study's analysis of investor behavior patterns suggests that these will ultimately reflect the deteriorating fundamentals in the metal.

Since 1980, when world consumption of silver in industrial uses and coinage fell by over 20%, there has been no improvement in total demand; nor does CRU expect very much in the future. Meanwhile, world mine production grew by over 10 percent in 1980-1981 and is likely to grow by almost as much again by the mid-1980s.

The study contains detailed analysis of present and future trends in East/West trade, government stockholding practices, supplies from India and the Far East, and of scrap.

CRU Consultants Inc, 33 West 54th St, New York NY 10019. Commodities Research Unit Ltd, 31 Mount Pleasant, London WC1X 0AD, England.

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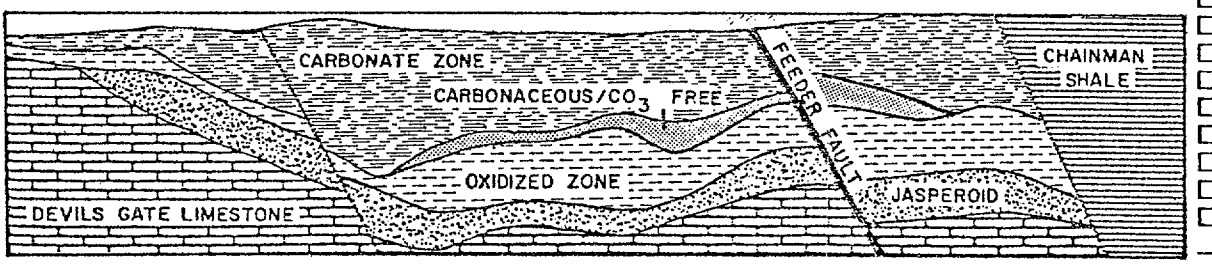
Your Abstract Number

№ 19402

CHEMICAL AND MINERALOGICAL ZONING ASSOCIATED WITH STRATAGENE, FAULT CONTROLLED GOLD MINERALIZATION AT ALLIGATOR RIDGE, NEVADA

AINSWORTH, Jack C., and BRIMHALL, George H Jr., Department of Geology and Geophysics, University of California, Berkeley, CA 94720

The Vantage ore bodies at Alligator Ridge are Carlin-type, sediment hosted, disseminated gold deposits. They occur within the lower Mississippian Pilot Shale, a finely laminated, calcareous to dolomitic, carbonaceous siltstone. The lateral and vertical distribution of alteration and chemical zoning in the Vantage 2 ore body can be attributed to lateral flow away from a feeder fault along the interface of the Pilot Shale and the underlying Devils Gate Limestone, coupled with the outward diffusion of this stratagene fluid. This resulted in zoning patterns that pinch out and converge on the Pilot-Devils Gate contact away from the feeder fault. Although the deposit at Carlin occurs in the Silurian-Devonian Roberts Mountain Formation, a similar model was used to describe gold mineralization there (Radtke et al., 1980), indicating that these ore forming systems operate within favorable host rocks regardless of age. The fluid-rock interaction resulted in the removal of the Ca, C, Mg, and CO_2, and the introduction of Au, Ag, As, Sb, and Si, a pattern similar to that found at Carlin. However, unlike Carlin, stratabound mineralization at Alligator Ridge was followed by [illegible]ate, deep oxidation that produced an advanced argillic assemblage of alunite-jarosite, kaolinite and hematite near the feeder fault.



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Your Abstract Number

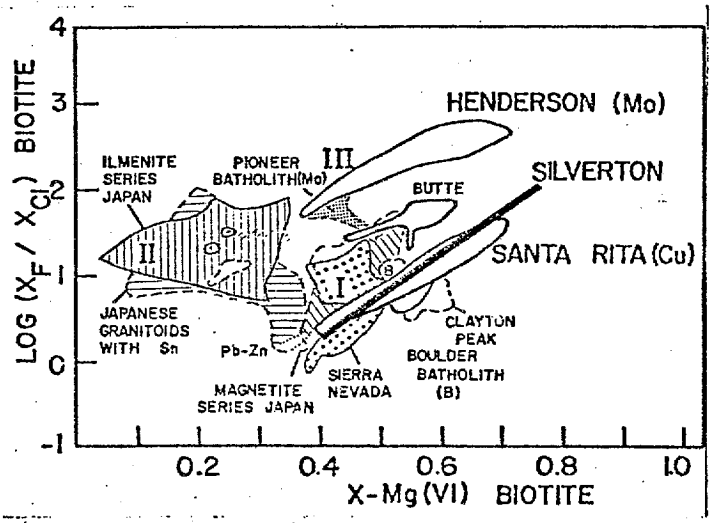
№ 19397

ALTERATION BIOTITE CHEMISTRY AND NATURE OF DEEP HYDROTHERMAL SYSTEM BENEATH SILVERTON DISTRICT, COLO.

GILZEAN, M. N. and BRIMHALL, G. H., Dept. of Geology and Geophysics, Univ. of Calif., Berkeley, CA 94720

Microprobe analyses of secondary biotites from beneath epithermal precious metal mineralization at Silverton suggest a deep hydrothermal system of relatively low F/Cl fluids and high Cu/Mo sulfides. The Silverton biotite compositions coincide with the Santa Rita porphyry copper trend on a log X_F/X_{Cl} vs. X-phlog diagram showing a distinctly lower F/Cl than the Henderson porphyry Mo system described by Munoz & Swenson (1981). Fracture-controlled disseminated pyrite and chalcopyrite ± sphalerite, galena and trace molybdenite occur.

Similar biotite analyses were used for the sake of comparison and indicate three major grouping of granitoids and related ores, I: Magnetite Series and "I"-type intrusives with associated Cu mineralization, II: Ilmenite and "S"-type granitoids with associated tin mineralization, III: High F/Cl Mo-rich systems (eg. Henderson). The Butte district in Montana is intermediate between types I and III consistent with the relatively high Mo content and nearby Pioneer Batholith Mo porphyries. Its slope is identical to the Santa Rita and Henderson trends. Mineralized type II granitoids with high whole rock SiO_2 and low Al_2O_3 indicate a correlation of increasing F/Cl with octahedral Al in biotites.



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Your Abstract Number

№ 21777

GEOCHEMICAL CONSTRAINTS ON THE GENESIS OF THE SUMMITVILLE GOLD-COPPER DEPOSIT

STOFFREGEN, Roger E., Department of Geology and Geophysics, University of California, Berkeley, CA 94720

The Summitville gold-copper deposit, located in the eastern San Juan Mountains of Colorado, is characterized by intense alunitic alteration of the quartz latite host. The mineralogy of the deposit suggests formation in close proximity to the surface, and evolution from a vapor dominated sulfate rich system to a more reduced, liquid dominated system.

Temperature and depth constraints are provided by a cristobalite manto which locally caps the deposit, and the occurrence of pyrophyllite at depth. The cristobalite manto probably formed at the surface by quenching of a geothermal fluid saturated with quartz. Where present, this manto thus defines the surface at the time of mineralization, which is roughly coincident with the present surface. The upper limit of pyrophyllite defines an isotherm of approximately 300°C near the base of the ore zone, at a depth of 1,500' below the cristobalite manto.

Wallrock alteration grades from an inner zone of quartz-pyrite ("vuggy silica"), to quartz-alunite-kaolinite, then to a kaolinite-illite zone. The vuggy silica zone, which diminishes with depth and disappears at about 1,000' below the cristobalite manto, is believed to have formed in fractures occupied dominantly by vapor, but through which acidic near surface condensates could drain. Adjacent to these zones liquid water, held by capillarity in smaller fracture spaces, remained predominant, as evidenced by locally abundant liquid-rich fluid inclusions in alunite.

A change to more reducing conditions, possibly caused when liquid water reoccupied major fractures and caused self-sealing of the system, isolating it from surface oxidation, is demonstrated by deposition of sulfide minerals in the vuggy silica zone. The sulfide paragenesis suggests control by decreasing temperature or increasing sulfur fugacity, terminating with local deposition of native sulfur.

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Placer mining and other projects for Queenstake Resources

Placer mining in the Yukon has been Queenstake Resources' stepping stone to involvement in a wide range of projects in western North America.

Starting with its unique dredging operation on Clear Creek, Yukon, the company now has a second placer project in production at Black Hills Creek, and has various interests in gold and hardrock prospects in Yukon, Alaska and the western US. As well, its 12.5% equity ownership in Texas-based Canyon Resources gives it an interest in several promising oil and gas plays in Texas.

After minor setbacks at the start of the 1982 placer mining season (severe spring ice conditions and discontinuities in the gold pay channel), Queenstake mined 2038 oz of raw placer gold with its bucketline dredge at Clear Creek, yielding 1650 oz of fine gold. Mining has got off to a good start this season (beginning late in May) and throughput of 225,000 yd^3 is anticipated. A program of sampling was conducted in 1982 which has determined the pattern for dredge mining throughout 1983 to ensure that the average gold grade of dredged gravel remains above 0.01 oz/yd^3.

Queenstake's second placer operation is located at Black Hills Creek, 60 miles southeast of Dawson City, Yukon. The property was acquired in September 1982 and mining, using two Caterpillar bulldozers, a front end loader and a sluice box got under way late in May. The company expects to process around 2000 yd^3/day at an average grade of 0.016 oz/yd^3.

Late in May 1983, Queenstake acquired another Yukon placer mining property, located on the tributaries of 60 Mile River. A total of 77 placer claims (seven miles) and two miles of placer leases were acquired for $171,000 cash and future option and purchase agreements which could total $721,000 over five years. Production will commence this season on a block of 16 placer claims.

Queenstake obtained $425,000 in mining equipment in exchange for a 22½% net profit interest on the 16 claims and entered into a 90-day option agreement in which investors can earn a 52½% net profit interest in the 16 claims in exchange for $900,000. In addition to this $1.3 million of project capital, Queenstake would also acquire an overriding production royalty and a 25% net profit interest in the 16 claims.

Over the next few months the company will conduct a drilling program on the balance of the nine miles of placer property the cost of which is to be 75% funded by investors in exchange for a right of first refusal to provide project financing. Contingent on the success of this exploration program, a second placer mining operation is planned on the 60 Mile properties in 1984.

Forecast production from all of Queenstake's Yukon placer gold mines this season is projected to be in excess of 6000 troy oz of fine gold. With 1000oz of this amount hedged at a price of $US511/oz, total revenue is projected at $3.25-million, a more than three fold increase over 1982.

In addition to its own placer operations, Queenstake has an interest in the Dublin Gulch project, optioned to Canada Tungsten Mining Corporation. (Queenstake retains a 10% prepayout and a 30% post payout net profit interest.) Drilling to date indicates the presence of high grade gold-tungsten bearing reserve containing approximately 8000 oz of placer gold in 140,000 yd^3 of gravel. In 1983 Canada Tungsten will commence a program to remove the overburden material in preparation for mining the gravels in 1984.

Queenstake is also participating with Canada Tungsten on the Mar tungsten deposit in the Yukon which has drill indicted reserves of 6-million tons grading 0.8% WO_3. Canada Tungsten can purchase half of Queenstake's 20% interest for $2-million on making a decision to place the property into production.

Other placer holdings include the Big Creek property, Yukon, and the Bullion Creek property in northern BC. The company's interest in Barker Creek, Yukon was sold in 1982 after disappointing results from a bulk sampling program.

In the US, Queenstake has an option on the Argus epithermal gold property northeast of Bakersfield, California. In February 1983 a joint venture agreement was reached with Anaconda Minerals Company under which the latter has spent $1.25-million on the property, paid Queenstake $350,000, and agreed to

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fund the next $10-million of exploration and development costs. Anaconda and Queenstake will share future costs and production proceeds in a 51:49 ratio. Geological mapping is under way prior to an initial drilling program.

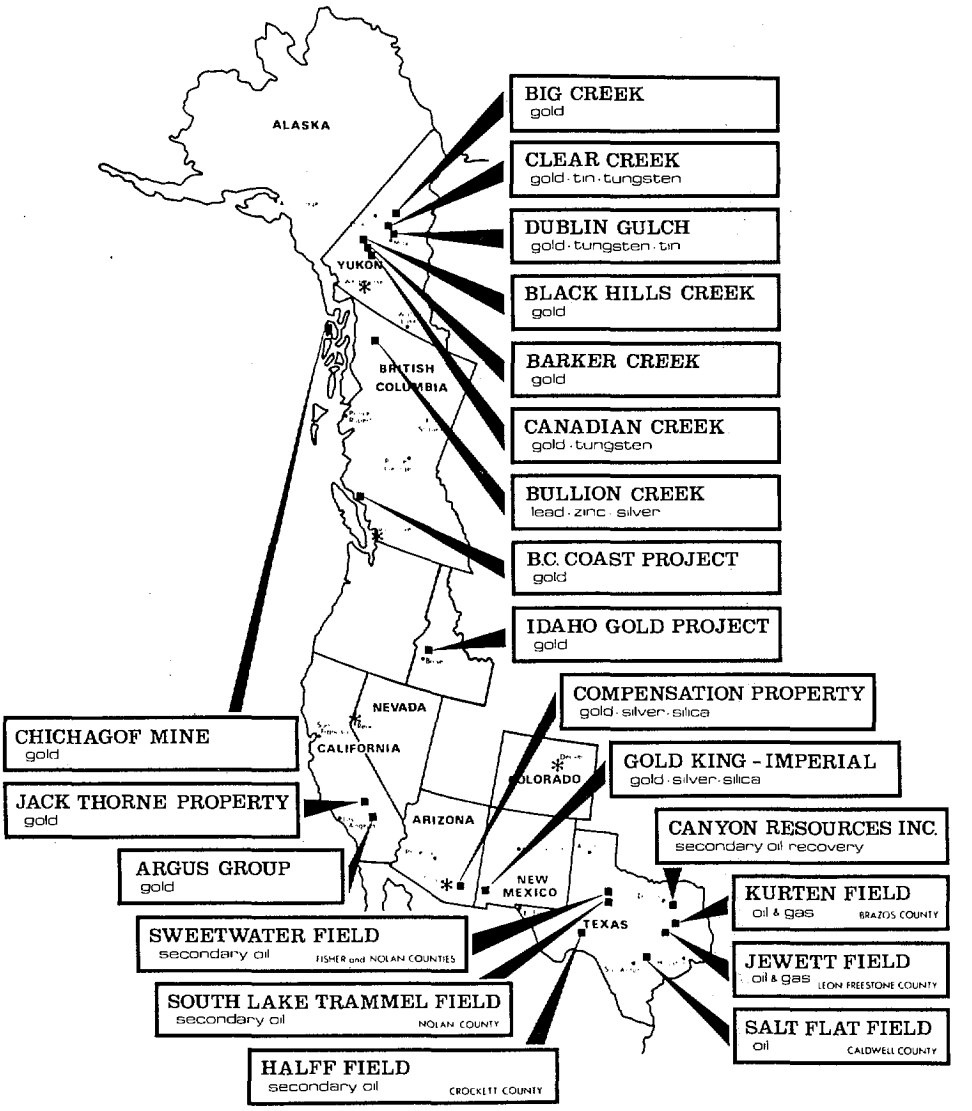
A production decision is expected in 1983 on the Gold King-Imperial gold-silver-silica flux operation in New Mexico. Queenstake is negotiating a smelter flux contract with nearby Phelps Dodge smelter and anticipates production at 100 tons/day.

Another gold prospect in Idaho has been optioned to Flow Resources and Eldorado Minerals. Queenstake also holds interests in the Cactus gold project west of the Argus mines in California; the Jack Thorne gold prospect near Fresno, California and the Compensation mine property in Greenlee County, Arizona. Recent improvements in smelter flux markets and the price of silver have made the latter property attractive as a source of silver-bearing fluxing ore.

Late in April 1983, the company acquired an option on the former-producing Chichagof mine in southeastern Alaska. It can earn a 50% interest by spending $1.3-million on the property. The 1983 exploration program will include rehabilitation and sampling of underground workings, surface vein sampling, rock geochemical studies and analysis of waste dumps and stamp mill tailings. Cost of the first phase is $200,000 (US). Depending on results, this will be followed by drilling.

As Queenstake continues to negotiate joint venture partnerships and new property acquisitions, it is evident that 1983 will be another year of rapid development for this dynamic junior mining company. Queenstake is owned 45% by Canada Tungsten Mining Corporation.

Queenstake Resources PROPERTY LOCATIONS



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ASARCO

Mining Department

July 21, 1983

MEMORANDUM

To : R.J. Kupsch

From: S.A. Anzalone

Re : Round Mountain Mine, Nevada - Sale of Felmont Oil Corp. Interest

At the request of Asarco's Exploration Department, a one-day visit was made to the Round Mountain Unit of the Smokey Valley Mining Division, Copper Range Company. The purpose of this visit was to review the ore reserve data plus supporting geological and drilling information submitted by Felmont Oil Corp. as part of an offer to sell its one-half interest in the property. The writer was accompanied by D.E. Crowell, D.F. Skidmore and J.R. Stringham.

Summary

1. Felmont's ore reserve figure of 157,600,000 tons assaying 0.034 oz. Au is acceptable.

	Tons	Grade oz. Au
Proven	109,178,000	0.034
Probable	41,471,000	0.033
Possible	21,538,000	0.034
Total	157,648,000	0.034

(D.A. Rhoades 5-3-83; 0.015 cut off, $500/oz. Au, 4.2:1 S/R)

Copper Range, using the same data base but a more sophisticated computer technique, calculated a reserve as follows:

Proven	133,076,000	0.038
Probable	89,559,000	0.036
Total	222,635,000	0.037

(2.68:1 S/R)

The Felmont calculation appears to be a bit conservative and the ultimate grade probably lies somewhere between the Felmont and Copper Range calculation.

2. Felmont's reserve calculation procedures are acceptable although somewhat outdated for a deposit of this type (See below). Their reserve appears to be conservative with regard to the total tonnage available. It may, however, be prudent to utilize the conservative figures until the drill hole spacing has been tightened in the outer portions of the deposit.

3. Drill hole spacing varies from 100 feet to 400 feet with the bulk of the drilling within the central ore zone being 200 feet or less. Approximately 600 percussion drill holes have been completed to date and while it is unlikely that additional drilling will affect the overall grade of the deposit, more drilling may be required to calculate a final minable ore reserve and design a pit. Golder Associates, the firm that calculated the Copper Range reserve, expressed the following concern over drill hole spacing:

> Drill Hole Density--the maximum drill hole spacing generally accepted to well define the reserves is 100 feet in Rock Type 1 and 200 feet in Rock Type 2. In excess of 50% of the reserves is located in areas where these constraints are not met. Additional drilling is needed in these areas. The drill hole spacing will need to be further reduced in areas with complex geology.

4. Proper drilling procedures and sample collection techniques were utilized during the various drilling programs.

5. The bulk of the gold mineralization at Round Mountain occurs in two distinct rock types. An upper sequence of hard, silicified, welded tuff designated Type 1 and a lower unit of softer, porous tuff designated Type 2. The current pit is located primarily in Type 1 rocks. The major portion of future reserves occur in Type 2 rocks. None of this ore type is exposed in the current pit; however, a small number of samples remaining from their 6" core drilling program were examined. Geological and mineralogical variations between ore Types 1 and 2 are sufficient to require that a bulk sample of Type 2 be obtained by large diameter diamond drilling to confirm that planned metallurgical processes proposed by Smokey Valley Mining for the future 40,000 tpd operation do, in fact, work.

6. A substantial tonnage of low grade gold mineralization amenable to bulk, low-cost mining methods exists at Round Mountain. The economic success of a joint venture on this deposit is contingent upon an increase in the current price of gold plus the development of an effective metallurgical process to recover the gold and minor silver values in Type 2 rocks.

Geology

Mineralization at Round Mountain occurs in a series of Tertiary volcanic units that were deposited on a basement complex of Paleozoic meta-sediments

and Cretaceous Shoshone granite. The volcanic units consist, from lowest to highest, of a basal lithic tuff, a poorly welded tuff (Type 2), a densely welded tuff (Type 1) and a small section of tuffaceous sedimentary rocks. These units, except for the capping tuffaceous sediments, were subsequently intruded by breccia pipes, then faulted, altered and mineralized. Mineralization occurs principally as pyrite, gold and electrum. The bulk of the deposit area was then covered by Quaternary gravels.

The geology of Round Mountain has been described in detail in a paper by Bradford A. Mills. A copy of this paper is attached for your ready reference.

Reserve Calculations

Felmont's reserve calculation utilizes a "combination of cross sectional and plan map methods." Reserve areas are first drawn on fence diagrams incorporating exploration drill holes and any other data available such as underground sampling from the old workings. The ore zones are then projected to level maps and adjusted for geological continuity. Polygons developed on the plan maps are then measured. Proven ore blocks are usually carried 50 feet beyond a perimeter drill hole in Type 1 ore and 100 feet beyond a perimeter hole in Type 2 ore. Proven ore is extended 100 feet toward another ore intercept in Type 1 ore and 200 feet toward another ore intercept in Type 2 ore.

Copper Range, on the other hand, utilized a computer to run their ore reserves. They used a geostatistical method known as "lognormal Krieg indicator with global adjustments." This reserve calculation procedure tends to be somewhat optimistic. Because of the relatively low variability shown in the Type 2 ores which constitute the bulk of future reserves, significant tonnages were classified as proven, even though the average drill hole spacing exceeded acceptable limits for these projections.

A final minable reserve will no doubt lie somewhere between the conservative Felmont method and the somewhat optimistic Copper Range calculation.

Type 2 Mineralization

A number of 6" diamond drill holes were completed to obtain bulk samples of Type 2 ore for metallurgical tests. Some of these holes were located to correlate with previously drilled rotary percussion holes. In the two or three cases examined, gold values in the diamond drill hole did not correlate well with the percussion hole. There appeared to be a 10% to 15% discrepancy in values on the low side. The gold values in the soft, porous Type 2 ore occur as randomly distributed blebs and grains of native gold associated with pyrite. When dealing with mineralization running as low as 1 ppm, the loss or gain of one small fragment of native gold will significantly alter the assay for a particular run. On the other hand, the average gold values of the numerous samples collected from Type 2 rock is probably a reliable figure.

Bulk Sample Test

A bulk sample test of Type 1 ore was conducted on portions of three benches in order to compare ore tonnage and grade figures developed from geostatistical projections from exploration drill holes against actual production based on blast hole assays. Mineralization in Type 1 ore is quite erratic and it is difficult to correlate mineralization from hole to hole. Gold values in Type 1 ore occur primarily as fracture face fillings associated with irregular shear zones. Based on this study, "approximately 94% of the gold projected to be in the bulk sample was confirmed as being there. The tons of ore actually mined was 73% of projected, but the grade of that ore was 28% higher than projected. Level by level comparisons are quite erratic." The sizable discrepancy noted is probably due to the large size of the ore blocks developed by the computer projections. If the irregular nature of the mineralization had been taken into consideration during the formulation of the variogram used in the reserve calculation, the ore blocks would have been smaller resulting in a drop in reserve tonnage. In addition, mining is more selective in Type 1 rock than anticipated; hence, there has been a corresponding increase in grade.

Mineralization is more evenly distributed throughout rock Type 2; hence, the overall irregularity will be considerably less in this unit.

Geological Mapping

Until recently, the pit geology was not mapped. Smokey Valley Mining has initiated a crash mapping program to record as much data as possible. It is unfortunate that this had not been done earlier in the life of the mine. The data would have been extremely useful in locating the mineralized trends for ore grade control and would have been extremely helpful as an aid in selecting the proper variogram for the computer ore reserve calculations.

Conclusion

A large tonnage of low grade gold mineralization amenable to open pit mining methods exists at Round Mountain, Nevada. The success of future mining activities at the deposit will depend on an increase in the price of gold and the development of a suitable metallurgical process.

S.A. Anzalone

S.A. Anzalone

SAA:llm

attachment

xc:	RLBrown	- w/attach.		DFSkidmore	- w/o attach.
	DECrowell	- w/o attach.		WLKurtz	- " "
	TEScartaccini	- " "		JRStringham	- " "

Getty dedicates its Mercur mine; to be 4th-largest gold producer

Getty Minerals Company dedicated its Mercur gold mine and mill complex 30 miles south of Tooele, Utah July 22nd. The new facility is processing ore containing micron-size gold in an area where gold was mined as early as the late 1800s.

About 2,800 tons of ore is being processed daily at the $90 million surface mine, making it one of Utah's largest mines and its largest primary gold producer. The seven-day operation employs about 200 people and has the capacity to produce 80,000 ounces of gold a year. With current production rates at other mines, that will make Mercur the nation's fourth largest.

The gold is leached by cyanide and recovered through a carbon adsorption process. The stripped material goes through electrowinning cells and a mercury retort before going to the electric furnace for molding into bullion.

Gold bars which resemble bricks and weigh about 34 pounds each are the end product.

Construction of the facility began in August 1981. Overburden removal began last winter and the five-story mill was completed in April. The first gold bar was produced April 21st. The mine's life is estimated at 14 years, based on known reserves using current technology.

Getty Minerals Company, a unit of Getty Oil Company, is operator. Gold Standard Inc. of Salt Lake City will receive 15 percent of the project's net profits after Getty recovers its costs.□

AMAX puts Kitsault plant into mothballs

As if to portend the further demise of molybdenum, AMAX of Canada said July 18th it was placing its moly mine and mill at Kitsault, British Columbia on a care and maintenance basis. Operations at Kitsault were suspended last November, affecting 375 workers. On January 31st, it was announced the suspension would be indefinite.

Since then, AMAX has maintained about 30 employees at Kitsault, but now that will drop to 10 or fewer. All facilities, including plant, equipment, housing and townsite, will be involved. Work should be completed by fall.

At capacity, Kitsault can produce 9 million to 10 million pounds of moly a year, significantly less than the 50 million pounds each of AMAX' Colorado properties — Climax and Henderson — can produce.

But with all of its properties not producing, AMAX began the year with an inventory of 78 million pounds, about four times the norm. AMAX suspended its list price in December, but the Metals Week dealer moly oxide price has been running at less than $4 a pound on sales of 1 million to 2 million pounds a week. Byproduct moly producers — mostly copper mines — have capacity to turn out some 100 million pounds of molybdenum a year. □

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Great Salt Lake gives brine producers woes

Even though the water level of the Great Salt Lake has peaked, problems related to its high water level haven't.

The lake reached its highest point on June 30th at 4,205 feet above sea level. Since that time, it has been dropping slowly and was recorded at 4,204.7 feet on July 15th.

The problem of how to handle the excess water, if it happens again, has yet to be settled. On July 21st, the Utah Legislature rejected a bill to cut a 300-foot gap in the railroad causeway that crosses the lake. It creates a higher water level on the south end of the lake than on the north. It is estimated that if the causeway were breached it would lower the lake's level throughout by 10 inches.

Two mineral companies along the lake haven't been flooded out, but both suffered. Damages of more than $9.5 million occurred in the past year at AMAX Magnesium Division and Great Salt Lake Minerals and Chemical Corporation.

Great Salt Lake Minerals has been increasing its dikes' height for several years, as each summer the water level has reached new highs. The company has spent $4.5 million on this, says Peter Behrens, president.

That company has given the state $200,000 to study pumping excess water into the surrounding desert. "The causeway was never proven it could help," he said. "Pumping the water into the desert seems to be the most feasible solution, and the most economical."

An AMAX spokesman, Lee Brown, says his company favors breaking the causeway. That "was the only method of quickly dropping the level of the lake and the most economical of current choices," he said.

"The problem for this year is over," said Behrens, because of the declining level. "The problem for next year is coming up." □

EARNINGS REPORTS

ASARCO Inc. reported net income of $27.6 million for the second quarter and $43.7 million for the half. That's up from losses of $24.8 million and $26.2 million in last year's periods. Better copper and silver prices helped as did stronger financial showings by overseas units.

Phelps Dodge Corporation earned $300,000 in the second quarter, including a $7.0 million contribution by its primary metals operations. It lost $3.6 million for the half. That compares with $19.4 million and $38.5 million losses in 1982. Better copper prices and significantly lower costs helped the company.

Hecla Mining Company earned $9.9 million in the second quarter and $17.5 million in the half. Income increased 29 percent in the second quarter despite increased development costs at several operations. Hecla lost $4.4 million in the 1982 half.

CF&I Steel Corporation lost $14.7 million in the second quarter and $32.5 million in the half as sales continued to decline. Those figures compare with losses of $5.9 million and $1.1 million a year earlier. In the half, sales declined 52 percent from the 1982 level.

Noranda Mines Ltd. posted a $9.2 million profit in the second quarter and $1.9 million in the half, seeing major improvements in markets and prices. A year earlier, it lost $57.1 million in the quarter and $75.4 million in the half. Noranda's metals and minerals segment contributed $29.3 million and $41.5 million in the quarter and half.

Inco Ltd. lost $39.7 million in the second quarter and $116.6 million in the half, compared with losses of $36.9 million and $74.1 million a year earlier. Compared with the first quarter loss of $76.9 million, the second quarter loss was reduced by almost 50 percent, due to reduced shutdown costs and higher nickel prices. The world's largest nickel producer hopes the U.S. recovery will improve its markets.

Rocky Mountain Energy Company, a unit of **Union Pacific Corporation**, reported earnings of $10.0 million in the second quarter, down from $14.1 million a year earlier. Its coal and soda ash operations haven't felt the effect of recovery.

Callahan Mining Corporation reported second-quarter earnings of $882,000, compared with a loss of $675,000 a year earlier. Revenues soared 82 percent to $6.3 million vs. $3.5 million in the 1982 period. For the six months, earnings were almost $2.0 million on almost $13.0 million in revenues, compared with a loss of $885,000 on $7.5 million a year earlier. Callahan is receiving higher silver prices and is continuing heavy spending on exploration projects — $4.2 million in the first half. The company is developing a major gold deposit in northern Michigan.

CXK→JWS

UNIT FACT SHEET FORMAT OUTLINE

UNIT: Getty Mining Company

LOCATION: Mercur Mine

GENERAL MANAGER: Frank D. Wicks

STAFF, TITLES: R.H. Migliaccio - Mine Superintendant
R.G. Maurer - Mill Superintendant
R.E. Bauer - Maintenance Superintendant
M.S. Staheli - Controller
S.A. Lawson - Human Resources
G.M. Eurick - Environmental & Occupational Health Coordinator
W.G. Shubert - Security Supervisor

UNIT DESCRIPTION:

A. Brief History Of The Unit:

The Mercur mining district is located at the south end of the Oquirrh Mountains of north-central Utah, about 35 miles southwest of Salt Lake City, Utah. The district lies in a topographic basin at an elevation of 6700 feet above sea level.

The Mercur district, originally known as the Camp Floyd district included three camps - Mercur, the most important; Sunshine, and West Dip. The district was first organized in 1870 with the silver excitement and received its name in 1879 with the discovery of cinnabar, i.e. mercury. The Gold ledge was discovered in 1883 but was soon abandoned when prospectors discovered they couldn't pan it. By 1889 all thoughts towards silver and cinnabar had vanished and the gold ledge became of interest for the few left in the district. In early 1890, an amalgamation mill was established in Manning Canyon and proved a failure. At this time the cyanide process was being developed and the Manning Canyon Mill was converted over to this process. At about the same time another mill was built in the Mercur area. The first successful application of the McArthur-Forrest cyanidation process had come true and in the next few years two more mills were built. The boom period of production followed which extended from 1898 through 1912. At this point the ore had diminished to where profitable operations could no longer be maintained and were terminated. The district was inactive until 1932 when modest mining and milling operations were initiated. These operations were terminated in 1942 by the Federal Gold Mine Closing Order. No production has been reported since 1942.

Total production to date for the Mercur district has been calculated at 1.2 million ounces.

Production in the Mercur district did not terminate because of depletion of the gold-bearing mineralization, but, rather the grade of the ore diminished to a point where an acceptable profit could not be maintained.

/3/83
XC: RLB
PGV
DEC
DMS
M.McC
WLK - Gold Dep Desc

B. Production:

1. Start Of Stripping: April, 1982 through November 1982 by contractor 5,000,000 Tons

2. Start Of Ore Production: January, 1983

3. Total Production (to May 1983 date) by Getty: Tons Of Ore - 484,000 Tons Of Waste - 3,542,000

4. Current Production Rate: 25,000 Tons/day

C. Deposit Geology:

The Oquirrh Mountains are a fault block range which trends north-south and is situated near the eastern margin of the basin and range physiographic province of the Western United States. The Mercur district is located on the east flank of a northwesterly trending anticlinal structure near the south end of the range. In this area, erosion has exposed marine carbonate sediments which are host to the gold mineralization.

The host rocks are Mississippian age, and consist of thin bedded silty carbonaceous bioclastic wackestones and packstones, silty carbonaceous mudstones and calcareous siltstones and fine grained sandstones which are overlain by silty, carbonaceous shales.

Intrusive rocks, which are present in the south half of the district, consist of irregular masses of finely porphyritic rhyolite which are barren of significant gold mineralization. The rhyolite rocks are reported to be of Oligocene age, i.e. 31.6 my.

The gold mineralization, which occurs as a very fine grained dissemination of micron size, forms stratiform bodies. The gold is found to be restricted to certain beds of wackestones and packstones which occur within a 230 feet thick sequence of strongly fractured carbonate sediments.

The gold mineralization was deposited in the carbonate host rocks by hydrothermal fluids and is associated with pyrite, marcasite, orpiment, realgar, barite and organic matter. The mineralization is found in zones of hydrothermal alteration consisting of decalcification of the carbonate rocks and the introduction of silica to form jasperoid, quartz-sericite and kaolinite.

A significant volume of the mineralized host rock is oxidized and, therefore, the gold is present in both sulfide zones and oxide zones. In the sulfide zone, gold occurs as minute inclusions in pyrite and marcasite, in organic complexes. and as free gold associated with quartz and calcite. In the oxide zone the sulfides and organic matter are destroyed and all of the gold occurs in the native state.

The Size of the native gold ranges from 10 microns to less than one micron in size with the majority of the gold being less than 5 microns in size.

There is no visible guide to gold mineralization since it is often observed that strong hydrothermal alteration, sulfide mineralization and carbonaceous matter may be present without gold.

Age is believed to be pre 31.6 million years. Mercur is a previously mined camp, all present reserves were previously mined. The majority of the drilling was by reverse circulation.

D. Illustrations:

1. None Available
2. See attachment #1.
3. None Available

E. Mining Operations:

1. Actual Daily Production Rate: 25,000 Tons/day

2. Two ten hour shifts per day

3. 355 days per year of operation

4. Open pit mining using the following equipment:

   a. Drilling - 3 DM25 Ingersoll Rand Blasthole Drills
   b. Blasting **
   c. Haulage - 7 85D Wabco Haulpaks, 85 Ton haulage trucks
   d. Other - 2 UH801 Hitachi Hydraulic Excavators (11 yd. Buckets)
      2 IH 570 International Front End Loaders
      2 75B Wabco Water Trucks
      4 TD25E International Dozers
      2 16 G Caterpillar Graders

   ** 20 Feet bench, 6¼" diameter hole, 14 ft. x 16 ft. pattern bulk ANFO

F. Milling Operations:

See attachment #2

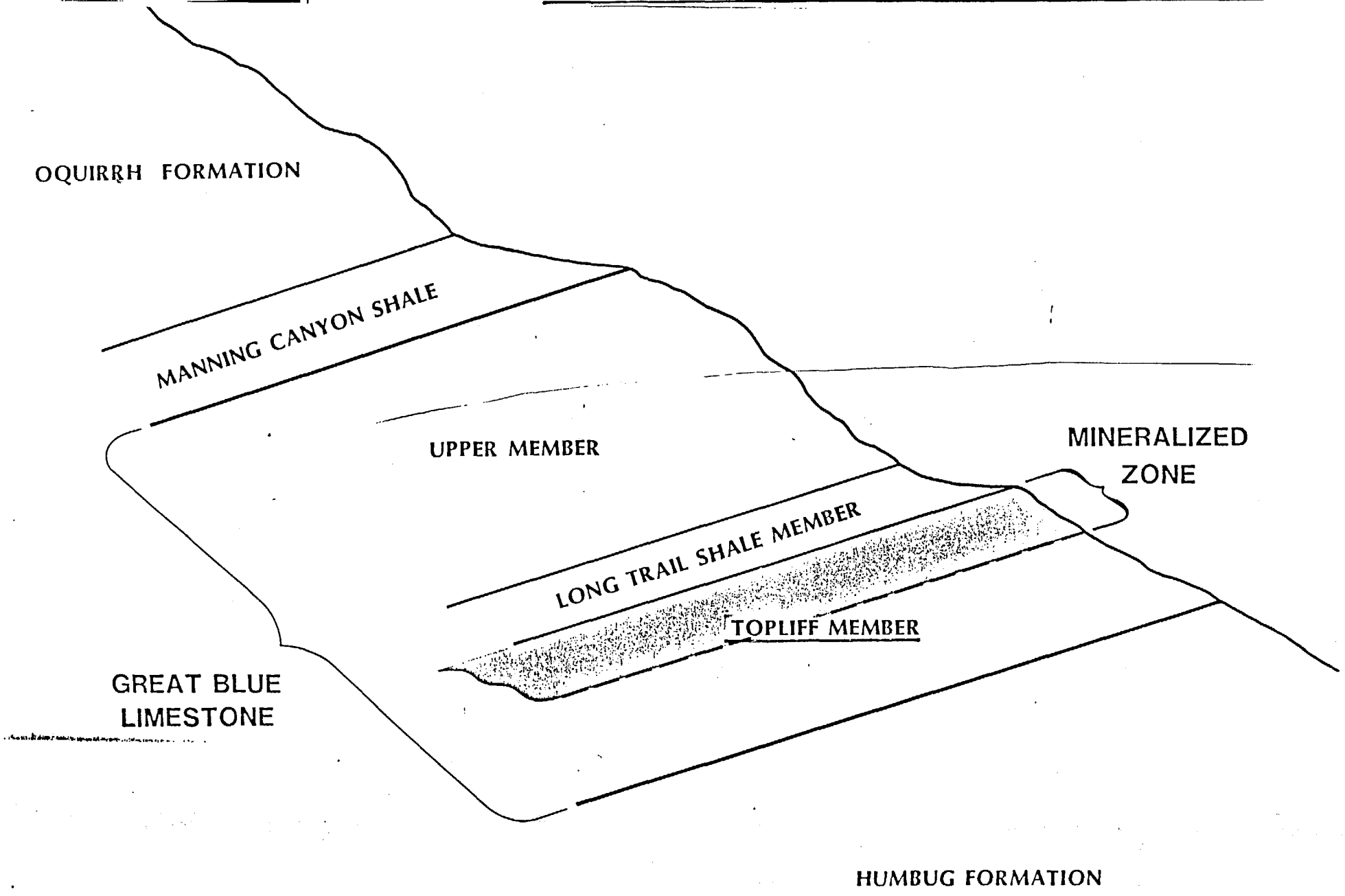
G. Refinery Operations:

See attachment #3

H. As of June 1, 1983 there are 203 employees in the Mine, Maintenance, Mill and Administrative Departments.

Attachment #1

GENERALIZED DISTRICT STRATIGRAPHY



Mill Operations

1. Flowchart see attached flowchart.

a)

	Daily	Monthly
Design Tonnage Rate	3000	87,000

Future Plans:

1. The ability to reject very low grade material from the SAG mill circuit will allow a 25 to 30% increase in the tonnage rate while upgrading the mill feed for easier extraction control.

2. Heap leaching of the low grade ore will provide a further degree of gold recovery. Heap leach testing programs are now underway.

b & c)

	Au oz/t	Ag oz/t	Hg ppm	Tl ppm	As %	Cu %	Pb %	Zn %	Fe %
Head Grade									
Design	.08-.09	.07	35	150	.1	.008	.006	.02	2.2
Recovery									
Design	70-85%								

d) Treatment methods, Crushing: Crusher and Conveyors

Grinding: Sag mill, Ball mill, Cyclones, Primary classifying screen, Thickener.

Leaching: Carbon in leach circuit.

Regeneration: Carbon reactivation rotary kiln.

Bullion: Acid washing of carbon, pre-conditioning of carbon, pressure stripping of carbon, electrowinning, melting.

e) Products: Gold and silver as doré product. Shipped to two refiners.

Mill Operations
Page Two

2. Problems

a) Recovery:

1) Recovered metal from ore at a given grade is as designed. Since the value of the head grade critically affects recovery, low recovery from low head grade material is expected.

2) The refractory (unamenable) characteristics of 10% of the ore causes higher than desired leach tails in some ore body campaigns.

b) Variability of feed:

The head grade, color, hardness, amenability and clay content are constantly changing.

Limestone ore:

More hardness= less grade, less amenability, less clay, dark color.

Shale ore:

Less hardness= more grade, more amenability, more clay, lighter color.

Orpiment and Realgar exist in both types of ore.

c) Oxide:

Oxide ore contains a considerable amount of clay. This fine sticky material causes high viscosity and minor thickening problems.

Sulfide:

Thiocyanate forms preferentially to the aurocyanide complex in the C.I.L. Heavy sulfide availability causes excessive cyanide consumption before the gold can be leached.

Carbon:

The indigenous carbon material, kerogens and asphaltines are prevalent in the refractory or lower amenable ores. They also exist in the oxide ores in smaller amounts. These carbons adsorb leached gold values before they can be adsorbed by the added carbon in the leach circuit. This condition eventually causes lower overall recoveries.

d) Crushing:

Some of the fine ore in wet weather becomes sticky and eventually causes obstruction of the crusher and conveyor chutes.

Grinding:

Variability of the rock content of the ore causes difficulty in ball charging to the SAG mill and eventually impedes operation.

e) Gangue:

Non-mineralized material in the ore does not cause operational milling problems. The quartz and limestone content in the gangue is harder, however, and requires more grinding power and steel consumption. It also reduces the ore grade.

f) Tailings:

Tailings gold values fluctuate with the amenability and head grade of the ore. This fluctuation causes undesired variations in the leach extraction.

g) Other problems:

Grade control remains to be the most serious problem. It must be noted that this is a problem at any mine site.

Machinery and specialized equipment have been modified where necessary and are performing as expected.

Attachment #3

Refinery Operations

Process Description - See Flow Chart

Loaded carbon from the first (or second) C.I.L. tank is pumped to the loaded carbon screen where it is separated from the slurry and washed with water. The slurry and wash is piped back to the C.I.L. feed launder. The washed carbon is discharged into one of two acid wash tanks.

In the acid wash tanks the loaded carbon is washed with a solution of nitric acid to remove calcium and thallium. After washing, the nitric solution is transferred to the neutralization tank where metallic elements are precipitated with sodium sulfide.

Loaded carbon from the acid wash tanks is educted to the preheat tank screen. Dewatered carbon from the preheat tank screen drops into the preheat tank which is partially filled with a caustic-cyanide strip solution. After the carbon transfer is complete, the preheat tank is heated by external coils.

The preheated carbon and caustic-cyanide solution is then dropped vertically into the carbon strip tank where it is heated with steam applied to external heating coils on the tank.

After the carbon strip tank is heated and stripping is accomplished by passing barren solution through the strip column.

The barren solution is preheated by passing it through a series of two heat exchangers. The first heat exchanger uses solution leaving the stripping columns to preheat the strip liquor. The second heat exchanger, using steam further raises the strip solution temperature column.

The solution leaving the column, high in gold concentration, is called pregnant liquor.

Stripped carbon is discharged from the carbon strip tank to the stripped carbon surge tank.

The pregnant liquor is pumped to the electrowin cells where gold, silver, and mercury are plated out on iron wool. The overflow from the cells - spent electrolyte (or barren) is used for elution of the next desorption batch.

The cathode baskets containing the gold are removed periodically from the cells. The iron wool, plated gold, silver and mercury are removed from the cathode basket.

And then placed in a mercury retort boat and put into the retort furnace. Mercury is volatilized in the heating section and is air swept into a condenser where it is condensed to liquid mercury.

The boats are removed from the furnace. And the iron wool, gold, and silver from the retort boats are then added to the crucible in the melting furnace. Fluxes are also added to the crucible. The fluxes combine with iron and other impurities to form a molten slag upon heating. The gold and silver melt and form an alloy.

The slag due to its low specific gravity floats on top of the molten gold-silver alloy.

The contents of the crucible is poured into a cascade series of molds. The gold-silver alloy settles in the bottom of the first mold and the slag floats and overflows the first mold and fills up the following molds.

Upon cooling, the slag and gold bullion can be easily separated. The solidified slag is brittle and can be broken up. The gold bullion is weighed and stored in the safe prior to shipment.

Mercury from the condenser will be collected in flasks and stored in the safe prior to shipment.

PROBLEMS

1. Wood Pulp in the ore comes over with the loaded carbon and causes numerous problems such as plugging tank overflow screens and lines.

2. The metal loading on cathodes is very critical for smelting. A lot of the iron will report to the doré and will not slag off if the loading on the cathodes is light.

3. Excess cyanide concentration in the strip solution causes wood pulp to turn into a very sticky material which in turn causes plugging of the strip column discharge line.

4. Insufficient cyanide and caustic concentrations in the electrowin cell feed causes poor plating on the steel wool, resulting in high barrens.

5. The metal loading on the carbon does not seem to affect the stripping efficiency. Lightly loaded carbon can cause wasteage, however, since it takes the same amount of time and chemicals to treat a lightly loaded batch of carbon as it does a heavily loaded batch.

REFINERY OPERATIONS

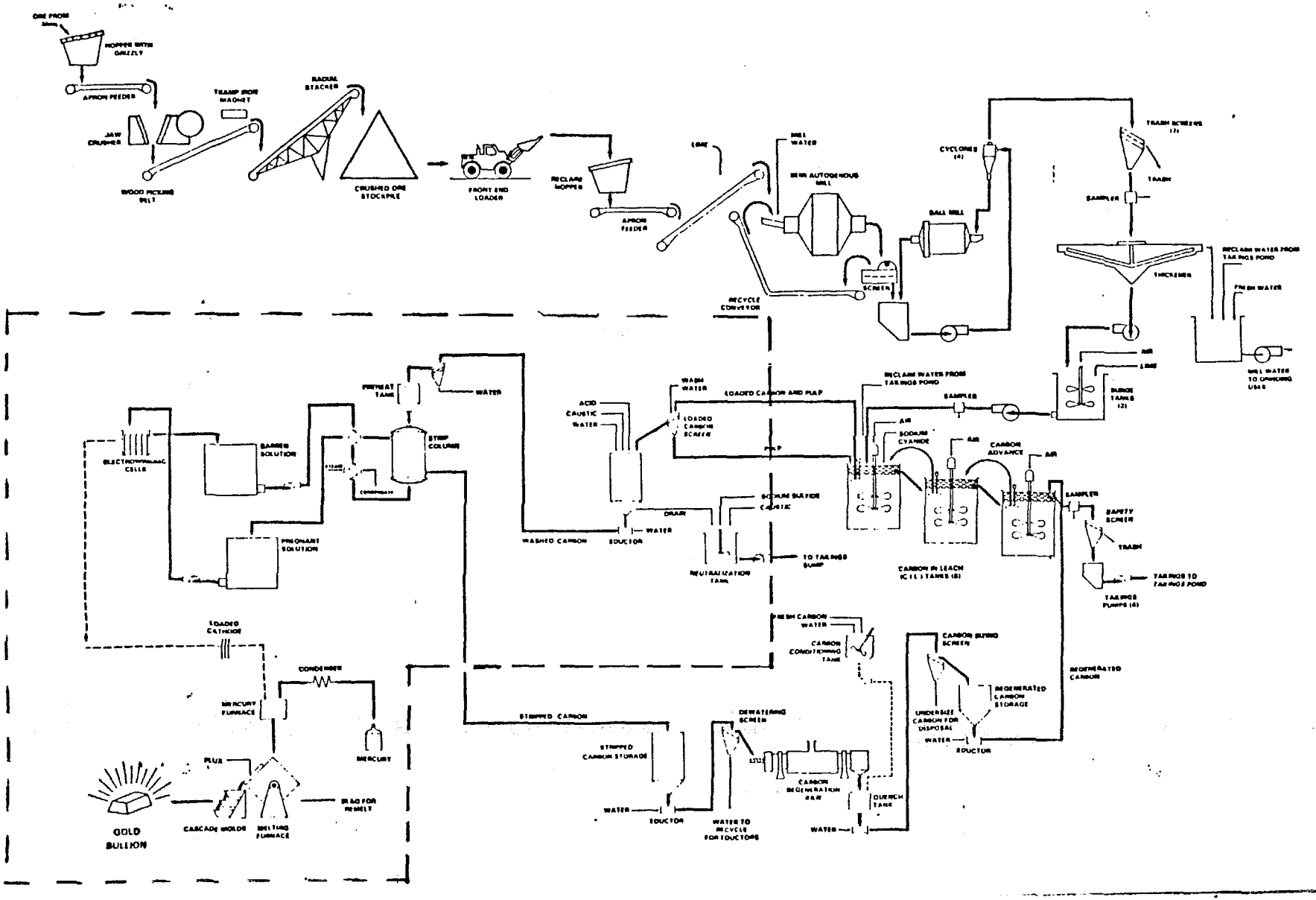
1. Operating Time,	Days per year	250
	Days per week	5
	Hours per day	16

REFINERY OPERATIONS

1.	Operating Time, Days per year	250
	Days per week	5
	Hours per day	16
2.	Carbon Loading, ounces gold per ton	
	Loaded carbon (heads)	75
	Stripped carbon (tails)	2
3.	Carbon Tonnage per day	5.38
4.	Daily Batch of Pregnant Liquor, Gals.	18,800
	Content of Metal	
	Gold, oz.	393
	Silver, oz.	25
	Mercury, lbs.	2
	Spent Electrolyte (Barren Sol.), oz/ton	0.30

MERCUR GOLD PROJECT

FLOWSHEET



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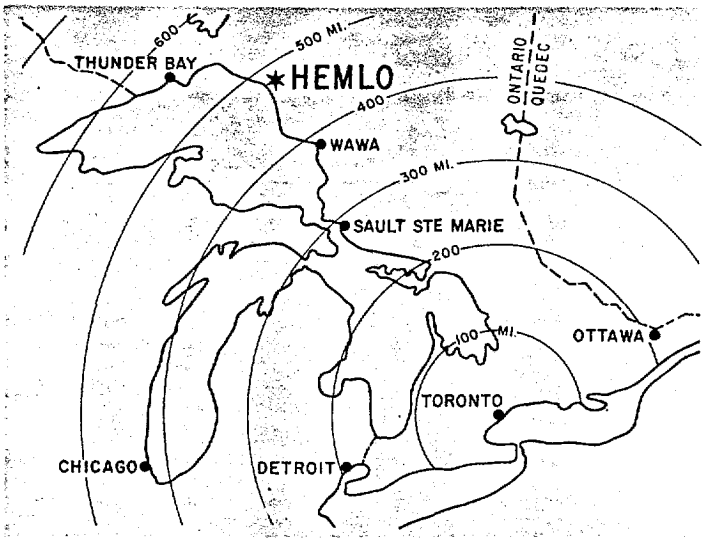

Noranda Proceeding with Golden Giant Gold Project

Major Canadian gold mine under development for 1985 start up in expenditure of more than C$200 million at Hemlo, Ont.

IN THE NEW HEMLO GOLD mining district, 155 miles east of Thunder Bay, Ont., the Golden Giant joint venture owned by Noranda Mines Ltd., Goliath Gold Mines Ltd. and Golden Sceptre Mines Ltd. is proceeding with a project requiring a capital expenditure exceeding C$200 million to develop the Golden Giant #1 deposit. This major development, situated less than 1 mile north of Trans-Canada highway 17 and 25 miles northeast of Lake Superior, will approach the level of Canada's largest gold producer, Campbell Red Lake at Red Lake, Ont., that had an output of 217,000 oz. in 1982.

At a site 30 miles north of Hemlo, the Geco division of Noranda has been active in production of copper, zinc and silver for about 25 years. During this period some exploration was conducted by the company at Hemlo.

Since Nov. 1982, Noranda Mines through its wholly-owned subsidiary Noranda Exploration Co. Ltd., has outlined more than 18 million metric tons of ore grading 9.54 grams of gold per ton. The ore also contains molybdenum as well as barite but silver is negligible. An initial 1000-ton per day production is planned for early in 1985, gradually increasing to 3000 tons per day by 1987.

Diamond drill delineation of the orebody is continuing simultaneously with the construction of a mill and the sinking of a 5-compartment production shaft to 1000 meters for completion by Nov. 1984.

At Golden Giant, ore will be crushed underground, and coarse ore will be loaded into skips for hoisting to surface in the main shaft, which will be equipped with a secondhand hoist used previously at Bethlehem Steel Corp.'s Grace iron ore mine in Morgantown, Pa. On surface, coarse ore will be reduced further in secondary and tertiary crushers, with fine ore fed to the processing complex including grinding, flotation, carbon-in-pulp, and

electrowinning facilities. Final product will be refined gold in the form of 31-kilogram bars.

History of Hemlo Exploration

The Hemlo-Heron Bay area has a history dating from 1869 in Heron Bay, and from 1920 in the Hemlo area. The station agent at Hemlo, J. LeCour, was the earliest recorded prospector in the area; he dug some trenches just north of the Hemlo station in the 1920s.

Geological work was conducted in 1931 and 1932 by J.E. Thomson of the Ontario Department of Mines, Geological Survey, who brought the area to the attention of the mining industry. He recommended two areas for exploration: Manitouwadge and northeast of Hemlo.

In 1944, Peter Moses of Heron Bay first discovered gold in what is now known as the Williams option, held by Long Lac Mineral Exploration Ltd. This discovery precipitated a minor staking rush in the district.

During the summer of 1947, Lake Superior Mining Corp. Ltd. performed a limited drill program of 13 holes and outlined a 900-ft. long zone averaging 0.25 oz. per ton of gold across a 6.5-ft. width..

Exploration and geological mapping were also conducted during 1947, and a report and geological map by Trevor W. Page in April 1948 found the results encouraging. He called for additional work on this main zone and a secondary zone a short distance to the south. Mr. Page's geological maps are valid to this day.

Between 1947 and 1951, the company completed 6000 ft. of drilling, and in 1951 the property was optioned to Teck-Hughes Gold Mines Ltd., which outlined 89,000 tons of 0.27 oz. of gold per ton. After additional drilling, the company was dissolved in 1965.

A geological report and map of the Hemlo area, covering a broader part of the synclinal structure, between the Black River and White Lake, was prepared by Trevor W. Page and approved by M.W. Bartley. It was prepared for the department of industry and development of Canadian Pacific Ry., dated Dec. 1, 1957.

After 1965, the Williams claims were restaked and worked a number of times, and in each case allowed to lapse.

In March 1980, two Timmins prospectors, John Larch and Don McKinnon, staked the now well known Lake Superior shear zone, subsequently optioning the claims to M. Pezim of Corona Resources in Oct. 1980. The successful drill program supervised by David Bell, consulting geologist, which followed, and coupled with major discoveries by both Goliath-Golden Sceptre and Long Lac in 1982, initiated one of the major staking rushes in recent Canadian mining history. In excess of 5000 claims were staked in the Hemlo camp, and an additional 10,000 claims have been staked elsewhere in northern Ontario.

Three key properties have been outlined in the camp:

1. Goliath-Golden Sceptre—Noranda-Golden Giant joint venture; deposit #1; about 62 hectares.
2. Corona Resources—Teck-Corona option; east zone and west zone; about 277 hectares.
3. Williams Patents—Lac Minerals; Williams deposit; about 159 hectares.

Williams Patents

The property, comprising 11 patented claims, was originally staked in 1945 by Mr. Ollmann of Heron Bay on the original Moses gold discovery made one year earlier near Moose Lake. Mr. Ollmann, in partnership with Dr. J.K. Williams of the state of Maryland, worked the ground between 1945 and 1948, eventually patenting the group. The claims were optioned by the Williams family to Lac Minerals in 1981. Subsequent drilling has outlined a significant gold deposit containing in excess of 4 million tons of ore grading 0.20 oz. of gold per ton.

International Corona Resources

This group originally was staked in 1945 by T. Page, M. Bartley, M. Fisher, A. Halliday and J.K. Williams. Lake Superior Mining Corp. Ltd. was then formed in 1947 to evaluate the property. It was dissolved in 1965 as previously noted.

Six separate ventures held the claims from 1965 to 1980 when D. McKinnon and J. Larche restaked the ground subsequently optioning the claims to Corona in Oct. 1980. In addition to further testing of the west zone, Corona's evaluation outlined the previously unknown east zone, the main ore zone on the property. The property was optioned to Teck Corp. in Nov. 1981. Initial reserve estimates ranged from 1.5 million to 2 million tons in June 1982. These reserves have been subsequently increased during the 1983 drill program.

Goliath-Golden Sceptre

Both properties were initially staked in 1945, Goliath by T. Page and Sceptre by C. Edwards. The Page claims were incorporated into the Lake Superior Mining holdings and patented in 1959. The patents were subsequently allowed to lapse, and the ground was eventually restaked in July 1980 by Messrs. McKinnon and Larche, who were grubstaked by H. Bonhomme and R. Schirelli. The Edwards claims lapsed in 1950 and the group was acquired by various interests during the period from 1960 to 1979. In Jan. 1980, the claims were restaked by McKinnon et al. The McKinnon-Larche claims were then optioned by Golden and Goliath of the Vancouver-based Hughes-Lang group, who combined both properties into the Molson Lake joint venture.

The Molson Lake joint venture under the direction of D. Bell & Associates conducted an intensive exploration program on the properties between Jan. and Nov. 1982. These efforts along with those of Lac Minerals on the adjoining Williams claims, resulted in the discovery of the Goliath deposit. Preliminary drill results on the downdip extension of the Williams zone were impressive. In Nov. 1982, a major joint venture between Goliath, Golden Sceptre and Noranda Exploration was made to place a 1000-ton per day mine into operation by Nov. 1984.

(Continued on page 6)

(LEFT) View of equipment used to clear plant site area at Noranda Mines Ltd.'s Golden Giant gold project at Hemlo, Ont. (RIGHT) Scene at Noranda Mines Ltd.'s Golden Giant gold project at Hemlo, Ont., showing exploration with diamond drill.

Noranda Proceeding with Golden Giant Gold Project

(Continued from page 5)

General Geology

The Hemlo gold deposits are situated within the Heron Bay metavolcanic belt of Archean age, 2.6 - 2.8 billion years. This succession of metamorphosed volcanic and sedimentary rocks is within the Abitibi-Wawa-Shebandowan subprovince of the Canadian shield. This same geological subprovince hosts the world class gold deposits of the Timmins, Val d'Or and Kirkland Lake camps and major base metal deposits of the Noranda and Mattagami Lake regions of Canada.

The mineralized formations outlined to date have been found on the south limb of the Hemlo syncline. This is an easterly plunging volcanic-sedimentary basin in the Heron Bay greenstone belt. The Hemlo basin or syncline was developed during deformation and metamorphism of the original volcanic-sedimentary assemblage in late Archean time. This syncline has been intruded by a series of younger granitic plutons and stocks. The largest of these late intrusives, the Cedar Lake Pluton, outcrops one mile north of the deposits. The Herrick Lake and Gowan Lake gneiss domes flank the syncline to the south and north, respectively.

The hinge zone or axis of the syncline has been highly sheared. A major northeast trending fault zone, the Lake Superior shear zone, bisects the basin along the Black River Valley, and is truncated by the Cedar Lake Pluton.

The Heron Bay greenstone belt can be subdivided into eastern and western domains. East and north of Hemlo, clastic metasedimentary rocks predominate. In contrast, the western half of the belt is underlain by coarse volcaniclastic or pyroclastic units. In overview, the belt is thought to have developed from an active volcanic pile near Heron Bay, which wedges out eastward through the deposit site into a major sedimentary basin.

The stratigraphy of the deposit can be broadly subdivided into three distinct east-west trending and steeply north dipping formations that have locally attained an upper amphibolite rank of metamorphism.

These formations are:

1. Playter Harbour formation. A basal succession of mafic to intermediate volcanics in which the base of the formation has been destroyed by intense granitization accompanying the development of the Herrick Lake gneiss dome.

2. Goliath formation. A central diverse group of felsic volcaniclastics, quartz and feldspar porphyries, chemical and clastic metasediments. This group of rocks is the host for the Hemlo stratabound ore deposits. The felsic volcanic members of this formation are calcalkaline in composition and may indicate renewed volcanism in the original Hemlo basin.

3. Cedar Creek formation. An upper thick succession of clastic metasediments directly overlying the ore deposits. The unit contains volcaniclastic interbeds and rapid increases in thickness to the east.

All the gold mineralization delineated to date in the camp has been found within the upper members of the Goliath formation near the contact with the overlying clastic metasediments of the Cedar Creek formation.

The mineralization can be further seen to have developed at a point where the felsic volcaniclastics of the Goliath formation are thinning rapidly eastward, essentially pinching out. These upper units of the Goliath formation appear to mark a significant synvolcanic mineralizing event.

Deposit #1 Geology

Deposit #1 strikes east-west 115° and dips north at 60° to 70° along a defined strike length of 400 meters.

The mineralized gold zone is open along strike and at depth below 1000 meters. Extensions at depth and to the east of the main zone are being tested by a major diamond drill program. The probability of increasing the ore reserves is considered to be good.

The main ore zone is composed of a diverse group of stratabound, pyrite-rich, siliceous and locally barite-rich metasediments. They are of probable chemical sedimentary derivation. The mineralized zone ranges from 3 to 40 meters and is characterized by sheet-like regularity and by remarkably consistent gold values varying from 4 to 16 grams per metric ton.

Gold occurs as finely disseminated native gold in the pyritiferous schists. The grade is not dependent on iron sulfide content.

The ore-bearing rocks contain appreciable molybdenite. Accessory minerals such as stibnite, realgar, and a light green mica thought to contain titanium, vanadium and/or barium, are characteristic of the deposit. The deposit displays a distinct vertical zoning of layering.

Hanging wall sediments grade downward into a molybdenum-rich siliceous schist, then into mixed siliceous, baritic, and locally sericitic schist. Then into a basal lower grade barite-rich horizon. A second subeconomic mineralized zone has been outlined from 30 to 80 meters stratigraphically below the main ore horizon. This lower zone has erratic gold values and rarely exceeds 4 grams per metric ton; however, the internal mineral zoning in the unit is comparable to that of the main deposit.

The main ore horizon can be traced laterally along strike into a pyritiferous volcaniclastic unit. This distinctive unit has stretched quartz-feldspar-porphyry fragments and fewer metasediment and cherty clasts, all set a quartz-feldspar-biotite matrix. It has high background gold values.

The deposit is underlain by an intensely altered succession of quartz-eye sericite schists with variable amounts of tourmaline, pyrite, rutile, and often abundant green mica. A series of feldspar and quartz porphyry sills and dikes intrude both the ore horizon and footwall schists.

The youngest intrusives in the area are diabase dikes that strike roughly north-south and have a steep easterly dip.

Mineralogy

The following table lists both major and minor minerals found within the deposit. Major minerals are in bold type.

SILICATES, SULFATES, CARBONATES, OXIDES

Quartz-SiO_2: total SiO_2 = 55%, 40%-70% range

Barite-$BaSO_4$: 12% average, 0-40% range

Micas-K, Al, Fe, Mg, (V), (Ti), OH, silicates: 5-30%

Feldspars-K, Na, Ca, Al, (Ba), silicates: 0-15%

Calcite-$CaCO_3$: 0-15%

Rutile-TiO_2, and Sphene-$CaO.TiO_2.SiO_2$ = 0.3-0.4%

Chlorite, Amphibole, -Fe, Mg, (Al), (Ca), (Na), OH, silicates: moderate to minor

Tourmaline, Diopside, Sillimanite, Garnet, Staurolite, Epidote, - silicates: minor

COMMON SULFIDES AND GRAPHITE

Pyrite-FeS_2: 8% average, 5-20% range

Molybdenite-MoS_2: 0.16% average, 0.03-0.40% range

Sphalerite-ZnS: Zn = 0.008% average, 0.003-0.024% range

Chalcopyrite-$CuFeS_2$: Cu = 0.002% average, 0.001-0.004% range

Pyrrhotite-FeS_X: traces

Graphite-C: 0.04% average, trace - 0.12% range

GOLD-BEARING MINERALS

Native Gold-Au (Ag = 0-20%): Au = 10 grams/ton

Aurostibite-$AuSb_2$: several grams/ton in part of drill hole NGG-14

ANTIMONY, ARSENIC AND MERCURY-BEARING MINERALS

Stibnite-Sb_2S_3: minor but widespread; Sb = 0.02% average, 0.01 - 0.16% range

Native Antimony-Sb: traces

Berthierite-$FeSb_2S_4$: traces

Gudmundite-FeSbS: traces

Jamesonite-$FePbS_2$: traces

Boulangerite - $Pb_5Sb_4S_{11}$: traces

Chalcostibite-$CuSbS_2$: traces

Tetrahedrite - $(Cu, Fe, Zn, Ag, Hg)_{12}(Sb, As)_4S_{13}$: minor to traces; 0-2% Ag

Arsenopyrite-FeAsS: trace

Native Arsenic-As: trace

A. Gold

Native gold occurs typically as fine disseminated grains on silicate mineral grain boundaries, is rarely associated with pyrite and is not known to occur with arsenopyrite. Visible gold is rare. Gold grades, though typically uniform, are higher in siliceous units of the ore zone.

B. Molybdenum

Molybdenum is found throughout the deposit but like gold, is most common in the siliceous units of the deposit. Again, the association is with silicate minerals along grain boundaries.

C. Barium

Barite is the major barium mineral in the deposit. It is most common at the base of each of the ore zones. The volume content ranges from 10% to 40%. A barite-gold association is a feature unique to Hemlo because it is part of the original geological environment.

D. Silver

The silver content of the deposit is low; averaging 1.5 grams/ton. The low silver values result in high gold to silver ratios, varying from 2:1 to 20:1.

Ore Reserves of Deposit #1

The current ore reserves for deposit #1 are estimated to be 18.5 million metric tons of ore grading 9.54 grams/ton gold, 0.10% molybdenum, 1.5 grams/ton silver, and 10% to 13% barite. The general parameters used in defining these reserves are as follows:

A. Cutoff grade—2.7 grams/ton gold

B. Minimum mining width—3 meters

C. Density—3.0 tons/cubic meter

D. Dilution—10% of zero grade

Ore reserve tonnage and grade is computer calculated at Noranda Mines' Don Mills data center. The program measures longitudinal areas of blocks, utilizes horizontal thickness, specific gravity, gold grade and other limiting constraints. Each block has a gold grade that represents the average assay value of a drill hole piercing that block. The maximum radius of influence from a drill hole intersection is 150 meters.

PLATTCO

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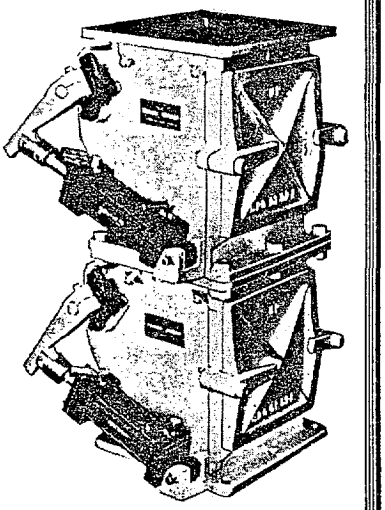

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October 19, 1983

FILE MEMO

PGE in Magnetite

Under Mineralogical Research the occurrence of platinum-group elements in magnetite from placer deposits at Goodnews Bay, Alaska (Geological Survey P.P. 1275, 1981). Analyses showed platinum in amounts as high as 1,100 ppm (35 oz./ton).

The authors state that "grain sizes of the various isometric phases range from less than one micron to about 65 microns across; the phases are irregular inclusions in magnetite grains that are 200 to 400 microns across." They conclude that "such PGE held in the magnetite lattice may represent a valuable resource."

Assuming recovery could be economic, there are many other placer occurrences where the magnetite may contain platinum, such as in the Tulameen river west of Princeton, British Columbia. Platinum metal deposits world-wide are described in USGS P.P. 630.

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difficulties in securing a market for its share. Reynolds Metals is expected to take up its 40% share of production for use at its U.S. smelters. Other shareholders are C. Itoh, Kobe and Nissho Iwai.

Cyanide Removal Process

Inco Ltd. has announced completion of licensing agreements with two more Canadian gold producers with regard to its SO_2/air cyanide removal process for effluent treatment. Carolin Mines will utilize the process at its Ladner Creek operations and Du Pont of Canada Exploration, will implement the method at its Baker mine facility. Scottie Gold Mines was licensed to use the method in 1982. All three operations are located in British Columbia. The process reduces total cyanide concentrations in mill tailings and should achieve operating savings because of lower reagent costs. In Ontario, the McBean mine project near Kirkland Lake — a joint venture between Inco and Queenston Gold Mines — will also be using the method when fully operational early next year (*MJ*, May 27, p.362).

Mesquite Update

According to a review of activities, published recently by Gold Fields America Corp., part of the Consolidated Gold Fields PLC group, the feasibility study being conducted upon the Mesquite gold deposit in Imperial County, California, will be completed in the second half of fiscal 1984.

Operating subsidiary, Gold Fields Mining Corp., which owns the mineral rights to more than 16,100 ha in the area, announced the establishment of a preliminary ore reserve of some 26 Mt at a grade of 2.4 g/t late last year (*MJ*, November 12, 1982, p.346). Since that time additional drilling has been conducted and a decline has been driven into the flat lying orebody enabling bulk samples to be taken for metallurgical and heap leach testing.

Completion of testwork and the feasibility will cost a further $6 million and will determine the most economic mining and treatment techniques.

PRODUCTION

Key Lake Production Start-Up

Destined to become the largest non-communist uranium operation, the Key Lake, Saskatchewan, mine entered commercial production at the end of September, at a cost of $C500 million. At full capacity, the mine will be producing 5.4 Mkg/y (*MJ*, July 8, p.32) and employing between 400 and 500 workers. It has a minimum estimated life of 15 years and recent drilling has indicated considerably more reserves.

The Saskatchewan Mining Development Corp. is the largest shareholder with 50%, Uranerz Exploration and Mining holds 33.3% and the remaining 16.7% is held by the Canadian Crown company, Eldorado Nuclear, through its subsidiary.

Rain Affects Copper Mining

Kennecott Corp.'s Ray Mines, Arizona copper division has been forced to close down because of heavy rain. Three other copper companies have reported reduced operations because of the resultant severe flooding. Railways and roads serving Phelps Dodge Corp.'s Morenci and Ajo facilities in the state were blocked by mud — ore deliveries to concentrators have been reduced to about 50% of capacity; mud slides have been given as the reason for Duval's partial shutdown of the Sierrita facility, although there has been a three day old strike affecting the mine. Asarco Inc. has suspended operations at its Hayden copper smelter in Arizona due to water supply problems ironically brought on by the flooding and 200 workers have been laid off for at least one week. Production is expected to resume when the smelter's water supply is restored.

First Gold from Detour Lake

The first gold bar has been produced at the Detour Lake gold mine in Canada which is jointly owned by Campbell Red Lake Mines and Amoco Canada Petroleum. Work has also started on the development of an underground mine scheduled for start-up in 1987 (*MJ*, July 29, p.75).

Dead Sea Potash

A feasibility study for the possible expansion of the Dead Sea extraction plant at the Arab Potash Co. operations in Jordan has been completed by Davy McKee, George Wimpey & Co. and Sir Alexander Gibb & Partners. The study suggests that doubling the plant's capacity would cost about $100 million.

First Production from Embalut

First production from the Embalut coal field in East Kalimantan, Indonesia, has been shipped to Taiwan. The 5,000 t shipment of steaming coal has gone to the Hsingta Cement Co. Ltd. and Far East Soda Co., a chemical producer.

Development of the Embalut coal field began in 1980 and production started in June this year. Reserves have been put at 20 Mt with most of the output going to Taiwan.

Initially an output of 15,000 t/month of steaming coal is planned by the joint venturers which comprise Jadestone Industrial Co. of Taiwan and P. T. Kitadin Corp. of Indonesia. Output will ultimately rise to 50,000 t/month.

Clarendon Refinery Unit to be Reactivated

Alcoa Minerals of Jamaica is to reactivate a second bauxite refining unit at its Clarendon alumina works in Jamaica in the middle of next year. Start-up of the unit will raise the plant's alumina output to more than 500,000 t from the current level of 365,000 t/y.

Finnish Barytes

Outokumpu Oy has reported that barytes concentration has begun at the Pyhasalmi copper-zinc-pyrite mine in Finland on a trial basis. The ore contains about 4% barytes, but in the past the $BaSO_4$ contained in the ore has not been recovered. About 30,000 t of barytes is contained in the annual output of ore from the mine. This year about 5,000 t of barytes concentrate will be recovered. If this proves economical commercial production will begin.

Czech Coal Output

Coal production in Czechoslovakia in 1983 is reported to be on target although costs of mining coal in the Ostrava mining area of northern Moravia are expected to rise in the future.

Last year output totalled 124.5 Mt, 558,000 t more than originally planned.

LABOUR

Mt. Newman Strike Over

Last week we pointed out that striking unions at the Mt. Newman operations in Western Australia were under severe pressure to return to work. This followed an ultimatum from the WA Industrial Commission and the substitution by Japanese customers of Indian iron ore for Newman ore, after Mt Newman declared *force majeure* on shipments through Port Hedland (*MJ*, September 16, p.208). The workers have now gone back to work and operations are returning to normal.

The strike originated with demands for a shorter working week and guaranteed employment for graduating apprentices. These matters are being settled by arbitration.

Bolivian Strike Losses

The strike by Comibol miners at tin operations in Bolivia has led to production losses of about $800,000 so far. Nearly half of Bolivia's 26,000 miners have responded to the strike call which has been staged to further demands for food supplies, machinery replacements and payment of a 41% salary increase (*MJ*, September 30, p.244).

Canadian Tungsten Pact

A tentative agreement has been reached with Canada Tungsten Mining Corp. and its employees for new labour contracts at its mining operation at Tungsten, Northwest Territories, Canada. The agreement has to be ratified by members of the United Steelworkers Union.

The mine closed down in January because of unfavourable market conditions but the company expects production to follow contract ratification (*MJ*, April 22, p.272).

Strike Threat Lifted

A work stoppage planned at Southern Peru Copper Corp.'s Cuajone copper mine in Peru has been suspended by workers. No details of settlement terms are available but workers at the Toquepala copper mine and Ilo smelter remain on strike (*MJ*, September 30, p.239).

GOVT. MEASURES

U.S.S.R. Wins Nickel Case . . .

According to *Reuters*, the U.S.S.R. mineral export organization Raznoimport

Gold production in the Superior Province of the Canadian Shield

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ABSTRACT

The Superior Province of the Canadian Shield is the largest gold producing area within Archean greenstone rocks in the world, with total production of over 140 million ounces of gold. One hundred and fifty-five mines have individually produced more than 8,000 ounces, although only a relatively limited number of deposits (33) produced more than one million ounces of gold. The output from these 33 deposits accounts, however, for 85% of total gold production. The majority of grades (68%) ranges between 0.10 and 0.30 ounce per ton, and there is no linear relationship between grade and tonnage.

Development of new gold mines reflects the history of gold prices. However, the response to the latest increase in gold prices was not as marked. It is concluded that gold deposits are becoming increasingly more difficult to find and new exploration techniques are urgently required.

Introduction

With most of its production coming out of the Superior Province, the Canadian Shield is the largest producer of gold from Archean greenstone rocks in the world. In total gold production, the Superior Province is surpassed only by the paleoplacers of Witwatersrand and the placers of the USSR (Anhaeusser, 1976). There has been no attempt so far to compile the historical production of the Province as a whole in a single document. The aim of this work is to fill this gap, not only for the convenience of having a complete list of gold producers, but, more importantly, as a basis on which to study the tonnage and grade characteristics of these deposits.

This comprehensive compilation started in 1979 (Bertoni, 1980) and involved the examination of many documents from various sources: annual reports of many corporations, issues of the *Northern Miner, Canadian Mines Handbook, CIM Bulletin, Canadian Mining Journal* and other publications; for Ontario, most of the information was taken from Ferguson *et al.* (1971), Beard and Garratt (1976), and Gordon *et al.* (1979); for Quebec, information was taken from the "Fiche de Gîtes" descriptions of the Ministry of Energy and Resources; for Manitoba, the reports by Stockwell (1937 and 1938) and Stewart (1980) were consulted. Other important sources included the mineral deposits file of the National Minerals Inventory Section of Energy, Mines and Resources Canada, and their report, "Canadian mineral deposits not being mined in 1980" (1980).

Two major problems were faced during this compilation: (1) some of the information is not available or has been irretrievably lost; and (2) some of the information conflicts when different sources are compared. In the second instance, the author used his judgment as to which source is more reliable. As a consequence, the end product is believed to be the best available compilation of its kind.

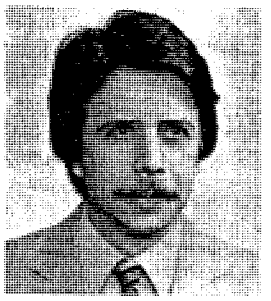
It is important to note that this compilation includes only deposits in which gold has been mined as the main commodity. It excludes gold that has been produced as a co-product from various base-metal deposits, such as Horne and Quemont in Noranda, Quebec. If the gold production from base-metal deposits is considered, the position of the Superior Province as a gold producer becomes even more enhanced.

Gold Production

Table 1 is a list of 155 gold mines of the Superior Province with production greater than 8,000 ounces of gold. This threshold was arbitrarily chosen mainly because the information about the very small producers is unreliable (there are about 55 mines with production of between 1,000 and 8,000 ounces). The some 140,840,000 ounces of gold produced by these 155 mines does not, therefore, represent the total gold production of the Superior Province, although it certainly comprises the majority of it. The list is organized in order of decreasing tonnage of ore milled to the end of 1981, and shows also location, periods of production, and recovered ounces of gold and grades. In many instances, operations were intermittent and the production period does not necessarily represent the actual number of producing years. The initial date recorded for each mine is that of the start of regular production, not the discovery date.

Table 1 also includes (in parentheses) published reserves. These are proven for current producers and estimated for former producers. It is apparent that the producing gold operations are working with fairly small proven reserves. There is little information that might be used to estimate the actual life expectancy of these mines.

Table 2 lists the distribution of gold deposits in the Superior Province in several intervals of size (tons of ore milled) and recovered grade over the mine life. Figure 1 shows the same deposits distributed into eleven intervals of total gold produc-



Carlos H. Bertoni

Carlos H. Bertoni was born and raised in Brazil, where he received his B.Sc. in geology at the University of Brasilia in 1974. After graduation he worked as an exploration geologist for Cominco Ltd. in Brazil until 1978. In 1980 he obtained an M.Sc. (mineral exploration) degree from Queen's University and has been working for Riocanex as a research geologist since then.

Keywords: Gold, Gold deposits, Gold production, Superior Province, Canadian Shield.

TABLE 1. Gold mines of the Superior Province (production to end of 1981)

Mine	Location[1]	Production Period	Ore Milled & Reserves[2] (tons)	Gold Recov. (oz)	Recov. Grade (oz/ton)
Hollinger	Tisdale Tp., Porcupine	1910-68	65,890,358	19,354,483	0.29
		1976-81 (tail.)	1,094,500	66,680	0.06
			(292,000)		(0.124)
Dome	Tisdale Tp., Porc.	1910-	39,355,914	10,625,383	0.27
			(2,146,716)		(0.207)
Kerr Addison	McGarry Tp., Larder Lake	1938-	36,156,631	9,988,743	0.28
			(682,194)		(0.147)
McIntyre (Schumacher)	Tisdale Tp., Porc.	1912-	36,044,627	10,641,658	0.30
			(196,000)		(0.130)
Pamour (No. 1)	Whitney Tp., Porc.	1936-	26,755,031	2,918,901	0.11
			(1,282,000)		(0.081)
Lamaque	Bourlamaque Tp., Quebec	1935-	25,626,347	4,595,269	0.18
			(293,000)		(0.109)
East Malartic	Fournière Tp., Q.	1938-81	19,938,303	2,852,250	0.14
			(1,062,500)		(0.09)
Sigma	Bourlamaque Tp., Q.	1937-	18,725,127	3,207,911	0.17
			(1,222,900)		(0.195)
Lake Shore	Teck Tp., Kirkland Lake	1918-65	16,620,903	8,499,199	0.51
			(330,000)		(0.42)
Can. Malartic	Fournière Tp., Q.	1935-65	10,941,950	1,196,601	0.11
			(304,000)		(0.09)
Beattie-Donchester	Duparquet Tp., Q.	1933-56	10,614,421	1,138,719	0.12
			(2,270,000)		(0.13)
McLeod-Cockshutt	Ashmore Tp., Geraldton	1938-68	10,337,229	1,475,728	0.14
			(837,000)		(0.10)
Malartic Gold Fields	Dubuisson Tp., Q.	1939-65	9,875,175	1,702,453	0.17
			(150,000)		(0.18)
Barnat	Fournière Tp., Q.	1938-70 & 79-81	9,749,128	1,244,083	0.13
			(110,000)		(0.07)
Wright-Hargreaves	Teck Tp., Kirk. Lake	1921-65	9,734,402	4,817,680	0.49
Teck-Hughes	Teck Tp., Kirk. Lake	1917-68	9,686,825	3,688,664	0.38
Campbell Red Lake	Balmer Tp., Red Lake	1949-	8,423,643	5,104,129	0.61
			(2,250,200)		(0.617)
Madsen	Baird Tp., Red Lake	1938-74	8,371,631	2,416,609	0.29
			(127,000)		(0.19)
Aunor (Pamour No. 3)	Deloro Tp., Porc.	1940-	8,294,574	2,479,894	0.30
			(248,000)		(0.150)
Camflo	Malartic Tp., Q.	1965-	6,642,690	1,353,545	0.20
			(2,321,076)		(0.130)
Preston	Tisdale Tp., Porc.	1938-68	6,284,405	1,539,355	0.24
Young-Davidson	Powell Tp., Matachewan	1934-56	6,128,272	585,690	0.10
Paymaster	Tisdale Tp., Porc.	1915-66	5,607,402	1,192,206	0.21
			(795,000)		(0.23)
Ross	Hislop Tp., Porc.	1936-	5,560,031	877,331	0.16
			(591,000)		(0.104)
Macassa (+ Tegren)	Teck Tp., Kirk. L.	1933-	5,551,492	2,465,455	0.44
			(781,394)		(0.45)
Sullivan Consol.	Dubuisson Tp., Q.	1934-68	5,085,518	1,184,926	0.23
			(344,000)		(0.22)
Sylvanite	Teck Tp., Kirk. L.	1927-61	5,049,536	1,667,520	0.33
Ankerite	Deloro Tp., Porc.	1926-53&78	4,993,929	957,292	0.19
Upper Canada	Gauthier Tp., Larder L.	1938-71	4,734,617	1,398,291	0.30
			(640,000)		(0.26)
Dickenson (+Robin R.L.)	Balmer Tp., R.L.	1948-	4,731,199	2,190,523	0.48
			(1,023,900)		(0.27)
Howey	Heyson Tp., R.L.	1930-41	4,630,779	421,592	0.09
San Antonio	Rice Lake, Manitoba	1932-68 & 82-	4,472,921	1,199,878	0.27
			(882,228)		(0.19)
Coniaurum	Tisdale Tp., Porc.	1913-18 & 28-61	4,464,006	1,109,574	0.25
Hallnor (Pamour No. 2)	Whitney Tp., Porc.	1938-76 & 81	4,226,419	1,645,892	0.39
			(50,000)		(0.098)
Delnite	Deloro Tp., Porc.	1937-64	3,847,364	920,404	0.24
			(109,032)		(0.24)
Renabie	Leeson Tp., Algoma	1947-70 & 82-	3,699,992	780,665	0.22
			(1,125,900)		(0.211)
Matachewan Consol.	Powell Tp., Matach.	1934-54 & 79-81	3,631,086	378,101	0.10
			(1,000,000)		(0.10)
Bevcon-Buffadison	Louvicourt Tp., Q.	1946-65 & 81-	3,493,243	454,122	0.13
(Mid Can. Gold)			(1,104,500)		(0.175)
			(814,000)		(0.11)
Siscoe	Dubuisson Tp., Q.	1929-49	3,258,306	881,858	0.27
Chesterville	McGarry Tp., Larder L.	1939-52	3,260,439	358,615	0.11
Pickle Crow	Connell Tp., Ken. Pat.	1935-66	3,237,572	1,446,214	0.45
Powell Rouyn	Rouyn Tp., Noranda, Q.	1937-55	3,141,001	380,729	0.12
			(260,000)		(0.12)

TABLE 1. (continued)

Mine	Location[1]	Production Period	Ore Milled & Reserves[2] (tons)	Gold Recov. (oz)	Recov. Grade (oz/ton)
Kirkland Lake	Teck Tp., Kirk. L.	1916-60	3,141,051	1,172,955	0.37
Stadacona	Rouyn Tp., Noranda, Q.	1936-58	3,052,420	467,180	0.16
			(137,340)		(0.15)
Mosher Long Lac	Errington Tp., Gerald.	1956&62-66	2,710,657	330,256	0.12
			(897,000)		(0.10)
Belleterre	Guillet Tp., Q.	1936-58	2,396,992	743,068	0.31
			(190,870)		(0.187)
Elder	Beauchastel Tp., Nor., Q.	1946-66	2,375,485	348,338	0.16
MacKenzie Red Lake	Dome Tp., R.L.	1935-66	2,353,833	651,544	0.28
Agnico Eagle	Joutel Tp., Q.	1974-	2,223,015	442,152	0.19
			(1,210,825)		(0.204)
Marban	Dubuisson Tp., Q.	1961-74	2,185,970	330,015	0.16
Reef	Whitney Tp. Porc.	1915-17&47-65	2,144,507	498,932	0.23
Cochenour Willans	Dome Tp., R.L.	1939-71	2,129,371	1,064,686	0.50
			(200,000)		(0.10)
Wasamac (No. 1)	Beauchastel Tp., Nor., Q.	1965-71	2,091,289	255,965	0.12
			(437,623)		(0.15)
Senator Rouyn	Rouyn Tp., Nor., Q.	1940-55	1,837,807	238,915	0.13
			(167,660)		(0.14)
Little Long Lac	Errington Tp., Gerald.	1934-56	1,780,516	605,449	0.34
Perron	Pascalis Tp., Q.	1933-51	1,775,711	438,779	0.25
Central Patricia	Connell Tp., Ken. Pat.	1934-51	1,729,248	621,806	0.36
Omega	McVittie Tp., Larder L.	1913&26-28, 36-51	1,615,081	214,098	0.13
Vipond	Tisden Tp., Porc.	1911-41	1,565,218	414,367	0.26
Hasaga	Heyson Tp., R.L.	1938-52	1,515,282	219,320	0.14
			(200,000)		(0.19)
Hard Rock	Ashmore Tp., Gerald.	1938-51	1,458,375	269,081	0.18
O'Brien (Darius)	Cadillac Tp., Q.	1930-56&78-81	1,399,475	593,856	0.42
			(300,000)		(0.16)
Toburn	Teck Tp., Kirk. L.	1912-53 (34 yr.)	1,186,316	569,432	0.48
Broulan	Whitney Tp., Porc.	1939-53	1,146,059	243,757	0.21
Norlartic	Vassan Tp., Q.	1959-66	1,139,443	145,610	0.13
			(120,000)		(0.11)
Francoeur (Wasamac No. 2)	Beauchastel Tp., Nor., Q.	1938-47 & 68-71	1,006,737	162,475	0.16
			(128,000)		(0.20)
Doyon (Silverstack)	Bousquet Tp., Q.	1980-	959,139	179,229	0.19
			(2,661,000)		(0.185)
Leitch	Eva Tp., Beardmore	1936-65&80	932,945	848,919	0.91
Starrat Olsen	Baird Tp., R.L.	1948-56	907,813	163,990	0.18
			(15,000)		(0.45)
Bousquet	Bousquet Tp., Q.	1979-	828,137	162,701	0.20
			(2,644,260)		(0.140)
Mic Mac	Bousquet Tp., Q.	1942-47	797,558	103,600	0.13
Uchi	Earngey Tp., Ken.	1939-43	757,074	114,467	0.15
Hoyle	Whitney Tp., Porc.	1941-44 & 46-49 & 81	725,494	71,843	0.10
			(967,400)		(0.11)
Eldrich	Duprat Tp., Nor., Q.	1956-62	717,654	100,076	0.14
			(300,000)		(0.16)
Donalda	Rouyn Tp., Nor., Q.	1948-56 & 70-71	727,375	120,086	0.17
			(480,000)		0.26
Chadbourne	Rouyn Tp., Nor., Q.	1979-	641,000	61,678	0.10
			(864,000)		(0.11)
Cons. Central-Cadillac	Cadillac Tp., Q.	1939-43&47-49	638,338	89,936	0.14
			(1,546,000)		(0.16)
Hugh Pam	Whitney Tp., Porc.	1926 & 48-65	636,751	119,604	0.19
Bidgood	Lebel Tp., Larder L.	1934-49&51	586,367	160,184	0.27
Upper Beaver	Gauthier Tp., Larder L.	1913-44&65-72	580,562	140,709	0.24
			(200,000)		(0.23)
Berens River	Favourable Lk., Ken.	1939-48	560,607	157,696	0.28
			(458,000)		(0.21)
God's Lake	God's Lake, Man.	1935-43	541,140	160,561	0.30
Arntfield	Beauchastel Tp., Nor., Q.	1935-42	529,480	55,662	0.11
Central Manitoba	Rice Lake, Man.	1927-37	456,047	159,252	0.35
Beardmore (N. Empire)	Summers Tp., Beard.	1934-41&49	425,866	149,493	0.35
Norbeau	McKenzie Tp., Chiboug., Q.	1964-69	419,029	163,362	0.39
Lapa Cadillac	Cadillac Tp., Q.	1938-43	(111,200)	47,263	(0.20)
			383,430		(0.12)
			(28,000)		(0.19)
McWatters	Rouyn Tp., Nor., Q.	1934-44	368,013	108,403	0.30
			(420,000)		(0.22)
Magnet Consolidated	Errington Tp., Gerald.	1938-43&46-52	359,912	152,089	0.42
			(105,000)		(0.27)
Bonetal	Whitney Tp., Porc.	1941-51	352,254	51,510	0.15

TABLE 1. (continued)

Mine	Location(1)	Production Period	Ore Milled & Reserves(2) (tons)	Gold Recov. (oz)	Recov. Grade (oz/ton)
Jerome	Osway Tp., Sudbury	1941-43	335,060	56,893	0.17
			(345,000)		(0.19)
Cline Lake	Jacobson Tp., Algoma	1938-43&47-48	331,842	63,328	0.19
			(20,000)		(0.25)
St. Anthony	Sturgeon L., Th. Bay	1905-41 (20 yr.)	331,069	63,310	0.19
Ankerite March	Deloro Tp., Porc.	1926-35 & 36-53	317,769	61,039	0.19
Moneta	Tisdale Tp., Porc.	1938-43	314,829	149,250	0.47
Courvan-Bussieres	Louvicourt Tp., Q.	1932-35&37-42	309,374	36,939	0.12
			(391,100)		(0.14)
West Malartic	Cadillac Tp., Q.	1942-46	308,332	35,631	0.12
Naybob	Ogden Tp., Porc.	1932-64 (14 yr.)	304,100	50,731	0.17
Joburke	Keith Tp., Sudbury	1973-75&79-81	302,561	28,440	0.09
Gunnar	Rice Lake, Man.	1936-42	298,829	101,463	0.34
Akasaba	Louvicourt Tp., Q.	1960-63	289,428	39,744	0.14
H.G. Young	Balmer Tp., R.L.	1960-63	288,179	55,244	0.19
Jason (Argosi)	51°15′ - 92°15′	1934-52 (13 yr.)	276,573	101,875	0.37
Barry-Hollinger	Pacaud Tp., Kirk. L.	1918-46 (16 yr.)	267,741	69,891	0.26
Louvicourt Gold Fields	Louvicourt Tp., Q.	1947-49	263,850	32,201	0.12
Bankfield	Errington Tp., Gerald.	1937-42&44-47	231,009	66,417	0.29
Crown	Tisdale Tp., Porc.	1913-21	226,180	138,330	0.61
Tyranite (Hedlund)	Tyrrel Tp., Gowganda	1939-42	223,810	31,352	0.14
Wendigo	Manross Tp., Ken.	1900, 36-43& 51	206,054	67,054	0.33
Bonwhit	Whitney Tp., Porc.	1951-54	200,550	67,940	0.34
De Santis	Ogden Tp., Porc.	1939-42	196,928	35,842	0.20
			(108,000)		(0.39)
Pandora - Amm	Cadillac Tp., Q.	1939-42	196,506	27,003	0.15
Wilmar	Dome Tp., R.L.	1967-71	193,435	48,359	0.25
			(3,150,000)		(0.08)
Tombill	Lindsey Tp., Gerald.	1938-42&55	190,622	69,120	0.36
Minto	McMurray Tp., Wawa	1929-42	184,600	37,678	0.20
			(108,000)		
Granada	Rouyn Tp., Nor., Q.	1930-35	181,679	51,476	0.28
Gold Eagle	Dome Tp., R.L.	1937-41	180,095	40,204	0.22
Wood-Cadillac	Cadillac Tp., Q.	1939-42	179,313	29,058	0.15
			(584,000)		(0.16)
Belmoral Ferderber	Bourlamaque Tp., Q.	1979-	177,445	27,000	0.164
			(1,490,000)		(0.204)
Beaufort	Pascalis Tp., Q.	1939-42	176,363	35,921	0.20
			(50,000)		(0.21)
New Alger (Thompson-Cad.)	Cadillac Tp., Q.	1936-39	175,816	16,466	0.09
Sand River	Eva Tp., Beard.	1937-42	157,870	50,065	0.32
Ashley	Bannockburn Tp., Mat.	1932-36	157,636	50,123	0.32
Canadian Arrow	Hislop Tp.	1981-	156,202	7,652	0.056
			(101,000)		(0.058)
McMarmac	Dome Tp., R.L.	1940-44&47-48	152,978	46,321	0.30
Annco	Dome Tp., R.L.	1965-71	149,243	55,220	0.37
Anglo Rouyn	Rouyn Tp., Nor., Q.	1948-51	147,377	33,193	0.23
Sturgeon River	Irwin Tp., Beard.	1936-42	145,123	73,438	0.51
Kerry	Moss Tp., Th. Bay	1932-36&42	143,724	29,948	0.21
Chimo	Vauquelin Tp., Q.	1966-67	141,900	58,434	0.41
			(650,000)		(0.20)
Ogama-Rockland	Rice Lake, Man.	1942-43&48-51	139,103	45,343	0.33
Shawkey	Dubuisson Tp., Q.	1936-38	137,978	25,414	0.18
Lake Beaverhouse	Gauthier Tp., Larder L.	1919-35&37-39	132,000	43,560	0.33
Morris Kirkland	Lebel Tp., Larder L.	1936-38&40-42	127,253	16,681	0.13
Parkhill	McMurray Tp., Wawa	1904&24-38 &40-44	125,778	54,301	0.43
Faymar Porcupine	Deloro Tp., Porc.	1940-42	119,181	21,851	0.18
Algoma Summit	Finan Tp., Algoma	1934-39&42-53	116,627	8,776	0.08
Quesabe	Duprat Tp., Nor., Q.	1949-52	113,614	28,252	0.25
			(225,000)		(0.30)
Kiena	Dubuisson Tp., Q.	1981-	111,588	15,019	0.13
			(5,811,500)		(0.18)
New Marlon	Rouyn Tp., Nor., Q.	1947-49	108,188	19,511	0.18
			(56,000)		(0.22)
Jackson-Manion	Dent Tp., Ken.	1934-40&47-48	105,357	27,142	0.26
Golden Gate	Teck Tp., Kirk. L.	1922-47	103,684	30,068	0.29
Porcupine Peninsular	Cody Tp., Porc.	1924-27, 40&47	99,688	27,354	0.27
			(1,480,000)		(0.12)
Duquesne	Destor Tp., Q.	1947, 49-50&52	90,249	24,968	0.31
			(80,000)		(0.37)
Red Lake Gold Shore	Dome Tp., R.L.	1936-38	83,949	21,100	0.25
Bras d'Or G.H. Dumont	Bourlamaque Tp., Q.	1980-	79,873	15,000	0.19

TABLE 1. (continued)

Mine	Location[1]	Production Period	Ore Milled & Reserves[2] (tons)	Gold Recov. (oz)	Recov. Grade (oz/ton)
			(665,702)		(0.203)
Sultana	Lake of Woods	1894-1906&49	77,481	15,977	0.21
Jacola	Dubuisson Tp., Q.	1933-37	71,504	15,159	0.21
Mikado	Glass Tp., Ken.	1896-1902, 10-11&31	57,813	28,335	0.49
Gillies Lake	Tisdale Tp., Porc.	1929-31&35-37	54,502	15,278	0.28
Sachigo River	54°30′ - 91°15′	1938-42	46,416	52,560	1.13
Strae Lake Beach	49°00′ - 93°15′	1938-41	33,662	11,568	0.34
Grace (Darwin)	McMurray Tp., Wawa	1902-10&23-44	45,528	15,191	0.33
Golden Star	48°30′ - 92°30′	1828-1901, 34, 38&41	19,345	10,757	0.56
			(20,000)		(0.42)
Jeep	Rice Lake, Man.	1948-50	17,562	13,629	0.77
Croesus	Munro Tp.	1915-18&31-36	5,333	14,859	3.43

(1) In Ontario, unless otherwise indicated.
(2) Proven for current producers and indicated for former producers.

TABLE 2. Tonnage-grade distribution of gold deposits in the Superior Province

Production Range to End of 1981 (mill. tons of ore)	Number of Mines	Percentage of Total	Number of Deposits by Range of Recovered Grades (oz/ton)				
			>0.40	0.30 to 0.40	0.20 to 0.29	0.15 to 0.19	<0.15
20 or more	6	4	0	1	3	1	1
10 to 20	6	4	1	0	0	2	3
5 to 10	16	10	3	4	5	2	2
3 to 5	16	10	2	3	4	2	5
1 to 3	21	14	3	3	5	4	6
0.5 to 1	15	10	1	1	3	5	5
0.1 to 0.5	61	39	6	15	13	16	12
0.01 to 0.1	14	9	5	2	6	0	0
Totals	155	100%	21 (13%)	29 (19%)	39 (25%)	32 (21%)	34 (22%)

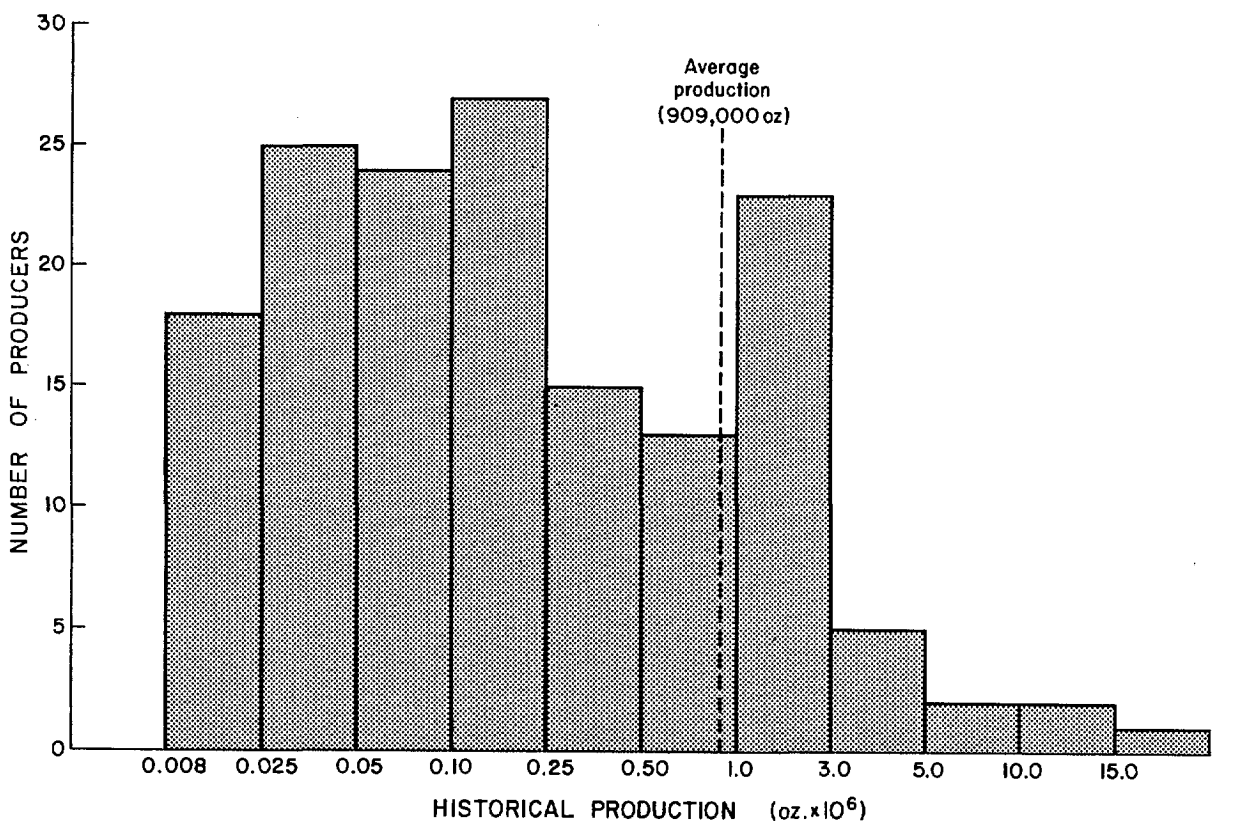


FIGURE 1. Gold production of the Superior Province (155 deposits greater than 8,000 oz).

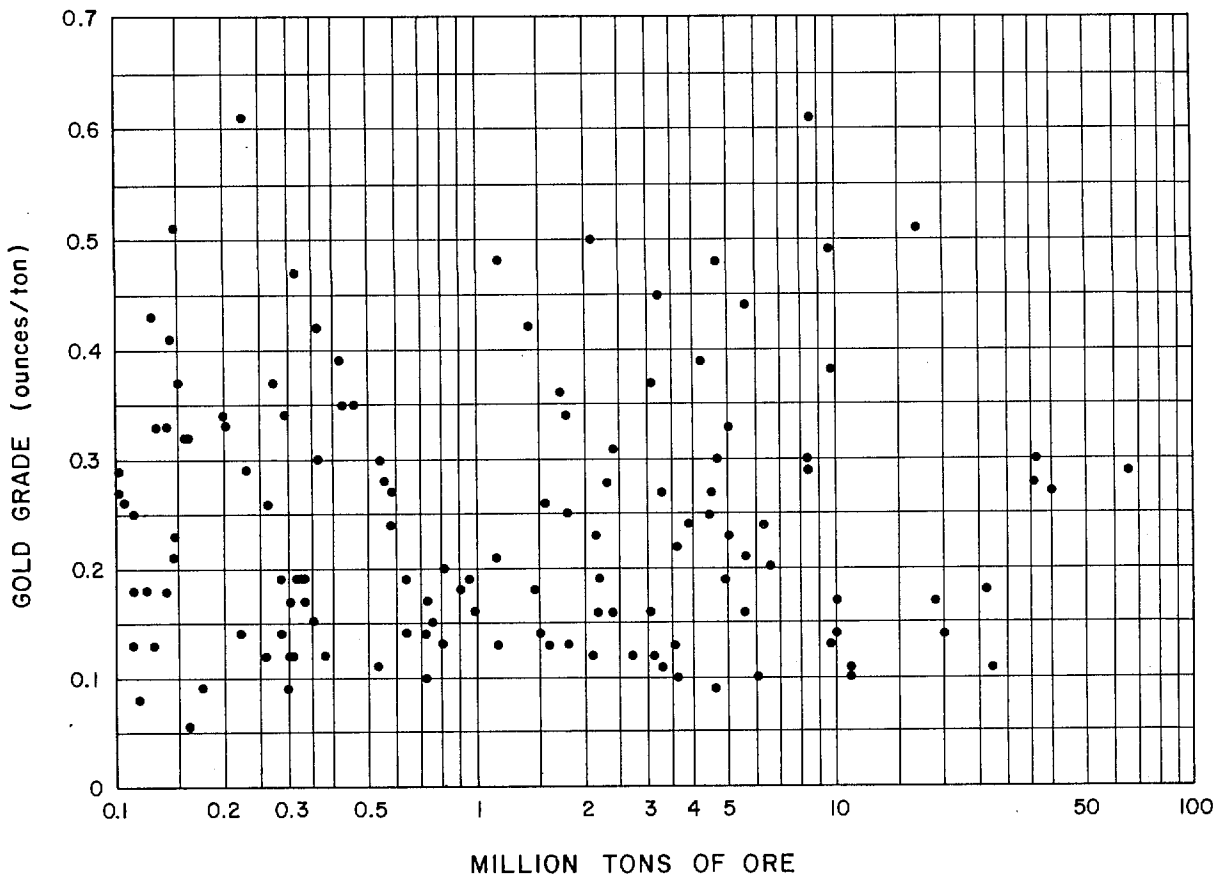


FIGURE 2. Tonnages and grades of gold mines in the Superior Province with production above 100,000 t of ore to the end of 1981.

tion. The total tonnage quoted for a particular mine does not necessarily represent the total tonnage contained by the orebody being mined. The three mines, for example, which have produced more than 50% of the total gold production of the Red Lake camp share the main vein system of the camp. It could also be argued that the Hollinger and McIntyre mines share a single 100-million-ton deposit, and that at Kirkland Lake a 51-million-ton orebody is subdivided among seven different mines. Such irrational distribution between mining properties and orebodies is a result of the small size of the claim groups at the time of exploration and discovery. The gold deposits of the Superior Province are, therefore, comparable in size to the base-metal deposits occurring in the same area.

Table 2 shows that there are only a relatively small number of large deposits. Of the 155 mines considered, almost half (48%) have individually produced less than 500 thousand tons of ore, and over half (58%) have produced less than one million tons of ore. Twenty-five per cent of the mines produced between one and five million tons, 10% between five and ten million tons, and only 8% above ten million tons of ore. As shown on Figure 1, only thirty-three mines (21%) have produced more than one million ounces of gold, and even fewer, just six (or 4%), more than five million ounces of gold. However, the gold production from those thirty-three mines, 120,194,000 ounces, is equivalent to 85% of the total gold production of the Superior Province. Moreover, the production of the five largest mines (59,176,000 ounces) accounts for almost half (42%) of the total production.

The tonnage and grade relationships are illustrated by Table 2 and Figure 2. The majority of grades (68%) are between 0.10 and 0.30 ounce/ton, with only few below 0.10 (3%) and above 0.50 ounce/ton (4%). The lower threshold of 0.10 ounce/ton seems to have been a particularly strong one, dictated by the economics of the gold price.

TABLE 3. Gold production and average grade of principal camps of the Superior Province

Camp	Gold Production to End of 1981 (ounces)	"Camp Grade" (oz/ton)
Porcupine, Ont	56,297,000	0.26
Kirkland Lake, Ont. (Teck Township)	22,911,000	0.45
Red Lake, Ont.	12,499,000	0.37
Larder Lake, Ont. (McGarry, McVittie, Gauthier & Lebel Twps.)	12,321,000	0.26
Cadillac/Malartic, Que. (Hosted by Pontiac and/or Piché Groups in Bousquet, Cadillac, Fournière and Dubuisson Twps.)	12,111,000	0.16
Val d'Or, Que.	9,087,000	0.17
Geraldton & Beardmore, Ont	4,090,000	0.22
Noranda, Que.	2,392,000	0.15
Rice Lake, & God's Lake, Man.	1,680,000	0.28
Matachewan, Ont.	1,010,000	0.10
Algoma, Ont.	959,000	0.21

Figure 2 is a plot of tonnage (production plus reserves) and grades for all mines of the Superior Province with production above 100,000 tons of ore. There is no linear relationship between grade and tonnage, but, instead, a scatter over a wide range.

Finally, Table 3 is the gold production and average grade of the principal camps of the Superior Province. The calculation of these figures was based on Table 1. The given production

for each camp is the majority, but not the total production of that camp, because the very small mines were left out. Similarly, the sum of the production of all listed camps is less than that of the Superior Province because several sizeable deposits are more or less isolated and cannot be included in a camp. The subdivision into camps is both geological and geographical. Geologically, it shows that a single orebody, as at Kirkland Lake, can be exceptionally rich, and that different geological environments have different "typical" grades. Geographically, it demonstrates the average grade which characterizes a certain camp.

Gold Mine Development and Gold Prices

Table 4 has the gold producers distributed into six tonnage intervals and four time periods for start of production. The time intervals were chosen to show the effect of the significant changes in price on the development of new mines.

The ups and downs in gold mine development reflect quite accurately the history of gold prices. From the 19th century until 1934, the gold price was fixed at (US)$20.67 an ounce. The United States, which controlled a major part of the world's bullion reserves after the first world war, revalued those reserves to (US)$35.00 an ounce in 1934 (Fig. 3). Table 4 shows that this increase in price resulted in the opening of many new gold mines, predominantly in the second half of the 1930s and, to a certain extent, after the end of the second world war. Although three deposits with reserves greater than 20 million tons were brought into production, the vast majority of the new mines had reserves of less than one million tons of ore at that time.

TABLE 4. Gold mine development

Production Range to End of 1981 (million tons of ore)	Number of Mines Developed			
	Before 1934	1934-1949	1950 - 1969	1970 - 1981
20 or more	3	3		
10 to 10	2	4		
5 to 10	5	9	1	
1 to 5	12	21	4	1
0.1 to 1	17	44	9	8
Less than 0.1	9	5		
Totals	47	85	14	9

Inflation, however, caused a decrease in real value of the gold and the Government of Canada passed the Emergency Gold Mining Assistance Act in 1948, which provided a subsidy to marginal producers. These subsidies did not have an effect comparable to the 1934 rise in price because the subsidies functioned only for the survival of existing mines. Only 14 mines were opened between 1950 and 1969, none of them with more than 10 million tons of ore. In constant 1979 Canadian dollars, from 1934 to 1939 Canadian gold mines actually were paid an average of about (Can)$225.00 an ounce for their gold (Fig. 3), and, because of inflation and foreign-exchange fluctuations, the price dropped to (Can)$72.00 an ounce by 1970 (Cranstone, 1980).

During the sixties, there was a concomitant diminution of the Russian gold supply, and an increase in demand by the private sector. In 1968, the United States government suspended sales to the industry, which then had to purchase gold in the open market. As a result, prices on world gold markets ranged between (US)$35.00 and (US)$45.00 an ounce until 1972, when they began to rise at a fast pace. The more recent history is known by everyone; the price of gold has reached previously unthinkable heights and has subsequently retreated to its current level.

Although it might still be early to judge, it seems that the reaction of the gold mining industry to the price increase after 1972 was not as great as that after the increase of 1934. In fact, no mine with total production of over 20 million tons of ore has been opened since 1938, when Kerr Addison started operations. The long period of financial difficulties experienced by the gold mines resulted in cuts in expenditures on development, maintenance and manpower among the surviving companies. As a result, few were in a position to react quickly to improved market conditions (Cousineau and Richardson, 1980). It is believed, however, that the recent and more dramatic fluctuations in gold price have been causing a greater im-

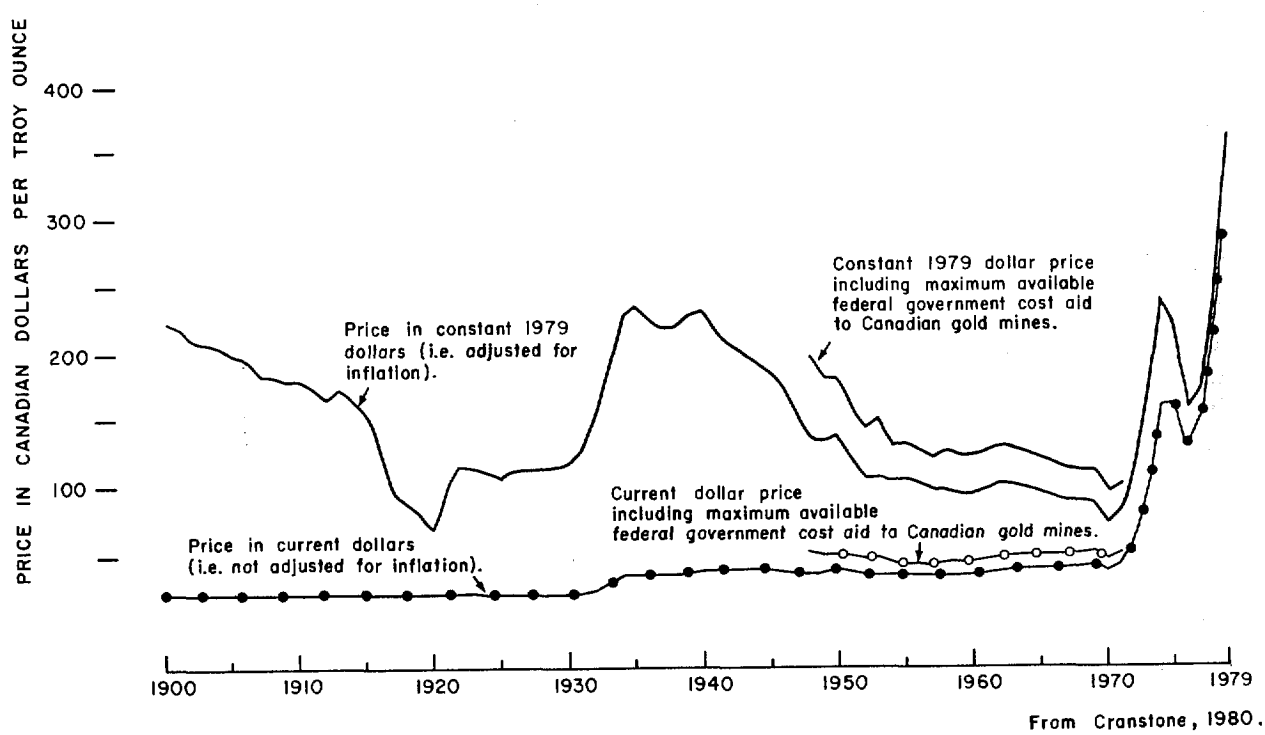


FIGURE 3. Average annual Canadian dollar gold prices, 1900 to 1979.

pact in the mining industry, with the development of new mines and reopening of old ones (Hamilton, 1981 and 1982).

From an exploration point of view, it is interesting to note that all discoveries since just before 1950 have been relatively small, with the exception of Detour Lake and Camflo. This fact could be interpreted as (a) due to insufficient exploration, reflecting a lack of incentive in the low prices, (b) discovery of all available deposits, after years of thorough investigation by several parties, and/or (c) ineffectiveness of the current exploration techniques in finding hidden gold deposits. It is believed that the first and the third possibilities, combined, offer the probable explanation. There is no study available to indicate the amount of exploration funds invested directly in gold prospecting over the years, and it is interesting that a few of the recent deposits (e.g. Detour Lake) were discovered during surveys for base metals and not specifically for gold. It is known, however, that, until recently, fewer and fewer companies have been directing any significant proportion or indeed any of their exploration efforts and dollars toward gold exploration.

It appears as if a significant number of mines or known deposits already have had their possible extensions verified through drilling, and the exploration attitude could be redirected toward the discovery of new deposits, or even camps, in relatively unexplored areas, with the intensive use of new concepts and models. New exploration techniques, using geochemical and geophysical tools, need to be designed and employed. Also as evidence for the argument that traditional exploration methods are becoming ineffective for gold exploration is the fact that most discoveries in the fifties and sixties were of base metals made with geophysical (and to some extent geochemical) and geological tools (Boldy, 1977) in areas with extensive glacial-drift overburden and away from known camps. Because most geophysical and many geochemical techniques are unable to detect gold ore under overburden or at depth in rocks, one of the best approaches would be to develop geological parameters, and better models capable of explaining the origin and evolution of known deposits. It is believed that the development of new and effective gold exploration concepts and techniques would be extremely rewarding, and a significant number of deposits can still be found.

Acknowledgments

I am grateful to Riocanex Inc. for granting permission to publish this paper. I am indebted to William M. Little of Cominco Ltd. for introducing the author to the subject of gold in the Canadian Shield. Officials of several companies are acknowledged for providing the author with information about their mining operations. The manuscript has been improved by critical reviews by Paul M. Kavanagh, Colin D. Spence, Terence J. Bottrill and Robert W. Hodder. I further appreciate the help of M. Shaw, who typed many versions of the tables, and S.J. Elsmore, who drafted the figures.

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Big North American representation expected at S.A. gold conference

The South African Gold and Investors Conference will take place in Johannesburg on **April 24 to 26, 1984** at the Sandton Sun Hotel.

It is anticipated that 800 foreign delegates will attend with most of them coming from North America.

Inquiries for detailed information should be addressed to the Organizing Committee, Kirkpatrick, Marais and Associates, P.O. Box 61653, Marshalltown 2107, South Africa, tel: (011) 836-6525, telex 89925 SA.

CIM

Process evaluation, modelling and optimization workshop

A SPOC workshop, organized by Université Laval - Centre de Recherches Minérales, will be held on **October 17-21, 1983,** at Auberge des Gouverneurs, Mont-Sainte-Anne, Québec. The workshop is a methodology transfer channel of the SPOC project coordinated by CANMET/EMR. It will include theory presentations, applied exercises and computer demonstrations on time sharing terminals. Sponsored by CANMET/EMR under the START (Short Term Aid in Research and Technology) program, attendance will be free, and travel and accommodation assistance is available.

For further information, contact:
Professor Gilles Barbery
Département Mines et Métallurgie
Université Laval
Québec, Qué. G1K 7P4;
Tel. 418-656-2555

CIM

JDS copy

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$500 for first 6,
then $70/each additional
(11/2/83 verbal)
Has presented this to 6 groups
ranging from 6-29 people.

have his "text outline" book in library.

October 22, 1983

Mr. James D. Sell
Manager--Southwestern Exploration Division
ASARCO Incorporated
P.O. Box 5747
Tucson, Arizona 85703

Dear Mr. Sell:

Thank you for your interest in my gold-silver seminar expressed during our October 19 chat at the Salt Lake A.I.M.E. meeting.

The promised information on this seminar is enclosed. Perhaps you would like to discusses it with me while I am in Tucson November 1; I will call you. As a matter of interest, I brought this to the attention of Fred Graybeal early this year, and he has invited me to bring the matter back up with him mid winter.

The comprehensive body of material proposed for discussion involves all known classes of bulk minable gold or silver deposits but focuses on the geology/exploration, including geochemistry, of those currently receiving attention. Discussions on bulk minable gold-silver history, mining, and metallurgy place the geologic discussions in a realistic perspective.

Importantly, the lecture on economics reveals some facts valuable both to prioritizing divisional level exploration decisions and corporate level financial policy making.

This might be an intense one-day presentation in your offices. A more expository two-day format is available as is a half-day presentation of selected topics.

Yours truly,

Stanley W. Ivosevic

Stanley W. Ivosevic

SWI:tbw

Enc

RECEIVED
OCT 26 1983
S. W. U. S. EXPL. DIV

FTG: As noted, I asked Wayne to send me an outline, as attached. I have met him 3-4 times but no nothing of his expertise. However, if interested, as WSK suggests, the meeting in Denver might be the time for him to give a days lecture to all the staffs, i.e. on Sunday? Jim

FTG 11/1/83 cf

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BULK MINABLE GOLD-SILVER--GEOLOGY EXPLORATION, DEVELOPMENT, ECONOMICS

A seminar prospectus prepared for

Mr. James D. Sell
Manager--Southwestern Exploration Division
ASARCO Incorporated

by
Stanley W. Ivosevic
1983

A practical approach to professional success

SUMMARY

Experienced practitioners and newcomers, alike, who attend this seminar will learn all phases of successful bulk minable gold-silver exploration and development. Attendees also will gain the understanding of precious metals economics needed to guide informed decision making.

The material presented is a synthesis of the unique geologic and economic concepts proven by your seminar leader in discovering ore deposits and generating minerals opportunities:

LECTURE 1. Introduction

LECTURE 2. Geologic Classification

LECTURE 3. Carlin-Type Gold Deposits

LECTURE 4. Back Arc Volcanogenic Silver or Gold Deposits

LECTURE 5. Island Arc Volcanogenic Silver or Gold Deposits

LECTURE 6. Gold-Silver Ratios

LECTURE 7. Production

LECTURE 8. Economics

LECTURE 9. Exploration

This seminar consists of nine lectures, with discussion sessions, delivered over an eight hour period. Lectures are illustrated with over 100 projected drawings, graphs, tables, and outlines.

Attendees are provided a 31-page set of course notes outlining each lecure in detail. Reference lists are included for those desiring to continue broadening their knowledge of bulk minable gold-silver.

The topics of the lectures are itemized above. Their contents are summarized below.

BULK MINABLE GOLD-SILVER--GEOLOGY, EXPLORATION, DEVELOPMENT, ECONOMICS

by
Stanley W. Ivosevic
1983

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--- Weight conversions

This seminar is designed around the success of your mineral ventures

JDS

ASARCO

Southwestern Exploration Division

December 29, 1983

File Memo

McLaughlin Gold Deposit
Napa County, CA

The occurrence of hydrothermal breccia in gold deposits, such as McLaughlin, Borealis and others, was expounded upon by Carl Nelson* at a recent AGS Meeting in Tucson.

The McLaughlin orebody, containing 22 million tons of .16 oz. Au (mineable by open pit with a w/o ratio of 2 or 3 to 1), occurs in a hot spring environment known principally for its mercury production. Sandy shales of the Cretaceous Knoxville Formation are the main host rocks, forming a wedge-shaped ore zone (see sketch attached) in fault contact with serpentine. Intrusive "hydrothermal" breccias, containing andesite and tuff fragments, K spar, silica, thallium and gold, have been identified in this zone, but make up only a small part of the whole. The ore in general contains .5 lb. per ton of mercury, plus pyrite, arsenopyrite and stibnite. However, the gold which is in the micron size range does not occur in the sulphides, but in the silica. The temperature of gold deposition was determined to have been around 150-200°C. The lava flows, which are pre-mineral range in age from .03 to 2.1 my.

Attached is a geologic plan of the area (after Paul Averitt, 1945, Quicksilver deposits of the Knoxville district, Calif. Journ. Mines and Geology, vol. 41, no.2) showing the location of the old Manhattan mercury mine which is (or will be) also the site of the McLaughlin gold mine.

Nelson considered Carlin to be in a separate category, although he did note that the host rocks there were sandy ---- similar to some beds in the Knoxville. Carlin, of course, is also in a hot spring environment, but no "hydrothermal" breccias have been found there.

From other sources, I have heard that due to encapsulation of the gold in silica, expensive autoclave treatment is required to obtain adequate recovery ---- adding up to a total cost of around $310.00 per ounce for projected production from the McLaughlin deposit.

J. H. Courtright

J. H. Courtright

JHC:mek

* Cimmaron Explorations, Lakewood, CO

cc: R. L. Brown
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R. S. Gray
H. C. Williamson
S. A. Anzalone

XC: HGK
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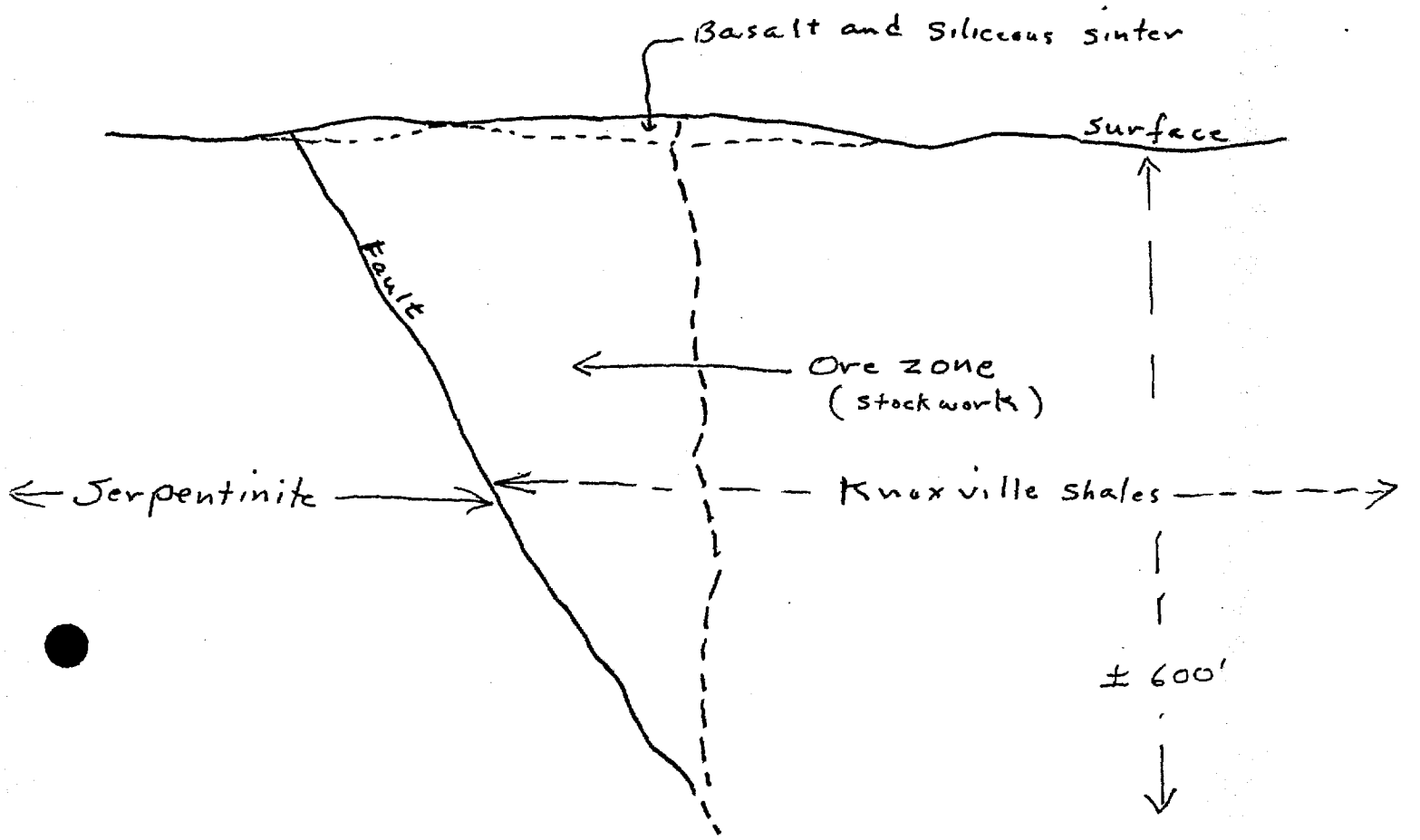
GJS, 1/3/84



McLaughlin Gold Deposit – Section, Looking North

Sketch – no scale T.H.C. 12-6-'83

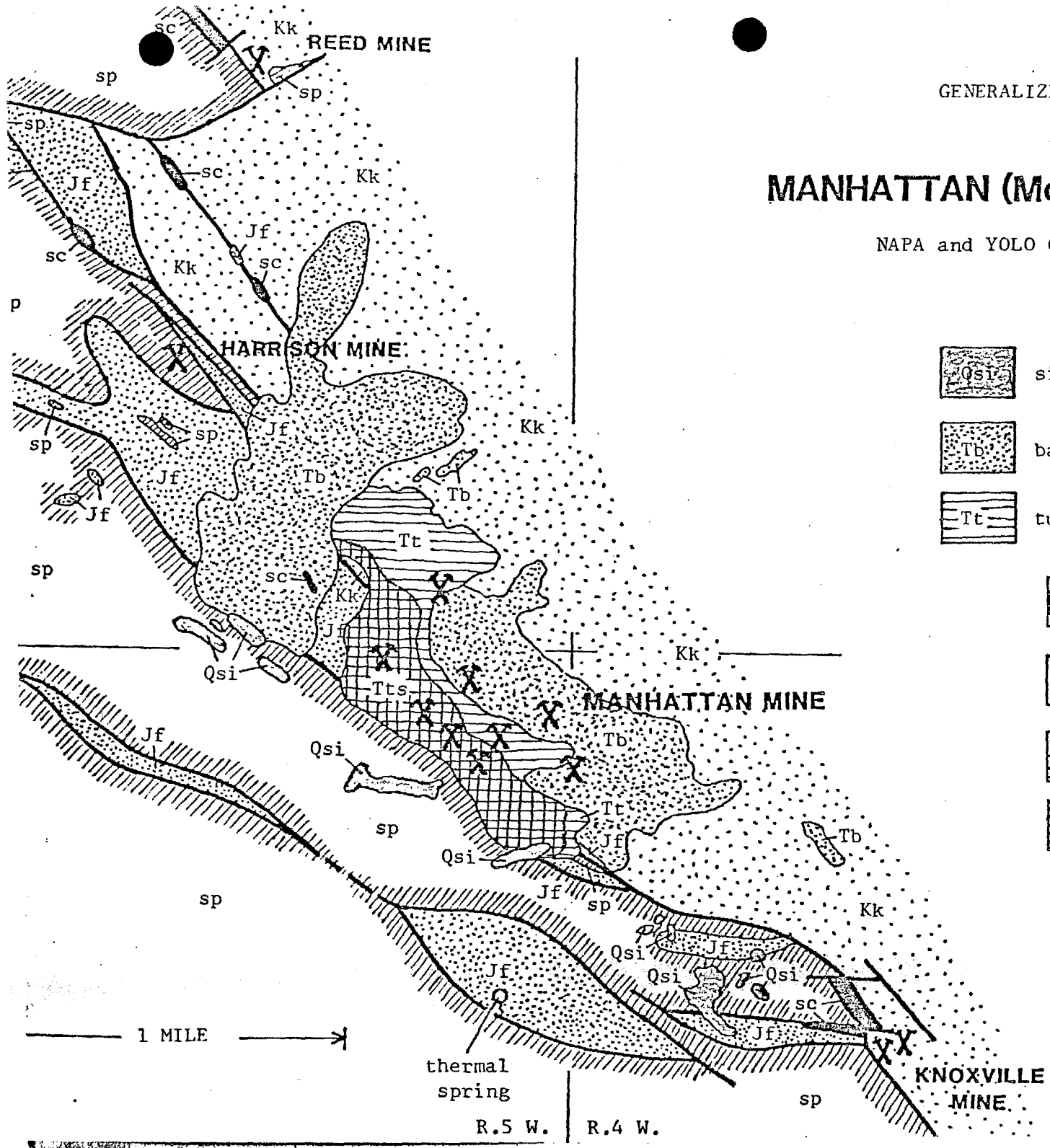
(from slide shown by Carl Nelson)

GENERALIZED GEOLOGIC MAP

of the

MANHATTAN (McLaughlin) MINE AREA

NAPA and YOLO COUNTIES, CALIFORNIA



Qsi — siliceous sinter deposits

Tb — basalt

Tt — tuff

Tts — silicified tuff ("opalite")

sc — "silica-carbonate" rock

Kk — Knoxville Formation

Jf — Franciscan Formation

sp — serpentinite

after Averitt, 1945

MLK 10/81

JDS

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Paul I. Eimon
Vice President-Exploration

J. R. S.
DEC 27 1983

PIONEER

December 13, 1983

Mr. J. R. Stringham
Assistant to General Manager
Southwestern Exploration Department
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1150 North 7th Avenue
Tucson, Arizona 85703

Dear Jim:

It was a pleasure to see you again in Tucson at the AIME meeting and especially to be updated on Bonanza with the talk that you gave. I look forward to further discussions with you on what you said and I am enclosing papers I have written or co-authored about epithermal systems. I do hope we are able to have a Bonanza reunion sometime in the future and I am glad that your episode earlier this year was nothing more than that.

Best wishes for a happy holiday season.

Sincerely,

Paul

Paul I. Eimon

PIE:ns

Enclosures

ASARCO Incorporated
DEC 19 1983
SW Exploration

EXPLORATION FOR EPITHERMAL GOLD AND SILVER DEPOSITS: THE EPITHERMAL MODEL

Paul Eimon, Mining Geologist
Denver, Colorado, U.S.A.

Paper presented at
The First International Symposium on Small Mine Economics & Expansion
Taxco, Mexico, May 17-21, 1981

EXPLORATION FOR EPITHERMAL GOLD AND SILVER DEPOSITS
THE EPITHERMAL MODEL

Paul Eimon
Mining Geologist
Denver, Colorado, United States

What and how geoscientists study, develop and act is - like other disciplines - based on the accumulation of a body of knowledge to which many predecessors have contributed. This paper will describe a new concept - based on research by some young geologists - that is an outgrowth of centuries of looking for and finding and developing mines. The epithermal model of 1981 owes something to early silver mining at Larium which helped float the Greek civilization. It owes more to silver and gold mining in the Americas, especially in Mexico during the past 400 years, and to observant geologists who left notes and reports on their observations.

The concepts owe much to earlier, creative geoscientists of this century. Ed Wisser, who was my professor at the University of California and who had previously been the chief geologist at Real del Monte at Pachuca, is one of these. He passed on to me not only his ardent love affair with Mexico but part of his library and files. Such files are crammed with the kinds of random, then-unexplainable or unexplained phenomena which a keen geologic observer would make. And they have a bearing on the new gold-silver model.

The 1980's resurgence of interest in gold and silver deposits is both economic and academic. This interest focuses on new low-grade deposits amenable to bulk mining, such as the gold mines of Carlin and Jerritt Canyon, Nevada, the gold-silver mines of Delamar, Idaho, the precious metal - base metal deposit of Real de Angeles in Zacatecas, and Homestake's new gold discovery in Northern California.

During the past fifteen years, new understanding of epithermal gold-silver deposits, combined with extensive earlier observations and knowledge (1890-1950), resulted in new genetic concepts. These concepts suggest exploration methods and ideas in the search for, the discovery of, and the evaluation of gold-silver ore bodies, ore deposits, and mining districts.

This paper will identify current ideas - new and old - and some of the men developing them, indicate how these ideas impact your precious-metals mining operation or exploration program, and describe how you can utilize the ideas.

While I will emphasize epithermal gold-silver deposits in volcanics, many of the same ideas and concepts apply to Carlin-type deposits.

Waldemar Lindgren originally proposed the term "epithermal" to describe those orebodies deposited from ascending solutions to shallow depths at low temperatures. He showed that such deposits bear a close relation to hot spring deposits and may be pictured as forming a short distance below the fumarolic outlet of the mineralizing solutions. Lindgren stated that epithermal deposits are generally restricted to areas of tertiary volcanism and are characterized by bonanza quantities of precious metals, shallow depth of formation and crustified or comb textures. Such deposits generally have a greater longitudinal dimension than a vertical one.

Of primary economic interest are the bonanza quantities of precious metals often found in these deposits. Many deposits have a relatively short but extremely productive life. Although these deposits may not have the life span of a porphyry copper deposit, the high return and the low capital investment required make the bonanza-type deposits highly profitable at current gold and silver price ranges.

Two generations of geologists have contributed to the model concepts and geologic knowledge for these types of deposits. The early work, in the first half of this century, was that by Waldemar Lindgren, A. F. Buddington, L. C. Graton, W. H. Emmons, Ed Wisser, Harrison Schmidt, Tom Nolan and others. They were involved in the development and mining of deposits in the western U. S. A. and Mexico and produced deposit descriptions and genetic theories which are valid today. Selected references from these men are included in the bibliography attached to this paper. Careful reading of what this first generation of epithermal geologists wrote indicates that many of our "new ideas" may not be so new after all.

The second, "now" generation of epithermal experts is currently researching the epithermal model and developing genetic and exploration concepts. Their work includes (1) research on the hydrothermal process in general; (2) laboratory experimental studies of ore, gangue, and alteration minerals (by many techniques such as fluid inclusions and stable isotope studies); (3) field and laboratory examination of active geothermal systems; (4) detailed research on deposits; and (5) the application of theory and concept in exploration and mine development. Such research is being done by: the United States Geological Survey, private exploration groups, and universities throughout the world - especially concentrated in the U. S. A., New Zealand and Japan.

A basic list of geologists - two generations of epithermal experts - is partially provided in the selected reference list attached to this paper. (You may be interested, too, in a longer list, provided in the bibliography to my paper on Bulk Silver Deposits, given at the CIM Meeting in Vancouver, B. C. in November, 1979.)

First on the list of vital, applicable research is by the United States Geological Survey, specifically: (1) investigations by Philip Bethke, Paul Barton, Robert Rye, James Ratte, Thomas Steven, Edwin Roedder, and others in the Creede District, Colorado, and (2) the career-long studies of hot springs, related to mineral deposition, by Donald White. These two U.S.G.S. efforts

have significantly contributed to our knowledge of genetic processes in the creation of epithermal deposits. Current U.S.G.S. work by B. Berger, R. Rye and others on Round Mountain and Carlin, Nevada, is vital.

Ellis and Mahan, New Zealand, have related the geology and geochemistry of geothermal systems. Their studies detail active systems corelated to epithermal gold-silver deposits.

Recent (1970-81) academic theses have contributed to our knowledge in this field. Among them are: John Dreier's thesis (Ph.D, University of Arizona) on Pachuca, Mexico; Larry Buchanan's thesis (Ph.D, Colorado School of Mines) on Guanajuato, Mexico; Fleetwood Koutz' dissertation (University of Arizona) on the Hardshell deposit, Arizona; Peter Vickre's dissertation (Stanford) on the Rochester Mine, Nevada; Phil Giudice's M.A. thesis (University of Arizona) on the disseminated silver mineralization at Creede, Colorado; Richard Robinson's Masters thesis (New Mexico Institute of Technology) on the Ore Mineralogy and Fluid Inclusions of the Southern Amethyst Vein at Creede, Colorado; and Darby Fletcher's upcoming dissertation (Stanford) on the Waterloo deposit, California.

Recent pertinent papers include: L. Buchanan, "Precious Metal Deposits Associated with Volcanic Environments in the Southwest"; Radtke, Rye and Dickson, "Geology and Stable Isotope Studies of the Carlin Deposit"; Barry Watson, "Bulk-Tonnage, Low-Grade Silver Deposits"; Fred Graybeal, "Characteristics of Disseminated Silver Deposits in the Western United States"; Byron Berger (upcoming paper), "Round Mountain, Nevada"; and, Michael Cruson, "Geology of the Cinola Deposit, Queen Charlotte Island, British Columbia".

Not yet in the literature, but significant to this body of knowledge, is the discovery of the McLaughlin Mine by Homestake in Northern California, and the development of the Enfield Bell Mine at Jerritt Canyon, Nevada, by Freeport.

THE EPITHERMAL MODEL

Understandings gained from trial-and-error development and sophisticated, directed research have gone into an Epithermal Gold Silver Model. This model has all of the problems of other geologic models, plus the capriciousness of rapidly varying processes in a near surface environment with low to moderate temperatures, densely fractured, tectonically active host rocks, and a mixed origin of hydrothermal solutions from metoric and magmatic waters.

Evolving knowledge of the epithermal environment is detailing a geologic model with the following characteristics:

1. Mineralization in sedimentary rocks or subaerial volcanics where late, multi-stage tectonic activity has created a plumbing system for entry of hydrothermal solutions. An extensional structural regime is favored to create the needed open spaces. The classic

horsetails, warps, simoids, and other structural irregularities are integral processes to the development of favorable environments.

2. Zones of permeable and porous host rocks accessible to fluids migrating away from fault systems.

3. A near-surface, low-temperature geologic environment related to a convecting geothermal cell with probable surface expressions of the cell such as silica or low-temperature alteration effects.

4. Varied chemical processes including:

   a. Fluids reacting with the wall rocks as they migrate away from the heat source.

   b. Temperature decreasing causing a decrease in solubility constants thereby complexing ions and precipitating metal bearing minerals from solution.

   c. Mixing of $\overline{H}$ fluids with more oxidizing ground water solutions near the surface.

   d. Boiling due to decrease in pressure near surface, causing fluid phase to be dominated by gaseous phase solutions which rise, condense and form acid waters, producing the argillically altered zones erratically overlying many of these deposits.

   e. Post-depositional supergene processes which often mask the primary processes but generally do not contribute to wholesale migration of silver and gold. (There are some notable exceptions to this observation, particularly the Charnacillo deposit in Chile.)

5. Erratic silver and gold distribution related to complicated fracture systems, silicification and diverse alteration assemblages. Precious metal deposition from the complex interaction of the processes will be highly erratic and difficult to explore and define. Metal distribution will be a geostatistician's dream and a nightmare of management.

This evolving model is not a fixed pattern into which a prospect may be fitted and validated. It is complicated, variable, capricious. The model is valuable when the explorationist studies the fundamental geologic processes embodied in it, processes which are now beginning to be understood.

Five deposits illustrate model consistencies and variations. These are (1) the U.S.G.S. conceptual genetic section at Creede, Colorado; (2) Larry Buchanan's model section at Guanajuato; (3) Byron Berger's genetic model sequence for Round Mountain, Nevada; (4) ASARCO's (F. Koutz) illustrative geologic section through the Hardshell deposit in Arizona; and (5) Bob Rye's conceptual genetic section for Carlin, Nevada.

ILLUSTRATION I

IDEALIZED NORTH-SOUTH SECTION THROUGH CREEDE MINING DISTRICT, COLORADO

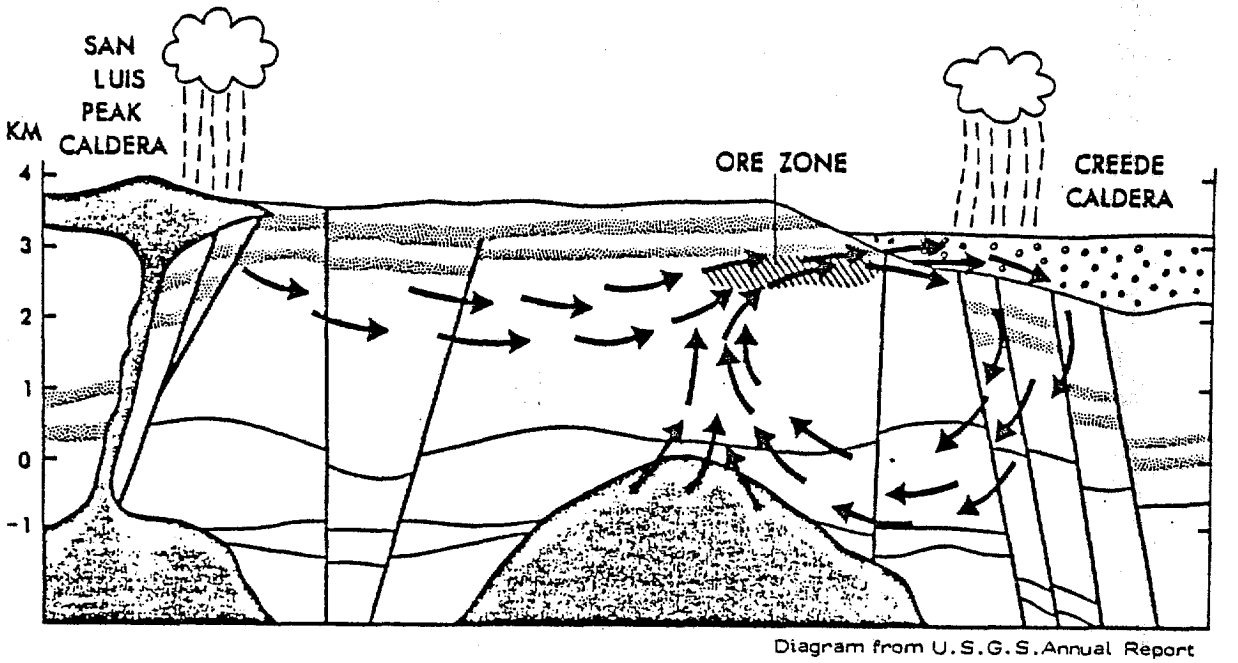
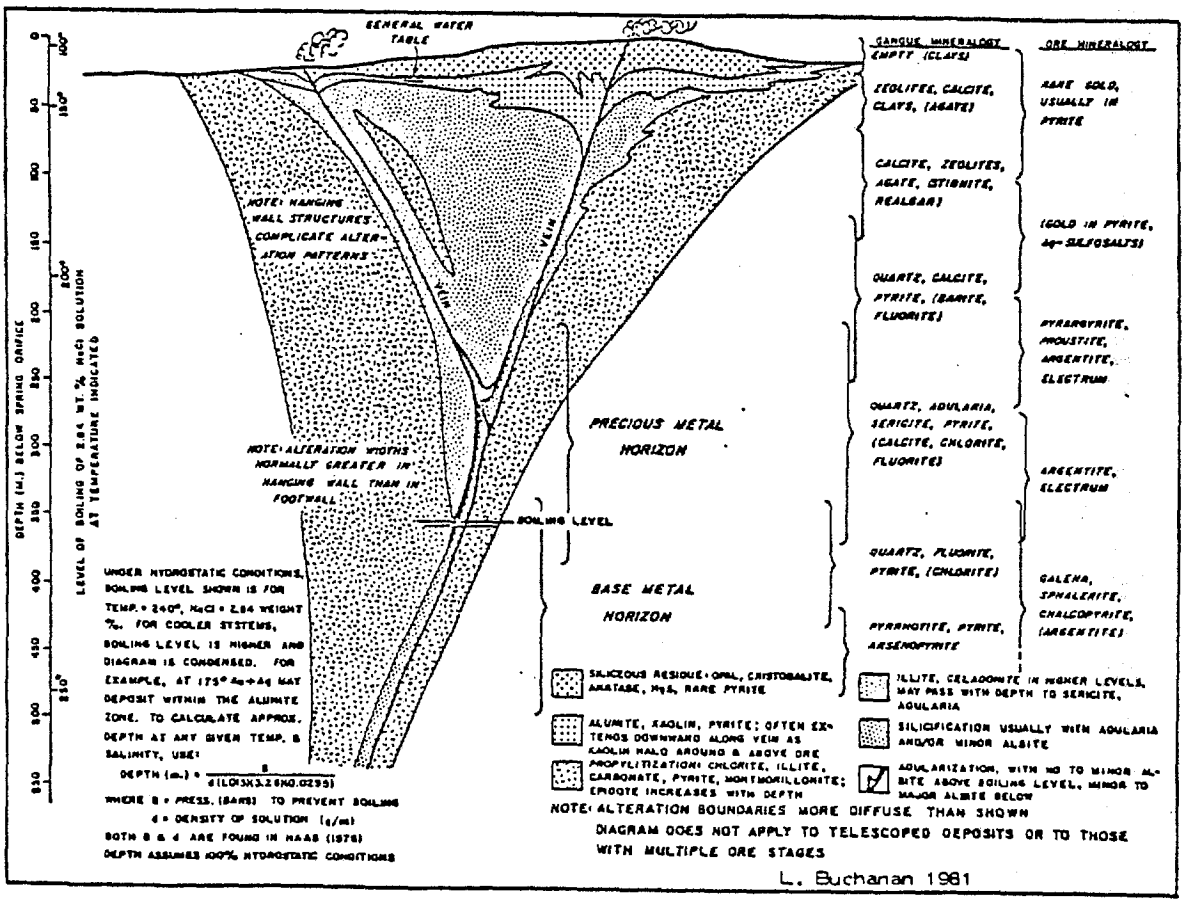


Diagram from U.S.G.S.Annual Report of 1980.

This generalized geologic and hydrological model by the United States Geological Survey shows the circulation and mixing of fluids in response to local heating by an igneous intrusion. Silver and other metals have been deposited in a boiling or ore zone in the Campbell Mountain Rhyolite.

ILLUSTRATION II

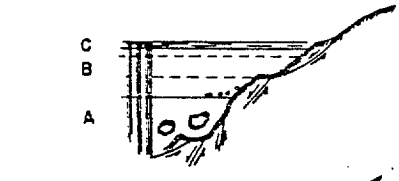
SECTION OF EPITHERMAL MODEL



Idealized Section of General Guanajuato Model showing alteration zones, mineral distribution, depositional conditions and metal distribution.

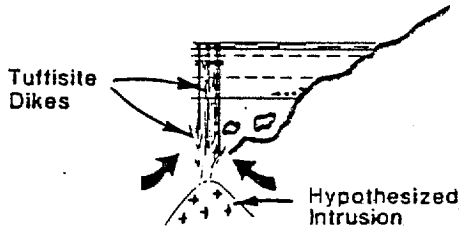
ILLUSTRATION III

GENETIC MODEL FOR THE DEVELOPMENT OF THE ROUND MOUNTAIN, NEVADA GOLD DEPOSIT



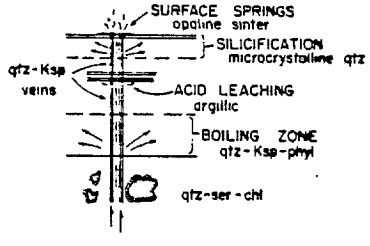
STAGE I

PRE HOST ENVIRONMENT
AND STRUCTURAL SETTING



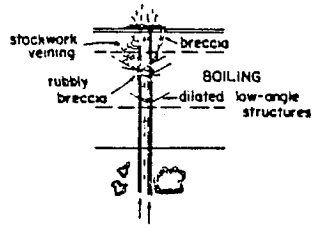
STAGE II

INTRODUCTION OF HEAT SOURCE



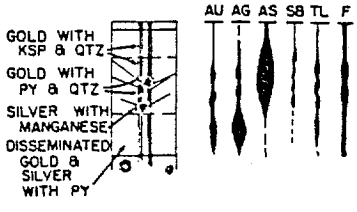
STAGE III

DEVELOPMENT OF THE
HYDROTHERMAL SYSTEM



STAGE IV

FINAL STAGE OF THE SYSTEM

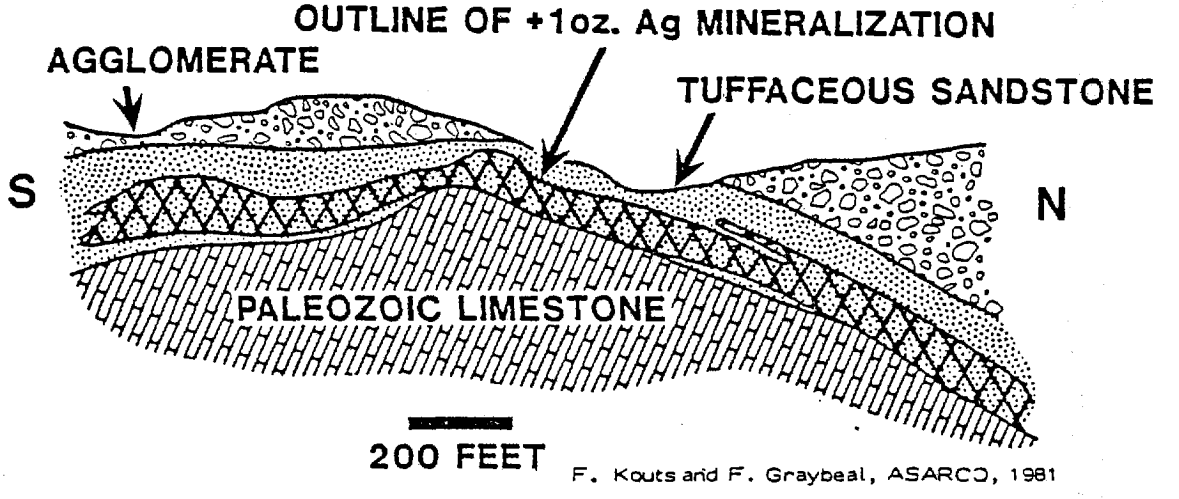


ZONING OF METAL DISTRIBUTION

Byron Berger
U.S.G.S. 1981

ILLUSTRATION IV

GENERALIZED SECTION THROUGH HARDSHELL DEPOSIT, ARIZONA

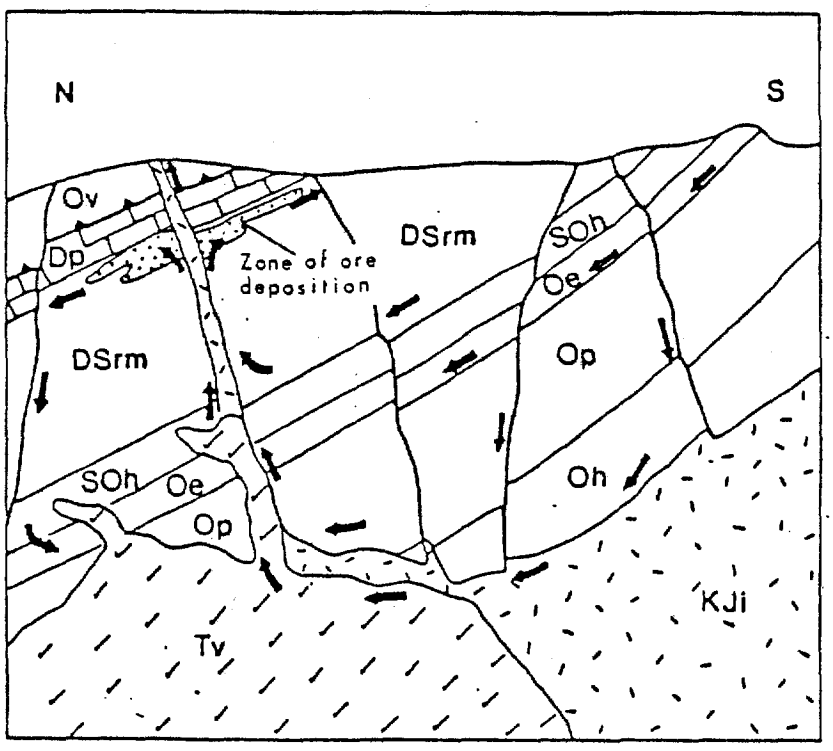


F. Kouts and F. Graybeal, ASARCO, 1981

This generalized section shows the relationship of plus one ounce silver mineralization and volcanic-sedimentary stratigraphy at the Hardshell silver, base metal, manganese oxide deposit in southern Arizona. This section illustrates (1) the lateral dimension of the epithermal model, (2) its relationship to a porous and permeable bed, and (3) the relationship of mineralization to the ground surface.

ILLUSTRATION V

HYDROTHERMAL SOLUTION PATHS AT THE CARLIN DEPOSIT, NEVADA



ROBERT RYE USGS, 1980

This generalized section shows the postulated path of hydrothermal solutions in the genesis of the Carlin Gold Deposit, Nevada. The zone of ore deposition is that favorable combination of a plumbing system for gold bearing solutions, favorable host rock with permeability and porosity, and depositional temperature and pressure relationships for gold deposition.

From the standpoint of practical mining exploration, mine evaluation and mine development, what do these epithermal models and genetic concepts mean?

We now more fully understand the geology of epithermal gold-silver deposits and the genetic processes that formed them. The model provides patterns or zones of mineralization, alteration, structure, and lithology which can tell us where we are, in the system, relative to high-grade and/or bulk tonnage gold-silver mineralization.

In the model, the enriched gold-silver zone has a top, a bottom, a shape, and relationships to structure and alteration patterns. The zones above, below and on the sides have mineralogic, geologic, temperature, pressure, and chemical features that are identifiable and characteristic. We determine the absence or presence of such features by field observations, geochemical sampling, and laboratory tests.

Epithermal deposits are characterized by late-stage, multi-stage tectonic activity which creates a plumbing system for entry of hydrothermal solutions. The deposits often have a roughly-undulating top that is related to ground surface at time of deposition and a bottom that may have an irregular root-like form.

These deposits are low-sulfide systems, related to structural features. Commercial grade gold and silver mineralization is found above the sulfide zone in a sulfur-poor, mixed-oxide system.

Higher grade gold-silver values are generally found in the upper part of the boiling zone. Depending upon depositional pressures and solution chemistries, gold and silver is deposited at temperatures in the range of 100-to-300 degrees centigrade. Fluid inclusion studies are a means to determine depositional temperatures, solution salinities, and the presence of a boiling phase. These temperatures become valuable as indications of the metal-rich zones in the system. Calcite silica relationships are also indicators of temperature and chemical relationships within the model.

The zone above a bonanza ore body may be very low in gold or silver but may contain a geochemical signature. Anomalous mercury, arsenic, flourine, antimony, barite, and other elements are important to exploration.

This zone has an alteration cap, containing illite, alunite, kaolinite, or other clay minerals.

These Gold-Silver Epithermal Model Concepts and new geochemical laboratory techniques are important to your:

1. Exploring for new deposits.

2. Expanding or verifying ore reserves at an operating mine.

3. Evaluating the potential of a new deposit or an operating mine.

In each case, increased knowledge of the size, shape, zoning and position of metal distribution, alteration and other geologic features, provides real or projectable data. Such data is crucial for mine planning and budgeting, for exploration budgeting and planning, and for new mine feasibility.

An understanding of the model is basic to an efficient selection of exploration methods.

Field observation and mapping are critical in area selection. One searches out and notes the surface expressions of the deposit.

Geo-chemical sampling and mapping are precious exploration tools, determining, high up in the system the absence or presence of target-indicative elements.

Because of limited sulfides and only minor magnetic variations, geophysical methods are not key. (Let me point out, however, that the systems are structurally-controlled and that alteration patterns are important . . . so, geophysical methods to ascertain structure or alteration, could be used.)

Drilling is the final exploration method. To be cost-effective, drilling should be employed only after extensive initial mapping, sampling, and data evaluation.

Having said this in a precise A, B, C manner, let me point out that the broken, altered nature of the tops of these systems means that potential deposits may be covered by valley fill. This encourages investigation of covered areas - but only where there are other ore indications.

Making the easy-to-state, difficult-to-do target determination and evaluation, demands top-quality explorationists who know the model concepts, open-minded exploration and management direction, and a flexible program for the search.

Here is some common-sense input for your gold-silver exploration, evaluation or mine development:

. Old information may be important. Centuries of search for gold and silver has left us old tracks which may lead to areas where new ideas can be applied.

. The genetic processes forming these deposits are consistent. The formational systems are near surface, low-temperature and moderate pressure. Because of this, surface expressions of epithermal deposits are variable.

. Metamorphism may have altered the expression of these deposits in older terrains. There is evidence that these deposits are not time dependent.

. Fluid-inclusion studies sound pretty esoteric. Because temperatures are key to whether or not ore was found in the system, and because these studies

determine temperatures at the time of deposition, fluid inclusion studies are as down-to-earth and important as older-and-long-accepted testers - assays, for example.

- Geologic observations and geochemical work are keys in the study of these deposits. There is no substitute for a geologist who knows these systems, particularly their surface and underground expressions, and how to correlate data.

- You who are not geologists and want to use this information in exploration, evaluation or mine development, should hire a knowledgeable geologist. If he cannot explain clearly to you how these concepts apply to your deposit, fire him and get one who can.

THE EPITHERMAL MODEL

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UNITED STATES DEPARTMENT OF INTERIOR GEOLOGICAL SURVEY

Comparative Anatomy of Epithermal Precious- and Base-Metal Districts Hosted by Volcanic Rocks: A Talk Presented at the GAC/MAC/GGU Joint Annual Meeting, May 11-13, 1983, Victoria, British Columbia

by

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Open-File Report 83-710

This report is preliminary and has not been reviewed for conformity with U.S. Geological Survey editorial standards and stratigraphic nomenclature.

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1/23/84
xc: JDS
FRK
HGK
GJS

Abstract[1]

In order to distinguish dissimilar from similar features of epithermal districts, lithotectonic, mineralogical and geochemical traits are compiled for 15 such districts. The districts occur in structurally complex settings associated with silicic to intermediate volcanics. Affiliation with subduction environments on a continental scale and caldera settings on a regional scale is common but is not demonstrable for all districts. Most deposits formed near the end of major volcanism, but some formed considerably later. Paleodepth to the top of the ore is 300-600m for most districts, although Au-rich districts appear to be shallower. The lateral extent of the ore zone is highly variable and far exceeds the limited vertical range (300-800m). Most ore was deposited from dominantly meteoric fluids ranging in temperature from 220°-290°C. Salinities ranged from 0-13 wt % NaCl equiv., and typical values were 1-3 wt %. Although noted for eight deposits, boiling is clearly associated with precious-metal deposition in only two deposits. Four districts, typified by Goldfield, Nev., are characterized by a highly sulfidized mineral assemblage, advanced argillic alteration, and ore deposition closely following emplacement of the host rock. The remaining eleven districts highlight a second, discrete type of deposit. They contain adularia, exhibit sericitic ± argillic alteration, and were mineralized significantly after emplacement of the host rock. The latter category includes two subgroups: Ag- and base-metal-rich deposits (e.g., Creede, Colo.), and Au-rich, base-metal-poor deposits (e.g., Round Mtn., Nev.).

[1] Abstract published in GAC/MAC/CGU Joint Annual Meeting, May 11-13, 1983, Program with Abstracts, vol. 8, p. A31.

List of Slides[1]

Slide	
1	Gold ore from Goldfield, Nev. (from Roger Ashley, U.S. Geological Survey)
2	Map showing locations of the epithermal precious- and base-metal districts studied.
3	Diagram showing groupings of the epithermal precious- and base-metal districts studied based on mineralogy, lithotectonic setting and geochemistry.
4	Two ternary diagrams showing relative production value based on 1973 prices for Au-Ag-(Pb+Zn) and Cu-Ag-(Pb+Zn). Some districts had no Cu production reported and so are not plotted on the Cu-Ag-(Pb+Zn) diagram.
5	Simplified geologic maps of Tonopah, Nev. and Guanajuato, Mexico shown at the same scale. Relative production is shown by the areas of the circles.
6	Slide showing outline of the talk.
7	Distinctive groupings of the epithermal districts studied based on mineralogy.
8	Examples of the paragenetic sequence for precious and base metals.
9	Ore from the Creede district, Col.
10	Typical alteration zoning for the highly sulfidized and less sulfidized districts.
11	Composition of rock(s) hosting the ore in the epithermal districts studied.
12	Timing of ore deposition with respect to volcanism.
13	Common structural settings of the epithermal districts studied.
14	Mineralized and unmineralized calderas in the San Juan Mtns., Col. (from Steven and Eaton, 1975).
15	Relative timing of emplacement of host rocks and ore deposition.
16	$\delta^{18}O$-δD diagram for the Creede district, Col. (modified from Bethke and Rye, 1979).

[1] Slides are at times slightly out of sequence because some color slides have been grouped together to reduce the reproduction cost to the public.

Slide

17 Temperature and salinity ranges from fluid inclusion studies for the epithermal districts studied.

18 Temperature and salinity ranges from fluid inclusion studies for the Colqui district, Peru (from Kamilli and Ohmoto, 1977).

19 Temperature and salinity ranges from fluid inclusion studies for the Sunnyside mine, Eureka district, Col. (modified from Casadevall and Ohmoto, 1977).

20 Paleodepths to the top of the ore bodies for the epithermal districts studied.

21 Similar characteristics of the epithermal districts studied.

22 Distinctive characteristics of the epithermal districts studied.

The data I am presenting are based on a detailed study of the characteristics of fifteen Tertiary epithermal precious- and base-metal districts hosted primarily by volcanic rocks. This slide (slide 1) from Roger Ashley shows the inspiration for this kind of study. It shows high-grade gold ore on silicified wall rock from Goldfield, Nevada. My coauthors and I have compared and contrasted the mineralogical, lithotectonic, and geochemical characteristics of these districts in order to identify unifying and discriminating themes through which models can be evaluated. We began the study with hopes of looking at 40 districts of assorted ages and geographic locations, but it very soon became apparent that this number was unrealistic. Not only are data limited with respect to separating observation from interpretation, much data is nonexistent or unavailable. So we limited this initial study to fifteen districts (slide 2) associated with Tertiary volcanic rocks. We view these data as a base to which additional near-surface precious-metal districts can and should be integrated as time and availability of data permit.

This slide (slide 2) shows the locations of the districts we studied. These specific fifteen were selected because they are typical, and in some cases, classic examples of near-surface, low-temperature ore deposits. Also critical was the existence and accessibility of published geological and geochemical data. Two of the districts are located in Peru, two in Mexico and the remaining 11 in the western United States. We have done an intensive field and laboratory study of the Creede district, Colorado and have gathered information on the other districts through study of the literature and personal communications with geologists who have worked in the districts.

Some districts which have been excluded merit a comment in passing. We have for now excluded districts in which telluride ores are dominant (e.g., Cripple Creek) because we think these may be a distinctive class of their own, and that documentation should be a subsequent stage in this study. Districts such as Park City, Utah, Providencia, Mexico, and Carlin, Nevada, were excluded on this round because they are primarily hosted by sediments. Undoubtedly, many of the ore-forming processes in these sediment-hosted gold deposits will be the same as for epithermal volcanic-hosted deposits; studies are needed to evaluate the specific influence of the chemical and physical parameters of different host rocks on these near-surface hydrothermal systems.

Based on the 15 epithermal districts studied, we have identified 2 major groups (slide 3), those which contain a highly sulfidized mineral assemblage and those which contain a less highly sulfidized mineral assemblage. Each of these groups contain a gold-rich and a silver-rich subset. For the highly sulfidized group, coded in warm colors, Julcani, Peru (in yellow) is representative of the Ag-rich member and Goldfield, Nevada (in red), the Au-rich. For the less sulfidized group, coded in cool colors, Creede, Colorado (in green) is the Ag-rich example and Round Mountain, Nevada (in blue) represents the Au-rich member. There appears to be a further breakdown of the Ag-rich less sulfidized group on the basis of higher and lower base-metal production, exemplified by Colqui, Peru and Tonopah, Nevada. However, because additional documentation is limited for this distinction, I will only be making brief mention of it.

The next slide shows some of these groupings with respect to production value (slide 4). This figure was prepared by taking available production figures and converting them to their dollar value using 1973 prices. Dollar value is plotted here; the normalizing numbers used are in the upper right hand corner. This figure is really two figures - one showing Ag-Au-(Pb+Zn) ternary diagram and the other showing a Ag-Cu-(Pb+Zn) ternary diagram. Districts for which the copper production reported is insignificant have not been plotted on the Ag-Cu-(Pb+Zn) ternary diagram. I believe these data are qualitatively significant in spite of numerous limitations of the figure. The Ag and Au production numbers should be pretty reliable; however, the base-metal numbers are much less reliable because many companies were not and are not interested in the less valuable base metals and so often did not report them. Also, readily available production data for five of the districts is only for a limited number of years. 1973 metal prices were chosen because they are prior to the wild fluctuations in precious-metal prices; using values from other years, however, does not significantly change the distribution of districts in the figure. The highly sulfidized districts, in red (Goldfield, Nev. and Summitville, Col.) and yellow (Julcani, Peru and Red Mtn., Col.), are all uniquely higher in copper. The gold-rich districts of both the highly sulfidized group (in red) (Goldfield, Nev. and Summitville, Col.) and the less sulfidized group (in blue) (Oatman and Round Mtn.) are well separated from all the Ag-rich districts. Amongst the silver-rich districts, there is a distinctive high base-metal group (Red Mtn., Lake City, and Eureka, Col., and Colqui, Peru). Lake City, Col. is shown in green and yellow because it contains two mineral assemblages, discrete in space and in time, which appear to fall into two different categories.

The areal size range of the 15 districts we looked at is large, from roughly 4 to 200 square km. The lateral extent of each district far exceeds the vertical extent, which is on the order of 200-1000 meters. So these districts are geometrically like pancakes. In general, areal size and production size of a district do not appear to have a linear relationship. This slide (slide 5) shows the mineralized veins in red at Guanajuato, Mexico and the productive zone at Tonopah, Nevada, outlined in red dots. Both districts are shown at the same scale. Guanajuato is obviously many many square kilometers larger than Tonopah but has only produced about five times in value what Tonopah has produced, as indicated by the relative areas of the circles.

One of the most critical distinctions among the epithermal districts we have evaluated is the mineralogy (slide 6). Unique to the highly sulfidized group, shown in red and yellow, is the occurrence of enargite (slide 7). Enargite is not present in any of the other districts, shown in blue and green. Advanced argillic alteration, which is characterized by extreme base leaching is also unique to these districts, in red and yellow. The minerals alunite and kaolinite, which are typical of this kind of alteration, are common within all these districts. Dickite and zunyite (also typifying advanced argillic alteration) are present at Red Mtn., Col. Very minor zunyite has also been reported in the Sunnyside mine, Eureka district which was surprising to us, because there are no other indications of advanced argillic alteration. Primary alunite is not

found in any of the other districts we looked at, although secondary alunite is present at Creede, Col. and Round Mtn., Nev.

Within the less sulfidized group of districts, shown in blue and green, Oatman, Ariz., and Round Mtn., Nev., in blue, are distinctive because they contain major native gold and silver and only rare base metals. In fact the native gold and silver are the only minerals of economic importance in these districts. The green subdivision shows districts containing not only native gold and silver, but also numerous sulfosalts, especially the tetrahedrite-tennatite series and also silver and gold selenides in those districts with fewer base metals. Base-metal bearing minerals, such as sphalerite, galena, and chalcopyrite, are common and form a major constituent of the veins at Colqui, Pachuca, Eureka, Creede and the earlier deposits of Lake City. With respect to gangue minerals, the presence of potassium feldspar, often identified as adularia, is characteristic of the green and the blue group--that is, all the less sulfidized districts--although it has not been identified at Colqui. Adularia is notably absent from any districts of the highly sulfidized group.

Quartz and pyrite are the only minerals that occur in all the districts we examined. Barite is present in most of the districts but is notably absent from the high-Au, less-sulfidized group, in blue. Fluorite and some variety of carbonate occur in many of the districts in all the color subsets. Hematite is invariably the iron-oxide stable in the veins, occurring in almost all the districts; in a few, it has been identified as secondary. Sericite and/or illite or illite-smectite mixed-layer clays have been identified in all the districts except Round Mtn., Nev.

The temporal and spatial variations in mineral assemblages in a district are fingerprints of the evolution of the ore fluid. These variations should provide insights into important variations in ore-depositing processes which result in differences among epithermal precious-metal districts. Unfortunately, detailed documentation of the temporal and spatial variations is often lacking, and generalizations are difficult to support. Even when a good paragenetic sequence is defined, it often can be applied only to one or two localities within a district, so much paragenetic data must be used with caution. In general, precious metals do occur later than most base metals in many of the districts we have studied (slide 8) (e.g. Guanajuato, Mexico and the Sunnyside mine, Eureka district, Col.). However, there is really no clear cut rule. At Creede, Col. and Colqui, Peru, for example, precious-metal deposition took place earlier than much of the base-metal deposition. This doubly polished thin section from the Creede district (slide 9) shows the early, relatively fine-grained generation of sphalerite, galena and silver-bearing tetrahedrite overlain by a coarse-grained sphalerite generation which is barren of silver. Of course, there is a real problem if there have been multiple mineral depositing systems such as at Lake City, Col. Although the precious metals appear to have been deposited late in the paragenetic sequence at Lake City, Slack in 1980 has also suggested that the precious-metal ores could have been an early part of a second distinctive ore-depositing system.

With respect to vertical spatial zonation, amounts of base metal tend to increase in depth in many of the districts, although this is well documented

in only a few of the 15 districts (e.g. Creede, Guanajuato). Gold and silver tend to be more abundant higher in the districts as compared with the base-metal distribution (e.g. Summitville, Goldfield).

Lateral spatial zonation in mineralogy is consistently distinctive with respect to alteration assemblages. Few generalizations can be made with respect to ore mineral distribution although several deposits display a lateral zonation of silver and base-metals (i.e., Colqui). With respect to alteration (slide 10), the most intense alteration is, of course, closest to the vein structures. In the case of the highly sulfidized group of districts, silicification occurs nearest to the vein, surrounded by advanced argillic alteration. This zone is surrounded by a zone of argillic alteration, which is characterized by clay minerals such as montmorillonite and kaolinite. Sericitic alteration (which denotes the presence of mica type minerals like illite) sometimes occurs within this argillic zone. Further outward, this alteration grades into a pre-ore regional propylitic alteration. For the less highly sulfidized group of districts, silicic alteration, often with K-feldspar present, occurs closest to the vein. This grades outward into sericitic alteration. Further from the veins, argillic alteration may be present enclosed by regional propylitic alteration. I have noted the possible presence of mixed-layer clays within the sericitic zone because sericite is such a broad field term and, at least in the case of Creede, Col., the supposed sericitic alteration is actually mixed-layer illite-smectite. In the case of Creede, the pre-ore regional alteration was K-metasomatism, which may have precluded propylitic alteration or may have been a substitute for it. This issue deserves further studies.

Lithotectonic setting does not appear to provide a useful means of distinguishing among the 15 epithermal districts we examined. The host rocks for these districts are rhyolitic to andesitic in composition (slide 11). The specific composition of the host rock appears to be important only in that it reflects that shallow precious-metal-bearing hydrothermal systems commonly develop in silicic, calc-alkalic volcanic environments. A rock unit of one composition frequently hosts all the ore, which probably mainly reflects the permeability of the unit and a fortuitous location. Among the 15 districts we looked at, rhyodacite is the most typical host rock.

In general, ore deposition took place close to the end of volcanism in the district (slide 12). For 10 of the 15 districts in this study, there was no volcanism or very minor volcanism subsequent to ore deposition. This timing could suggest that only when the magmatic system has waned can an ore-bearing hydrothermal system maintain itself long enough, and at the appropriate low temperatures, to form a significant ore deposit. On the other hand, deposits from hydrothermal systems which may have developed earlier in the volcanic cycle may be less common because they have less likelihood of preservation due to continued explosive volcanic activity.

The structural setting is not a determinant for distinguishing among the epithermal precious-metal districts evaluated. For example, the most common setting in our study is the caldera (slide 13), and both the highly sulfidized group (red) and the less sulfidized group (green and blue) occur within that setting. Both groups also occur in areas of silicic dome

development, exemplified by a more highly sulfidized district, Julcani, and a less sulfidized district, Silver City, Idaho. I do not want to overemphasize the importance of the caldera setting, because as Ed McKee pointed out in 1979 in Nevada, not all calderas, indeed, relatively few percentage-wise, appear to have been mineralized. Jim Rytuba in 1981 also documented the lack of known mineralization associated with most calderas in the western United States. This slide (slide 14), taken from Steven and Eaton's 1975 paper, shows the calderas developed in the San Juan Mtns. of Colorado. Only 1/3 of them have associated economic mineralization, as indicated by the stippled pattern. Pete Lipman, Tom Steven and co-authors in 1976 noted that the role of the calderas with respect to mineralization in the San Juan Mtns. was one of formation of zones of weakness above a magma chamber. The point really is that calderas can and do provide an excellent plumbing system for ore-depositing hydrothermal systems when they do develop. The ores at Goldfield, Nevada, for example are approximately 8 m.y. younger than the caldera along whose ring fracture they occur.

Where we do see a consistent distinction among the epithermal districts we have examined is in the relative timing of emplacement of the host rock and of ore deposition (slide 15). Deposition of the more highly sulfidized ores such as at Goldfield and Julcani appear to have very closely followed emplacement of the host rock. The genesis of the host rock and of the ore-bearing hydrothermal system appears therefore to be related for the more highly sulfidized group. In contrast, deposition of the less sulfidized ores, exemplified by Creede, Colorado, usually took place more than 1 m.y. subsequent to the formation of the host rock. In the Eureka district, Colorado, the time gap is as long as 10 to 15 m.y. Ambiguities in some districts remain as indicated by the less than 1 m.y. category; two less sulfidized districts, Silver City and Round Mtn., have times of mineralization and of host emplacement differing by about 1/2 m.y., but there is overlap in the dates if analytical uncertainty is taken into account.

The geochemical parameters of the ore fluid as determined mainly through isotopic and fluid inclusion studies[1] are fairly similar between the highly sulfidized and less sulfidized groups. Among the fifteen districts, three very detailed oxygen and deuterium isotope studies and five limited studies have been done. The studies suggest that the ore-bearing fluids were made up dominantly of meteoric water. This does not preclude the presence of a magmatic component of 5 or 10%. Some evidence does exist for this magmatic component. This slide (slide 16), for the Creede district, Col., is an oxygen/deuterium diagram and includes the meteoric water line and the box where waters of deep seated origin are conventionally believed to fall. The circles represent samples for which both $\delta^{18}O$ and δD have been determined; the triangles are samples for which only $\delta^{18}O$ have been determined. The early rhodochrosite (yellow) appears to have had a deep-seated, possibly magmatic origin, although the sphalerite (red), quartz (green) and alteration minerals (blue) have meteoric water signatures.

[1] The compiled geochemical data for the fifteen districts is available in USGS open-file report 83-450.

Temperatures of mineral deposition (slide 17) range from about 140°C to 325°C but during metal bearing stages typically range between 230° and 270°C. Late-stage, cooler, metal-poor fluids typically deposited minerals at temperatures between 200° and 140°C. Salinities range from near 0 to 13 wt. % NaCl equiv., with typical values close to 1 to 3 and 6 to 8 wt. % NaCl equiv. This slide (slide 18) shows the temperatures and salinities determined from fluid inclusions plotted against paragenetic sequence for the Colqui district, Peru, from a paper by Kamilli and Ohmoto in 1977. It shows the typical temperature range from early temperatures close to 300°C down to 140°C late in the evolution of the system. The salinity range is from 0 to 12 equiv. wt. % NaCl and is clearly bimodal in nature. Recent work by Foley, Bethke and Rye (1982) in the Creede district Col., where the salinities have a similar bimodal nature, shows that the lower salinity waters for Creede come from pseudosecondary inclusions resulting from periodic flushings of the system with barren groundwaters. Therefore, the ore-depositing fluids at Creede are those with the higher salinity, averaging between 4 and 8 equiv. wt. % NaCl. The bubbles in slide 18 represent boiling, which in the case of Colqui, occurred during precious-metal deposition as shown by the "$" sign. Boiling certainly is an effective mechanism for depositing ores. However, boiling unequivocally associated with precious-metal deposition has been demonstrated for only two of the fifteen districts. Precious-metal deposition due to boiling has been proposed for other districts based on less definitive evidence (Guanajuato, Tonopah). It remains to be seen whether it is always a requirement for precious-metal deposition. Of course, the common fine-grained nature of many epithermal precious metal ores makes documentation of boiling difficult. Boiling also has been noted at numerous deposits during times other than precious metal deposition (slide 19), as at the Sunnyside mine in the Eureka district, in the later barren stage. This suggests that caution should be used when applying evidence for boiling as an indicator of precious-metal deposition.

This slide (slide 20) shows paleodepths to the top of the ore, estimated either from a geologic reconstruction of the area (shown by the solid circles) and/or from pressure-temperature-composition conditions determined from fluid inclusion studies (shown by the open circles). Both methods have considerable uncertainties. Depth-pressure estimates based on fluid inclusion studies require assumptions with respect to the pressure-controlling medium (i.e., hydrostatic or lithostatic); in addition, pressure estimates from fluid inclusions could be too low if small amounts of CO_2 or other noncondensible gases are present. Bodnar and Kuehn (pers. comm., 1982) have shown that small amounts of CO_2 typically would not be visible in fluid inclusions or detected by freezing studies. Most deposits formed at depths between 300 and 600 meters. Colqui, and possibly the Sunnyside mine, Eureka district and Red Mtn. are thought to have formed deeper. At the Sunnyside mine, Eureka, the deeper value is for the first five stages and the shallow value is for the final, sixth stage. Casedevall and Ohmoto (1977) proposed that this large change in paleodepth to the top of the ore may have been due to extensive erosion--although 1000 meters is a little eyebrow raising. Another possibility that they recognized is that pressure conditions may have changed from lithostatic to hydrostatic. This kind of geologic dilemma in the data is just one of many which still need to be addressed for epithermal districts. The most gold-rich districts - the

less sulfidized Oatman and Round Mountain districts (in blue) and the more highly sulfidized Summitville and Goldfield districts (in red) - appear to have formed generally at shallower depths, usually less than 300 meters. Round Mtn. may have formed as shallow as 50 meters and sinters are still preserved there. Delamar is the only apparently shallow silver-rich district, for which I have no explanation. Most of these shallow values are based on geologic reconstruction estimates which have large uncertainties. Fluid inclusion studies would help to document these shallow paleodepths better.

In summary, these 15 epithermal districts studied have some similar (slide 21) and some distinctive characteristics (slide 22). They occur in similar petrotectonic settings (although, of course, one of the selection criteria was a volcanic host), have similar temperature and salinity ranges, have similar vertical ranges of ore, have fairly similar calculated paleodepths to the top of the ore, and were deposited from dominantly meteoric waters. Other unique characteristics distinguish a highly sulfidized group and a less sulfidized group, each with gold-rich and silver-rich members. The highly sulfidized group is distinctive with respect to the higher sulfur vein-mineral assemblage, typified by enargite, the presence of advanced argillic alteration, the close timing of ore deposition and emplacement of host rock and noteworthy Cu production. The less sulfidized group has, obviously, a lower sulfur vein-mineral assemblage; enargite is absent and tetrahedrite is the common Ag-bearing sulfosalt. Alteration in this less sulfidized group is typified by sericitic and argillic alteration, and adularia is a stable vein mineral; primary alunite is absent. Ore deposition took place considerably later than emplacement of the host rock, and Cu production is low.

Not surprisingly, this study raises more questions than it answers. For example, why are some hydrothermal systems richer in sulfur? Does the more recent emplacement of the host rock in the history of the ore deposit permit a higher magmatic component, resulting in more sulfur? Why do some late intrusions into the ring fractures of calderas develop associated ore-bearing hydrothermal systems while other apparently identical intrusions do not? If 220 to 290°C is an optimum temperature range for ore deposition, how do such fluids get to within 50 meters of the surface (i.e., Round Mtn.)? And just how critical is boiling for the deposition of precious metals. Need I say more than "more work is indicated" except to add that a critical part of the search for answers is to seek the right questions to ask and this can be accomplished by looking in depth at the facts that we already have.

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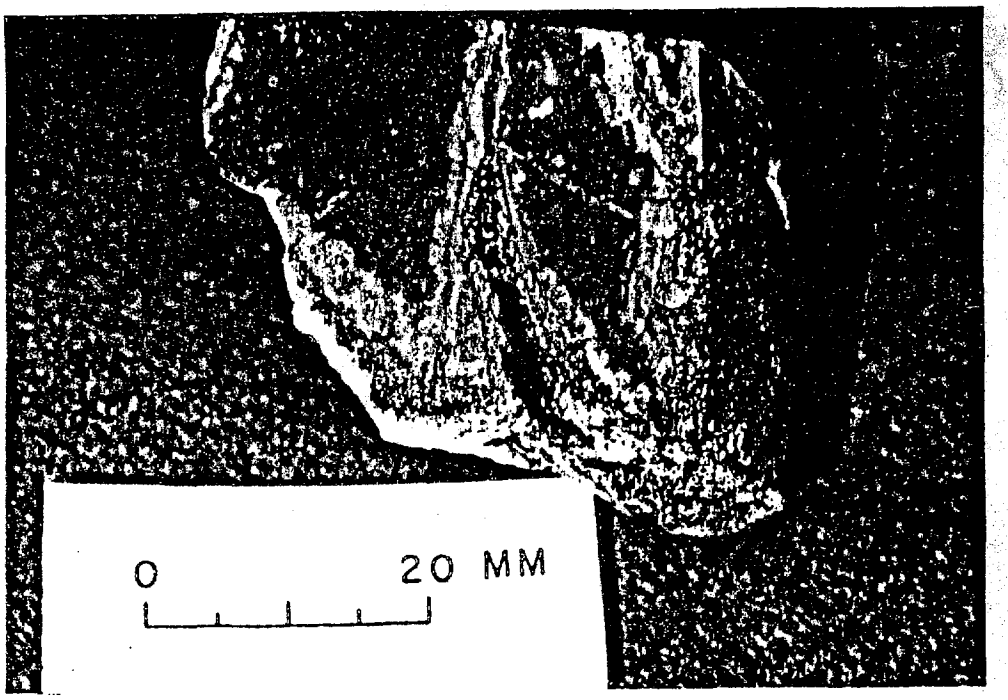
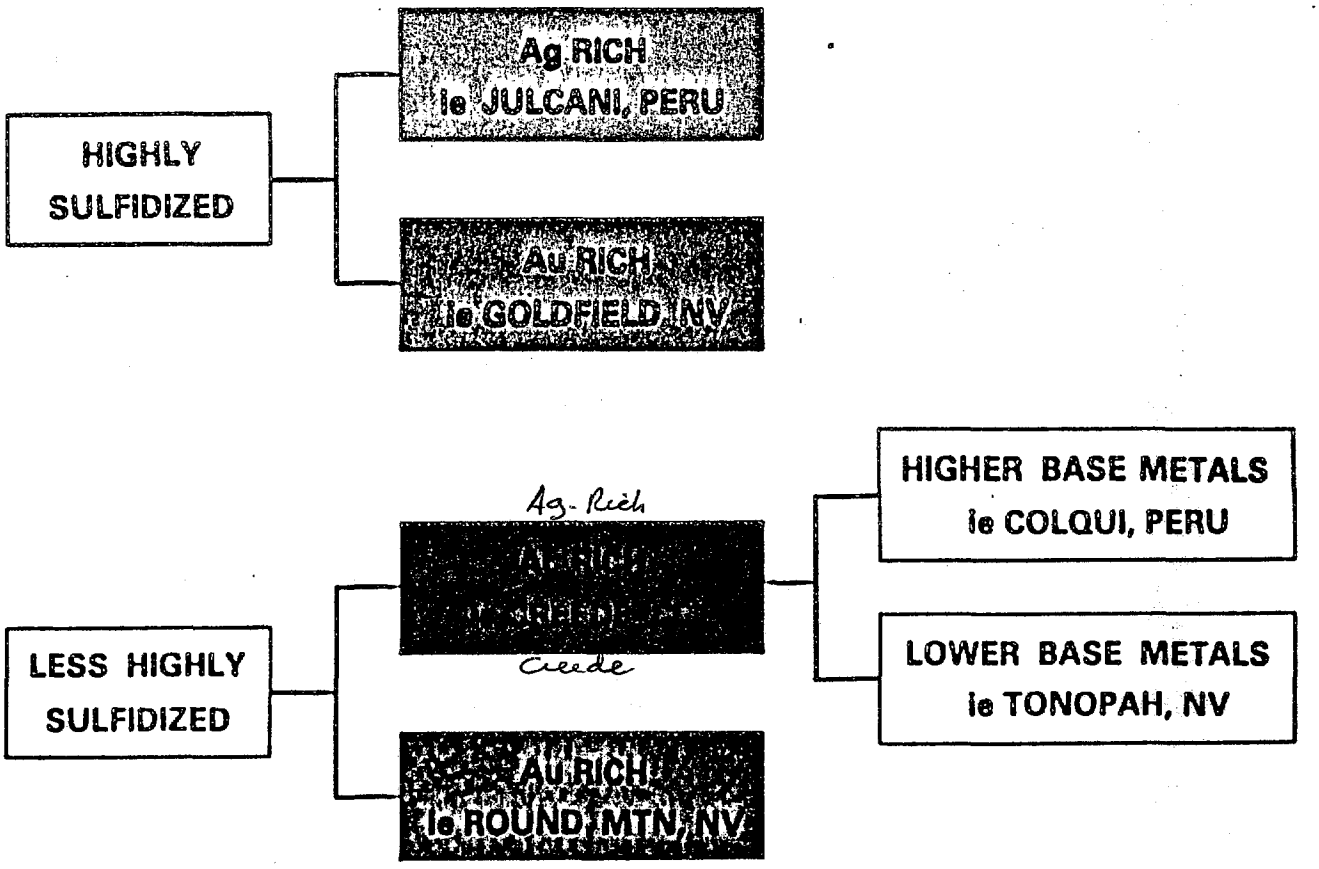
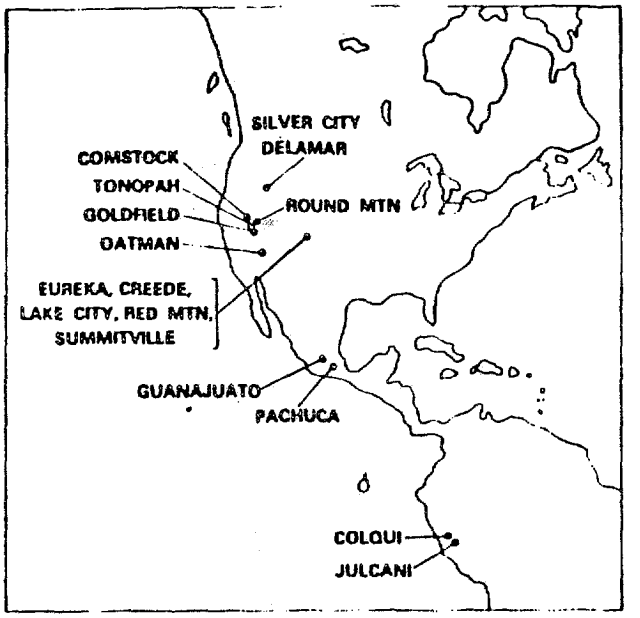


SLIDE 1



SLIDE 3



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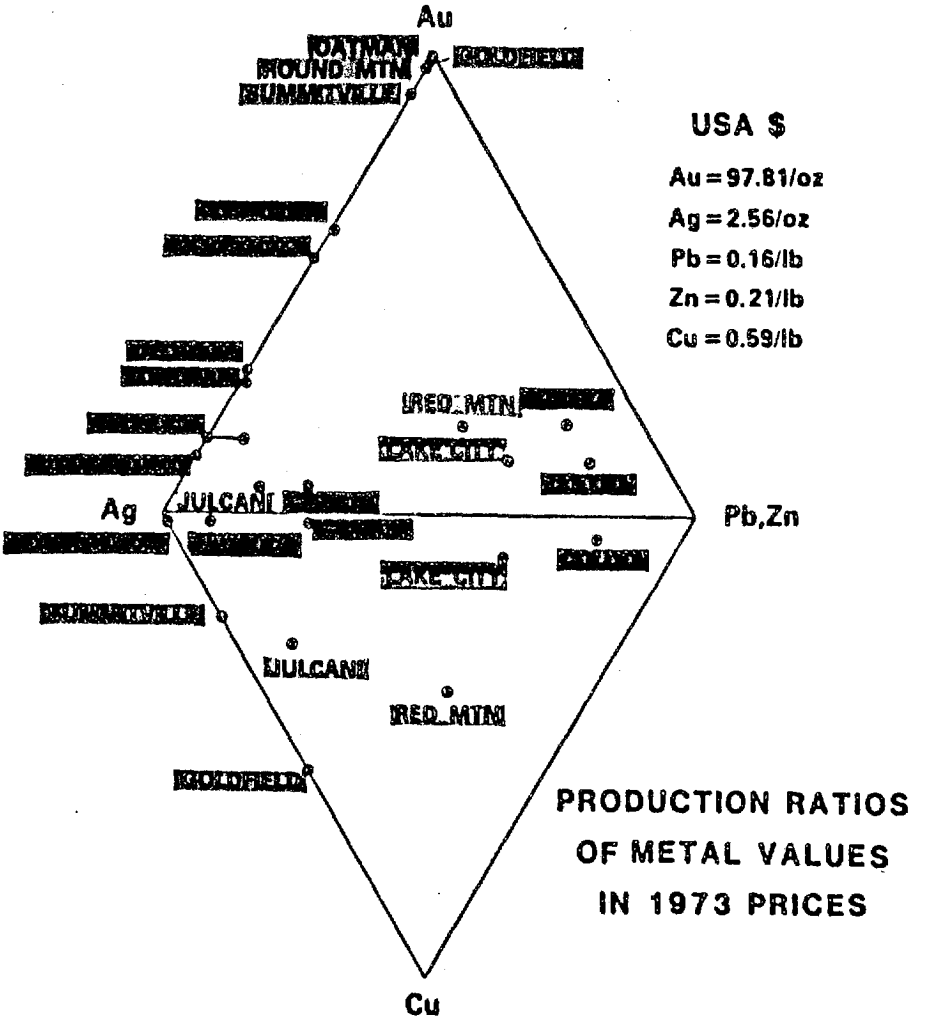

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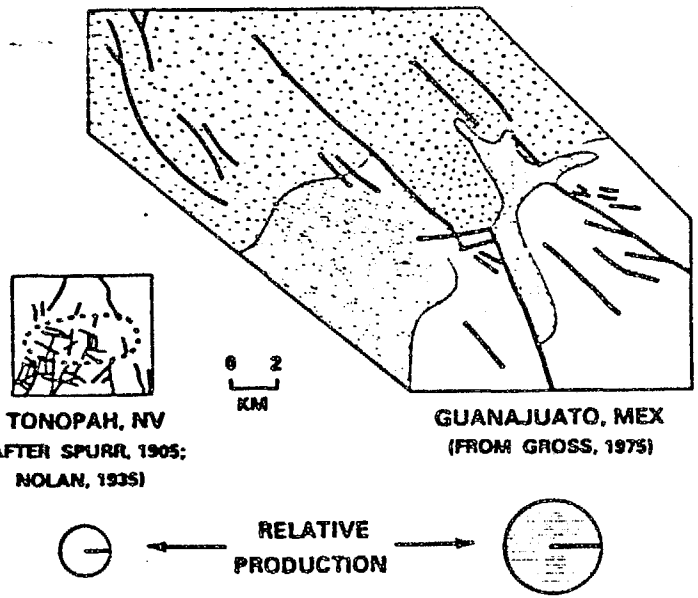


SLIDE 5

ENARGITE
HYPOGENE ALUNITE
MAJOR KAOLINITE

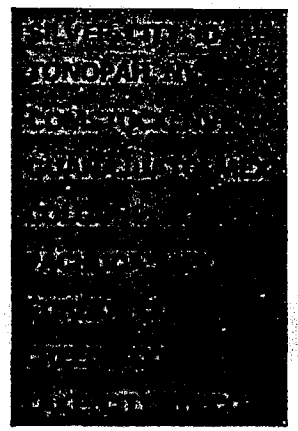
GOLDFIELD, NV
SUMMITVILLE, CO

RED MTN, CO
JULCANI, PERU
LAKE CITY (II), CO

MAJOR
Au° AND Ag°

OATMAN, AZ
ROUND MTN, NV

ADULARIA
Au°, Ag°, AND
BASE METALS



Silver City, ID
Tonopah, NV
Comstock, NV
Guanajuato
Colquiri, Peru
Pachuca, Mexico
Eureka, CO
Creede, CO
Lake City, CO

SLIDE 7

SLIDE 9

MINERALOGY

LITHOTECTONIC SETTING

GEOCHEMISTRY

SLIDE 6

PARAGENETIC POSITION OF PRECIOUS METALS

LATE: ROUND MTN
GUANAJUATO, MEX
EUREKA, CO
LAKE CITY, CO
SUMMITVILLE, CO

EARLY: CREEDE, CO
COLQUI, PERU

SLIDE 8

HIGHLY SULFIDIZED	LESS SULFIDIZED
SILICIC	SILICIC ± ADULARIA
↓	↓
ADVANCED ARGILLIC	SERICITIC (± MIXED-LAYER CLAY)
↓	↓
ARGILLIC ± SERICITIC	± ARGILLIC
↓	↓
REGIONAL PROPYLITIC	± REGIONAL PROPYLITIC

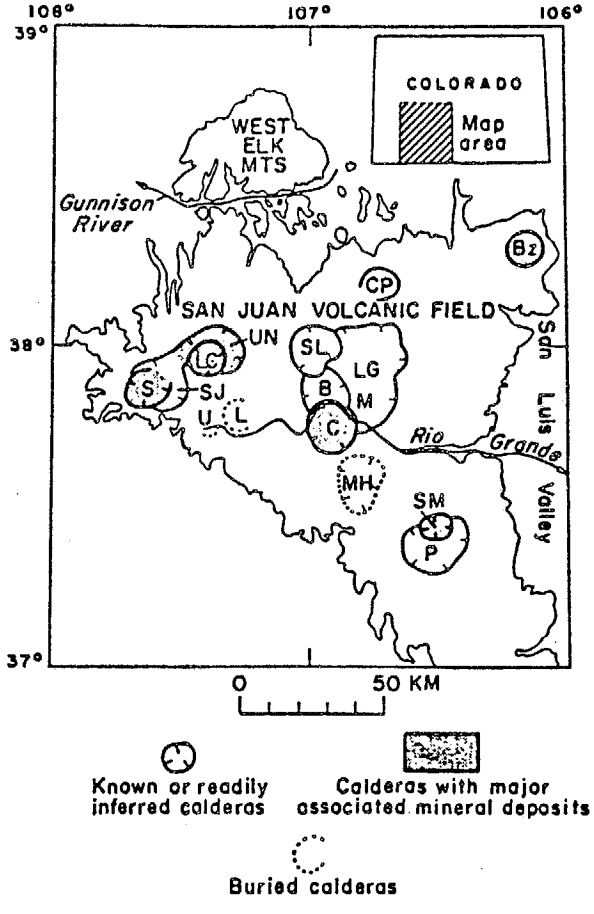
SLIDE 10

HOST ROCK	DISTRICTS	EXAMPLE
RHYOLITE	2	CREEDE, CO
RHYODACITE	5	GOLDFIELD, NV
LATITE	1	OATMAN, AZ
ANDESITE	2	TONOPAH, NV
FELSIC TO INTERMED. VOLCS.	5	PACHUCA, MEX
VOLCS. AND SEDS.	2	GUANAJUATO, MEX

SLIDE 11

VOLC. SUBSEQUENT TO ORE DEPOSITION	DISTRICTS	EXAMPLE
NONE	6	CREEDE, CO
BAS. ± RHY.	4	GOLDFIELD, NV
ANDES. ± BAS.	3	JULCANI, PERU
BAS., ANDES., RHY.	1	TONOPAH, NV

SLIDE 12



SLIDE 14

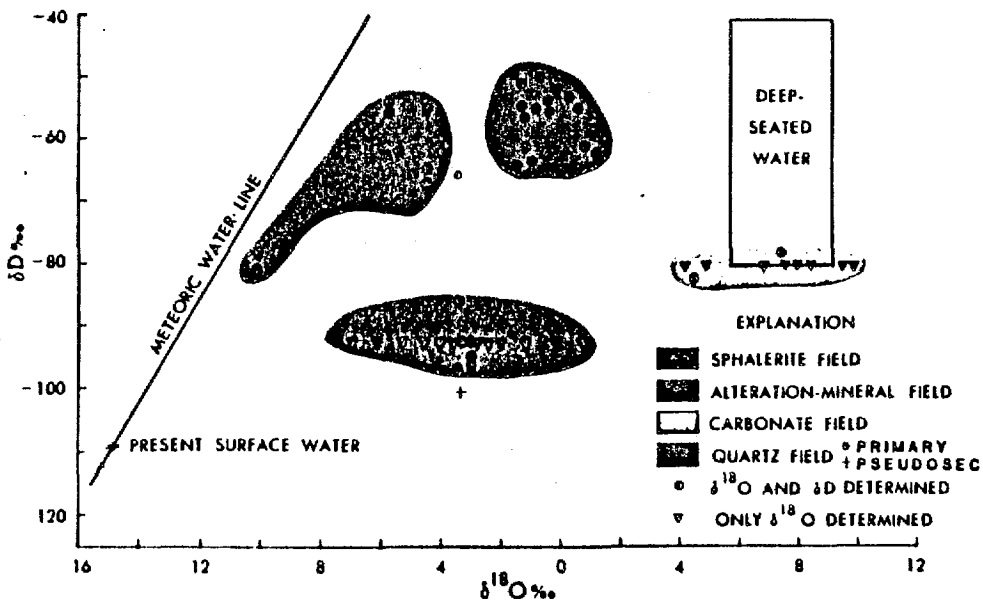
TECTONIC SETTING	DISTRICTS	EXAMPLE
CALDERA	6(+37)	GOLDFIELD, NV
		CREEDE, CO
		ROUND MTN, NV
SILICIC DOMES	2	JULCANI, PERU
		SILVER CITY, ID
EXTRUSIVE CENTER	1	COMSTOCK, NV
FOLDS AND FAULTS	1	COLORADO, [illegible]

SLIDE 13

RELATIVE TIMING OF DEPOSITION OF HOST ROCK AND ORE

TIME RELATIONS	DISTRICTS	EXAMPLE
VERY CLOSE TIMING	4	GOLDFIELD, NV
		JULCANI, PERU
< 1 MY	2	SILVER CITY, ID
		ROUND MTN, NV
> 1 MY	5	CREEDE, CO

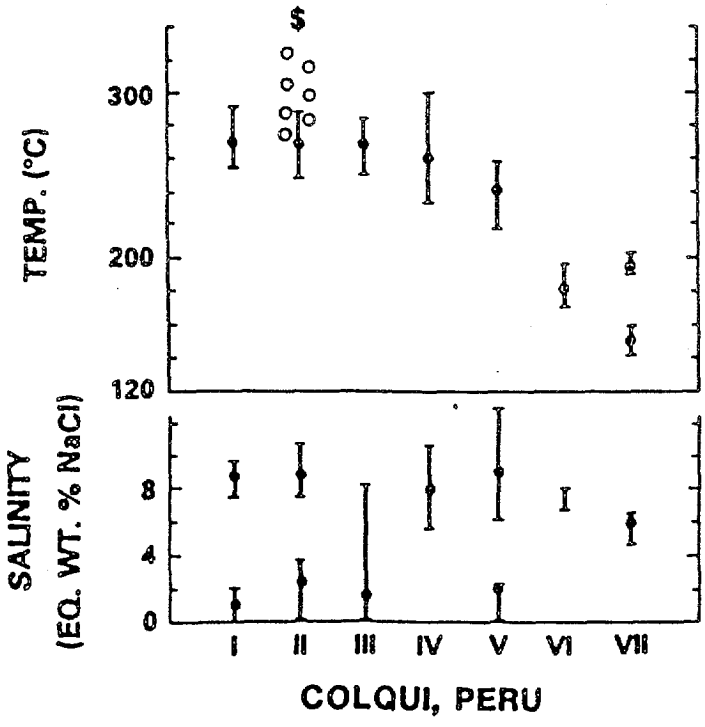
SLIDE 15



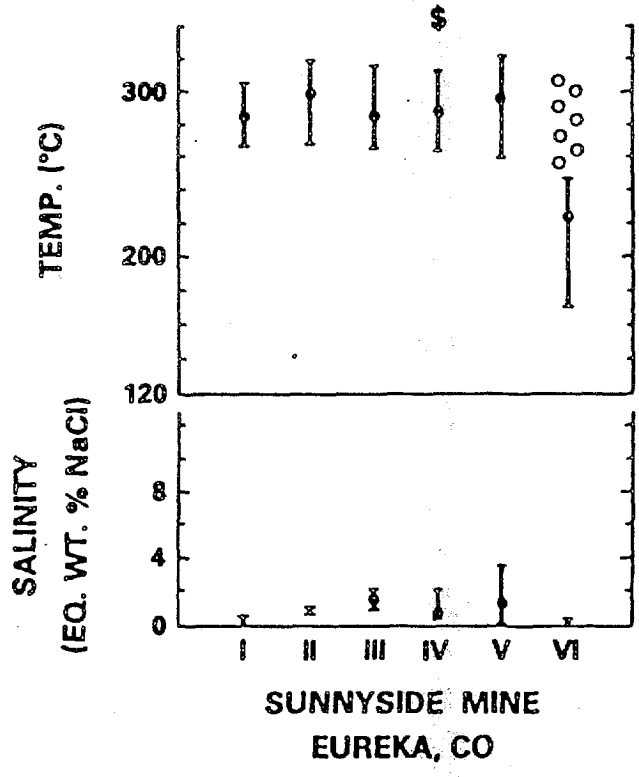
SLIDE 16

	OVERALL RANGE	TYPICAL RANGE DURING PRECIOUS-METAL DEPOSITION
TEMPERATURE (°C)	140°-325°	230°-270°
SALINITY (EQ. WT. % NaCl)	0-13	1-3 6-8

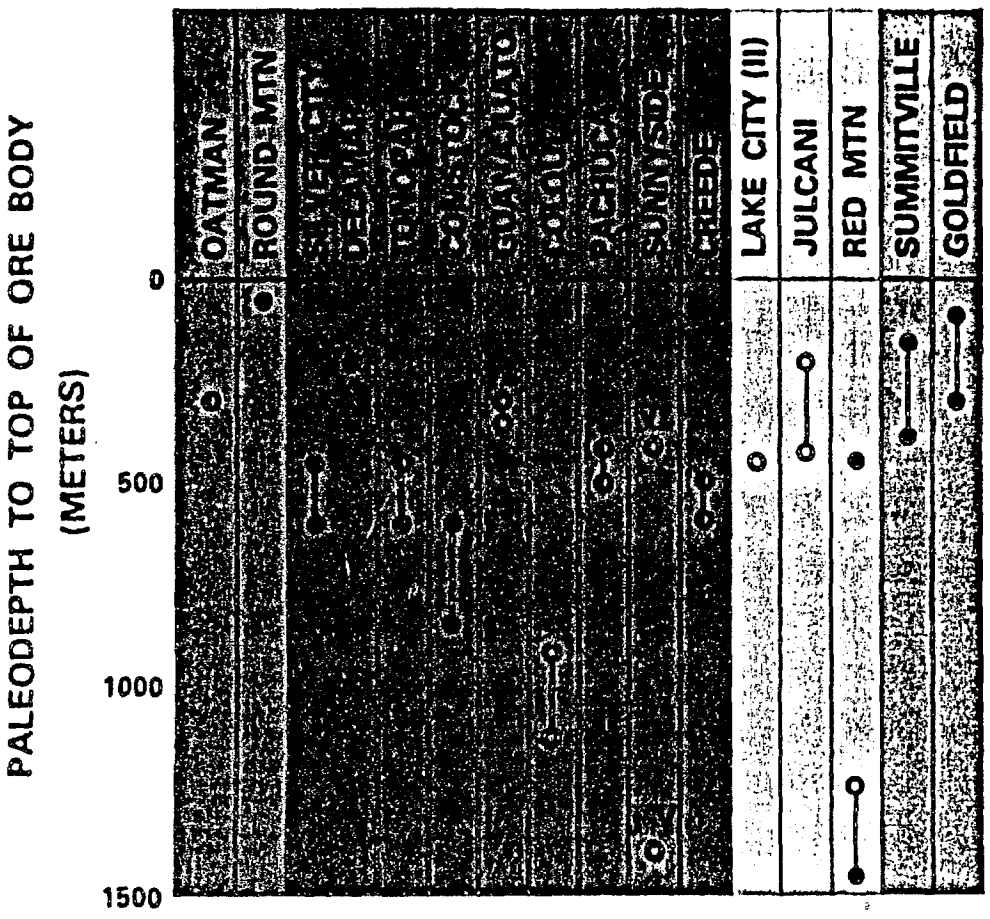
SLIDE 17



SLIDE 18



SLIDE 19



SLIDE 20

SIMILAR CHARACTERISTICS

- LITHOTECTONIC SETTING
- TEMPERATURE RANGE
- SALINITY RANGE
- VERTICAL RANGE OF ORE
- PALEODEPTH TO TOP OF ORE
- METEORIC WATERS DOMINANT

SLIDE 21

DISTINCTIVE CHARACTERISTICS

HIGHLY SULFIDIZED

- GOLDFIELD, NV.
- JULCANI, PERU

HIGHER SULFUR MIN. ASS.

ADVANCED ARGILLIC ALT.

CLOSE AGES OF HOST AND ORE

NOTEWORTHY Cu PRODUCTION

LESS SULFIDIZED

- ROUND MTN, NV.
- [illegible]

LOWER SULFUR MIN. ASS.

SERICITIC-ARGILLIC ALT.

AGES OF HOST AND ORE DISTINC

LOW Cu PRODUCTION

SLIDE 22

Paper #16

JPS

TOKIM SPRINGS GOLD MEGABIT

Russell Hardisty

Geologist

Precambrian Exploration, Inc.

Wheat Ridge, Colorado

Presented at the 89th Annual
Northwest Mining Association Convention
Spokane, Washington
December 2, 1983

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No Fig. 3

INTRODUCTION

The Tonkin Springs gold project is located in west-central Eureka County, Nevada, approximately 60 miles northwest of the village of Eureka (see Figure 1). The property is easily accessible by means of a 2-wheel drive vehicle from Eureka by traveling north on Nevada State Highway 20 approximately 40 miles, thence west-southwest 20 miles across a county maintained dirt road. The property was staked February, 1980 on the basis of several anomalous drainage sediment samples collected during the summer of 1979.

The topography of the Simpson Park range is typical of the Basin and Range province, with long narrow valleys and north trending mountain ranges. Vegetation covers varies from sagebrush covered hillsides to dense forest growths of pinon pine, juniper and mountain mahogany (see Figure 2). The climate is semi-arid with severe winters and mild to hot summers. The annual average precipitation is about 15 inches, the major portion of which is at higher elevations. Ideal field working conditions begin in early May and continue through late October.

REGIONAL GEOLOGY

The host rock for the gold mineralization at Tonkin Springs is a jasperoid within the lower to mid-Ordovician Vinini Formation (Merriam and Anderson, 1942). Merriam and Anderson divided the formation into two units; (1) the lower Vinini, represented by an interbedded sequence of sandy, dolomitic limestone, carbonaceous shale, calcarenite, siltstone, quartzite. The Vinini, as a whole, is included in the western siliceous assemblage, or upper plate rocks of the Roberts Mountain thrust fault (Roberts and Lehner, 1955). The thrust fault carried this siliceous assemblage east-southeast over the eastern carbonate assemblage, (see Figure 3), culminating the Antler Orogeny in late Devonian, early Mississippian (Roberts and others, 1958).

The thrust fault is not exposed in the immediate Tonkin Springs vicinity. Although Murphy and others (1978) mapped windows in the thrust plate near the gold deposit, no conclusive evidence of these windows has been found. The sole of the thrust zone is believed to be beneath the mineralization, within a few hundred feet. Anticlinal folding at the deposit is believed to be related to the thrust system, and may have produced fracturing that later became important during gold deposition.

The Tonkin Springs property is situated in the Eureka-Battle Mountain mineral belt (Roberts, 1966), approximately halfway between the two population centers (see Figure 3). The mineral belt is historically, one of great economic importance, especially in precious metal production.

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To date, there are three mining companies producing gold in the belt: Duval's Copper Canyon and Copper Basin open-pit mines, porphyry copper deposits, now mined only for gold; Placer Amex's Horse Canyon deposit; and the Windfall mine near Eureka. Inactive deposits include: the original Cortez silver district; Placer Amex's Gold Acres and Cortez low-grade gold deposits; and the Keystone massive sulfide deposit, to name a few. Deposits in the belt with future production planned include Cominco's Buckhorn gold mine, Exxon's Mount Hope porphyry molybdenum deposit, and the Ruby Hill gold/silver mine near Eureka (see Figure 3).

The Eureka-Battle Mountain mineral belt has also been recognized as being tectonically active, and has been traced for approximately 700 km along a northwest trend from central Oregon (Stewart and others, 1975). This linement has been referred to as the Oregon-Nevada linement (Stewart and others, 1975), northern Nevada rift (Zoback and Thompson, 1978), and the Cortez rift (Barrash and Venkatakrishnan, 1982). The linement is characterized by a belt of closely spaced, NW trending en-echelon faults, dike swarms, and mid-Miocene volcanism thought to mark the inception of Basin and Range rifting in Nevada (Zoback and Thompson, 1978). The belt is also outlined by an aeromagnetic anomaly interpreted as being related to the deep-seated feeder system for the basaltic andesite flows and dike swarms in the Cortez and Roberts Creek Mountains (Stewart and others, 1975).

A parallel belt of low grade gold deposits occur in the Lynn-Railroad mineral belt (Roberts, 1966), made up of Newmont's Carlin, Maggie Creek, Boot Strap, Blue Star, Gold Quarry, and Rain deposits, and is located approximately 60 miles northeast of Tonkin Springs. In addition to Newmont's mines, two newly discovered low-grade gold mines along the trend include Dee Mining Company's Boulder Creek deposit and Western States' Gold Strike deposit.

STRATIGRAPHY

The Vinini Formation is the oldest exposed formation in the vicinity of the Tonkin Springs deposit. The formation has been subdivided into two units on the geologic map (see Figure 4), the lower and upper Vinini. The lower Vinini is represented by an interbedded sequence of sandy dolomitic limestone, calcarenite, and carbonaceous shale. This unit is inferred to represent a near-shelf facies, transitional to a true eugeosynclinal depositional environment, on the basis of the terriginous clastic fraction present in these sediments. Sandy, dolomitic limestone and calcarenite appear to be the most favorable lithologies for silica replacement or jasperoid development, and also for gold mineralization (see Figures 5, 6, 7). Overlying this lower sandy sequence, but still included within the lower Vinini, is an interbedded sequence of thin-bedded limestone, carbonaceous shale, quartzite, and minor

greenstone. This sequence is inferred to represent a deeper water facies as compared with the sandy sequence. The large volume of carbonate material in the lower Vinini is suggestive of being more closely related to the transitional assemblage (Hotz and Wilden, 1955), than to the siliceous assemblage. The lower Vinini is in fault contact with the upper Vinini at all localities on the map where the contact is exposed. The contact is often concealed, but where exposed is always brecciated and silicified.

The upper Vinini is represented by quartzite, bedded chert, and black organic shale, lithologies more characteristic of the western siliceous assemblage. The base of the upper Vinini is represented by brecciated quartzite. Primary depositional bedding has been largely obliterated by brecciation and silicification. The quartzite is overlain by bedded chert that displays primary bedding that varies in thickness from 1 to 4 inches, with a few beds up to one foot or more, separated by thin-shaly partings.

The Devonian limestone, shown on the map (see Figure 4), as Dl, is represented by a gray fine-grained, crinoidal limestone, that also contains horn corals and brachiopods, and is in thrust contact with the Vinini Formation to the northwest of the Tonkin Springs deposit. The limestone was deposited in a miogeosynclinal environment and belongs to the eastern carbonate assemblage (Roberts and others, 1958). The limestone and jasperoid klippes are thought to be metagravity slides rather than tectonic thrust faults (Murphy and others, 1978). A cross-cutting, northeast trending high angle normal fault is responsible for silicification or jasperoid development within the limestone; fossils are preserved after silicification aiding in correlation.

TERTIARY IGNEOUS ROCKS

A clay-altered, biotite, hornblende, alkali feldspar intrusive is exposed on the surface, adjacent to the mineralized deposit on the west, and at several localities to the northwest. Feldspars are totally altered to clay, and biotite and hornblende are usually altered to chlorite or epidote, making identification and classification difficult. The rock has an abundance of cross-cutting calcite veins, and usually contains 2 to 3% disseminated pyrite.

The intrusive bodies appear to be sill-like in habit. Most are tabular and were emplaced parallel to bedding, possibly along thrust faults. However, intrusive bodies may occur as vertical dikes to the north of the mineralized body. The intrusives are important to the mineralization in as much as they appear to have influenced location of channel ways for the late mineralizing fluids. The intrusives themselves, however, are barren of gold. Age of the intrusion is uncertain, but field relationships show

it to predate Tertiary volcanism and mineralization.

Tertiary volcanics, undifferentiated on the map, are made up of two units, a rhyolitic crystal-lithic tuff and a set of porphyritic andesite flows. The white to gray, crystal-lithic tuff contains phenocrysts of quartz and biotite, plus various lithic fragments set in a fine grained ground mass of ash and pumice. The reddish brown to purple, porphyritic andesite contains phenocrysts of alkali-feldspar and hornblende set in an aphanitic ground mass. K-Ar age dates on the rhyolite indicates ages of Eocene to Oligocene or 37.5 ± 0.4 m.y., and age dates on the andesite indicate an Oligocene age or 33.4± 2.6 m.y. (Murphy and others, 1978).

STRUCTURE

Two major sets of high angle normal faults have been recognized at Tonkin Springs, an older NW set that appears to have been the more important set during mineralization. The younger NE trending faults give rise to large soil and rock-chip anomalies, but as of yet have failed to prove to be economically important (see Figures 17, 18). The anomalies are thought to represent hydrothermal fluid leakages along these fractures in conjunction with jasperoid development and may have been sealed by silica prior to the main gold mineralizing event.

The NW trending fractures are thought to be Basin and Range faults, associated with the 17 to 14 m.y. rifting. Drill hole data within the main deposit defined to date, indicate three parallel fractures containing higher grade mineralization, up to 0.5 oz/ton Au, separated by zones of lower grade mineralization (see Figure 8). This would indicate that the age of the mineralization may be late miocene, or roughly contemporaneous with volcanism and plutonism in the Roberts Creek Mountains to the southeast and in the Cortez Mountains to the northwest.

The NE trending faults truncate the NW set at different localities on the map. The fractures have been interpreted as being tear faults, associated with Basin and Range oblique extension (Zoback and Thompson, 1978).

MINERALIZATION

Drilling to date has delineated a deposit that trends roughly northwest-southeast approximately 1500 feet, and laterally approximately 1000 feet. Reserve calculations indicate a deposit of approximately 2.5 m.t., with an overall average grade of 0.09 oz/ton Au. The stripping ratio has been calculated to be 2.4 tons of waste to one ton of ore. A deeper zone of mineralization has

been determined to contain approximately 0.5 m.t. of 0.09 oz/ton Au material, but would have a stripping ratio of 14.7 to 1, making it uneconomical to extract at today's gold price (see Figures 9, 10). Chances for additional reserves along the trend appear to be excellent as indicated by the geology and geochemistry.

Mineralization occurs in both oxidized and unoxidized zones, with the bulk of the mineralization in the unoxidized zone. Microprobe work performed in conjunction with metallurgical studies carried out by Bacon, Donaldson, and Associates of Vancouver, B.C., has determined that approximately 75% of the gold mineralization in the unoxidized zone occurs as micron-sized particles in sulfide minerals, namely pyrite and arsenopyrite. Pyrite is found in two forms, as framboidal pyrite in silica veinlets and as normal cubic pyrite disseminated throughout the rock (see Figures 11, 12, 13). This type of mineralization has been documented at the Carlin gold deposit (Radtke and Rye, 1980), and at the Cortez gold deposit (Wells and Mullens, 1973). The remaining 25% probably occurs as micron-sized free gold particles in silica veinlets, although there have been no reports of visible gold. Other sulfide minerals associated with the deposit include realgar, orpiment, cinnabar, and stibnite (see Figures 14, 15).

Thin-section analyses indicated the following paragenesis: decalcification ⟶ silicification (jasperoid development) ⟶ silicification (silica veinlets with sulfide and Au mineralization ?) ⟶ calcification (calcite veinlets with realgar, ⟶ cinnabar, and barite fracture filling) ⟶ microfracturing ⟶ carbonization. There does not appear to be any significant gold mineralization associated with the late stage introduced hydrocarbon, which is found along microfractures and as a black oily film coating sulfide crystals.

Ore controls at Tonkin Springs include the following: Tertiary intrusive on the west, Tertiary volcanic capping on the east, northwest trending fractures, and a favorable host rock. The Tertiary intrusive is inferred to have intruded along a bedding plane thrust fault (see Figure 9). This conclusion is based on the following observations: geometry of the base of the intrusive; cross-sectional geometries of ore pods perpendicular to the intrusive's basal contact; and milled and fractured sedimentary inclusions within the intrusive. The sill-like intrusive body acted as a conduit as well as a dam for hydrothermal fluids. The Tertiary volcanics, namely the crystal-lithic tuff, also acted as an impermeable barrier, as there is ore-grade mineralization just below the silicified sedimentary contact with the volcanics, but only weak geochemical mineralization within them. The northwest trending high-angle fractures appear to be important conduits for the hydrothermal fluids, with subsequent lateral penetration into a favorable host.

Porosity and permeability appear to have been developed by different mechanisms: primary or intergranular porosity, developed as a result of deposition of terriginous clastic material; secondary intergranular porosity, developed after decalcification of sandy carbonates and calcarenites; tectonic fracturing produced by thrusting and associated folding; northwest trending fractures, associated with Basin and Range rifting and continental extension; and brecciation adjacent to intrusives.

GEOCHEMISTRY

Discovery of the main gold deposit at Tonkin Springs, is directly attributable to a well-designed geochemical survey and to quality geochemical analyses performed on approximately 4500 soil, rock-chip, and drainage sediment samples. The samples were analyzed by Mikron Geochemical Laboratory, Wheat Ridge, Colorado. In addition to geochemistry, detailed geological mapping and perserverance played an important role in the discovery. The Tonkin Springs vicinity has been an area of intense exploration over the past 16 years by seven major companies. All attributes combined, lead to targeting the first five drill holes where they intercepted ore grade mineralization and thickness.

The decision to stake the claims was taken after a gold anomaly was detected in drainage sediments during 1979. The samples defined an anomalous area of approximately 6 square miles (see Figure 16). Gold values ranged from 0.02 to 0.10 ppm. Drainages in the immediate vicinity of the Tonkin Springs deposit contained gold contents varying between 0.03 and 0.06 ppm.

Soil sampling, conducted on a 400 x 200 feet grid system revealed a number of large and intense gold anomalies within the gold bearing district defined by the drainage sediment geochemistry (see Figure 17). The soil gold anomalies are accompanied by elevated values for Hg, As, and Sb. The concentration ranges for these elements in the Tonkin soils were found to be:

Au	undetectable to 1.6 ppm
Hg	26 to 968 ppb
As	50 to 3000 ppm
Sb	1 to 52 ppm

Rock-chip samples collected over the Tonkin Springs deposit were analyzed for Au only. The gold contents varied from undetectable to 28 ppm. Figure 18 is a map showing rock samples collected by a competitive company that clearly defines the northeast trending fault shown on Figure 4, and also defined by soil geochemistry in Figure 17. The data in Figure 17 and 18 suggests a northeast trend for the mineralization, which drilling has since proven to be erroneous.

SUMMARY

Drilling at Tonkin Springs has delineated a mineralized deposit containing approximately 2.5 m.t. of 0.09 oz/ton Au. The stripping ratio has been calculated to be 2.4:1. A deeper zone of mineralization contains approximately 0.5 m.t. of 0.09 oz/ton Au material at a strip ratio of 14.7 to 1.

The mineralization occurs in the lower to mid-Ordovician Vinini formation, a near-shelf eugeosynclinal facies. Gold mineralization has been determined to occur as micron-sized particles in pyrite and arsenopyrite, and as micron-sized free gold in silica veinlets. The mineralization is controlled by Tertiary volcanic capping, Tertiary clay-altered sill-like intrusive bodies, northwest trending fractures, and a favorable host rock.

The property lies within the Eureka-Battle Mountain mineral belt (Roberts, 1966), a belt that has been tectonically active since at least 15 m.y. B.P. and was a locus for mid-Miocene volcanism and plutonism (Stewart and others, 1975). This is evidenced at Tonkin Springs by three northwest trending fractures containing higher grade mineralization. Fractures with this orientation have been interpreted as being related to Basin and Range rifting, suggesting a late-Miocene age for mineralization or roughly coincidental to volcanism and plutonism in the Roberts Creek Mountains to the southeast and the Cortez Mountains to the northwest.

Discovery of the mineralization is attributed to a well-designed geochemical survey and quality geochemical analyses. This in conjunction with detailed geological mapping over a three-year period, targeted the first five drill holes in ore-grade mineralization and thickness.

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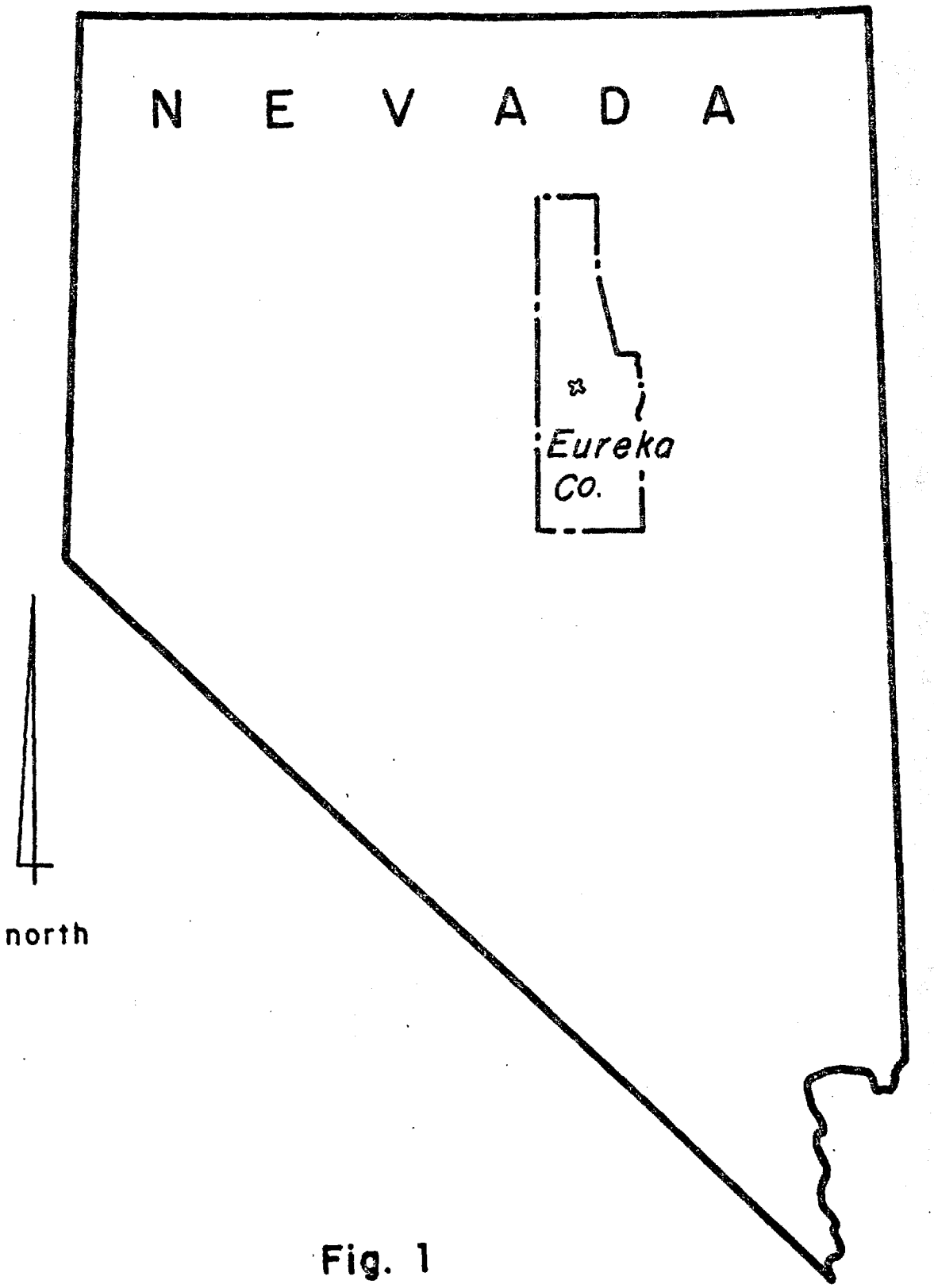


Fig. 1

LOCATION MAP, TONKIN SPRINGS.

Eureka County, Nevada

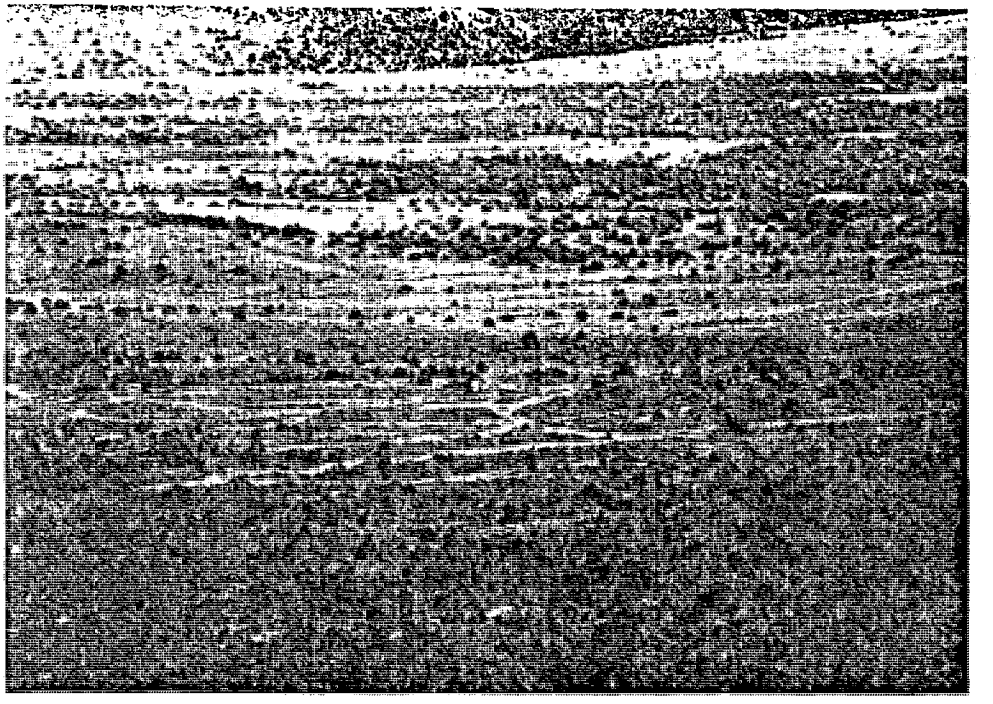
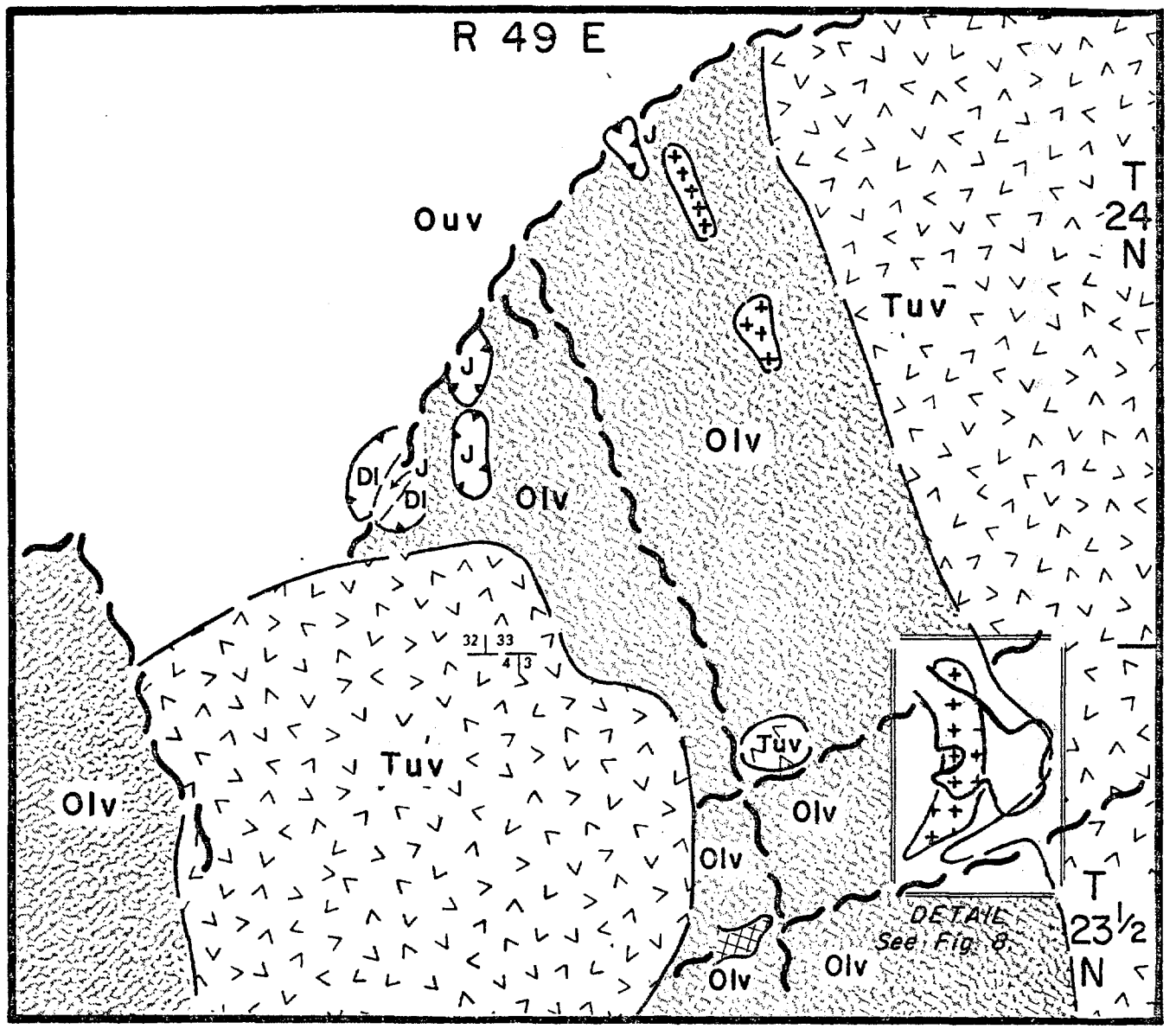


FIGURE 2 -- Photo of Tonkin Springs main deposit from above. Note drill rig in center of photo, and variation in vegetative growth.



EXPLANATION

north

- Tuv — Tertiary Volcanics
- Intrusive
- J — Jasperoid
- Contact
- Shear
- Dl — Devonian Limestone
- Ouv — Ordovician Upper Vinini
- Olv — Ordovician Lower Vinini
- Mineralization
- Thrust Fault (teeth on upper plate)

Fig. 4. GEOLOGIC MAP

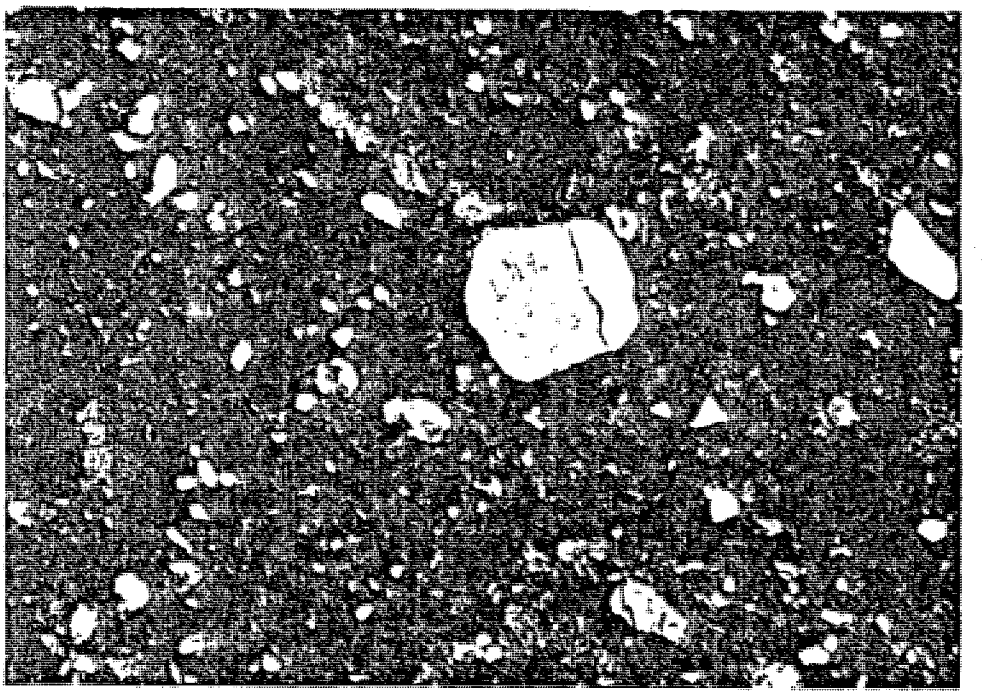


FIGURE 5 -- Photomicrograph, x-nichols, 4x, calcarenite, rounded quartz grain in center of photo is approximately 1 mm across. Quartz grains embedded in calcite matrix.

|———| represents 1 mm.

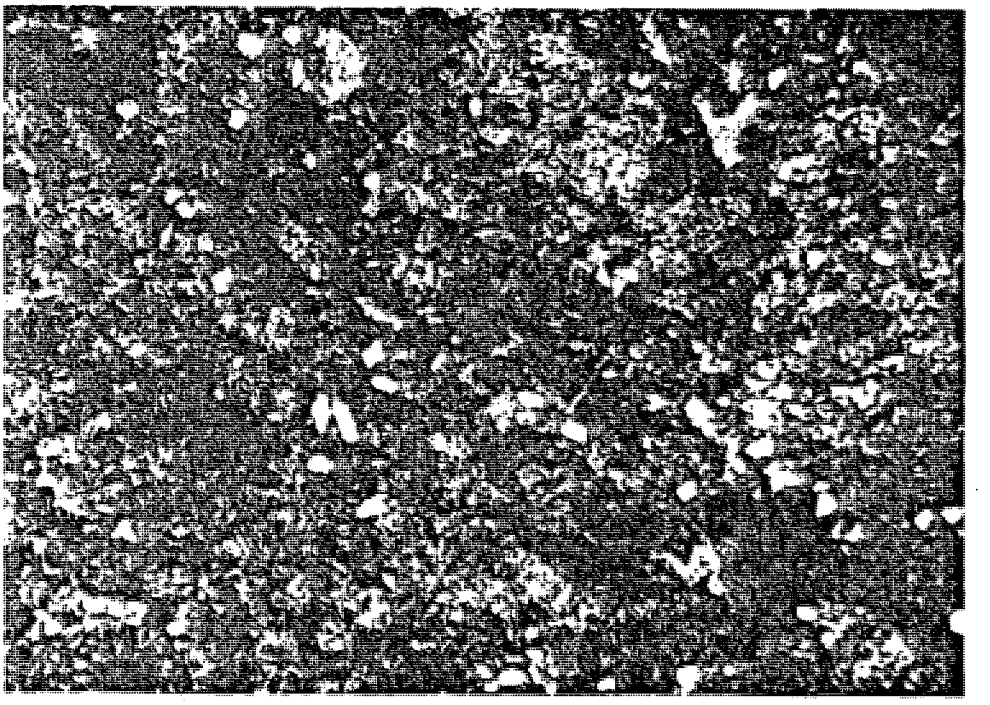


FIGURE 6 -- Photomicrograph, x-nichols, 4x, silty, dolomitic limestone. Fine grained to silt size quartz grains, embedded in calcite, dolomite matrix.

|———| represents 1 mm.

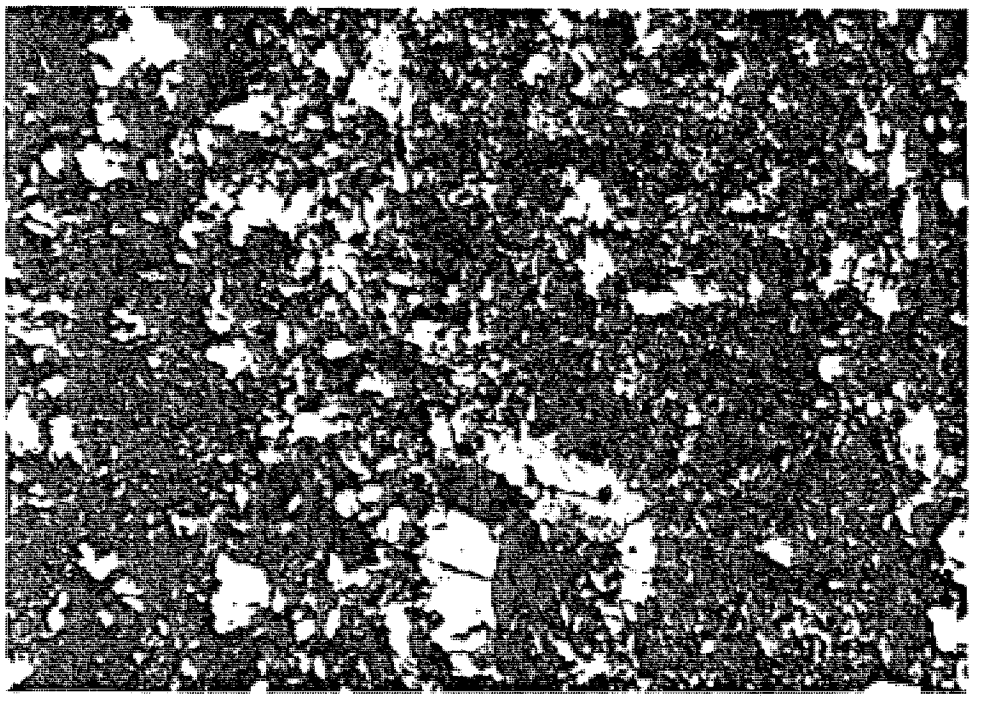


FIGURE 7 -- Photomicrograph, x-nichols, 4x, jasperoid, jig-saw texture, quartz grain bottom of photo, approximately 1 mm across, embedded in fine grained jasperoid matrix. Inferred to represent a silica replaced calcarenite.

|———————| represents 1 mm.

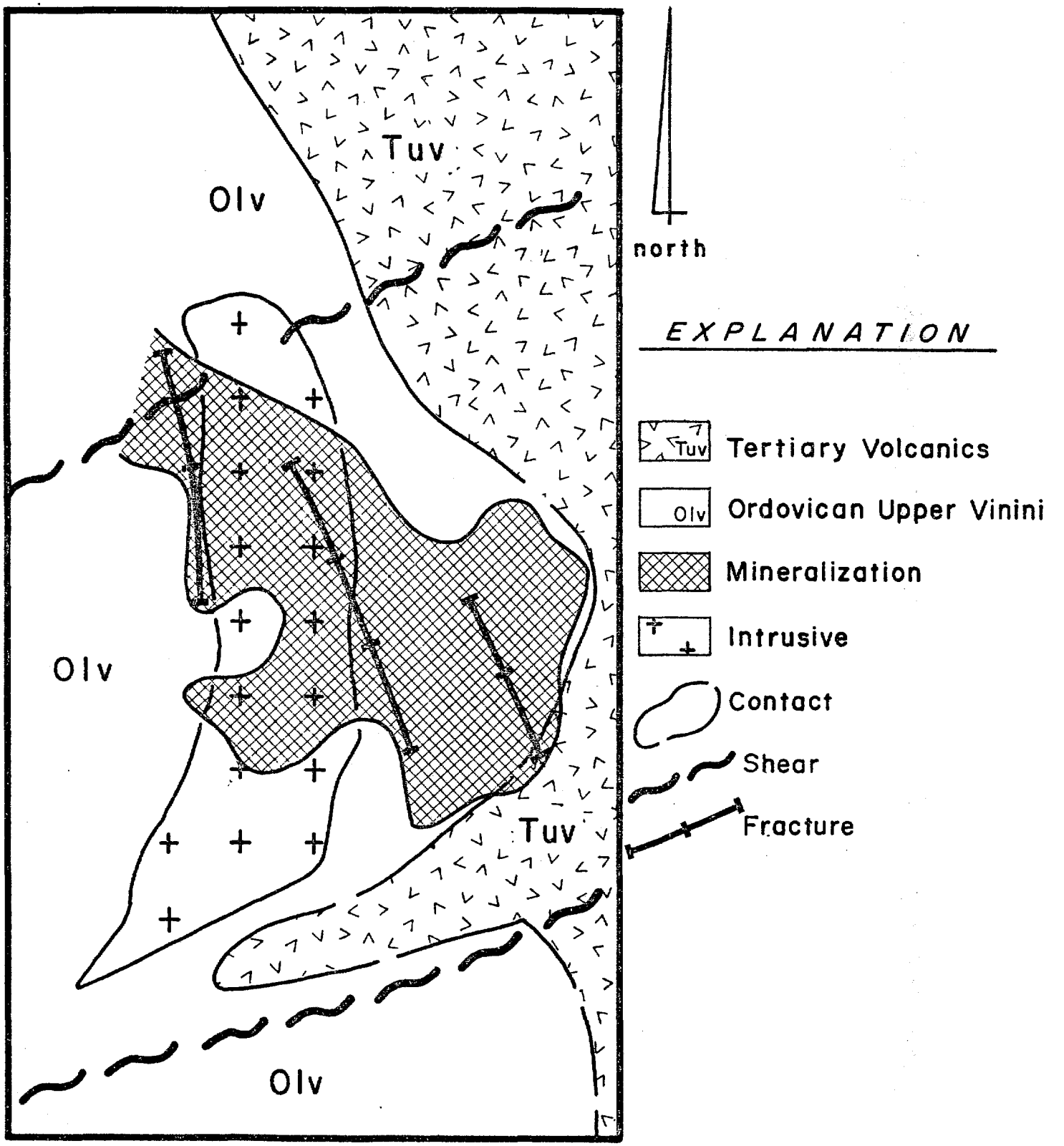


Fig. 8. DETAIL OF GEOLOGIC MAP

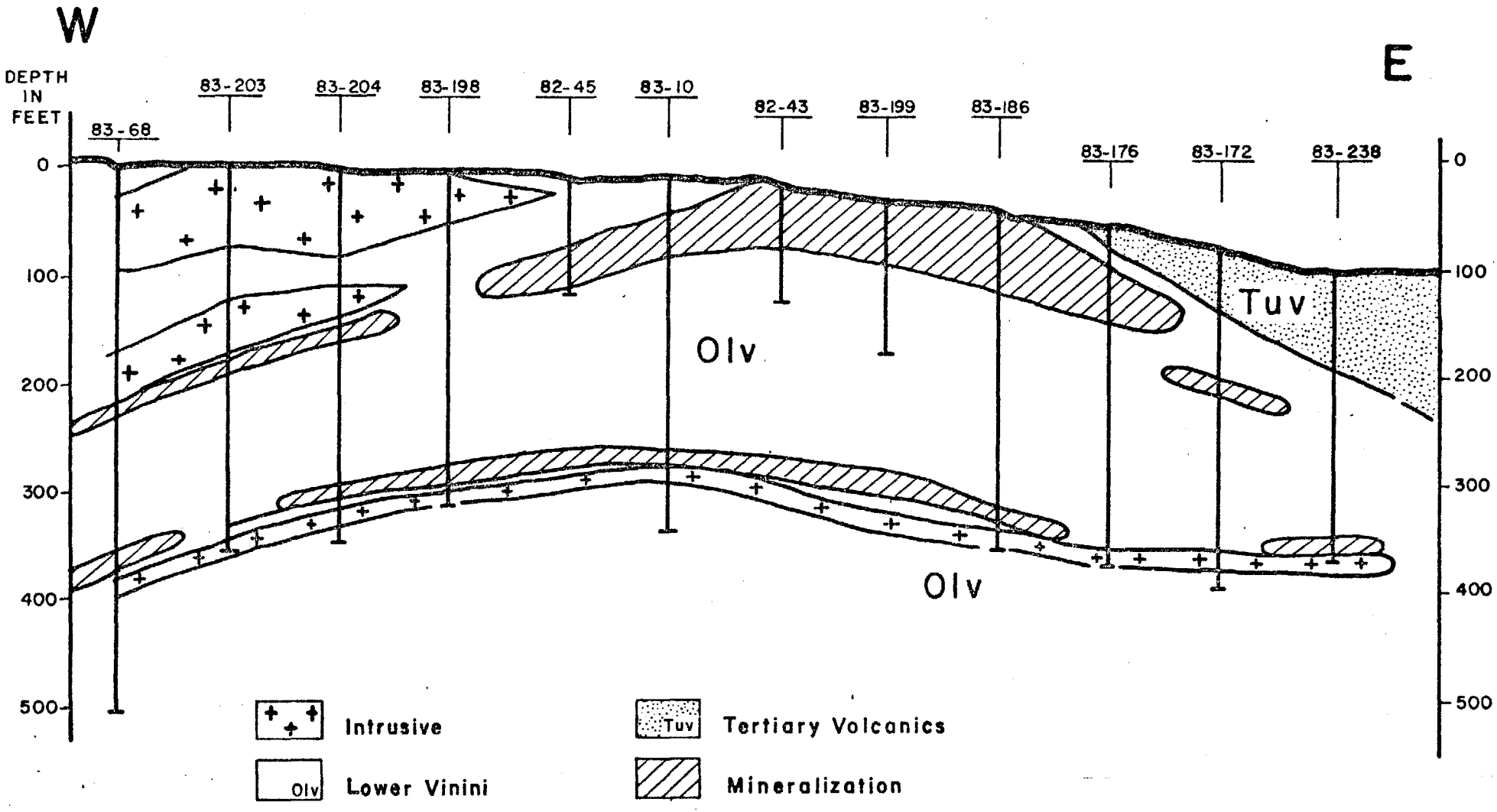


Fig. 9. EAST - WEST GEOLOGIC SECTION ACROSS TONKIN SPRINGS MAIN DEPOSIT

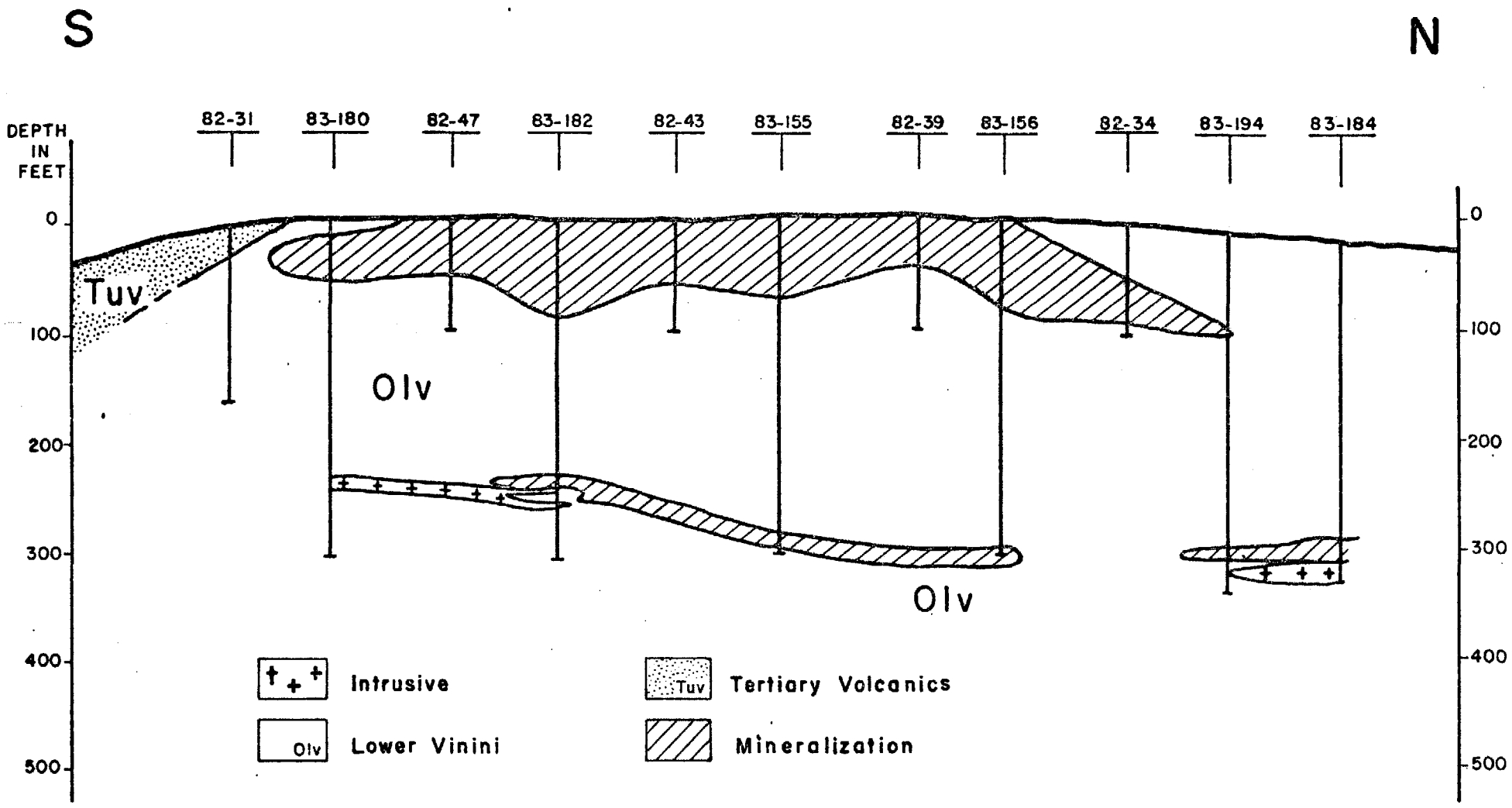


Fig. 10. NORTH - SOUTH GEOLOGIC SECTION ACROSS TONKIN SPRINGS MAIN DEPOSIT

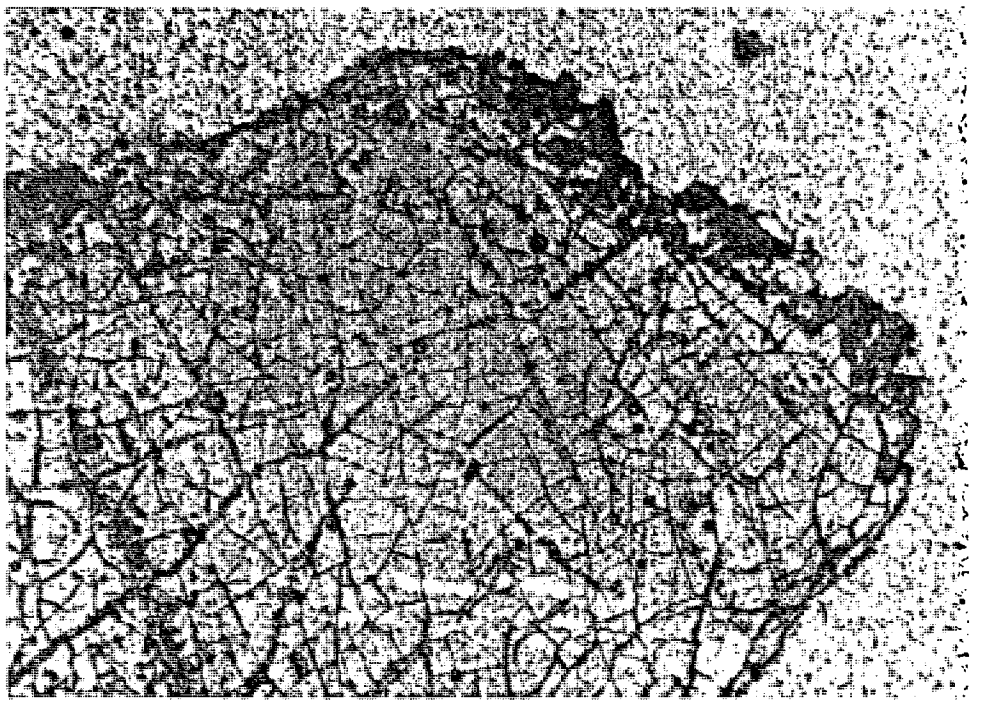


FIGURE 11 -- Photomicrograph, plane polarized light, 4x, fine grained jasperoid, black opaque mineral is framboidal pyrite, along a silica veinlet. Note micro-fracturing.

|———————| represents 1 mm.

FIGURE 12 -- Photomicrograph, plane polarized light, 4x, jasperoid, framboidal pyrite along silica veinlet, and disseminated throughout matrix.

|———| represents 1 mm.

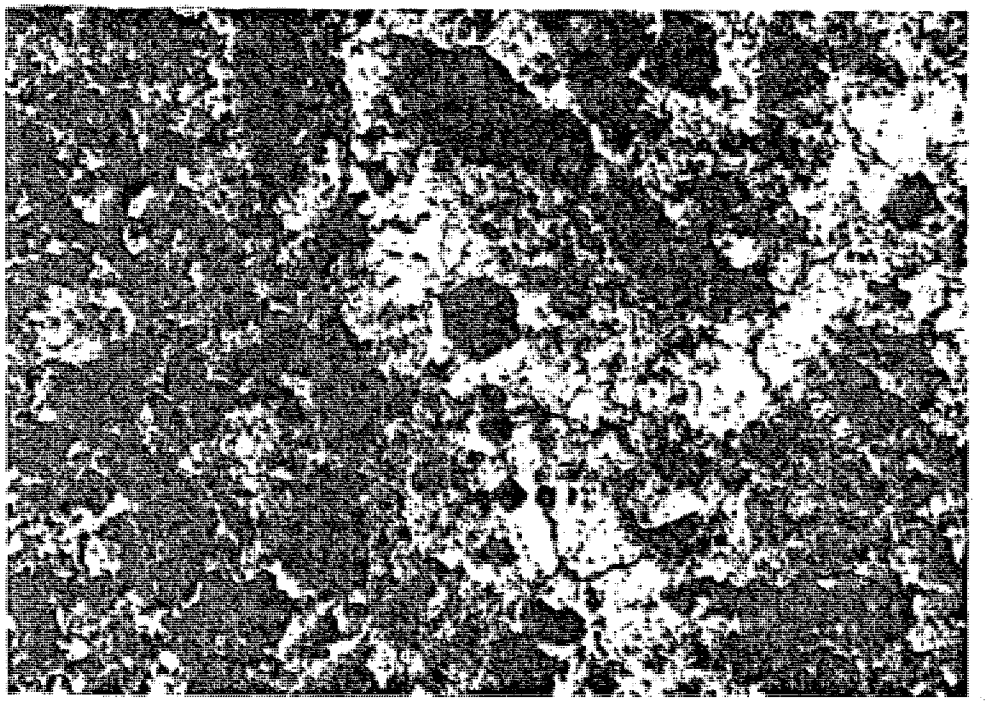


FIGURE 13 -- Photomicrograph, plane polarized light, 4x, opaque mineral is cubic pyrite, note pyritohedron in center of photo. Pyrite embedded in jasperoid matrix.

|———| represents 1 mm.

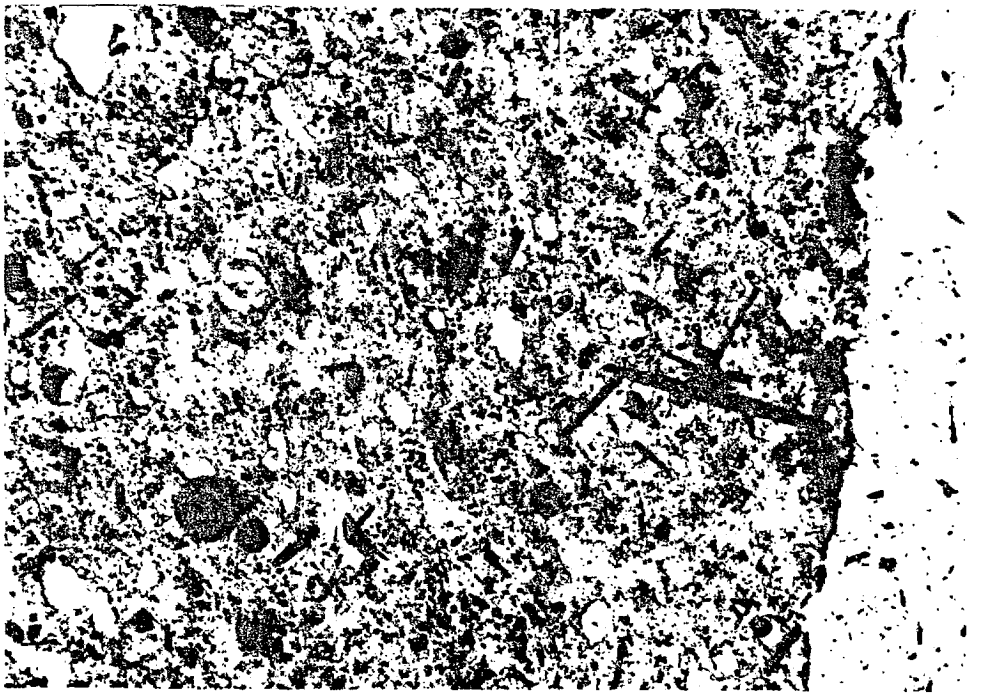


FIGURE 14. Photomicrograph, plane polarized light, 4x, opaque mineral is stibnite, blade is approximately 1 mm long, embedded in jasperoid matrix.

|———| represents 1 mm.

FIGURE 15. Photomicrograph, plane polarized light, 4x, opaque mineral cluster of stibnite, embedded in fine-grained jasperoid matrix. Unreplaced dolomite rhombohedrons show apparent relief.

|———| represents 1 mm.

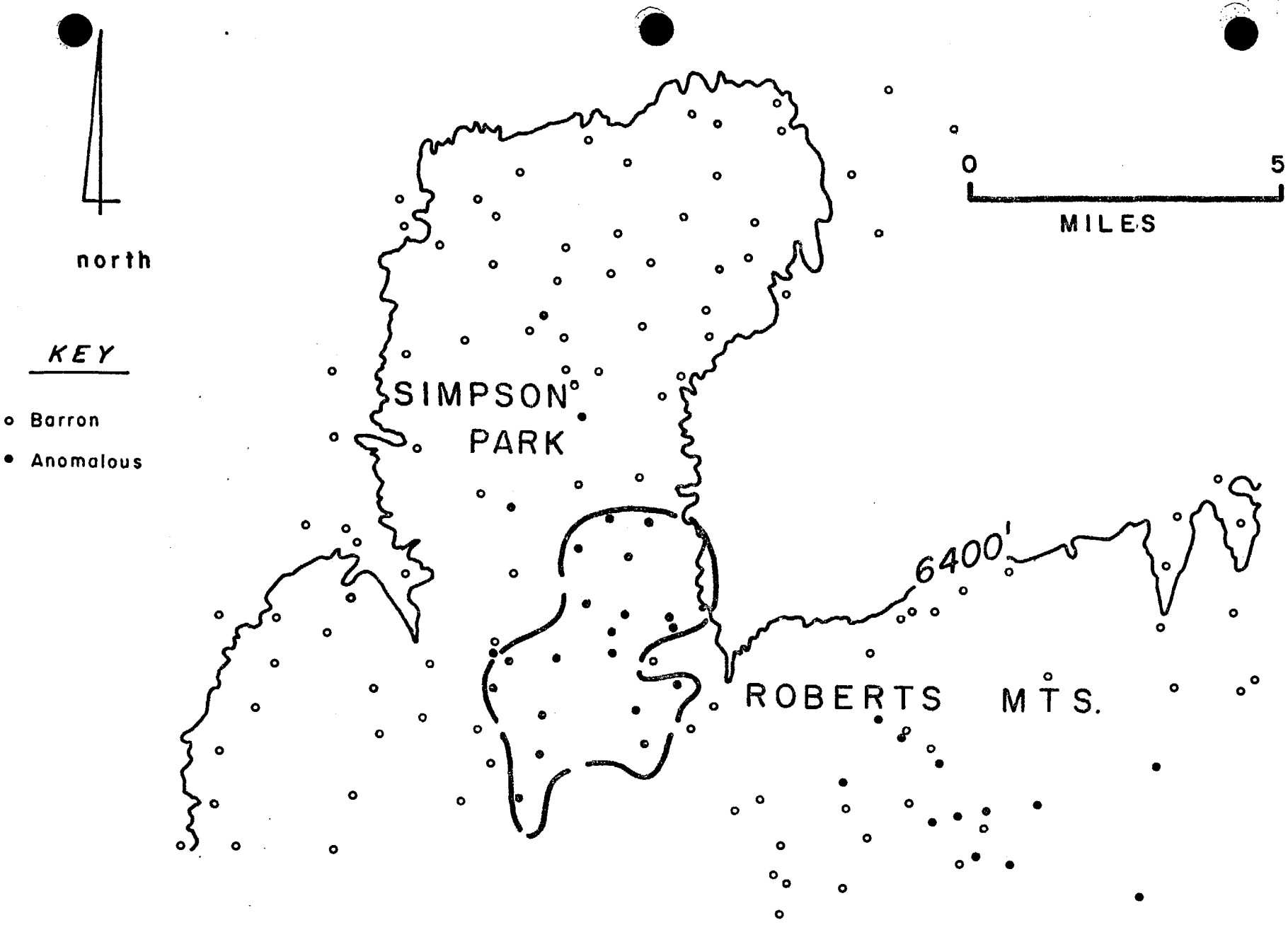


Fig. 16. GOLD DISTRIBUTION IN STREAM SEDIMENTS
Northern Simpson Park and Roberts Creek Mts.

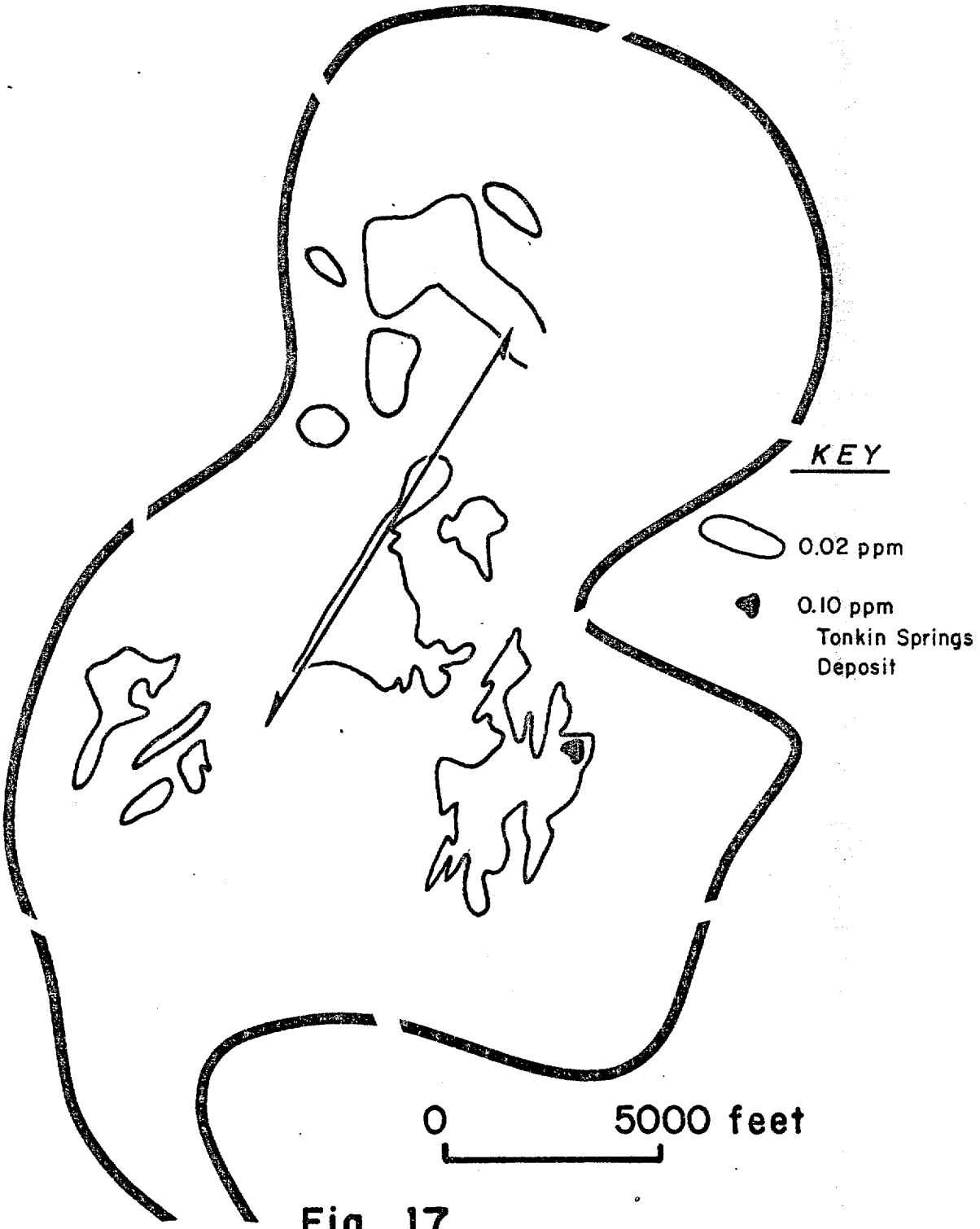


Fig. 17

Distribution of Anomalous Au Values in drainage sediments & soils

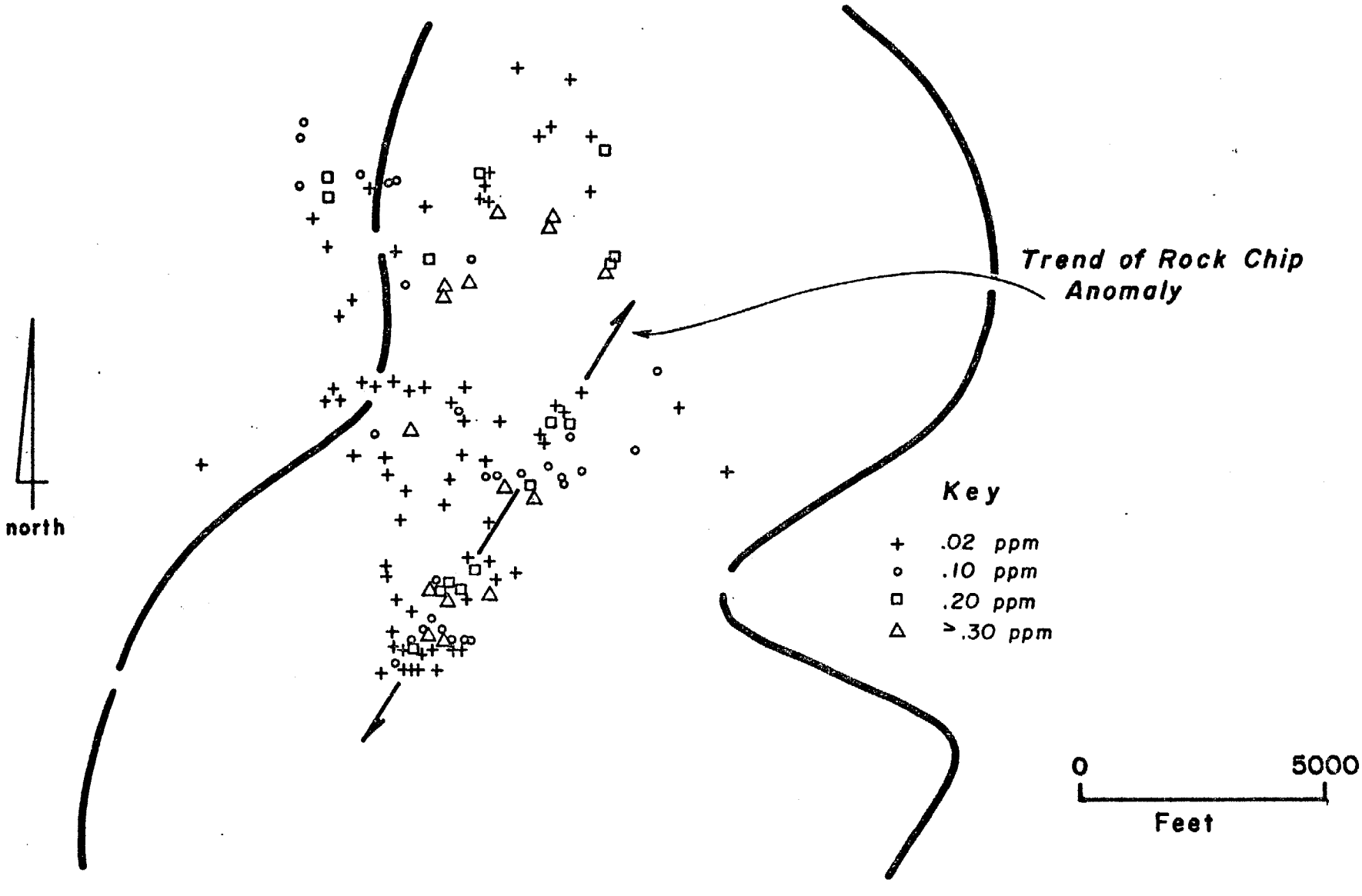


Fig. 18.

Au MINERALIZATION ROCK CHIP

JDS

Paper #31

GEOLOGY AND DEVELOPMENT OF THE RELIEF CANYON GOLD DEPOSIT, PERSHING COUNTY, NEVADA

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NORTHWEST MINING ASSOCIATION CONVENTION
SPOKANE, WASHINGTON
NOVEMBER 1983

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ACKNOWLEDGEMENTS

Before introducing my topic regarding the Geology and Development of the Relief Canyon Gold Deposit, I wish to acknowledge the contributions of certain individuals to the project:

For their valued advice and direction:

Lacana Chairman Dr. Bill Gross, President Ed Thompson, Chief Engineer Colin Marshall, Vice-President Harvey Sobel, and Exploration Manager Mike Easdon.

Additionally, Steve Barnett, Don Hopkins, Wilma Hatter, Bert Jeffries, and Doug Bruha have all contributed greatly to the project management and geological evaluation.

Of course, my wife's continued support has proven to be invaluable, and I thank her greatly.

INTRODUCTION

The Relief Canyon Project involves development of open-pittable, disseminated, epithermal gold metallization. The project site is roughly 20 miles northeast of Lovelock, Nevada, along the southwestern margin of the Humboldt Range. The Rochester Silver District is 5 miles north of Relief Canyon and the Antelope Springs Mercury District is to the immediate south.

Gold occurs within and proximal to jasperoidal silicification developed principally within a sedimentary breccia unit underlain by Late Triassic platform margin carbonates and overlain by Late Triassic pelitic deltaic rocks.

Lacana and another company are both developing this deposit which contains greater than 8 million diluted ore tons with an average grade of 0.04 oz Au/ton. At a waste to ore ratio of 1.5, the anticipated annual ore production will approximate 1 million tons.

Results of both bench scale metallurgical tests and a comprehensive heap leaching project suggest that gold recoveries exceeding 65% may be sustained without crushing and agglomerating the ore.

The project is very well located with respect to the availability of water, power, and labor. The topography is excellent for a mining and milling operation. Projected capital and operating costs are favorable. Final feasibility analysis and design engineering will commence in February, 1984.

REGIONAL GEOLOGY

Lithology

Six major rock groups are exposed in the southern Humboldt Range. In order of decreasing age, these are volcanics of the Koipato Group, marine carbonates of the Star Peak Group, terrigenous clastics of the Auld Lang Syne Group, a gabbro and diorite intrusive complex, a quartz monzonite intrusive complex, and young alluvium and volcanics.

Permian-Triassic volcanics of the Koipato Group form the regional basement. The upper portion of the Koipato consists of rhyolitic tuffs intruded by boron and fluorine enriched leucogranite and rhyolite porphyry considered by Vikre (1981) to be of the same origin as the tuffs. These rocks contain more than 100 million tons of low grade, disseminated, and vein controlled precious metals mineralization in the Rochester Silver District.

Early to Late Triassic calcareous marine strata of the Star Peak

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Group were deposited upon islands or peninsular basement highs of the Koipato volcanics. Nichols and Silberling (1977) indicate that nearly 4000' of basinal, slope, shelf, and platform carbonates and subordinate terrigenous clastics and volcanics record eastward marine encroachment upon the Koipato.

Approximately 25,000 feet of Late Triassic argillite, arenite and subordinate limestone of the Auld Lang Syne Group were rapidly deposited unconformably upon the Star Peak Group as regional uplift and deltaic sedimentation occurred, according to Burke and Silberling (1973).

Following deposition of the Auld Lang Syne Group, orogenic activity resulted in widespread intrusion of Jurassic gabbro, diabasic quartz diorite, and amphibole-pyroxene-magnetite diorite, as described by Johnson (1977).

Younger intrusions of Cretaceous granodiorite and quartz monzonite are rarely present in the southern Humboldt Range, but Vikre (1981) concluded that these rocks probably underlie the Rochester Silver District.

Post-Mesozoic rocks exposed in this region include Tertiary alluvial deposits and Quaternary basalt flows and younger alluvium.

Structure

The Jurassic-Cretaceous Nevadan Orogeny is reported by Johnson (1977) to have caused extensive low grade regional metamorphism of rocks exposed in the southern Humboldt Range. Complex folding and minor thrust faulting was probably coeval with the emplacement of Jurassic gabbro and related intrusives. The folding event is obvious in the field, but the thrust faults, where examined in the field, are obscure, and in places, dubious.

Two intersecting major structural zones of Late Mesozoic to Tertiary age are present in the region. The possibly older of the two zones is exactly coincident with the Rye Patch lineament described by Rowan and Wetlaufer (1981) as a major northwest trending right lateral strike-slip system. It forms a topographically pronounced, sub-parallel, linear fault belt of at least 5 miles width and transects the southernmost portion of the Humboldt Range and the northernmost portion of the West Humboldt Range.

The other and probably younger fault system is the northeast trending Humboldt Structural Zone considered by Trexler and others (1979) to be a left lateral shear which transects most of northern Nevada. It forms the western margin of the West Humboldt Range. A parallel structure, the Black Ridge fault, defines the western edge of the southern Humboldt Range and is considered by Vikre (1981) to be a normal fault of great displacement dipping westerly.

Examination of the Pershing County geologic map by Johnson (1977)

suggests that a significant right lateral component to the Black Ridge fault might exist. If so, it is possible that the mineralized rocks exposed at Relief Canyon may have once been positioned vertically and horizontally proximal to the Rochester District.

LOCAL GEOLOGY

Lithology

The gold deposit at Relief Canyon occurs at the contact between the Cane Spring Formation, the uppermost unit of the Star Peak Group, and the overlying Grass Valley Formation, the lowest unit of the Auld Lang Syne Group. Strata-bound and conformable mineralization exists principally in a local sedimentary breccia. This unit appears to be a polymictic debris flow of intraclastic and possibly fluxoturbidite breccia of marine origin.

Drilling indicates that the highly irregular debris flow occasionally exceeds 200' in thickness and is unconformable with the underlying platform-margin carbonates of the Cane Spring Formation.

The Cane Spring Formation contains thinly bedded, silty, carbonaceous, and bioclastic limestones in its lower portion and thickly bedded, occasionally recrystallized micrites and weakly bioclastic limestones in its upper portion.

The debris flow appears to be unconformably overlain by micaceous, feldspathic, and quartzitic subphyllitic argillites and arenites of the Grass Valley Formation. Burke and Silberling (1973) conclude the Grass Valley to be the basal portion of a large, westerly prograded deltaic complex. Soft sediment deformation textures are present in the Grass Valley, particularly near its basal contact with the debris flow.

The debris flow comprises crudely stratified and rarely graded beds of subangular to subrounded clasts of greatly variable sizes trapped within a clay and mud matrix. Common fabrics of the breccia include both interlayered and massive zones of dominantly matrix supported, rarely imbricated clasts in clayey, silty or sandy matrix material. Fragments of micrite with rip-up clasts are common. Slabs and blocks of limestone several hundred feet in maximum dimension occur as mega-clasts surrounded by finer grained breccia or resting directly upon the Cane Spring Formation.

Clast lithologies are identical to strata of the Grass Valley and Cane Spring Formations. Evidence of multiple episodes of transport of the debris flow, prior to its complete lithification, is common. Large and small scale soft sediment deformational textures are occasionally recognizable.

Within the drilled area, the debris flow appears to be an irregular triangular wedge or bell-shaped mass in plan, the apex of which is located at the center of the east margin of the gold deposit. The Black Ridge fault probably terminates the breccia to the west, but drill confirmation of this does not yet exist. The debris flow is absent east of the apex and it is terminated to the northeast by strongly altered diorites which occupy an older, high angle, northwest trending fault system. Footwall limestones of the Cane Spring Formation are present on the northeast contact of the diorites.

Structural contours of the base of the Grass Valley may reflect the Jurassic-Cretaceous folding event of the Nevadan Orogeny. However, many small, irregular, divergent folds are superimposed upon a broad, open, northeast trending antiform which plunges southwesterly. It is currently unclear whether any or all of these folds might reflect original emplacement geometries of the debris flow.

Origin of the Debris Flow

Our current working hypothesis regarding origin of the debris flow fundamentally involves minor movement along the fault now occupied by diorite intrusives as very rapid loading of deltaic sediments occurred upon poorly lithified micrites. Such movement may have induced sub-horizontal shear failure of the water-rich, unlithified or poorly lithified calcareous oozes. The importance and mechanics of earthquake induced slumping, liquifaction, and plastic deformation as a result of rapid sedimentation is discussed by Morgenstern (1967), Dott (1963), McIlreath and James (1978), and other referenced authors. It is unknown whether failure along the delta front occurred or if failure occurred along the margin of a submarine channel, or other similar escarpment.

Because slumping and detachment involved both the Cane Spring and Grass Valley Formations, it is likely that the poorly lithified deltaic sediments were left largely unsupported on their western basinward margin. This concept coupled with the observations of soft sediment deformation in the base of the Grass Valley and its apparently unconformable relation to the debris flow suggests that it was locally emplaced by gravity sliding upon the debris flow. Renewed deltaic sedimentation then continued.

Mineralization and Alteration

Economically recoverable gold mineralization, principally as native gold or electrum, occurs most commonly as thick strata-bound lenses within the debris flow proximal to its contact with the Grass Valley Formation. Irregular zones of mineralization are present in minable quantities in the Cane Spring Formation also, but rarely present in the Grass Valley.

Permeability of the debris flow was a major factor in the mineralizing process. Hydrothermal fluids were effectively trapped under basal shales of the Grass Valley and migrated laterally under low hydrostatic pressures, favoring the silty and sandy zones of the debris flow. Mineralization of the larger limestone clasts occurs along their margins and in small fracture networks within the clasts.

The bulk of the mineralization occupies the crest of the broad northeast trending, southwest plunging antiform described earlier, but smaller subsidiary folds also provide important structural controls.

Two visually obvious alteration features present at the surface include large, resistant outcrops of jasperoid and widespread iron oxide staining. Where not intensely silicified, the breccia is mostly sericitized and weakly silicified. Argillization appears to be uncommon, but small quantities of kaolinite and montmorillinite have been identified.

Except in rare instances, the breccia is strongly and completely oxidized, probably by hypogene fluids. Most of the Cane Spring Formation is also well oxidized.

At the surface, the diorite intrusives are not only extensively oxidized, but are also propylitized. Quartz, calcite, serpentinite, talc, hematite, and pyrite are common alteration products observed in drill cuttings of the dikes.

Paragenesis and Geochemistry

By way of thin section, polished section, and scanning electron microscope analyses, Bruha (1983) concluded that at least five stages of mineralization have occurred at Relief Canyon.

The first stage involved minor brecciation and widespread introduction of quartz, pyrite, and sericite. These minerals initially nucleated on calcite shell fragments in the limestone clasts and along cleavage planes in the siltstone clasts, but were finally deposited as broad disseminated bands throughout the clasts and matrix.

The second stage involved minor brecciation and localized veining of quartz, MnOX (?), talc, and amorphous primary hematite.

The third stage involved brecciation and fracturing with introduction of flaky primary hematite and drusy quartz.

The fourth stage involved brecciation and very severe hypogene oxidation of all previously deposited minerals. This produced hematitic selvages on breccia fragments and hematitic banding in the matrix. Deposition of amorphous hematite upon previously developed pseudomorphs of hematite after pyrite also occurred as extensive solution banding

of original breccia clasts developed.

Deposition of zoned fluorite may have begun as early as the third stage of mineralization, but this is unclear.

The timing of gold deposition is also unclear, but it may have followed the third stage and terminated by the end of fluorite deposition. Electrum commonly fills tiny vugs or occurs with amorphous hematite. Clear evidence of gold associated with pyrite does not exist, nor has silica encapsulation of gold particles been observed. Most gold particles are 2-4 microns in maximum dimension.

Elements commonly associated with gold are silver, fluorine, arsenic, antimony, and mercury. One peculiar feature observed in polished section is an apparently late stage mineraloid containing minute crystals of sulfides of Cs-La, Pb, Zn, and Cu.

Homogenization temperatures of saline fluid inclusions in fluorite range between 160-187 C, indicating a minimum hydrostatic head of less than 300 feet.

Age of the mineralization is unknown and datable minerals have not been found. Field relations indicate a range of possible ages from Late Cretaceous to Late Tertiary. An interesting question is whether the mineralization at Relief Canyon is related to that of the Rochester Silver District, determined by Vikre (1981) to be Late Cretaceous in age.

DEVELOPMENT

History

The Relief Canyon Project site was originally known as the Emerald Spar fluorite prospect and minor exploration of surficial fluorite occurrences was done in the early 1940's according to Papke (1979).

In 1978, Falconi and Associates, an engineering firm based in Auburn, California, staked 4300 acres of claims at the site of the Emerald Spar prospect. Falconi's innovative intention was to develop a cement plant to service anticipated construction of the federal government's MX missile project.

In 1979, Jim McKee of Duval Corporation initiated an aggressive precious metals exploration program in the Humboldt Range, which involved mapping and stream sediment sampling. A single sample assaying 0.45 ppm Au was collected at the mouth of the main canyon transecting the gold deposit. With detailed mapping and sampling, Jim identified large zones assaying .01-.06 oz Au/ton. In 1981-82, Duval drilled 38 reverse circulation percussion holes and confirmed the existence

of a low grade, but potentially minable zone 2400' long by 1800' wide.

In the fall of 1982, Lacana Vice President Harvey Sobel and Reno Exploration Manager Mike Easdon reviewed the project data at Duval's Tucson office and immediately recommended the property for acquisition. Lacana President Ed Thompson negotiated a well-balanced option agreement with Duval and we immediately commenced bench scale metallurgical tests. Lacana now owns 100% of the original Falconi farmout.

Drilling and Metallurgical Tests

After receiving re-assay confirmation of Duval's drilling results and numerous cyanide leaching tests on the drill cuttings, we proceeded to cut a series of deep trenches for large metallurgical samples. Upon collecting approximately 2.5 tons of various ore-grade material, a battery of cyanide bottle roll, column leach, barrel leach, assay screen analysis, agglomeration, crushing, and gravity concentration tests were conducted. The principal labs involved included Dawson Metallurgical Lab in Salt Lake City, Kappes-Cassiday Associates in Reno, and Hunter Mining Lab in Reno.

The following results were indicated: cyanide leach recoveries ranged between 50-90% at a rapid leach cycle with low to moderate reagent consumption; a particle gold problem was not observed; agglomeration might enhance percolation; point to point assay variation among samples could be large, but the overall average grade indicated by Duval's drilling was preserved.

With this encouragement in hand, we immediately commenced detailed geological mapping and systematic drilling on 200' centers to evaluate the indicated geological reserves. 48 reverse circulation percussion holes were completed by Eklund Drilling Company of Elko by the spring of 1983. Upon compiling the drill assay data, we learned that our potential reserve calculations very closely matched our geological reserve calculations based on Duval's work.

Upon completion of our first drilling phase, we decided to proceed with a pilot-scale metallurgical test in the field. Many well known metallurgists and experienced operators including Lacana's Chief Engineer Colin Marshall, Don Duncan, Robert Shoemaker of San Francisco Mining Associates, Harris Salisbury of Dawson Metallurgical, Bruce Thorndycraft of Pinson, and Bruce Brogoitti and Ash Patwardan of Mine and Mill Engineering contributed to the process engineering and pilot plant design.

We decided to mine and cyanide heap leach two 5000 ton ore blocks. The mining sites of these ore blocks were chosen to be representative of not only the bulk of the gold deposit, but also of material we knew would be mined during the pay-back period of the project.

Bob Hoover of Bo-Ter Construction Company was contracted to mine the ore and construct the leaching facility, crushing plant, and recovery circuit according to final design engineering provided by Mine and Mill Engineering.

Four potential mining sites were chosen and drilled on 25' centers by reverse circulation. Two of these potential sites were then selected as being adequately representative and overburden stripping began simultaneously with plant construction. Upon completion of the strip, all of the in-place heap leach ore was drilled on 3'-5' centers for blastholes. A composite sample for each blasthole was collected and assayed. Additionally, a composite head sample split of approximately 1000 tons was cut from the original 10,000 tons of mined ore as it was delivered to the pilot plant.

Ore from the two mining sites was mixed and roughly 4700 tons of run-of-mine ore was loaded on the first pad by front end loader and dozer. Dilute cyanide solution was immediately applied to this heap.

The remaining 4300 tons of mined ore was crushed to nominal -3/4" and periodically sampled. Approximately 7 lbs. of Portland cement and 1 lb. of NaCN per ton of ore were added to the crushed rock. 12 weight percent water was added to this mixture as it was rotated in a drum agglomerator. Upon curing of the cement, the agglomerated ore was loaded onto the second pad by front end loader and leached.

Performance of the agglomerated ore heap has been excellent in terms of rate of gold extraction, percolation, and gold recovery. Recovery to date has exceeded 75% of the contained gold.

Similarly, performance of the run-of-mine ore heap has been outstanding also. Slight ponding occasionally occurred, but solution channelling is not evident. To date, this heap has an indicated recovery exceeding 65% of the contained gold.

Fill-in drilling on 100' centers commenced in late summer of this year and we expect to be finished with our final drilling program by the end of the year. This work has allowed us to refine both the geometries of the ore blocks and our geological interpretations. Preliminary evaluation of the newer drilling data suggests that both grade and tonnage of our original potential reserve calculation have decreased by a factor of approximately 5%.

We estimate that by the end of this year, approximately 400 holes will have been drilled by Duval, Lacana, and an adjacent property owner to fully define the deposit.

Production Plans

As earlier mentioned, Lacana and another company are developing the deposit. Joint venture negotiations will soon be finalized. Lacana will operate the venture. Production could commence as early as summer of 1984.

The trend of the deposit remains open to the southwest, but it is currently untested. It appears likely that as mining proceeds, development drilling of this region might expand the estimated 8-year mine life.

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