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GOLD Reprints 1984 - [illegible]

1. Dixie Comstock, NV.
2. US MX Mid-Year Report, Jan. 15, 1984.
3. Borealis, NV Solution Recovery Techniques
4. Homestake to buy Felmont Oil 25% of Round Mtn
5. Kesler Proposal.
6. Excerpt - NV Bureau Report 36
7. Gold Fields Mesquite Property April 84
8. Kappes, Cassiday & Assoc. Papers.
   1. Leaching Small Au-Ag Deposits
   2. Heap Leaching - Single ?
   3. Sol-ex Experience
   4. Sterling Mine JV.
   5. New Developments
   6. Advances in Sext. for Cu.
   7. Bid for Plant.
   8. Small & Large Heaps.
   9. Consideration of Choice of Carbon Adsorption.
9. Alligator Ridge, NV June 84
10. Carlin Exploration Program.
11. GSA Abstract June 84 - Alligator Ridge, NV
12. GSI News - Thallium - A Pathfinder
13. Aquarius Mine
14. Noranda's Teesma Deposit, NV.
15. Osceola Placer - Continental Gold. Oct 84
16. Ropes, Project
17. Operating Costs & Cash Flows Notes. 4 pages.
18. EG - Alaska Landforms & Ore Deposits Sept Oct 84

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19. Sedimentary Hosted Precious Metal Deposits.
20. Relief Canyon, NV Brochure.
21. A Proposal to Explore for Disseminated Gold Deposits in NV & Adjacent States, by RC Leeche
22. Note on Hayden Hill, CA.
23. McLaughlin Deposit Geochemistry. Nov '84
24. Precious Metals — Boom or Bust!
25. Atlas' Gold Bar.
26. Final EA — Paradise Peak, NV.
27. Gold Costs, USA.
28. Mesquite Project (BLM) Jan '85
29. Taylor Mine, NV.
30. Canada — Mining Tax Gold Concerns.
31. Cortez caldera + gold mineralization
32. Freeport - McMo - Ron 4th Qtr report.
33. Some Gold Note — 4 Pages
34. The Mining Industry — Au + Ag. Einar '85
35. Cortez, NV Au mineralization
36. SMC Report 85-94 Petrographic Studies - Eight Mile.
37. Amax Announces "Sleeper": Feb '85
38. Relief Canyon, NV
39. GSA Abstracts: Verdstone Deposit
~~40. 1st Round H~~ Cordilleran Deareation etc.
40. 1st ~~Round~~ Hemlo Gold.
41. Selection of Methods of Gold Analysis.
42. Minor Metals in Focus.
43. Alligator Ridge, NV — 5th year

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34. The Gold Mine VS The Quail at Dawn!
35. Mesquite Gold Deposit
36. Margarita Property
37. Sergeson Gold – A Possible Exploration Lead by NC
38. OF 84-890. Controls on Base & Precious Metal Mineralization in Deep, Epithermal Environments.

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44. NV Gold Placers. INC. Maps
45. Newell Lab.
46. The Gold Mine vs The Quail at Dawn
47. Hemlo & Other Prospects '86
48. The Harp-Lachung Granulation - Fred Nolte.

WRK — Thanks, looks like the approach needs to be followed more o more. Jim

January 4, 1984

MEMORANDUM FOR: Peter G. Vikre, Manager
Great Basin Exploration Division
Reno, Nevada

Dixie Comstock Project
Nevada

Dear Mr. Vikre:

Upon your return from vacation the top priority job for you and your staff will be to assemble a complete report on the Dixie Comstock exploration to date. A detailed list of what exactly should be in the report is as follows:

1. The report should include a geologic map which covers the Dixie area along the lines of the one that Barney mason previously submitted. Any new information should be included.

2. Please provide a drill hole map showing the number of the hole above the circle which locates the hole and the total depth of the hole below the circle. This should be on a crude contour map which shows the location of the mill, all pits, roads, tailings, bed rock, shafts, and the range front fault.

3. Maps of the underground workings should also be provided, although if it is convenient to composite them onto a single plan, that's fine. Any underground assay data which is available should be plotted on the map(s) of the underground workings.

4. Please provide cross sections at a scale of 1"=50' primarily along east/west and north/south orientations. If you want to draw some orientations in some other directions, that's fine too, and you probably can't draw too many sections. Each section should show the hole number at the top and the footage at the bottom of the hole. Individual assays should be written on the left side of the drill hole and selected average intercepts for gold on the right. The position of bed rock should be shown as well as the base of oxidation. Holes projected more than 10' to the section should be drafted with a long dash if they are in front of the section and a short dash if they are in back of the section with a notation showing how far the holes are projected to the section. Please send a set of sections with the gold assays and averaged intercepts separately from the sections described in the next paragraph.

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5. Using a xerox copy of the sections showing gold please contour on the cross sections all gold assays and also contour all assay data for arsenic, mercury, silver, antimony or other metals for which sufficient data are available. You do not need to draft on the individual geochemical values. It might be easier to use the sections with the gold assays and simply contour mercury or other metals on the gold base.

6. Using a separate xerox (or copy) of the drill hole surface plan, please contour the following:

   a) Average gold, silver, arsenic, mercury and antimony, in alluvium where assays are available.

   b) Contour average values of gold, silver, arsenic, mercury and antimony in bed rock rock where available.

   In addition to contour maps showing metals and bedrock and alluvium, you might also want to contour gold at a 0.03 oz. Au cutoff and contour the average thickness of 0.03 oz. Au intercepts.

The accompanying report which should be written by you should briefly discuss the rock types, structure, alteration, mineralization, accuracy of the assays, review of check assay work, metallurgy, and a brief discussion of the various contour maps. Finally, your report should contain a specific section which discusses the potential of the property including where the potential mineralization occurs, how much of it is there, the grade, and reasons. All drafting can be done by hand as long as it is legible. You may want to involve everybody in your office to at least accumulate the basic information which then can be interpreted by you or others you might select, but in any case, preparation of this report should be one of your top priorities and will be required in New York before any further drilling can be contemplated. Your report should conclude with a section on recommendations and you might wish to locate drill holes which you would like to see drilled in order to fully test the potential of the property.

If you have any questions, please give me a call.

Very truly yours,

F. T. Graybeal

F. T. Graybeal

cc: W. L. Kurtz

USMX

U.S. MINERALS EXPLORATION COMPANY

MID-YEAR REPORT

January 15, 1984

To Our Shareholders:

Precious metal exploration expenditures by our joint venture partners and USMX in the past six months have provided encouraging results on several USMX properties. Achievements in petroleum exploration and financing have also been very positive during the same period.

USMX's cash position has doubled from $1.5 million to $3.1 million and assets have increased from $3.7 million to $5.5 million since year end.

During the first two quarters of fiscal 1984, the Company's cash position and net working capital have doubled from $1.5 million to $3.1 million and our asset position has increased from approximately $3.7 million to $5.5 million. This increase is primarily due to the private placement sale of the Company's restricted stock to Centennial Minerals Ltd. and several leading European financial institutions at prices ranging from $ .50 to $ .66 per share. Centennial Minerals now owns 4.5 million shares of USMX, or approximately 8.5% and USMX owns 400,000 shares of Centennial, or approximately 8.5%. In November, Centennial completed its public offering at (C) $5.00 per share. Our investments in Centennial and our affiliate, Grayhill Exploration are listed at cost or equity basis in the financial statements; therefore, present market value, which is substantially higher, is not reflected in shareholders' equity.

At the Montana Tunnels, further drilling has upgraded probable and possible reserves to proven. Proven reserves amount to 28.1 million tons grading 0.04 oz/ton gold and 0.36 oz/ton silver. (1.1 million oz. gold and 10.1 million oz. silver).

On the Montana Tunnels property, results from 49 additional drill holes completed by our joint venture partner, Centennial Minerals, have been recently received. These results combined with the results of 95 previous drill holes (144 drill holes totalling 72,000 feet) have revised and upgraded a substantial portion of probable and possible reserves into the proven category. Proven reserves currently amount to 28.1 million tons grading 0.04 ounces per ton gold and 0.36 ounces per ton silver based on a cut-off of 0.02 ounces per ton (gold equivalent). Combination of both gold and silver values yields a gold equivalent grade of 0.048 ounces per ton. Base metals grading approximately 0.30% lead and 0.90% zinc may add significant credits to the overall profitability of the project.

Additional probable and possible reserves have been discovered and an extensive drilling program is planned for the Spring of 1984 in an attempt to establish additional proven reserves.

These reserves are based only on the core area of metallization drilled on average 130-foot centers to the depth of 600 feet proposed for the first open pit. Deeper portions of the drill holes (between 600 to 1000 feet) and step-out drill holes outside the core area have discovered further ore grade gold and silver mineralization. Many of the deeper holes have terminated in ore. An extensive drilling program is planned for the Spring to upgrade additional probable and possible reserves into the proven category.

A leading precious metals expert has compared the Montana Tunnels diatreme potential to a number of diatreme deposits in the world, including Cripple Creek, Colorado.

Dr. Richard Sillitoe, a leading precious metals consultant, and members of the United States Geological Survey recently studied the Montana Tunnels area. Dr. Sillitoe concluded that the Montana Tunnels property is a large diatreme structure with a strongly mineralized core area containing ore grade gold, silver, lead and zinc values. He confirmed that much of the property remains unexplored and excellent potential exists for additional high grade mineralization. He has compared the Montana Tunnels diatreme in size and potential with other large, ore-bearing diatremes in the world such as those at Balatoc, Philippines and Cripple Creek, U.S.A. The Cripple Creek diatreme has a recorded production of over 21 million ounces of gold and is still producing. Gold ore ranging between .03 to over 1 ounce per ton has been mined near the surface by open cut and by underground methods to depths of over 3000 feet over a large area of the Cripple Creek diatreme. Although the parallels between the two diatremes are striking, a much deeper and extensive drilling program will be required to establish the similarities and potential of our property.

At the Montana Tunnels, a number of the drill holes have intersected gold mineralization ranging between .50 to over 2.5 ounces per ton over true thicknesses ranging from 10 to 50 feet. The Montana Tunnels is principally a large low grade deposit amenable to low cost open-pit mining methods. Within the low grade deposit, discreet pods and veins of high grade mineralization are erratically distributed which may add an additional parameter of potential profitability.

The engineering frim of Dolmage Campbell & Associates Ltd. have recently completed a report on the project. They have acknowledged that the program conducted by Centennial is being carried out in a well executed and professional manner and confirmed that ore reserve estimates are accurate. Dolmage Campbell has verified that the operating and capital costs for the project are very favorable at an initial production rate of 10,000 tons per day. The overall stripping ratio is less than 1:1 during the early years of production and current reserves will support this rate for a period exceeding 13 years. The project is well situated with respect to the availability of water, power, labor and housing. The topography and large land position provides an excellent site for a large open-pit mining and milling operation.

The next phase of development includes preparation of the final feasibility study, receiving the required environmental operating permits, and obtaining the financing to proceed with construction. Major engineering firm proposals are now being reviewed to prepare the final feasibility study. The Colorado School of Mines Research Institute and Hazen Research Inc. are completing the final metallurgical studies on a 350-ton bulk sample mined from the deposit in November. These pilot scale tests will provide optimum plant design criteria for the mill system and are scheduled for completion by late April. ELM Consultants of Portland, Oregon, has commenced Phase I of the environmental permitting process. Several large banking and investment institutions are currently considering financing proposals for the project.

The requirements to commence production are progressing satisfactorily and it is expected that the final feasibility analysis and design engineering will be complete by late 1984 and construction is expected to commence in early 1985.

Gold and silver mineralization has been discovered on seven other USMX properties.

The company has been involved in seventeen additional mineral properties with the exploration financed by other joint venture partners and USMX. As of November 30, 540 drill holes had been completed on these properties and additional drilling is in progress.

Gold and silver mineralization of possible economic significance has been discovered on the surface and in drill holes on the following properties currently under joint venture agreements. In Nevada, the SB property - AMSELCO; the GL property - COMINCO; the GS property - SAGE; the Silver Lode property - SOUTHERN PACIFIC; in Oregon, the Coy property - CHEVRON. AMAX and NICOR have recently formed a joint venture to explore the B property in Nevada and GOLDSIL is continuing to evaluate the SG property in Montana. Due to discouraging drilling results, ASARCO terminated the agreement on the DC property in Nevada and LACANA has terminated the agreement on the BC property in Montana.

USMX expended $120,000 over the last six months drilling eight 100%-owned properties located in Nevada. High grade silver mineralization (5 to 300 ounces per ton) has been discovered on the surface and in close-spaced shallow drill holes on the HV property. Low grade silver mineralization (ranging from 1 to 7 ounces per ton) has been encountered near the surface in wide-spaced drill holes at the JS property. Forty-five drill holes have been completed on these two properties. Near surface gold mineralization (ranging from .01 to .24 ounces per ton) has been intersected on the G, Maggie Creek and W & K properties. A total of 179 drill holes have been completed on these three properties. Drill hole results on the A, O, and PR prospects were negative. USMX is currently conducting metallurgical studies and entertaining development proposals from industry partners on the properties which have proven precious metal mineralization.

Recent discoveries in Kansas include two producing oil wells and one producing gas well.

Positive results have been forthcoming in our petroleum activities. Two wells were drilled in December, 1983 in Barton County, Kansas. Both wells have been completed as shallow oil producers and are currently being equipped with pumping units and tank batteries. Based on completion tests the #1 Yeakley is expected to initially produce at a rate of 100 barrels of oil per day and the #1 Amerine is expected to initially produce at a rate of 20 barrels of oil per day. The U.S. Minerals Exploration/Grayhill Exploration Joint Venture (USJV) owns 50% working interest in these wells and approximately 520 surrounding acres. The development potential of the Yeakley-Amerine area is very promising.

The #1 Davis well was completed in Pawnee County, Kansas in March of 1983, gas sales at the rate of 250 MCF gas per day began on January 13, 1984. The USJV owns 25% working interest in the #1 Davis and in 640 surrounding acres known as the Davis Unit. The development potential for the Davis Unit looks very encouraging.

The #1 Wirth well, drilled and completed in Morgan County, Colorado in October of 1983, tested oil and gas but in non-commercial quantities. The information gained from this well indicates very good potential in undrilled portions of the lease. USJV is evaluating the possible courses of action which may be undertaken to further test this property with the least risk to the USJV. USJV currently owns 37.5% working interest in 640 acres surrounding the #1 Wirth.

At our annual meeting held in October, Gary Grauberger, David Jonson, Michael Mehrtens, John Shannon and Gregory Pusey were re-elected to the Board of Directors. Management has since retained Price Waterhouse Inc. to replace Fox & Company as our independent auditors. In addition, we have retained Kanan, Corbin, Schupak & Aronow, Inc. of New York, a public relations firm, to represent the Company in the investment community. USMX has opened an office in Reno, Nevada and employed Earl Abbott as regional geologist to supervise and continue our exploration activities in the southwestern United States.

Centennial Minerals Ltd. and leading European financial institutions have recently acquired 10% of USMX's outstanding shares.

In summary, the Montana Tunnels property has a proven precious metal reserve which ranks as one the largest new discoveries in North America. From this foundation, USMX is poised for growth as a significant precious metals producer. Our exploration expertise has attracted over $7.5 million in exploration expenditures from industry partners allowing the Company to remain debt free while solidifying our capital base. Present capital commitments by our partners for development of USMX properties currently totals $27 million. The recent acquisition by several leading European financial institutions and Centennial Minerals of approximately 10% of the Company's shares underscores the confidence placed in USMX's income prospects. Your Company intends to capitalize on this recognition and develop beneficial relationships with leading North American financial institutions.

USMX now believes it is solidly positioned for growth and looks forward to future increases in revenues, cash flow and earnings. We appreciate your continued support and loyalty.

Gary L. Grauberger, President

BALANCE SHEETS

ASSETS

	November 30, 1983 (Unaudited)	May 31, 1983 (Audited)
Current assets:		
Cash, including certificates of deposit	$3,115,000	$1,530,000
Trade accounts receivable	65,000	28,000
Interest receivable	20,000	18,000
Notes receivable	4,000	4,000
Other	4,000	4,000
Total current assets	3,208,000	1,584,000
Property and equipment, at cost:		
Mineral properties	687,000	539,000
Oil and gas properties (full cost)	651,000	805,000
Vehicles	42,000	30,000
Equipment	21,000	21,000
	1,401,000	1,395,000
Less accumulated depreciation and depletion	200,000	309,000
	1,201,000	1,086,000
Investments and other assets:		
Investment in Grayhill Exploration Company	724,000	722,000
Investment in marketable equity securities, at cost aggregate fair market value approximating $1,336,000 at Nov. 30, 1983 and $312,000 at May 31, 1983	434,000	312,000
	1,158,000	1,034,000
	$5,567,000	$3,704,000

LIABILITIES AND STOCKHOLDERS' EQUITY

	November 30, 1983 (Unaudited)	May 31, 1983 (Audited)
Current liabilities:		
Accounts payable	$45,000	$5,000
Payroll taxes payable	5,000	1,000
Total current liabilities	50,000	6,000
Commitments	—	—
Stockholders' equity:		
Common stock, no par value		
Authorized - 80,000,000 shares		
Issued and outstanding - 49,050,000 at May 31, 1983 and 52,500,000 at Nov. 30, 1983	5,514,000	3,626,000
Retained earnings	3,000	72,000
	5,517,000	3,698,000
	$5,567,000	$3,704,000

STATEMENTS OF OPERATIONS
(Unaudited)

	Three Months Ended November 30, 1983	Three Months Ended November 30, 1982	Six Months Ended November 30, 1983	Six Months Ended November 30, 1982
Revenues:				
Property conveyances, rentals and option fees	$ —	$ 40,000	$ 12,000	$ 78,000
Income from oil and gas sales	—	7,000	—	7,000
Interest income	61,000	27,000	110,000	59,000
Sales of oil and gas wells	—	—	16,000	—
Miscellaneous income	5,000	6,000	8,000	8,000
	66,000	80,000	146,000	152,000
Costs and expenses:				
Prospecting	6,000	41,000	21,000	41,000
General and administrative	120,000	105,000	196,000	174,000
	126,000	146,000	217,000	215,000
Income (loss) before equity in net income (loss) of affiliate	(60,000)	(66,000)	(71,000)	(63,000)
Equity in net income (loss) of affiliate	(16,000)	3,000	2,000	5,000
Net income (loss)	$ (76,000)	$ (63,000)	$ (69,000)	$ (58,000)
Weighted average number of common shares outstanding	51,452,000	47,030,000	50,324,000	47,030,000
Net income (loss) per share	$(.00)	$(.00)	$(.00)	$(.00)

Contact: Gary L. Grauberger
Judy R. Jones
12189 Ralston Road, Suite 200
Arvada, Colorado 80004
(303) 420-3547

I have no interest in subscribing KUPER April 4 issue Send to ~~WK~~ JDS

HEAP & DUMP LEACHING

N012/F

JANUARY 1984 VOL. 1 - NO. 1

AN INTERNATIONAL NEWSLETTER

Borealis

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FROM: F. T. GRAYBEAL

TO: DM Smith - Denver
PG Vikre - Reno
WLKurtz } - Tucson
JD Sell }

Kesler Proposal

Do you see any merit in the attached? I guess I'd prefer to support university theses research. *

1/3/84

W. copy SEK proposal

* I agree that thesis work is best — tho I guess Kesler would try + work the above proposal into further thesis funding. Use $25 000 to be used elsewhere.

The University of Michigan

DEPARTMENT OF GEOLOGICAL SCIENCES
1006 C. C. LITTLE BUILDING
ANN ARBOR, MICHIGAN 48109-1063
(313) 764-1435

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December 22, 1983

F. T. G.
JAN 3 1984

Dr. Frederick T. Graybeal
Chief Geologist
ASARCO Inc.
120 Broadway
New York, NY 10271

Dear Fred:

I am on sabbatical leave for calendar year 1984 and am trying to line up exploration-oriented projects. I am particularly interested in following up on the work we have done using gas analysis in exploration for epithermal vein and hot spring deposits. The method is based on analysis of gases released from fluid inclusions by decrepitation and we have already used it to delineate strong gas anomalies around Pueblo Viejo and greenstone gold deposits in the Porcupine district.

No real work has been done on epithermal systems however, and it is here that the method has the most promise. If ore shoots and zones is these deposits form because of boiling, as is generally agreed, we should be able to detect zones of unusually high gas concentrations in barren vein or rock material above boiling (e.g. ore) zones. If this idea is right it could guide both regional and mine-scale exploration. To test this possibility we need to carry out a detailed survey in a mineralized vein system analyzing several hundred samples collected above, in and below ore zones. My first choice for a test area would be the Silverton-Lake City area in the western San Juans, but we would be glad to work anywhere else that looked better.

It would take about $20,000 and 6 months to do the job including $4,000 field costs, $1,000 sample prep costs, $10,000 analytical costs and $10,000 for two month's salary. Would ASARCO be interested enough to support all or part of an effort of this type or in suggesting any other line of supportable exploration research? Best regards.

Yours very truly,

Steve

Stephen E. Kesler

SEK/mis

Ag

Au

NEVADA BUREAU OF MINES & GEOLOGY
REPORT 36

PAPERS GIVEN AT THE PRECIOUS-METALS SYMPOSIUM SPARKS, NEVADA NOVEMBER 17-19, 1980

Victor E. Kral, Editor
March, 1983

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GOLD DEPOSITS AT COPPER CANYON, LANDER COUNTY, NEVADA

DAVID W. BLAKE Duval Corporation, Battle Mountain, Nevada 89820
EDWARD L. KRETSCHMER Pinson Mining Company, Winnemucca, Nevada 89445

ABSTRACT

Application of geologic concepts and a concentrated drilling program resulted in the discovery of the Tomboy and Minnie gold orebodies adjacent to Duval's Cu-Ag-Au open pit mine at Copper Canyon. These gold deposits were put into production during the Winter of 1978-79. The two gold deposits are located about 2 km south of the Copper Canyon open pit and represent a unique gold occurrence that has relations similar to the Cu-Ag-Au ore deposits in the district.

Gold occurs in a 30 meter thick, sulfide-bearing basal portion of the lower member of the Middle Pennsylvanian Battle Formation. In both the Minnie and Tomboy gold deposits, the Battle Formation consists of conglomerate and calcareous sandstone and shale that have been thermally metamorphosed to a calc-silicate bearing conglomerate and quartz-tremolite hornfels. Within the basal conglomerate, the matrix has been replaced primarily by pyrite and pyrrhotite, and lesser quantities of sphalerite, galena, marcasite, and chalcopyrite. This high sulfide-bearing, metaconglomerate contains the gold now being mined. The gold ore occurs mainly in closely spaced fractures and as replacement sulfides in the metaconglomerate. A north-south striking silicified granodiorite dike exposed in the Minnie deposit is believed to be genetically related to the deposition of gold-bearing sulfides and the hydrothermal alteration products include silicification, epidote replacing tremolite, chlorite, and clay minerals. Quartz veining is rarely observed in the deposits.

Geochemical study of surface media was undertaken to determine if anomalous gold concentrations in rocks and soils would outline the individual deposits in the Tomboy-Minnie area. Anomalous gold concentrations greater than 1.7 ppm in the first interval of drilling outlines both deposits. Anomalous gold in soil (greater than 1 ppm) corresponds with the known occurrence of gold in outcrops.

INTRODUCTION

The gold deposits at Copper Canyon are located in the southeastern part of the Battle Mountain mining district about 150 km northeast of Reno (Figure 1). These gold deposits were put into production in late 1978, because of dwindling Cu reserves and a price increase in gold. Duval Corporation, a wholly owned subsidary of Pennzoil Company, also operates an open pit leach copper mine at Copper Basin, 10 km north-northeast of Copper Canyon.

Since 1866, when the first claims in the district were located, various mining companies and lessees were active. Most of this activity prior to Duval's operations, was between 1916 and 1955 when the Copper Canyon Mining Company controlled the property at Copper Canyon. Most of the production during this period was from the underground Copper Canyon Mine. Natomas Company operated a dredge at the mouth of Copper Canyon between 1947-1955 and according to Johnson (1973, p. 37-38), recovered 100,000 ounces of gold.

In the early 1960's, Duval Corporation acquired claims at Copper Basin and Copper Canyon from American Smelting and Refining Company. Exploration work conducted by Duval Corporation from 1962 to 1965 outlined a low grade deposit of Cu-Ag-Au at Copper Canyon and a Cu deposit at Copper Basin. The first concentrate was produced at Duval's mill located at Copper Canyon in 1967.

Total production for the district up to 1961 includes 150,000 ounces of Au, 2.1 million ounces Ag, 15,000 tons of Cu, 5,000 tons of Pb, and 1,500 tons of Zn. Minor amounts of Sb have also been produced. Morrissey (1968, p. 13) indicated that approximately $1 million in rough turquoise was produced from Copper Basin up to 1962.

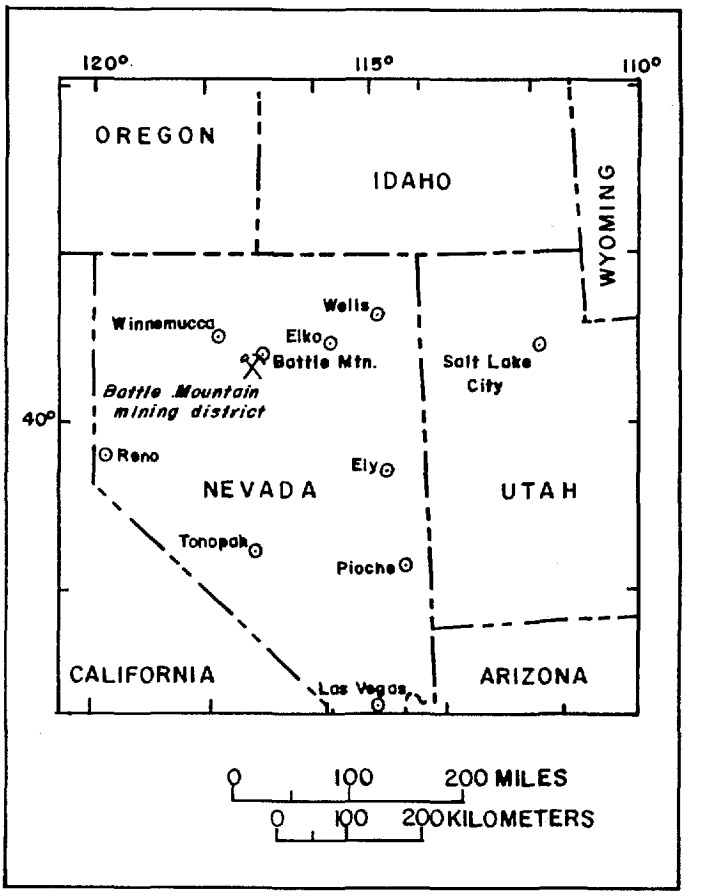


FIGURE 1.

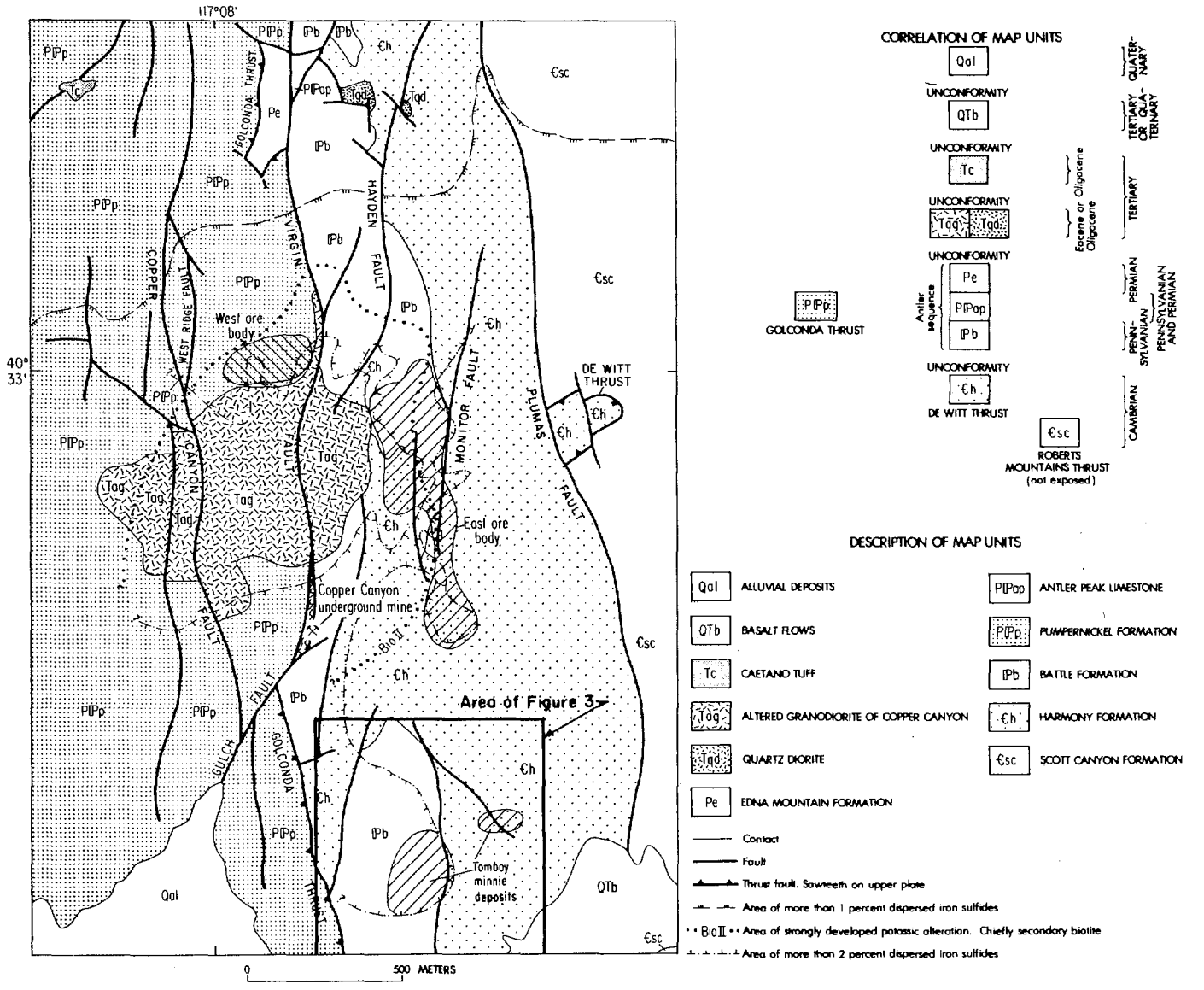


FIGURE 2. Geology of the Copper Canyon area.

In late 1978, Duval Corporation converted its Cu-Ag-Au mill at Copper Canyon to a cyanide Au operation. This gold operation, now in full production, culminated from comprehensive geologic study which was combined with a three year drilling program. Genetic models of ore deposits in the district were fully utilized in developing these deposits.

A detailed evaluation of the gold potential in the Battle Mountain mining district was initiated in 1973. This study coordinated past and present geologic data that had been collected by many individuals, including publications by Theodore and Blake (1975 and 1978); Blake, Theodore, and Kretschmer (1978); Roberts (1964); and Roberts and Arnold (1965). Various unpublished company reports served as an aid in our evaluation of Au occurrences for those inaccessible underground workings. The geologic information available and studies conducted by the authors led to the discovery of gold-bearing units in the Tomboy, Minnie and Independence mines. Presently ore is being produced from the Tomboy Mine, and pre-stripping is underway at the Independence Mine. Production at the Minnie Mine ceased in December of 1979.

This report includes background information on the geology, mineralization and alteration of the Copper Canyon Cu-Ag-Au and Au deposits. Geochemical distribution of gold in rocks and soils over and adjacent to the Au deposits is included to demonstrate our exploration usage of this metal to outline these relatively small gold deposits.

GEOLOGY AND MINERALIZATION AT COPPER CANYON

Sedimentary rocks of Paleozoic age are host for the various ore deposits at Copper Canyon. Intruded

into these Paleozoic sedimentary rocks and spatially associated with Cu-Ag-Au deposits is a small pluton of mid-Tertiary altered granodiorite porphyry.

The two deposits that were mined from 1967 to 1978 at Copper Canyon include the east and west orebodies (Figure 2). These two deposits were mined primarily for their Cu content, but they also contain significant Au and Ag as by-products.

Economic hypogene sulfide mineralization is confined to favorable former calcareous-bearing sedimentary units of the Middle Pennsylvanian Battle and Pennsylvanian-Permian Pumpernickel Formations. Supergene concentrations of Cu also occur in these units but such secondary copper constitutes ore only in the Upper Cambrian Harmony Formation and altered granodiorite where major faults have localized chalocite.

Mineralization in the east orebody is mostly confined to the lower member of the Battle—a metamorphosed calcareous conglomerate. In the west orebody, mineralization occurs in the lowermost part of the Pumpernickel Formation. The Golconda thrust at Copper Canyon separates tectonically the Pumpernickel from the underlying Battle Formation.

Thermal metamorphism, mineralization, and hydrothermal alteration are genetically and spatially related to the altered granodiorite porphyry (Figure 2). The sulfides in the two orebodies exhibit zonal relations with granodiorite porphyry (Figure 2). Pyrite, pyrrhotite, chalcopyrite and lesser quantities of galena, sphalerite, marcasite, molybdenite arsenopyrite, covellite, and chalcocite, are the recognizable sulfide minerals associated with the ore deposits at Copper Canyon. These minerals occur as disseminations, replacements, fracture fillings and in quartz veins and veinlets. Most ore grade mineralization occurs with a halo of potassic alteration and within an area of greater than two volume percent iron sulfide (Figure 2).

Siliceous and calcareous conglomerate in the basal part of the lower member of the Battle Formation in the east orebody hosted a fairly uniform body of Cu-Ag-Au mineralization (Figure 2). Alteration minerals within the ore zone consists of varying quantities of quartz, K-feldspar, biotite, and tremolite, depending on the overall original composition. Depth of oxidation ranges from 0 to 20 meters over an irregular 20 meter thick chalcocite bearing zone. Malachite, azurite and chrysocolla in the oxidized zone indicate the former presence of copper sulfides. In and near faults in all rock units, oxidation and chalcocite development are extensive.

The west orebody occurs in a garnetiferous and calc-silicate assemblage of rocks directly north of the granodiorite porphyry (Figure 2). The pyrite and pyrrhotite with lesser concentrations of chalcopyrite are the dominant sulfides in this 35 meter thick ore zone. Total sulfide content ranges from 10 to over 75% and decreases northward away from the granodiorite porphyry. Chalcopyrite is best developed near the intrusive contact. In addition, Au and Ag concentrations decrease northward.

GOLD DEPOSITS

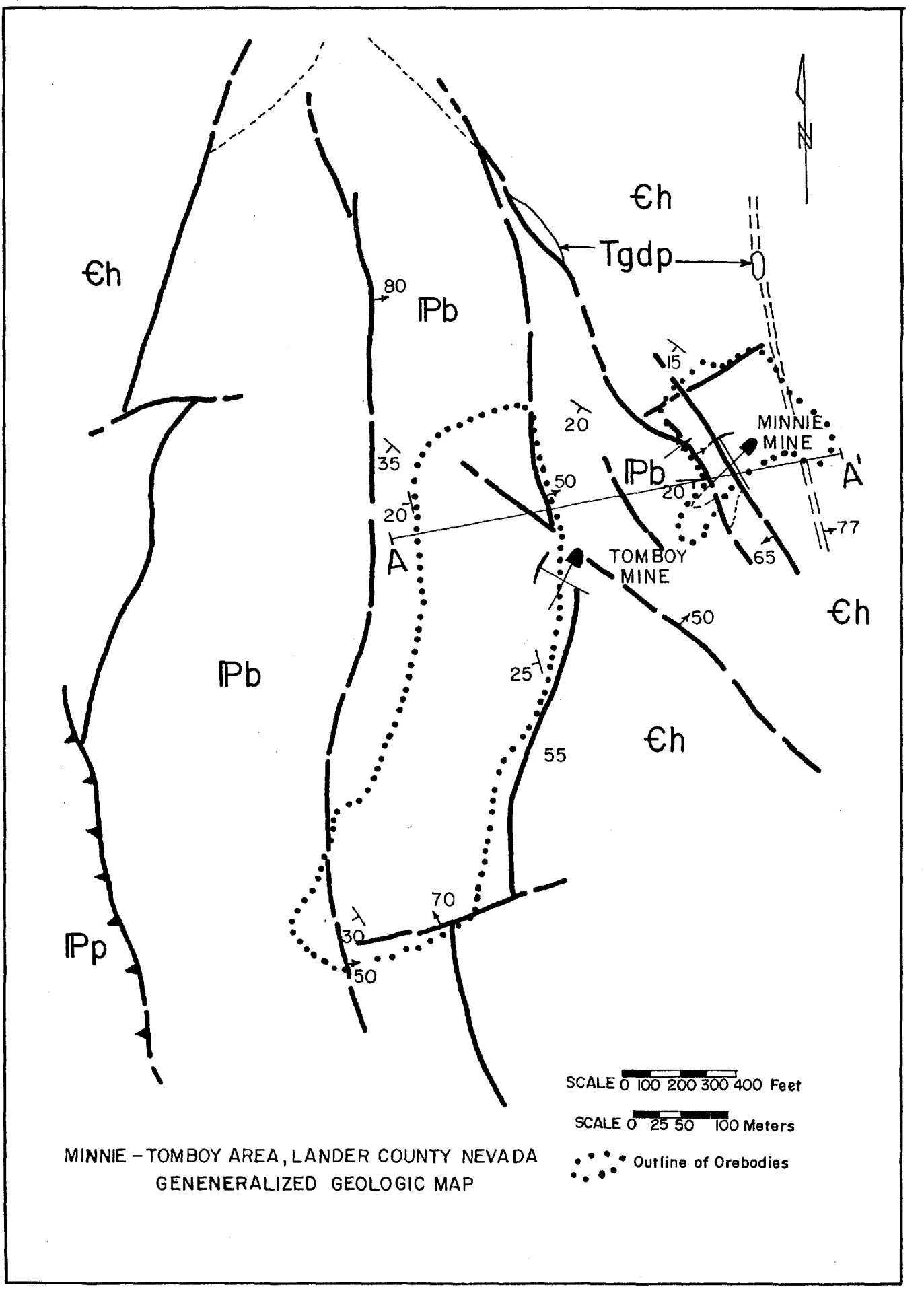
Duval Corporation started producing gold ore from the Tomboy and Minnie open pits in late 1978. Since that time the geologic setting for these two deposits has been studied using standard pit-mapping techniques. Detailed studies pertaining to the genesis of these gold deposits have only recently been undertaken. Therefore, much of the description of these orebodies in this report is based on pit mapping and drill cuttings.

The Minnie and Tomboy gold deposits are located about 1200 meters south of the Copper Canyon Cu-Ag-Au deposits (Figure 2). Mineralization in the gold deposits occurs in a 30 meter thick basal part of the lower member of the Battle Formation that overlies weakly metamorphosed sandstone and shale of the Harmony Formation (Figure 3). Along the southwestern part of the area (Figure 3), chert and siltstone of the Pumpernickel overlies the Battle Formation above the Golconda thrust. Along the eastern limits of the Minnie orebody, a north-trending granodiorite porphyry dike crops out.

The Minnie deposit has been down-dropped along a northwest striking normal fault with an approximate dip slip offset of 200 meters (Figure 4). A northwest striking reverse fault has offset these older normal faults and displaced gold-bearing conglomerate. A series of northwest mineralized fractures dipping moderately to the northeast have served as channel-ways for the fluids associated with gold deposition in the conglomerate of the Tomboy deposit.

Gold is uniformly restricted to the basal part of the Battle in both deposits. Economic concentrations of gold are not found in the Harmony below the unconformity. In the Tomboy mine, the conglomerate attains a total thickness of 100 meters and dips westward 30 degrees. Within the two gold deposits, the conglomerate consists of subangular to subrounded clasts up to one-half meter across. Clasts are chert, quartzite and rare limestone. Matrix for these clasts consists of metamorphosed medium-grained tremolite-bearing sandstone. Throughout the Battle Formation, lenses of former calcareous shale and limy units have been converted to tremolite. A few limestone clasts have been metamorphosed to garnet, quartz, diopside and tremolite.

Sulfide minerals recognized in the gold deposits are pyrite, pyrrhotite, and lesser quantities of sphalerite, galena, marcasite, and chalcopyrite. The majority of these sulfides occur as disseminations and replacements of the tremolite-bearing matrix. Vertical zoning of sulfide minerals is recognized in the deposits. Above the Au zone, pyrite and rare pyrrhotite are the only sulfides having a total combined sulfide content of 1.0 to 1.50 volume percent. Pyrrhotite is more abundant than pyrite in the gold-bearing metaconglomerate and the total combined sulfide content ranges from 10 to 50 volume percent.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Tgdp
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80
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15
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MINNIE
MINE
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Pb
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50
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€h
Pb
25
€h
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Pp
30
50
SCALE 0 100 200 300 400 Feet
SCALE 0 25 50 100 Meters
Outline of Orebodies
MINNIE - TOMBOY AREA, LANDER COUNTY NEVADA
GENENERALIZED GEOLOGIC MAP

FIGURE 3.

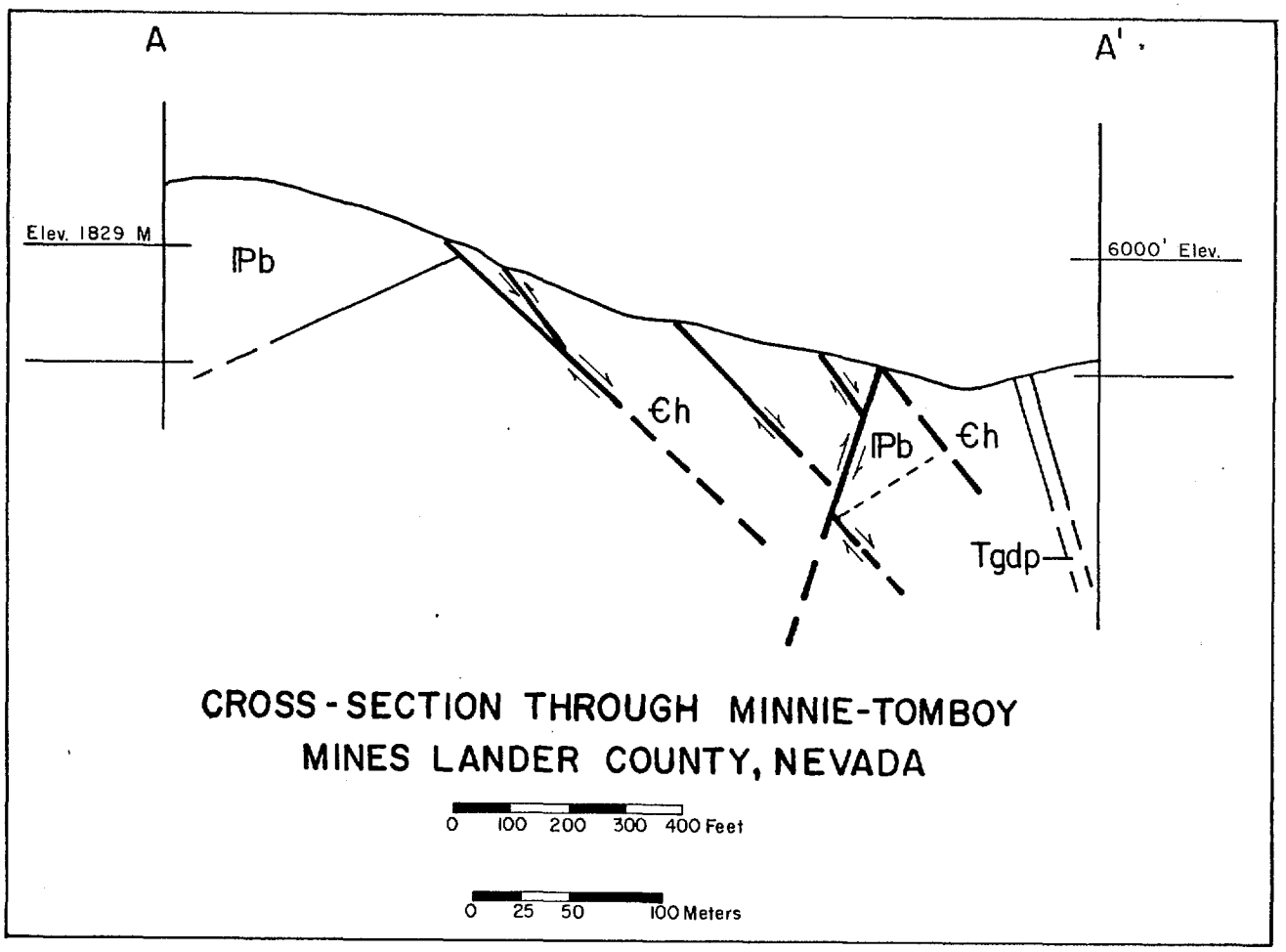


FIGURE 4.

The deposition of Au in these deposits is related to the introduction of pyrite and pyrrhotite and possibly to subsequent alteration of pyrrhotite to marcasite. Gold occurs as free minute grains, commonly less than 0.5 mm. Disseminated and replacement sulfides carry the gold as well as fracture-filling sulfides in the conglomerate. Silver mineralization is low throughout the two deposits, but high Ag assays correspond with high Pb concentrations.

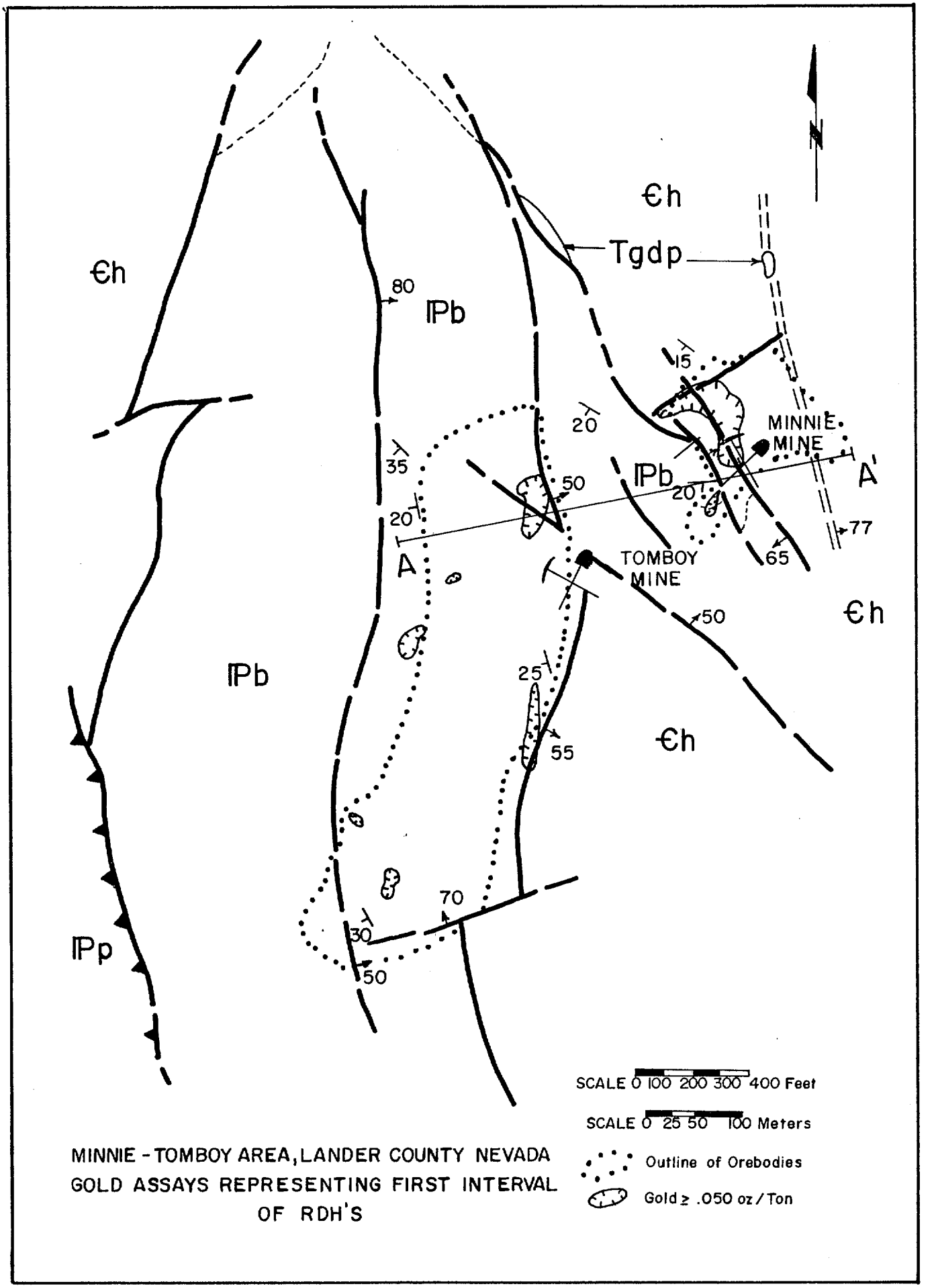
Thermal metamorphism of the conglomerate has produced a tremolite-bearing siliceous conglomerate. Hydrothermal alteration associated with gold deposition is reflected in the replacement of tremolite and other metamorphic products by epidote, chlorite and clay minerals. Epidote also occurs in fractures. Most of the clay is restricted to faults and as thin selvages along mineralized fractures. Aside from the silicification of the conglomerate and underlying clastic units of the Harmony Formation, quartz as a gangue mineral is rarely observed in veins or veinlets containing sulfides. The north-trending granodiorite dike is weakly silicified and contains minor alteration products typical of the propylitic assemblage.

GEOCHEMICAL STUDIES

Geochemical sampling was carried out after most of the drilling had terminated and during the initial pre-stripping stage. Therefore, we attempted to utilize what data was available and supplement it with randomly selected soil samples to correlate with the results of assays from drill cuttings. Figure 5 illustrates the Au assays from the first interval of bedrock drilled by rotary drill (RDH). Figure 6 shows the results of our soil sampling. We selected the B horizon here because geochemical results for gold were most consistent. All soil material analyzed represented the minus 80 mesh fraction. The dotted line outlines the surface projection of the two gold deposits on figures 5 and 6.

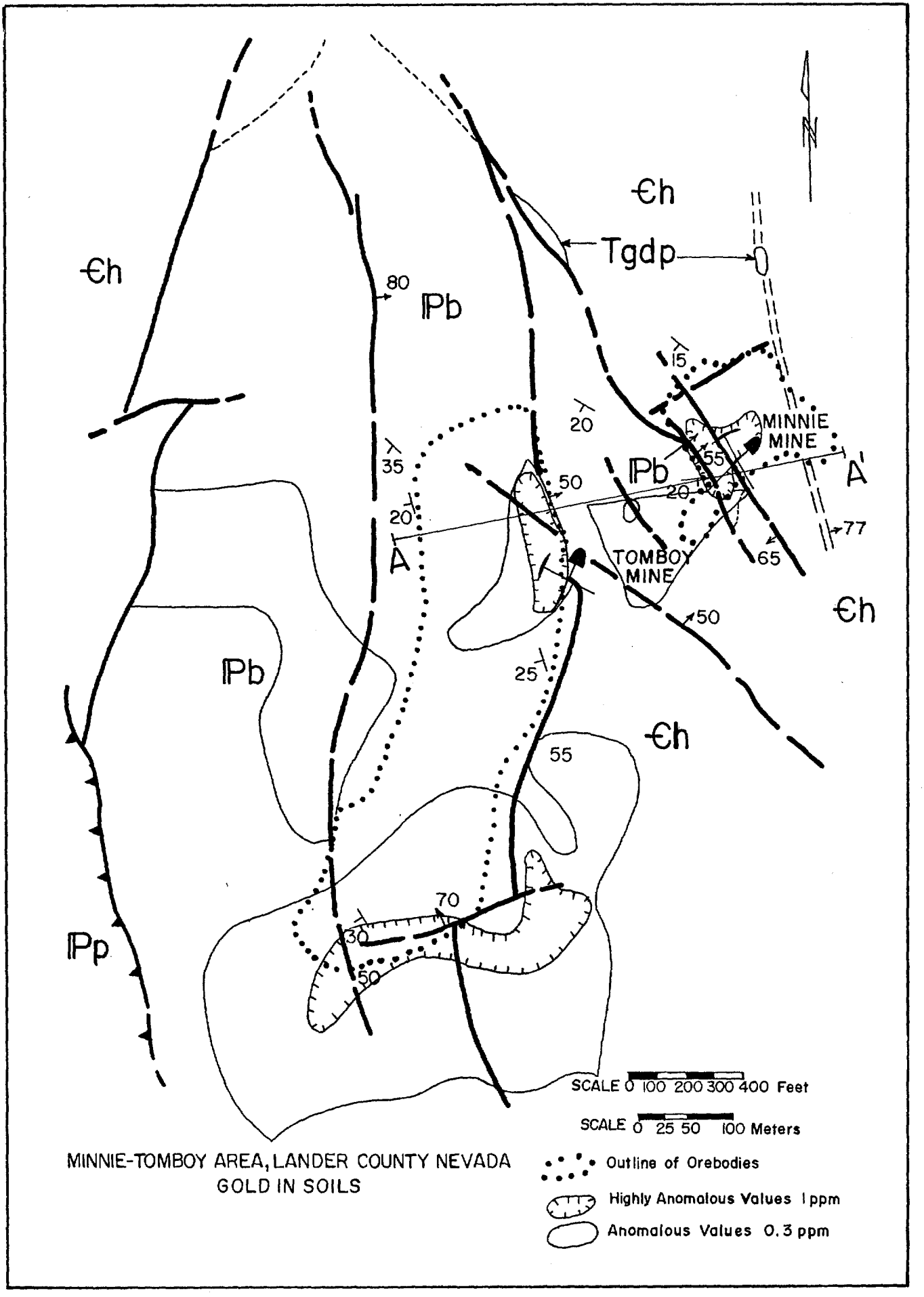
Gold concentrations greater than .050 oz/ton (1.7 ppm) clearly reflect the two deposits based on rock material assayed from the first interval of drilling (Figure 5). The broken north-south elongated pattern of the gold distribution in the Tomboy orebody reflects primarily the exposed basal Au-bearing member of the Battle Formation that dips westward about 30 degrees. In the Minnie deposit, anomalous gold was found in rock samples directly over the ore deposit.

Figure 6 shows the results of our soil geochemistry for gold, contoured using two sets of Au concentrations. Those samples considered highly anomalous, or greater than 1 ppm, overlie the original discovery for both the Minnie and Tomboy mines. The highly anomalous area along the



MINNIE - TOMBOY AREA, LANDER COUNTY NEVADA
GOLD ASSAYS REPRESENTING FIRST INTERVAL
OF RDH'S

FIGURE 5.



MINNIE-TOMBOY AREA, LANDER COUNTY NEVADA
GOLD IN SOILS

FIGURE 6.

southern edge of the Tomboy deposit reflects structural control for Au deposition. The area of 0.3 ppm Au in soils is significantly more extensive than the area of 1 ppm Au and encloses most of the high concentrations of Au. Further, the zone of 0.3 ppm Au in soils west of the Tomboy probably indicates downslope transport of gold by weathering from the main geologic occurrences at the Tomboy.

Based on our limited geochemical work at these two gold deposits, geochemical sampling using soils and bedrock material would have detected and outlined the ore zones. The most important geologic exploration guide is that the economic concentrations of Au are restricted to a favorable stratigraphic unit within a well-defined Au-Ag zone established on a district-wide scale. Evaluation of a previously known Au occurrence in this district using sound geologic concepts of ore genesis in conjunction with limited geochemical sampling successfully outlined the deposits.

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GEOLOGY AND ORE DEPOSITS OF THE TAYLOR MINING DISTRICT, WHITE PINE COUNTY, NEVADA

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ABSTRACT

The Taylor Mining District is located on the west flank of the Schell Creek Range, 15 miles south of Ely, Nevada. Rocks in the range consist of easterly-dipping Paleozoic limestone, dolomite, and shale, which have been intruded by dikes and sills of mid-Tertiary rhyolite. Silver bearing formations in the district consist of the Devonian Guilmette Limestone and its transition with the overlying Pilot Shale; and the Mississippian Joana Limestone. Known commercial deposits in the district are restricted to the upper 400 feet of the Guilmette.

The most prominent structural feature in the district is a north-northwest trending asymmmetrical anticline, having a vertical west limb and a gently dipping east limb. Two prominent fracture systems are associated with this anticline, one striking north-northwesterly and the other northeasterly. Some of the north-northwesterly fractures have a fault offset, down on the west; original movement was pre-ore, but re-activation occurred during Basin and Range tectonics.

The ore consists of argentite, native silver, and perhaps cerargyrite in jasperized Guilmette Limestone. The silica and silver appear to be contemporaneous, and are slightly older than the mid-Tertiary intrusive rhyolite. The solutions entered along the fracture systems and deposited the minerals in crackle breccia on and near the axis of the anticline, at or near the Guilmette-Pilot contact. A period of calcite deposition followed the ore-forming period. The source of the silver may have been a deep-seated intrusive body, though evidence for such a body is lacking; other possible sources were the Chainman Shale and the mid-Tertiary intermediate flow rocks. Some supergene redistribution has occurred, giving the deposit its present uniform, blanket-like form. The deposit occupies about 40 acres, averages 50 feet thick, and lies near the surface (average overburden is 30 feet, and the deposit crops out in many places). The deposit contains seven million tons averaging three ounces silver per ton, and is haloed by a like tonnage of lower grade material.

Ore was discovered in the district in 1868, and about 60,000 tons averaging 20 ounces silver per ton were mined before 1885. An additional 100,000 tons averaging 10 ounces silver per ton were produced from 1920 through 1960. Silver King Mines explored and developed the district in the 1960's and 1970's. A 1200 tpd, CCD cyanide plant became operational on April 1, 1981.

LOCATION

The Taylor Silver District is located near the eastern border of Nevada, in White Pine County (Fig. 1). The district is 14 miles south of Ely, and four miles east of Highway 6, 50, 93.

Silver King Mines, Inc., and Agnew Enterprises are joint-venture partners in the property, with Silver King as operator. The companies own four patented mining claims in the district, eight claims for which patent proceedings are underway, and 108 unpatented claims, all in the Humboldt National Forest.

GEOLOGY

GENERAL

The district lies on the western slope of the Schell Creek Range, at elevation 7500 feet. Rocks in the range consist of Paleozoic sediments, mainly limestone, dolomite, and shale. The sedimentary rocks were intruded by a few small bodies of mid-Tertiary rhyolite. Remnants of mid- to late-Tertiary intermediate flows and pyroclastic rocks occupy the western foothills of the range.

STRATIGRAPHY

The oldest formation exposed in the central Taylor district is the Guilmette Limestone, Devonian in age (Fig. 2). The Guilmette is about 2,000 feet thick. It consists of massive, fine-grained limestone in the upper few hundred feet, grading downward into sandy limestone and sandy dolomite. The lower contact is gradational into the Devonian Simonson Dolomite. The Guilmette is resistant to erosion, especially in its upper part, and forms massive cliffs.

The Guilmette is overlain by a 100 foot thick section of thin-bedded limestone and siliceous shale. This section represents a transition between the Guilmette and the overlying Pilot Shale.

The Pilot Shale is Devonian and Mississippian in age, and consists of 300 feet of banded, siliceous shale. The Pilot forms distinctive tan-weathering talus slopes.

The Joana Limestone disconformably overlies the Pilot Shale. The Joana is about 300 feet thick. It is a typical Mississippian limestone, consisting primarily of thick-bedded to massive coarse-grained limestone which is composed mainly of crinoid fragments and other fossil remnants. The upper one-third and lower one-third of the Joana forms massive, monument-

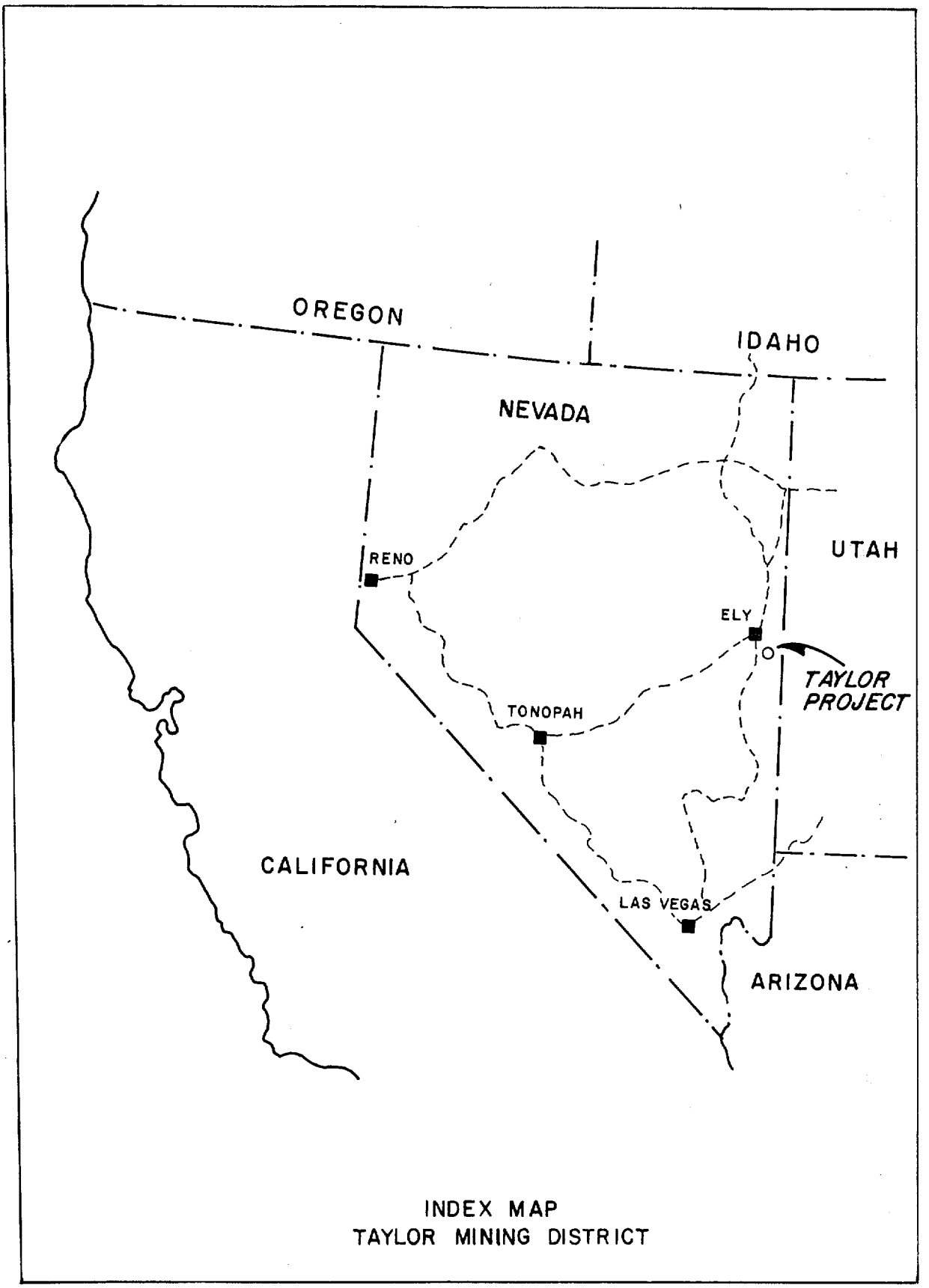
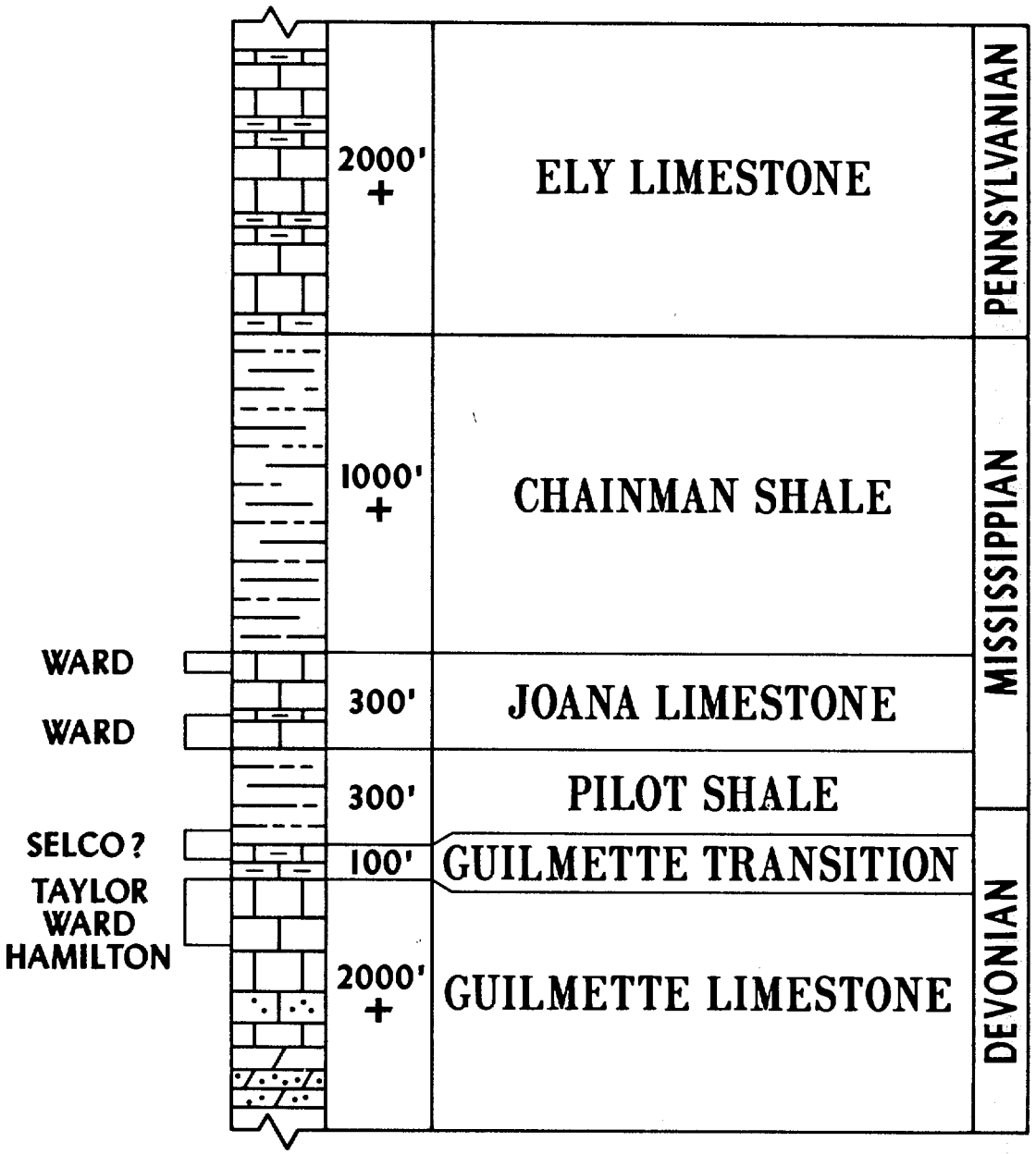

OREGON
IDAHO
NEVADA
UTAH
RENO
ELY
TAYLOR PROJECT
TONOPAH
CALIFORNIA
LAS VEGAS
ARIZONA
INDEX MAP
TAYLOR MINING DISTRICT

FIGURE 1.



TAYLOR MINING DISTRICT
STRATIGRAPHIC SECTION

FIGURE 2.

like cliffs in eastern Nevada. The central one-third is thinner-bedded, less resistant, and forms talus slopes.

The Mississippian Chainman Shale disconformably overlies the Joana Limestone. The Chainman consists of incompetent, black, euxinic shale; the formation typically forms strike valleys which exhibit chaotic, landslide topography where younger rocks have slid over the Chainman. Numerous sandstone and quartzite beds, mainly in the upper part of the formation, form small ridges.

The depositional thickness of the Chainman is difficult to determine because of poor exposures and because of the tendency of the Chainman to fold or flow under stress, thus causing missing or repeating sections. Sadlick (1958) measured 23 sections in Eastern Nevada and Western Utah, and estimates the Chainman was about 1,000 feet thick. In the Taylor district, the maximum observed thickness is about 1,000 feet. Ten miles to the west, in the Ward Mining District, the average thickness in 80 core holes is 600 feet.

The Pennsylvanian Ely Limestone overlies the Chainman Shale. The Ely consists of more than 2,000 feet of cyclicly deposited thin-bedded limestone, shaly limestone, and calcareous shale. The limestone beds are predominantly fine-grained, though a few beds are composed of crinoid and brachiopod fragments.

It is interesting to note the number of places in which this section of the stratigraphic column (uppermost Guilmette to basal Ely) is the locus for mineral deposits in Eastern Nevada.

The top of the Guilmette contains silver ore at Taylor, Hamilton (Treasure Hill bonanza mined in the 1870's), and several other smaller occurrences; these beds are also the host for the very extensive copper-zinc-silver deposits in the Ward District (Heidrick, 1963).

The transition beds above the Guilmette are the host for the gold deposits at Alligator Ridge; these beds and the overlying Pilot Shale are under intensive exploration in eastern Nevada at present.

The Joana Limestone is the host for silver mineralization at Taylor and Hamilton, silver-gold ore in deposits which are satellitic to the Ruth copper porphyry deposits; and extensive zinc-silver-copper deposits at Ward.

The Chainman Shale and Ely Limestone are the host rock for about 20% of the copper ore mined at Ruth (the remainder of the ore is in quartz monzonite porphyry). The Ely is the host for lead-silver-zinc deposits at Ward.

IGNEOUS ROCK

Intrusive activity was confined to the mid-Tertiary (35 M.Y. ±), when a few rhyolite dikes, sills and irregular bodies intruded the sedimentary rocks. The rhyolite bodies contain xenoliths of silver-bearing jasper, but are themselves intensely hydrothermally altered to clay minerals. We infer that the rhyolite was emplaced only slightly later than the silver and silica.

STRUCTURE

The Schell Creek Range is a north-trending, eastward tilted horst typical of the Basin and Range geomorphic province. Rocks in the range have been subjected to three distinct periods of diastrophism:

1) Mid-Mesozoic compressional phase ("Antler" or related movement) which resulted in north trending tight folding and decollement thrust faulting, and extensive deformation of incompetent units such as the Chainman Shale. These stresses in the Taylor District formed a north-trending, asymmetrical anticline (Fig. 3 & 4); the competent Guilmette Limestone fractured to crackle breccia on the crest and flanks of the anticline; and this breccia became the host for the Taylor ore deposits.

2) "Laramide" uplift and intrusive phase, which extended intermittently from 120 to 35 million years ago. Numerous north to north-northwest trending, high-angle, predominantly normal faults with small displacement were first activated in the Taylor District during this phase. A complementary east-northeast set also developed at this time. These faults were the conduits for the hydrothermal silver-bearing fluids which deposited the silver ore in the Guilmette crackle breccia.

   Notably few igneous bodies were intruded into rocks of the Schell Creek Range during this phase. They are limited to a few rhyolite plugs and dikes in the vicinity of the Taylor District. Near the close of this diastrophic phase, the intermediate flows and pyroclastics which now occupy the foothills of the range were extruded.

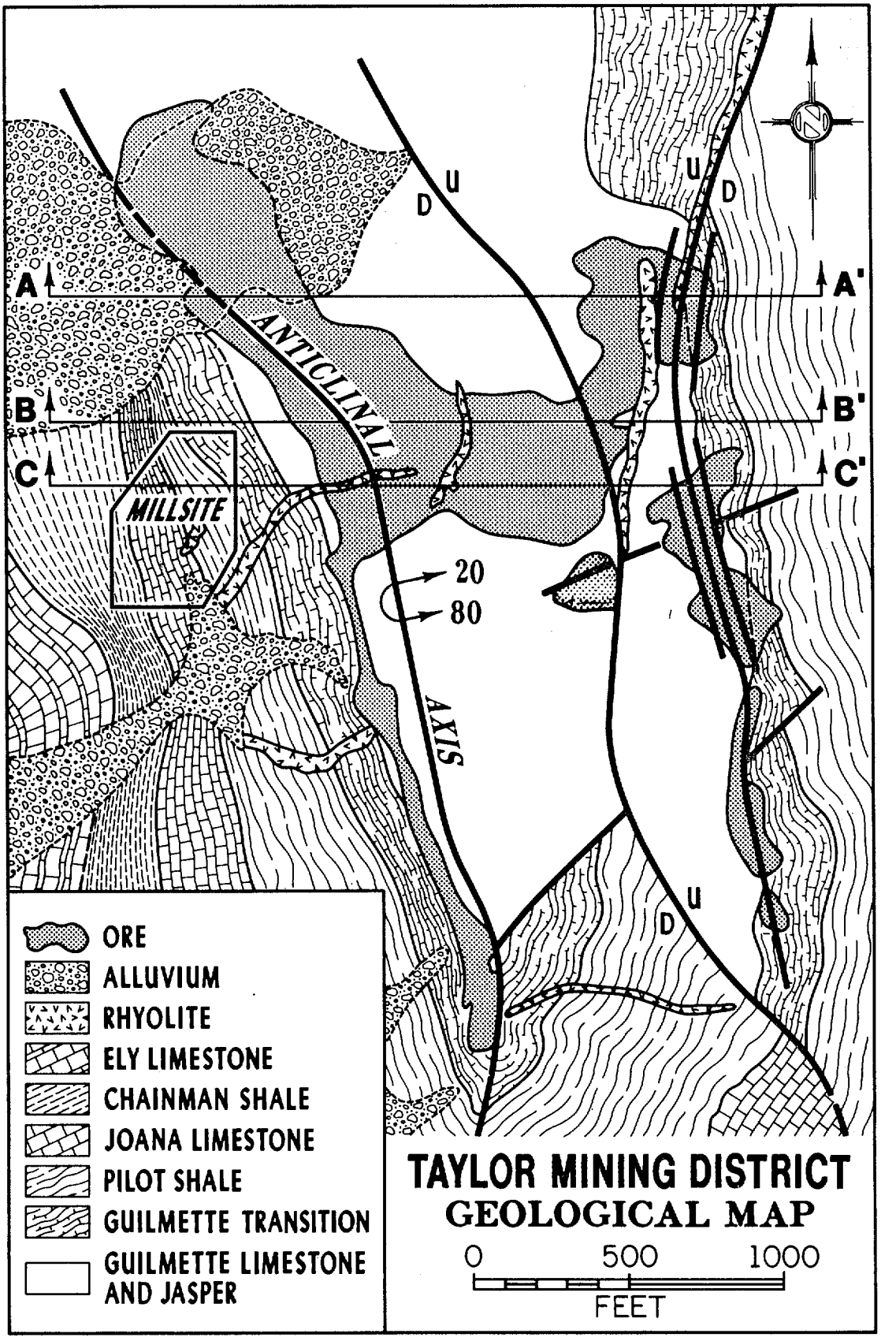
3) Late Tertiary to recent "Basin and Range" phase. This continuing orogeny has formed the present geomorphology of Eastern Nevada. Total structural relief from range to valley is typically several thousand feet. Many of the north-trending faults in the Taylor District were re-activated during this period.

ORE DEPOSITS

MINERALOGY

The ore in the Taylor District consists of finely disseminated crystals of argentite and clots of native silver in a gangue of silicified limestone which Lovering (1974) has described as jasperoid. Much of the jasperoid consists of breccia fragments. Accessory minerals include limonite pseudomorphing pyrite, calcite and quartz as late-stage veins and as the matrix cementing the jasperoid breccia, and rare purple fluorite.

Lovering has identified other sulfide minerals such as stibnite, sphalerite, tetrahedrite, chalcopyrite, galena, and pyrargyrite (?), but these are rare and are of no economic importance.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ANTICLINAL
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ALLUVIUM
RHYOLITE
ELY LIMESTONE
CHAINMAN SHALE
JOANA LIMESTONE
PILOT SHALE
GUILMETTE TRANSITION
GUILMETTE LIMESTONE AND JASPER
TAYLOR MINING DISTRICT
GEOLOGICAL MAP
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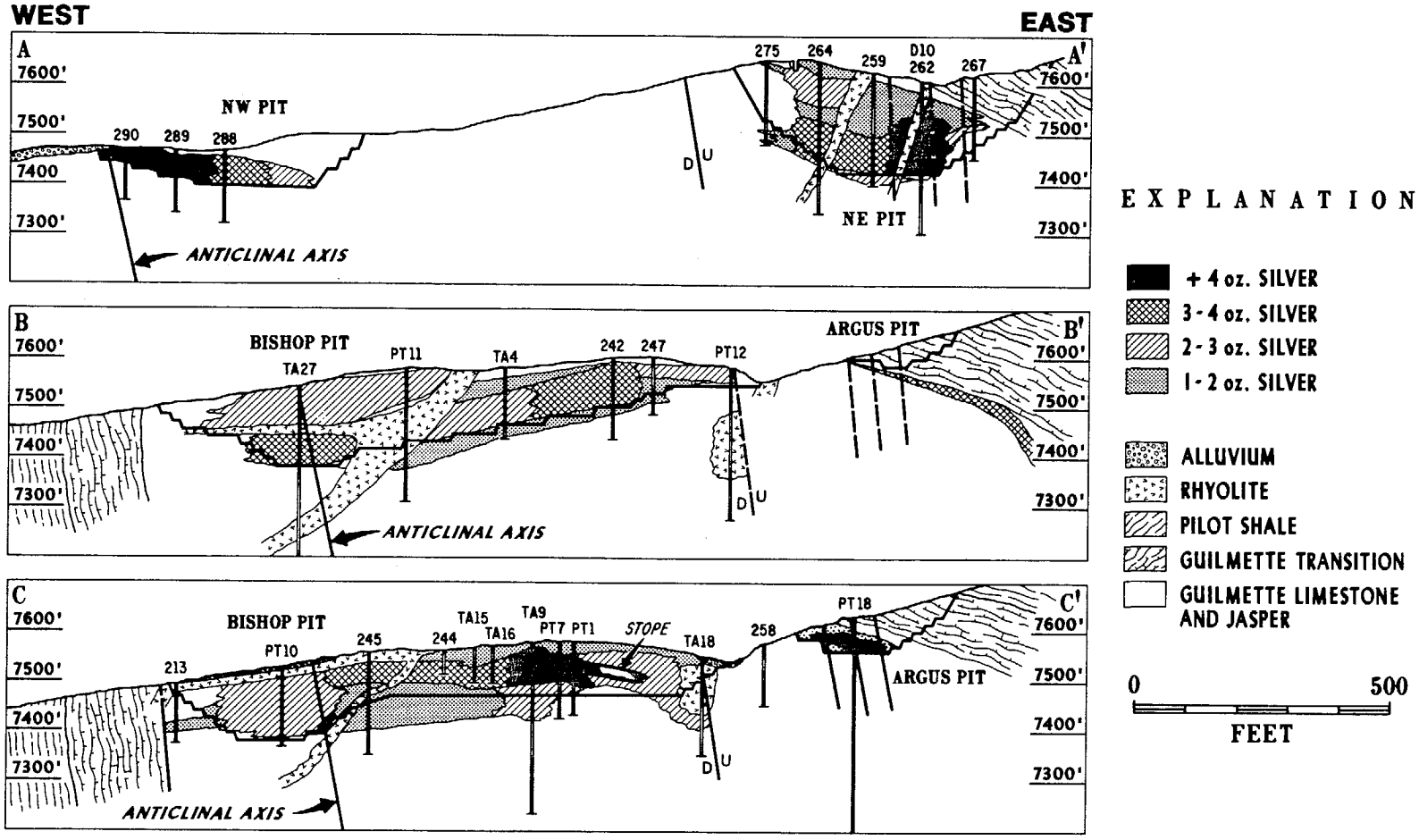
FIGURE 3.



TAYLOR MINING DISTRICT
REPRESENTATIVE CROSS SECTIONS

FIGURE 4.

FORM

The ore deposits consist of large tabular masses of argentian jasperoid which occur at the top of the Guilmette Limestone. The ore occupies the crest and flanks of the north-trending asymmetrical anticline which is the dominant structure in the district (Fig. 4). The ore is flat-lying on the crest of the anticline; dips vertically on the west flank of the anticline; and dips gently to the east on the east flank. The form of the ore bodies has been modified by late movement of small magnitude along the north and east-northeast trending normal faults. The ore body averages about 50 feet in thickness, based on a grade cutoff of two ounces silver per ton; occupies about 40 acres, and contains about seven million tons of ore averaging 3 ounces silver per ton with a like amount of waste which averages about .75 ounces silver per ton.

PARAGENESIS

The ore solutions were introduced along the normal faults, entered the crackle breccia zone at the top of the Guilmette Limestone, were impeded by the capping shale of the Guilmette transition zone and replaced the limestone, depositing the silver as argentite and also depositing pyrite and silica. Drewes (1967), Lovering (1974) and the Silver King staff believe that the silica which formed the jasperoid was deposited contemporaneously with the argentite because of their ubiquitous association.

Rhyolitic dikes and sills were intruded along the same channelways as the ore solutions, but at a later time; fragments of argentian jasper occur commonly within the rhyolite bodies.

This ore-forming period probably occurred during late stages of the mid-Mesozoic orogenic phase and/or during the "Laramide" orogenic phase. Later movement re-brecciated the jasperoid; this breccia was then re-cemented with quartz and calcite. Some further modification of the ore deposits occurred during the recent Basin and Range orogenic activity.

SOURCE OF SILVER

Both Drewes and Lovering relate the argentian hydrothermal solutions to a deep-seated igneous body which they believe was emplaced under the Taylor District during Cretaceous or Tertiary time.

The existence of such a stock is open to question in my mind. The Schell Creek Range is distinctive for its paucity of intrusive bodies or thermal effects related to such bodies, and aeromagnetic surveys flown at various elevations over the District reveal no anomalous magnetism which might reflect a deeply buried intrusion.

Another possible source for the silver (and the silica) is the Chainman Shale. Robert Boyle (1970) and others who have sampled black euxinic shales such as the Chainman report silver contents of one to ten ppm. If the Chainman Shale overlying the Taylor District contained one ppm silver, then each square mile of Chainman would have, upon deposition, contained as much silver as is presently contained in the Taylor deposit.

The Chainman was subjected to intense diastrophic forces during the mid-Mesozoic compressional orogeny. Because it was much less competent than the overlying and underlying strata, it was folded, deformed, and in many areas has been thinned to a fraction of its original thickness. Much heat must have been generated during this period, and the connate water in the Chainman would have dissolved increasing amounts of silver (which is very soluble in water even at standard temperatures—Scherbina, 1970, p. 1136). Much of this water would have been squeezed from the formation during the mid-Mesozoic and the "Laramide" diastrophism, and would have found its way as a hydrothermal solution into the favorable loci of the jasperoid breccia.

Another possible source of the silver was the mid-Tertiary intermediate flows and pyroclastic rocks which covered the District at one time; the flows may, however, be younger than the silver, as are the intrusive rhyolite bodies.

SUPERGENE ENRICHMENT

Several lines of evidence suggest that the present form of the Taylor deposit has been at least partially determined by supergene redistribution and enrichment of the silver values:

1) The deposit is unique in its uniformity; within the ore outline as shown on Figure 5, the occurrence of either high grade zones or barren zones is rare, and silver content feathers out both vertically and horizontally.

2) A composite of the 130 ore holes in the deposit (Fig. 6) shows a distinct, gradual increase in silver values from the surface downward to a maximum at a depth of about 60 feet, followed by a gradual decline to a depth of 180 feet; at this point, the jasperization of the limestone rapidly decreases as does the silver content.

3) The jasperoid near the surface has a leached appearance, being lighter in color and somewhat more "spongy" than the typical jasperoid below a depth of 20 feet.

4) A sequential analysis, as follows:
   A) The rhyolite is younger than the argentian jasper, because fragments of the jasper are common within the rhyolite bodies, having been plucked from the channel walls during emplacement.
   B) Silver values are typically higher just above rhyolite sills. This paragenesis suggests that silver was remobilized after the rhyolite was intruded.

There are several ways by which the primary silver, in the form of argentite and perhaps also as the native metal, could have been dissolved in the

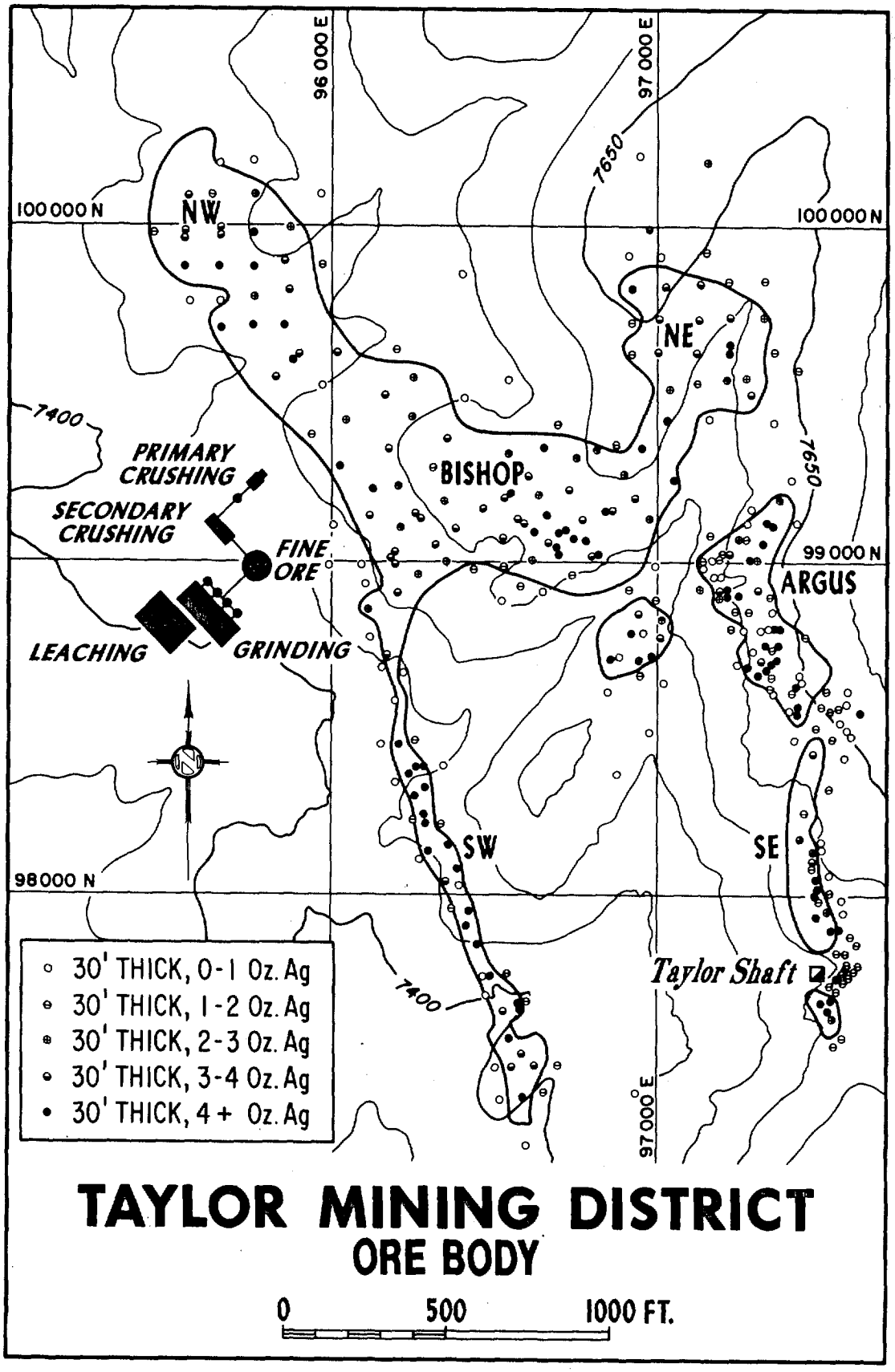
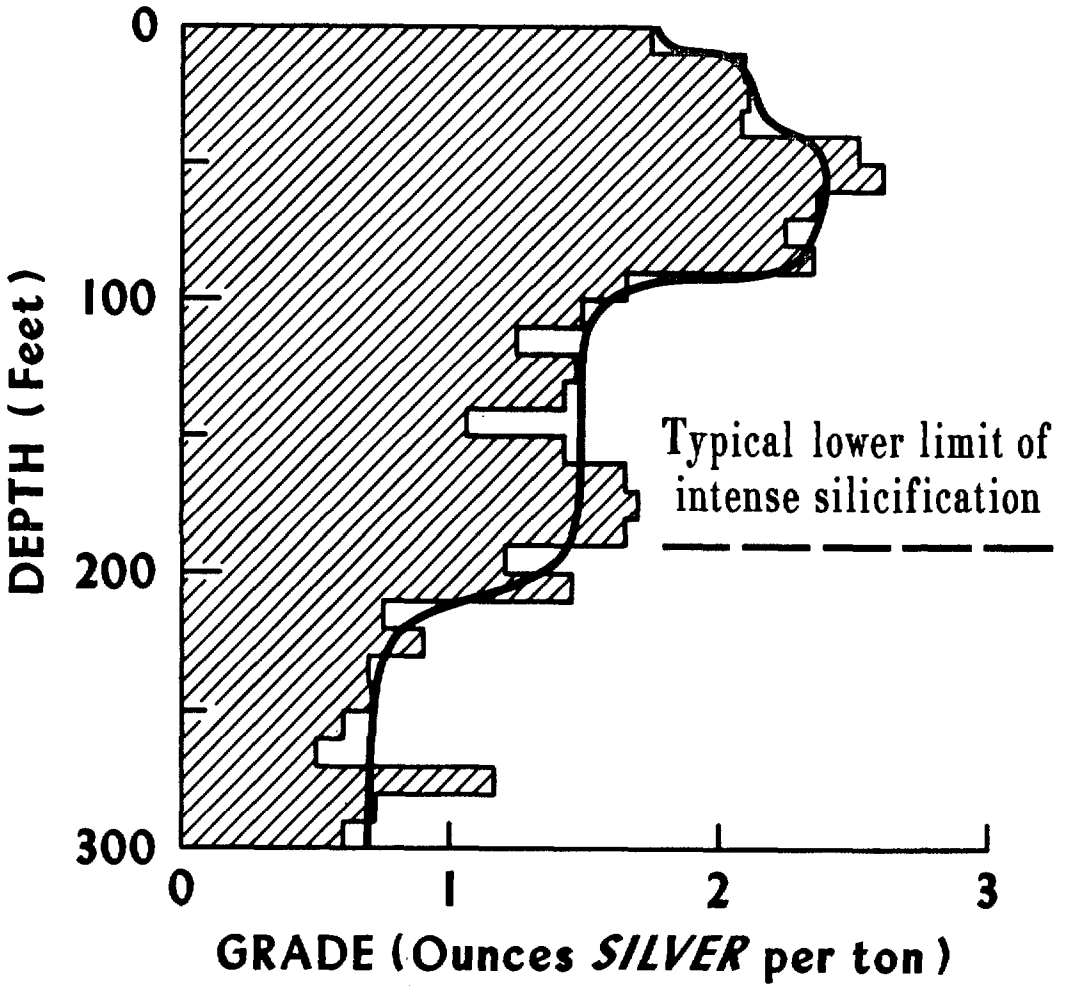

96 000 E
97 000 E
7650
100 000 N
100 000 N
NW
NE
7400
PRIMARY CRUSHING
SECONDARY CRUSHING
FINE ORE
BISHOP
7650
99 000 N
ARGUS
LEACHING
GRINDING
SW
SE
98000 N
Taylor Shaft
7400
97 000 E
30' THICK, 0-1 Oz. Ag
30' THICK, 1-2 Oz. Ag
30' THICK, 2-3 Oz. Ag
30' THICK, 3-4 Oz. Ag
30' THICK, 4+ Oz. Ag
TAYLOR MINING DISTRICT
ORE BODY
0
500
1000 FT.

FIGURE 5.



TAYLOR DISTRICT
HISTOGRAM

AVERAGE GRADE vs DEPTH
of 130 drill holes within the *TAYLOR OREBODY*

FIGURE 6.

zone of oxidation, and re-precipitated at the redox interface. Boyle and Scherbina, as well as many others, have studied and reported on this process. Their findings are briefly summarized as follows:

1) Dissolution of silver minerals in the zone of oxidation
   a) Oxygenated water will dissolve argentite, forming soluble silver sulfate.
   b) If pyrite is present, the process is enhanced; the pyrite is oxidized first, forming sulfuric acid as well as ferrous sulfate which further oxidizes to form ferric sulfate. The sulfuric acid and ferric sulfate then attack the argentite and/or native silver, accelerating its decomposition to silver sulfate.
   c) In a carbonate environment, sulfuric acid may react with the carbonate to form a soluble silver bicarbonate complexion.
2) Transportation of dissolved silver downward in the surface waters probably as silver sulfate at Taylor.
3) Precipitation of silver as native metal or as secondary argentite, caused by the following agents:
   a) A reduction of the acidity of the solution, because of contact with carbonate wall rocks, will cause the silver to precipitate as an unstable oxide which rapidly breaks down to form native silver.
   b) A reduction in the oxidation potential of the solution, because of proximity to the water table, will reduce ferric ion to ferrous ion. The ferrous ion will precipitate native silver, re-oxidizing ferrous ion to ferric ion.
   c) Sulfides in the zone of reduction (e.g. pyrite) will cause silver in solution to precipitate, probably as secondary argentite, according to Schurmann's reaction series.
   d) The reducing action of hydrogen sulfide, generated by decaying organic matter, will cause silver in solution to precipitate, probably as argentite.

HISTORY

The Taylor District was discovered in 1868; during the following 20 years, about 60,000 tons of ore averaging 20 ounces silver per ton was mined, primarily from the Argus shear zone. This is one of the north-trending normal fault zones which acted as feeders for the mineralizing solutions. Formations are displaced downward on the east, bringing unmineralized Pilot Shale in contact with the ore-bearing Guilmette; the fault zone marks the eastern limit of surface silver mineralization. In this area a six-foot thick bed at the top of the Guilmette, or perhaps a solution breccia zone parallel to bedding, was stoped preferentially. Ore was also stoped from fault zones and from irregular bodies.

The ore from this mining phase was taken by wagon to Steptoe Valley where two smelters treated it. Because of the long haul and the primitive ore processing facilities, the ore cutoff grade was high, about 10 ounces per ton. The District became idle about 1892.

Around 1920, a cyanide treatment plant was built at Taylor, but there is no recorded production from it.

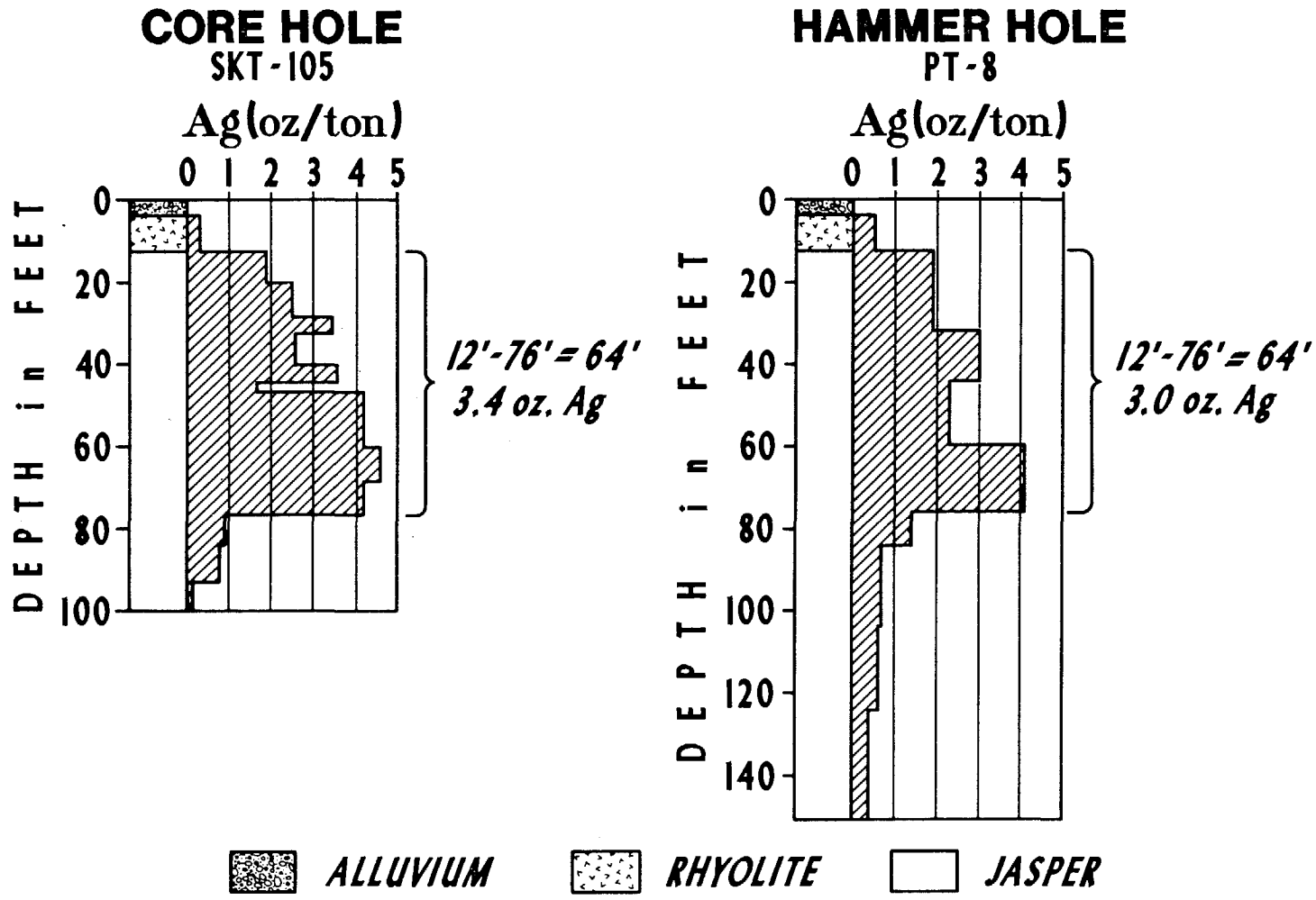
During the 1930's, the District was revived; additional ore was mined underground in the Argus area and the Monitor area to the west. Also, some ore was mined by open cut methods in the Argus and Monitor areas and on Bishop Hill, and most of the old dumps from the 19th century mining, which averaged about eight ounces silver per ton, were also processed. Much of the ore during this period was sold as siliceous flux to the Kennecott smelter at McGill. Total production during this period was about 100,000 tons which averaged 10 ounces silver per ton.

About 1960, K. L. Stoker acquired the existing claims in the Taylor District from several individuals and companies, and when Silver King Mines, Inc., was formed in 1961, the claims became Silver King property. Additional claims were staked to form the present property holding of 120 mining claims totaling about three square miles. Four of the claims are patented, and patent has been applied for on eight others.

Silver King in 1962 began a program of deep percussion drilling along the Argus shear zone in an attempt to locate deep ore below the old mine workings. A small pod of high grade ore was discovered, and in 1964 the Taylor shaft was sunk to a depth of 400 feet (Fig. 5). The pod produced 4,000 tons in 1965 averaging 30 ounces silver per ton. Further exploration drifting and underground drilling failed to discover significant additional reserves and the mine was closed.

Many of the percussion holes drilled in the early 1960's penetrated substantial thicknesses of low grade silver mineralization at or near the surface. The mineralization was too low grade to constitute ore at prevailing silver prices; however, when silver prices began to escalate in 1973, Silver King resumed its exploration and development drilling program. Several core holes were drilled 10 to 20 feet from existing hammer-drill holes to check the validity of the chip samples. The core data confirmed the chip data (core samples assayed, on average, 5% higher than comparable chip samples, Fig. 7). This check drilling also demonstrated the unique uniformity of Taylor ore deposit; mineralization within the exterior ore boundaries is ubiquitous, and there exist few barren or high grade areas (Fig. 5).

The percussion drilling program resumed in 1974, and has continued intermittently to the present. In all, about 450 holes have been drilled in the District since 1962, of which 22 are core holes. The drilling has outlined an area of about 40 acres underlain by ore grade mineralization, which is defined as at least a 30 foot thickness containing at least two ounces silver per ton. Using this cutoff, the ore averages 50 feet in thickness (up to 200 feet thick in a few



SAMPLE COMPARISON-CORE vs. HAMMERDRILL

FIGURE 7.

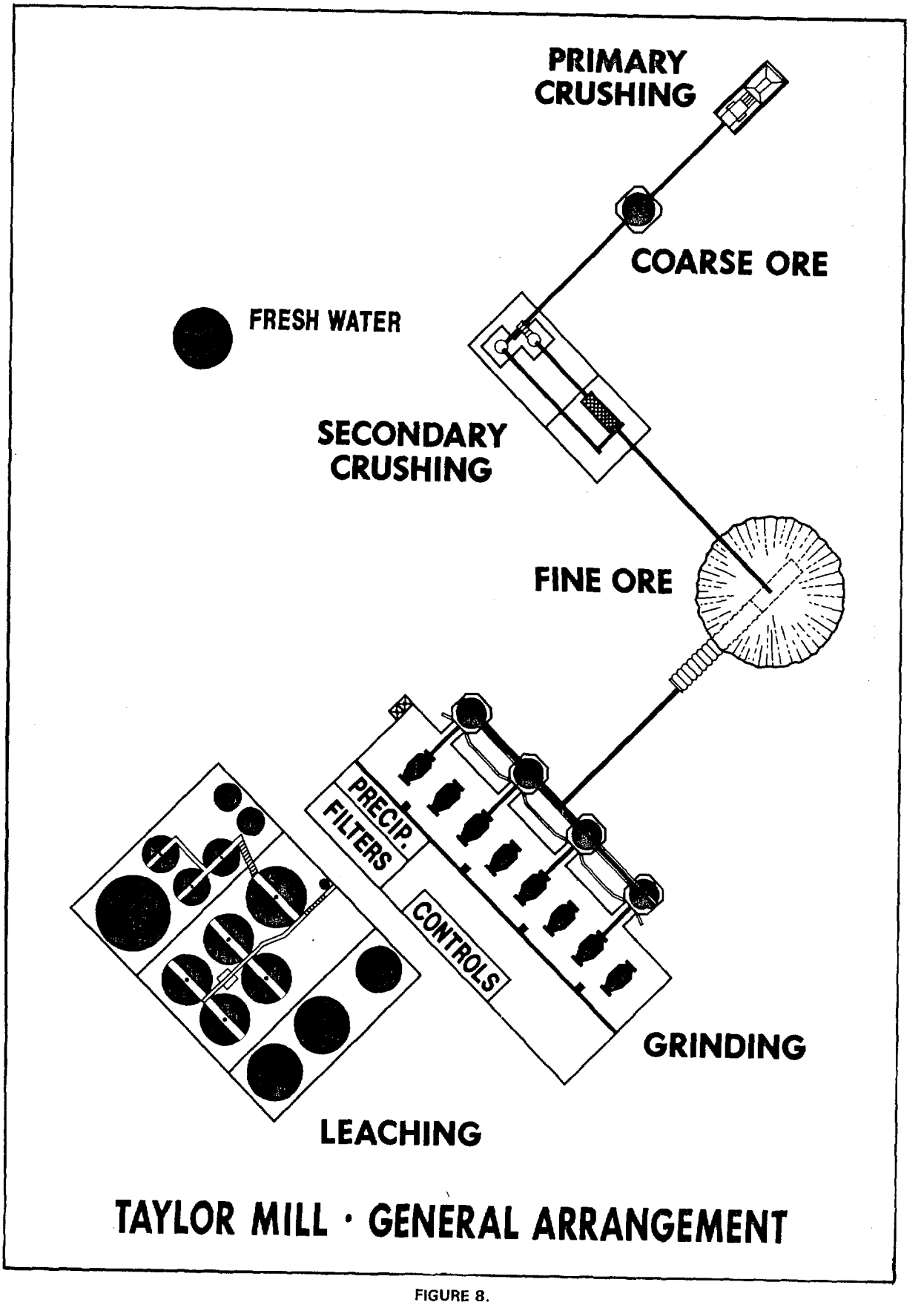

PRIMARY CRUSHING
COARSE ORE
FRESH WATER
SECONDARY CRUSHING
FINE ORE
PRECIP.
FILTERS
CONTROLS
GRINDING
LEACHING
TAYLOR MILL · GENERAL ARRANGEMENT

FIGURE 8.

places), with average overburden of 50 feet (waste to ore ratio is 1.7:1).

Measured diluted reserves using the two ounce cutoff are presently five million tons averaging 3.3 ounces silver per ton. Much of the overburden is low grade mineralized rock; if all overburden which exceeds one ounce silver per ton is "stripped to the mill", diluted reserves become seven million tons averaging 3.0 ounces silver per ton. Inferred reserves based on reasonable geologic projection will add at least three million tons of ore to the reserve.

Financing for a concentrator was secured in 1979. Construction of a 1200 ton-per-day countercurrent decantation cyanide leach plant began immediately, and was completed in early 1981 (Fig. 8). Total expenditure in the District, including drilling, mine development, and mill construction, was slightly less than $10 million.

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BULK-TONNAGE, LOW-GRADE SILVER DEPOSITS—UPDATE 1980

BARRY N. WATSON Southwest Regional Manager, U.S. Borax

INTRODUCTION

In December 1976, the writer presented a paper at the Northwest Mining Association convention in Spokane, Washington which was a first attempt at describing the large tonnage silver deposits as a deposit type. Entitled "Bulk-Tonnage, Low-Grade Silver Deposits in Western North America," the talk was based principally on personal observations and private file data, as publications on disseminated silver deposits were virtually non-existent. The talk was later published in *World Mining* (March 1977, p. 44-49).

Since then, Delamar (Idaho) has become a viable producer, Candelaria (Nevada) has come on-stream, and important exploration and development work towards production is proceeding at Creede (Colorado), Real de Angeles (Zacatecas, Mexico), and in the Silver District of Arizona, among others. I am indebted to those geologists (and their companies) who have shown me some of the properties over which I have been speculating, and, in this *Update 1980,* I will attempt to refine and to expand upon some of the ideas previously presented.

DEFINITION OF "BULK-TONNAGE SILVER DEPOSIT"

A problem 4 years ago that is even more of a problem today is that of defining what should be considered a "bulk-tonnage, low-grade silver deposit." My previous criteria were that 1.) the tonnage must exceed 5 million short tons and the deposit must, thusly, be amenable to bulk mining methods, and that 2.) silver must be the most valuable extractable metal or mineral. In retrospect, there is nothing magical about the 5 million ton cut-off, and, with higher silver prices, open pit and underground LHD operations are being considered on smaller tonnages. Also, as everyone is aware, the greatly increased silver price relative to the prices of such metals as copper, lead, and zinc has made silver deposits out of occurrences previously known for their other metals.

For example, let's consider the Spar Lake, or Troy, stratabound orebody in the Belt Supergroup of Montana. Tonnage is given as 64 million tons and grades are 0.74% Cu and 1.54 oz. Ag (*Mining Record,* Apr. 18, 1979, p.5). Ten years ago this month, Troy was a copper deposit containing a total metal value per ton of $8.73 in copper and $2.69 in silver (metal prices from *E & MJ,* Nov. 1970). Today, with $1/lb. Cu and $19/oz. Ag, Troy is a silver orebody containing a total metal value per ton approaching $30 in silver and only $15 in copper.

Again, let's look at one of the several volcanogenic massive sulfide deposits I could have used to make this point. The Iron King mine near Prescott, Arizona produced 5 million tons with grades of 2.50% Pb, 7.34% Zn, 3.69 oz. Ag and very minor copper based on metal recovered (Hutchinson, 1973, Econ. Geol. v. 68, p. 1227). Ten years ago, an intact Iron King orebody would have been considered a zinc mine with zinc accounting for 58% of the dollar value per ton, lead 19%, and silver 17%. Today, such a deposit would be a silver lode with silver accounting for 46% of the dollar value per ton, zinc 37%, and lead 15%.

In 1976, I made the mistake of including a particularly silver-rich volcanogenic massive sulfide—the Sam Goosly deposit—among the bulk-tonnage silvers discussed. (At least, my acquaintances within the British Columbia Department of Mines assure me it is of such derivation.) As I consider the "bulk-tonnage, low-grade silver deposits" to be a *specific* deposit type in the same sense as are the "porphyry coppers," I now have no wish to include "stratabound, syngenetic massive sulfides" or any other occurrences obviously belonging to other deposit types in this category. At the moment, and although I have not personally seen the Hog Heaven or Hercules projects, I would consider all those deposits shown on Figure 1 to be bulk-tonnage, low-grade silvers related to one or the other of two types that will now be discussed.

DISSEMINATED-TYPE BULK-TONNAGE SILVER DEPOSITS

Most explorationists have an inclination to build conceptual models of ore deposits—the Climax moly model, the Carlin gold model, the Lowell-Guilbert porphyry copper model, etc. Experience shows us that these models aid our exploration efforts as long as we don't cling *too tightly* to any one of them. Herein, I am going to propose a somewhat simplified two-fold categorization of bulk-tonnage, low-grade silvers—1.) the disseminated-type and 2.) the stockwork-type—and show how these types can, at times, be spatially and genetically related. The disseminated-type generally embodies what I called the "Waterloo-type" in my original paper and will be reviewed first.

The truly "disseminated" bulk-tonnage silver deposit so far seems restricted to a volcanic or subvolcanic environment of Cenozoic or Mesozoic age. It is hosted principally by volcanoclastic rocks and by the fluviatile and lacustrine sediments so often related to volcanic piles. The mineralization can occur as a variety of silver-bearing primary or secondary minerals, usually associated with silica and often with manganese. Primary mineralization is *finely* disseminated, and the ores are usually refractory.

Known deposits are in the 5-50 mt range, grading 2-7 oz. Ag and with possible by-product barite, gold,

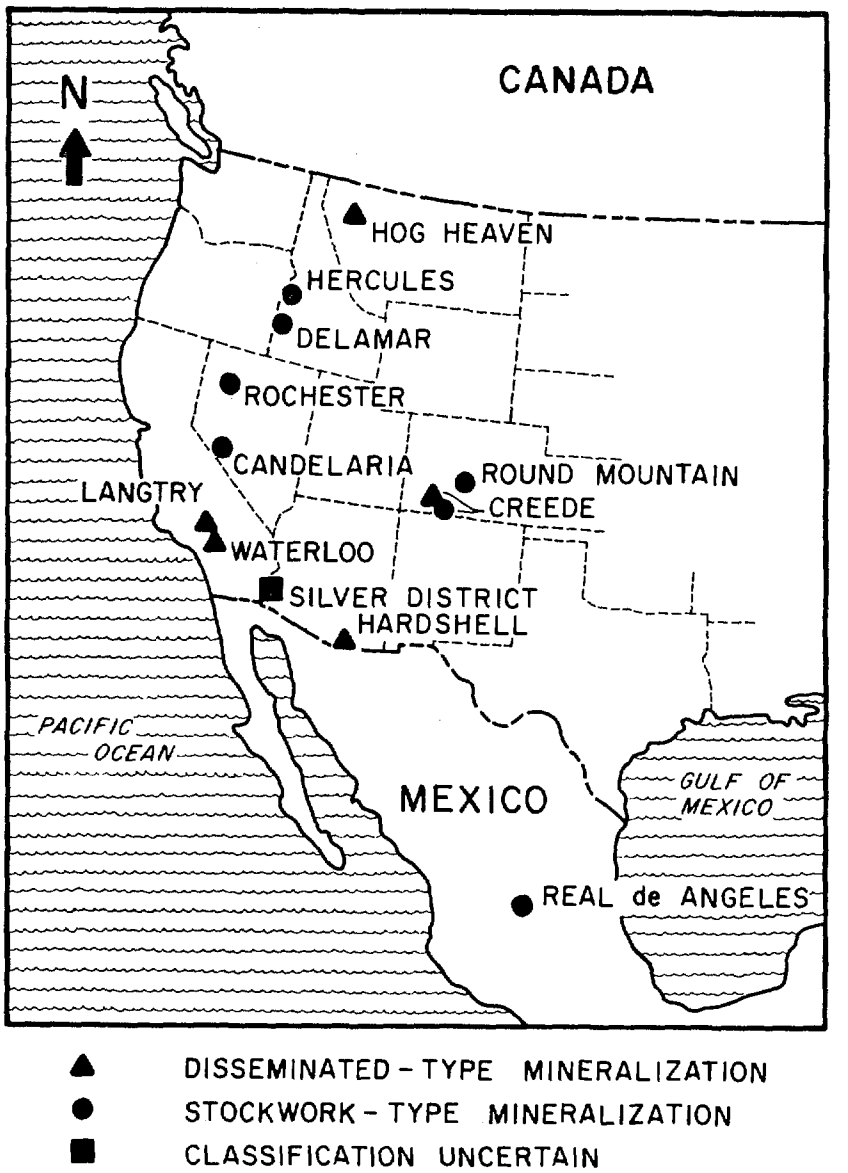


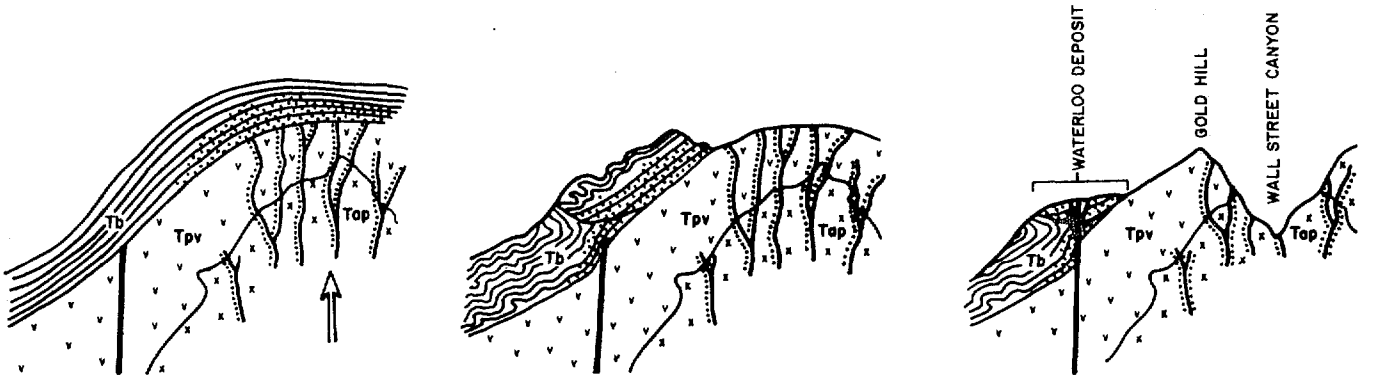
FIGURE 1. Location of some bulk-tonnage silver deposits.

lead, or zinc. Pyrite is ubiquitous but to varying degrees of intensity. Mineralization can *loosely* be described as stratiform, and deposits are thusly "manto-like," or "tabular" or "lensoid." Permeability of the host rock is critical. Mineralization seems to be of a sub-aerial hot-spring sort, and I consider *none* of the deposits that I have seen to be truly syngenetic in origin.

Deposits in this category include the Waterloo and Langtry on the southwestern flank of the Calico Mountains, California, the Hardshell in Arizona's Patagonia Mountains, the Creede Formation on Batchelor Mountain at Creede, Colorado, and probably the Hog Heaven near Kalispell, Montana.

The Waterloo and Langtry are erosional remnants of the same slide block that has moved a mile or so off the top of the Wall Street Canyon vein zone. Combined tonnages here are on the order of 50 million with grades in the 2-4 ounce range. The silver-silica-barite mineralization occurs in the somewhat limy base of the 2000-foot thick middle Miocene Barstow lakebed sequence. The "root zone" veins were tightly confined in andesitic volcanic and hypabbysal rocks. ASARCO may not be too far away from a production decision on the Waterloo, and, if it goes, the significant barite byproduct will be an important factor; particularly, considering that the silver recovery through cyanidation will probably be somewhat less than 65% (Val Kudryk, Feb. 16,1970, talk at AIME Ann. Mtg., Denver, Colo.).

The Hardshell deposit lies in the basal portion of a Mesozoic volcaniclastic sequence, and root zone veins are seen in underlying Paleozoic limestones. The mineralization seems to be spatially and genetically related to the Laramide Thunder Mountain porphyry center. The Hardshell is smaller but higher grade than the Waterloo, carries a higher stripping ratio, and suffers metallurgically from both silica encapsulation and manganese association. Fluid inclusion data indicates a depth of burial of about one kilometer at the time of mineralization (Koutz, 1978, *Econ. Geol.* v. 73, p. 311-312).



1. Andesite porphyry intrudes Pickhandle volcanics and overlying Barstow lakebeds, doming up these rocks along a NW-trending axis. Subsequent silven mineralization cuts competent igneous rocks as veins but disseminates under lower temperature, pressure, as moist and permeable lakebeds are encountered.

2. Barstow lakebeds on the anticlinal rise—now laden with silica, silver, barite—slide down the southwest flank of the anticline, triggered by the range-front (Calico) fault and under the influence of gravity. Deformation of unmineralized lakebeds is by folding, of mineralized lakebeds by shattering and brecciation.

3. Grossly simplified cross-section of present topography. More recent movement on the Calico Fault (including late strike-slip) has reflected upward as several strands through the mineralized breccia. The effects of probable southwestward tilting of the Calico Mountains block are not shown.

FIGURE 2. Suggested origin and structural emplacement of the Waterloo, California bulk-silver deposit. (From Watson, 1977.)

The MECO-Homestake deposit on Batchelor Mountain is hosted by fluvial sediments which fill a graben-like breach in the Creede caldera rim and which are laterally related to the late Oligocene Creede Formation moat fill. At least 10 million tons of 4-ounce silver is distributed somewhat irregularly through the center and lower portions of the graben fill. Silica encapsulation is a problem, as would be stripping ratio and grade control for an open pit operation. A much smaller but considerably higher grade tonnage might be mined by underground methods from the carbonaceous trashy base of the Monkemeyer sandstone unit (J. Beeder, personal communication). The root zone mineralization in volcanic rocks could well be more important than that in the Creede Formation and will be discussed below.

Before leaving the disseminated-type of silver mineralization, I would like to briefly point out at least two other metals that can occur in this sort of geologic environment. At McDermitt, Nevada, Placer Amex is mining mercury values contained in moat sediments of the McDermitt caldera. The mercury-bearing veins in rhyolitic rim rocks once mined at Cordero are just a stone's throw away. "Disseminated mercury" in permeable lakebed sediments essentially underlain by cinnabar-bearing veins in caldera volcanics evokes the similar scene at Waterloo or at Creede.

And, as might be expected, gold is a candidate for the disseminated-type environment. The Cinola deposit in the Queen Charlotte Islands, British Columbia contains an estimated 35.7 million tons averaging 0.055 oz. Au and 0.1 oz. Ag (Champigny and Sinclair, Aug. 1980, CIM Bull., p. 62; *E & MJ*, Oct. 1980, p. 31). Micron-size gold has been precipitated through the highly porous and permeable Miocene Skonun Formation, an alluvial plain facies in a braided river system. The mineralization appears to be related in some way to a late Miocene rhyolite stock which intrudes the sediments.

STOCKWORK-TYPE BULK-TONNAGE SILVER DEPOSITS

Stockwork-type bulk-tonnage silver mineralization can occur in volcanic rocks as at Delamar (Idaho), Rochester (Nevada), and Creede or in sediments brittle enough to shatter as at Candelaria and Real de Angeles. Like the disseminated-type silvers, they can be quite large, depending on cut-off grade. Silica encapsulation is not nearly as pronounced as in the disseminated types, but grade control, necessitated by erratic mineral distribution, can be critical.

Earth Resources and the Superior Oil companies christened the $22.4 million Delamar project just in time to catch escalating precious metal prices. Ore reserves at one time were up to 10 million tons grading a little less than the originally announced 4.2 oz. Ag and 0.046 oz. Au (Wright, Dec. 1976, talk to NMA Conv., Spokane, Wash.; *Min. Eng.*, Jan. 1978, p. 42). Reserves have since been expanded. Economic mineralization exists in Miocene porphyritic rhyolite tuff units and rhyolite breccia beneath thick clays of pre-mineral, low-angle fault zones. The most competent rhyolites shattered best for mineralization, and it is vein density along with local stockworks that make ore grade in at least 3 operating or proposed pits. Grade coincides with degree of silicification, and minor silica encapsulation problem was seen in 1978 mill recoveries which were 85-90% for silver and over 90% for gold (T. Weitz, personal communication).

At Rochester, ASARCO seems to have a very large tiger by the tail. A published "probable" reserve estimate for Nenzel Hill gives 70 million tons running 1.39 oz. Ag and 0.0072 oz. Au with a further 30 million tons categorized as "possible" (*E & MJ*, Jan.

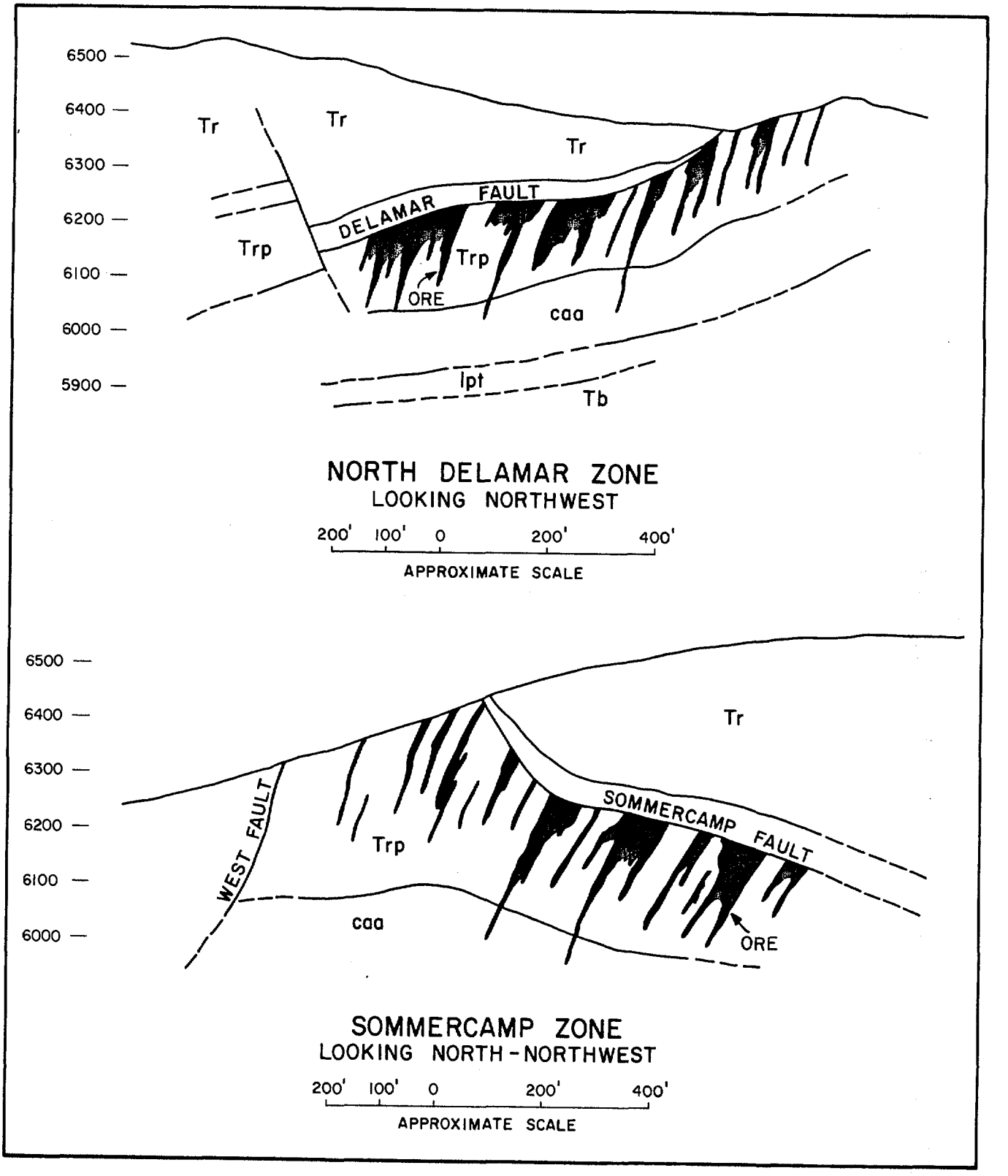


FIGURE 3. Diagrammatical cross-sections showing relationships between structure and ore in rhyolitic volcanic rocks at Delamar, Idaho. (Adapted from Delamar Mine handout sheets.)

1980, p. 170). Permo-Triassic tuffs, flows, and epiclastic rocks host the mineralization which fills an extensive unoriented fracture system between larger veins. Silver sulfosalts and various argentiferous sulfides provide for good metallurgical recoveries, but the finding of a richer nucleus of mineralization for early investment payback purposes has been a real challenge. Work on fluid inclusions, mineral stabilities, and both stable and radiogenic isotopes suggests that mineralization took place in late Cretaceous time and under conditions deeper than epithermal (Vikre, 1978, Stanford U. PhD diss., 404 p.).

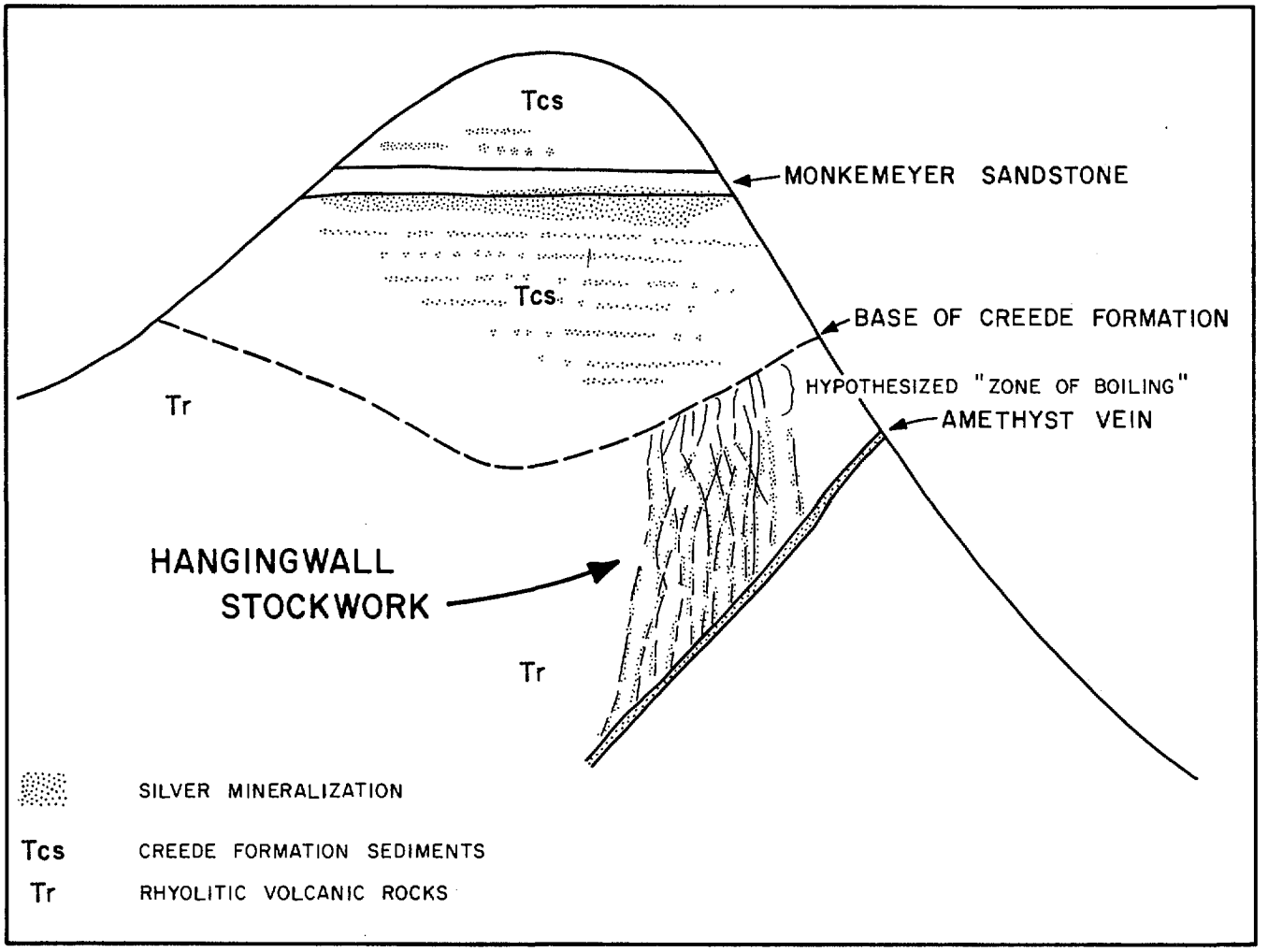


FIGURE 4. Diagrammatical cross-section through Batchelor Mountain, Creede, Colorado, showing spatial and probable genetic relationship between disseminated-type silver in the Creede Formation sediments underlain by stockwork-type silver in rhyolitic volcanic rocks. (Not to scale.)

MECO's Creede Underground silver mineralization is presently undergoing intensive exploration by Chevron. Stockwork ore with some disseminated values occurs in rhyolitic ash-flow tuff hangingwall of the 50°-65° dipping Amethyst vein and is separated from overlying Creede Formation mineralization only by a hypothesized "zone of boiling" just below the sedimentary-volcanic contact. Here we see clearly a spatial and genetic association between disseminated and stockwork types of bulk-tonnage silver mineralization! Chevron is said to be seeking a sizeable tonnage of 4-7 ounce rock amenable to an open pit operation (*World Mining,* Apr. 1978, p. 11). Tonnages to be developed could be quite large, but only a relatively small portion would seem to be reasonably subject to an open pit. Metallurgical recoveries are said to be good because of the sulfide nature of the mineralogy, and barite might be worthy of by-product consideration both here and in the Creede Formation above.

Oxymin has just put Candelaria into production at a cost of $30 million, although the press reports the operation to be already $119 million ahead on silver futures trading (*Paydirt,* Oct. 1980, p. 58). With mine pre-planning like this, who needs a high-grade sweetener for quick payback! Oxymin is not yet giving out tonnage and grade information, but original reserves several years ago were announced as 12-13 million tons at 3.15 oz. Ag and 0.002 oz. Au (*E & MJ,* Jan. 1980, p. 158). Recovery in the cyanide leach may be only on the order of 50% for the silver. Much good geologic study remains to be done at Candelaria, but the bulk of the mineralization appears to occur in a quartz veinlet and fracture stockwork within silicified and dolomitized shaly sediments of the Triassic Candelaria Formation. Locally, disseminated mineralization is found in the thin underlying Permian Diablo grit. Silica encapsulation, manganese association, and grade control are all problems. This hydrothermal and zoned mineralization seems to be related to early Cretaceous felsic igneous activity.

The Mexican Government, in conjunction with Placer and Frisco, plans to bring the Real de Angeles open pit operation on-stream by January 1982 at a cost of $147 million (*World Mining,* July 1980, p. 11). Reserves announced at 59 million tons grading 2.6 oz. Ag, 1.0% Pb, and 0.92% Zn are principally in the form of sulfides and give good metallurgical recoveries. Most of the mineralization is in a network

of fractures in Upper Cretaceous graywacke, but minor amounts are disseminated in thin siltstone interbeds. Strong veins mined in the past crosscut the strata in the ore deposit. Host rock alteration is minimal except immediately adjacent to veins and mineralized fractures where chlorite and carbonate envelopes are seen. A carbonate halo surrounds the deposit. No intrusive rock is known in the district. district.

At least brief mention should be made of Anglo-Bomarc's Hercules deposit in western Idaho which would tentatively seem to fit the stockwork class of silver mineralization. Host rocks are silicified andesitic breccia and rhyolite (Anglo-Bomarc's Hercules property Progress Report, June 1976), and reserve announcements seem quite varied as to tonnage and grade. Geologists who have visited the property report the mineralization to be rather severely faulted.

Two deposits that I originally considered to be bulk-tonnage, low-grade silvers have yet to be mentioned. The Round Mountain deposit north of Silver Cliff, Colorado contains maybe 10 million tons grading 2-3 oz. Ag. It has been proposed that very low grade primary stockwork mineralization in Oligocene rhyolites has undergone supergene enrichment above an impervious horizon of black volcanic glass (USBM R.I. 7486, 32 p.). Values are tied up in manganese and iron oxides.

In the Silver District north of Yuma, Arizona, Gulf and Western Industries is proceeding toward production on an announced reserve of 7 million tons grading about 4.5 oz. Ag *(E & MJ,* Feb. 1980, p. 11-12). Barite would be an important by-product here, but there are no present plans for recovery of the fluorite, lead oxide, and zinc silicate that occur in the ore (J. Teet, personal communication). The mineralization occurs in veins up to 200 feet in width which cut metamorphic rocks, phaneritic intrusive rocks, and middle Tertiary volcanics. The veins are composed essentially of quartz-carbonate-fluorite-barite, and the fine-grained silver is only about 55% recoverable. Mining would proceed in several elongate pits. The Silver District mineralization is somewhat of an enigma as regards modelling, but its mineralogy, age, and probable volcanic association suggest a relation with disseminated-type bulk-tonnage silver.

MANTO- OR REPLACEMENT-TYPE SILVER DEPOSITS

At least one further category of bulk-tonnage, low-grade silver mineralization might be suggested—a manto- or limestone replacement-type. Candidates would include the Taylor and Ward Districts in eastern Nevada where Silver King owns skarn and jasperoid "replacements" or "mantos" at Paleozoic shale-limestone contacts. The Taylor District contains at least 6 million tons grading 3.2 oz. Ag with a 1:1 stripping ratio *(E & MJ,* Oct. 1979, p. 163). At Ward, joint-venturer Gulf Minerals is evaluating possible underground exploitation of around 17 million tons of silver-bearing ore in 3 different horizons. The suspected causative monzonitic intrusion at Ward cut by deeper drilling is 34 m.y. old. There seems to be close similarities between this silver-bearing environment and the not-too-distant Alligator Ridge gold environment.

Also fitting a "replacement" or "manto" bulk-tonnage silver category might be Goldfields' recent discovery at Shafter, Texas. A rumored 7-10 million tons of silicified and oxidized ore grading 6-7 oz. Ag and with significant gold occur in a favorable Permian limestone unit 10-14 feet thick *(Skillings,* Nov. 24, 1979, p. 22; *World Mining,* Apr. 1979, p. 11). The discovery is essentially the downdip continuation of mineralization at the Presidio mine beyond a dike-occupied, post-mineral fault.

SILVER-GOLD RELATIONSHIPS

In conclusion, it is interesting to note that grade and tonnage figures on several new operations and recent drilling projects suggest that the entire spectrum might well exist between bulk-tonnage silver deposits with little gold and bulk-tonnage gold deposits with litte silver. Gold-silver co-product and silver by-product properties appear to include the Landusky and Zortman in Montana, the Sunbeam and Dewey in Idaho, and the Borealis, Buckhorn, Con-Imperial Comstock, Cornucopia, and Santa Fe in Nevada. Smoky Valley's new discovery at Round Mountain is touted to have half as much recoverable silver as it has gold *(E & MJ,* Oct. 1980, p. 31), and word from Northumberland concerns very significant silver mineralization in what has been considered a bulk-tonnage gold situation. This all, however, is grist for papers in the future.

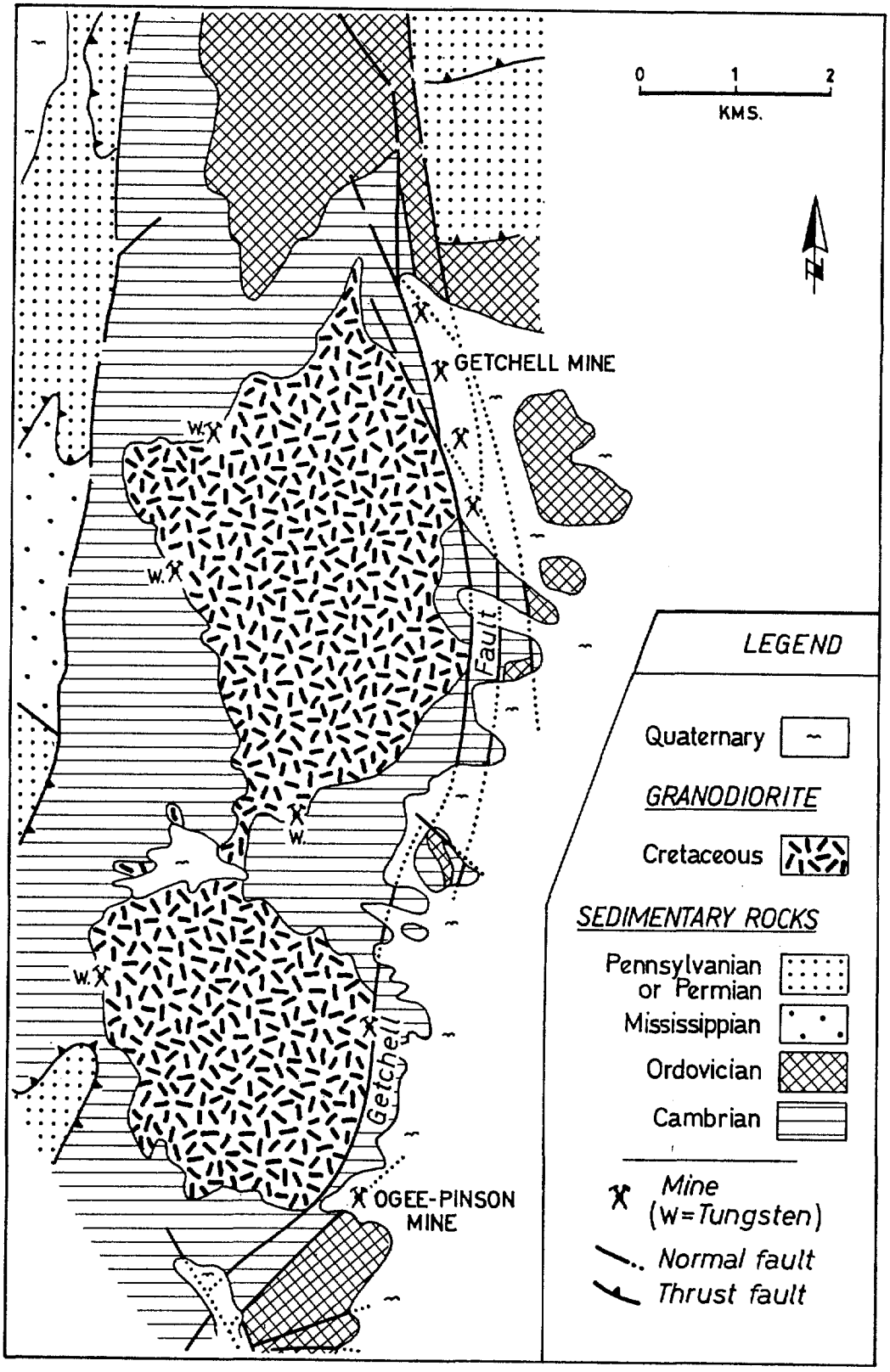


FIGURE 6. Geological setting of the Getchell gold deposit and nearby tungsten mineralization, Humboldt County, Nevada (taken from Silberman et al., 1974 and Berger, 1975).

There are no characteristics, other than the ubiquity of pyrite, which are common to all the low-grade gold categories outlined above. However, pervasive silicification and the presence of organic carbon are widely noted features: Silicification is particularly widespread in the volcanic-hosted epithermal, massive enargite-luzonite-bearing, and Kuroko-type massive sulfide deposits, but also occurs more locally in the porphyry copper-gold, Carlin-type and vein deposits. Silicification is believed to be more abundant in the higher-level volcanic environments because there compressed temperature gradients are more likely to result in dumping of silica. Accompaniment of gold by organic carbon is commonly stressed for Carlin-type deposits, but is also a feature of sedimentary rocks involved in the intrusive-hosted deposit at Porgera, the breccia-hosted deposits at Wau-Edie Creek, the volcanic-hosted deposits at Pueblo Viejo and Cinola, and even the Kuroko-type massive sulfide at Rosebery. As proposed by many workers (e.g., Boyle, 1979), carbonaceous matter may contribute to an effective environment for precipitation of gold complexed in aqueous solutions.

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Thanks are due to the numerous geologists who have accompanied me on visits to most of the above-mentioned low-grade gold deposits and prospects in the western USA, British Columbia, Costa Rica, the Dominican Republic, Colombia, Chile, Argentina, eastern Australia, New Zealand, Papua New Guinea, the Philippines, Sabah, Pakistan and Yugoslavia. Much of my thinking on low-grade gold deposits has developed during engagement by Gold Fields Exploration Pty. Ltd. of Australia; their Exploration Manager, Roger A. Shakesby, sanctioned publication of this general summary. Discussions with H. F. Bonham of the Nevada Bureau of Mines and Geology and J. Paul McKibben of Gold Fields were particularly valuable. H. F. Bonham and Donald C. Noble reviewed the manuscript.

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OPERATION OF AN OPTICAL ORE SORTING SYSTEM

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ABSTRACT

This paper describes the installation and operation of an optical ore sorting system currently in use at Black Pine Mining Company's Black Pine operation, located ten miles from Philipsburg, Montana.

The Black Pine is an underground vein-type operation employing room and pillar mining methods and diesel load-haul-dump equipment. Approximately 500 tons per day of ore grade material, containing values primarily in silver and copper, are mined from underground and processsed in an optical sorting plant before shipment to a nearby smelter. Of the 500 tons per day production approximately 340 tons per day are shipped as ore, with the remaining 160 tons per day being dumped as waste after sorting.

This paper presents a brief history of the discovery, exploration, and development of the Black Pine deposit, and the need for, and implementation of a sorting system for this particular ore deposit.

The intention of this paper is to describe the installation and operation of the sorting plant in use at Black Pine, and it therefore deals only briefly with the theoretical and technical aspects of the sorting machine itself. Detailed technical information on the machine is available from the manufactures representative.

HISTORY OF THE BLACK PINE MINE

The Black Pine Mine was discovered in 1882 and produced 2,135,000 ounces of silver from 1885 to 1897 when the mine was closed because of low silver prices.

There was only small scale activity on the property during the intervening years, until, in 1932 Black Pines Inc. was organized, the mine was rehabilitated and a 200 ton/day mill was erected.

The mine closed again in 1941, and Inspiration Consolidated Copper Company acquired the property in 1970. After completion of an extensive exploration program, Black Pine Mining Company, a wholly owned subsidiary of Inspiration, was formed and the mine was put into production in May of 1974.

GEOLOGY OF THE BLACK PINE DEPOSIT

The area of the Black Pine Mine is located in the Spokane Formation, which is part of the Missoula Group of the Pre-cambrian Belt Series rock. In the Black Pine locality, the Spokane formation consists of a series of thin-bedded red to reddish-brown quartzites and thin-bedded sandstone and shale. It is believed that the quartzite may be the result of metamorphism or hydrothermal alteration of siltstone and sandstone units.

The strike of these sedimentary beds varies from N 15° W to N 45° W and the dip ranges from nearly horizontal to 25° SW. The ore bodies are localized in small folds or wraps in the east limb of the regional Marshall Creek syncline.

The four veins which have been recognized on the property are The Upper, Tim Smith, Combination and Onyx. The Combination Vein, with the best economic potential, is the only vein currently being mined extensively. The Combination Vein has an approximate strike length of 3000 feet and is projected from its outcrop 3500 feet down-dip with an average of 15° SW.

The major minerals that have been identified in the Combination Vein are pyrite, chalcopyrite, galena, sphalerite, tetrahedrite, argentite, pyrargyrite, and hubnerite. These minerals are found in a quartz vein and occur as either layers paralleling the vein structure or as large masses. The large masses of minerals are usually restricted to anticlinal flexures of the vein.

Secondary minerals resulted from oxidation of pyrite and sphalerite into limonite and hydrozincite, also, minor amounts of the bromides and copper oxides also can be observed throughout the mined area.

The vein in some places cuts across bedding planes, but, on the whole, the vein follows opened bedding planes in quartzite and should be classified as a fissure vein.

MINE DEVELOPMENT AND CURRENT OPERATIONS

The Combination Vein is currently being developed and mined using an inclined room-and-pillar mining system utilizing diesel load-haul-dump equipment. A main haulage drift has been driven in ore from a point on the outcrop of the vein, oblique to the dip, on approximately a 10° decline. The ten foot high by fifteen foot wide haulage has been driven to a point approximately 3000 feet from the portal to the bottom of an old shaft on the property at a point 450 feet below surface. Development drifts are driven six feet high by fifteen feet wide on 100 foot centers north and south from the main haulage and the ore is developed in 100 foot square blocks.

Ten foot high by fifteen foot wide development drifts connect the stope blocks on 400 foot centers. The ore is mined in the stope blocks using jack-leg drills and one yard diesel front-end loaders at a six foot mining height. Broken ore is transferred to draw points in the stope development drifts where it is loaded onto ten ton diesel trucks by a three yard front-end loader and hauled to surface through the main haulage drift.

ORE DILUTION PROMPTED SORTING STUDIES

The Combination Vein in the Black Pine area varies from two feet to six feet in true vein width, with an average calculated width of four feet throughout the current mineable reserves. Vein continuity is exceptional, but vein width is variable within very short distances on the structure.

The minimum practical mining height on the vein has proven to be six feet using currently available equipment, and the ore dilution factor can become quite high in stoping areas of lower vein widths.

Ore dilution is minimized in the larger development headings by using a "double-blast" or resuing technique where practical, however, dilution in the stoping areas is inherent to the operation.

Early in the life of the operation studies were undertaken of methods to decrease the ore dilution in the stoping areas, and these studies led ultimately to the installation and operation of the present sorting system.

A milling alternative was also examined, however, several factors—capital costs, investment to proven reserves ratio and numerous environmental problems—precluded this alternative. The practical solution seemed to be to devise some method or technique of separating as much waste from the run of mine material as possible before shipping the remaining product to the smelter.

By sight, this solution seemed feasible because of the color variation in the run of mine material—the waste consisting of pink to brown quartzite and the ore being composed of white quartz hosting black to gray-metallic tetrahedrite and other minerals.

It was known that in the early days of mining the Black Pine deposit, much hand sorting was done to up-grade the shipped product. However, this being economically unfeasible at the tonnages required for the current operation, it was decided to investigate various mechanical means which were available for effecting a separation between different colored materials in a common stream.

PRELIMINARY SORTING STUDIES

Investigation of the sorting alternative led to contact with Ore Sorters of Canada, a member of the Rio Tinto Zinc Group of Companies, located in Peterborough, Ontario.

Preliminary testing of Black Pine run of mine material on their Model 16 Photometric Sorter was very encouraging, and bulk sample testing was undertaken in May of 1977.

Initial testing on a run of mine sample of 3530 pounds of material crushed to −2″, screened to +½″, indicated an upgrading of 36% and 40% in the silver and copper heads, with an overall recovery of 82% and 85% respectively. (See Fig. 1)

Second phase testing using a run of mine sample of 3255.5 pounds of material crushed to −4″, screened +½″, and split into two feed sizes, +½″ to −2″ and +2″ to −4″, indicated an upgrading of 46% and 45% in the silver and copper heads, with an overall recovery of 87% and 86% respectively. (See Fig. 2)

Calculations based on the second phase testing indicated a 60% to 40% split between accept and reject material.

Using the parameters established in the second phase testing, and the current Black Pine ore reserve calculations, a proposed flow sheet (See Fig. 3) and a projected production sheet (See Fig. 4) were established. Economic analysis of the projected production indicated a viable operation, and installation and construction of a sorting plant were subsequently undertaken in July of 1978.

The Black Pine sorting plant was put into full operation on January 1, 1979.

PHOTOMETRIC SORTING PRINCIPLES

The basic components of a sorting system, as incorporated in Ore Sorters Model 16 Photometric Sorter, are a rock presentation system, a scanning system, and a separation system.

Rock presentation is accomplished by feeding onto a slide plate where separation takes place and then onto a belt moving at slide plate exit velocity. At the end of the slide plate, particles are squeezed between the main belt and a soft roller which can distort to accomodate particles up to a certain size. Forced acceleration takes place here, which breaks up the groups, and rock instability is dampened by a very loose belt running above the main belt.

The roller and stabilizing belt can be adjusted to accommodate different size particles.

Photometric techniques are based on the two principles that photon counts obtained from a rock surface exposed to light are proportional to the light intensity and color is basically a function of the non-uniformity of reflectance with wave length in the spectrum.

Adequate illumination is accomplished by means of a laser light source and a rotating mirror drum, and reflectance data is gathered by a photo multiplier tube.

Separation is achieved mechanically by means of air blast valves controlled by an electronic processor collecting and analyzing reflectance data. The machine can be set to blast either ore or waste depending on the ratio of each in the material stream.

PLANT DESIGN AND OPERATION

Test work, as seen in the plant flow sheet, indicated that the best overall results could be obtained by sorting the run of mine material in two separate size ranges and combining the accept streams for shipment. The plant was therefore designed using this test criteria.

Run of mine material is fed from a stockpile by a Cat 950 loader into a 16″ x 24″ jaw crusher set for a 4″ top size. The crushed material is then screened

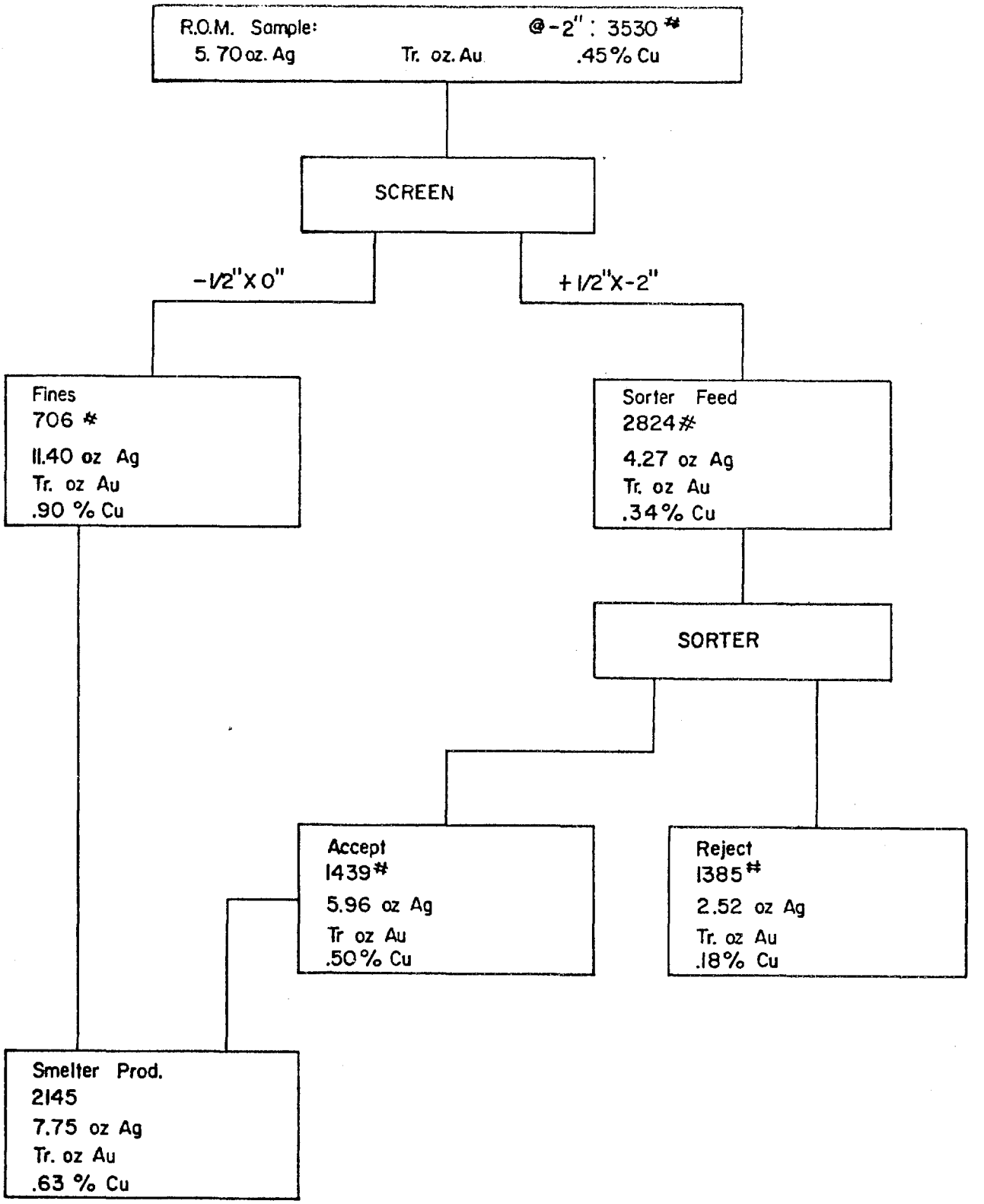

FIGURE 1. Schematic first test, ore sorters assay bulk test.

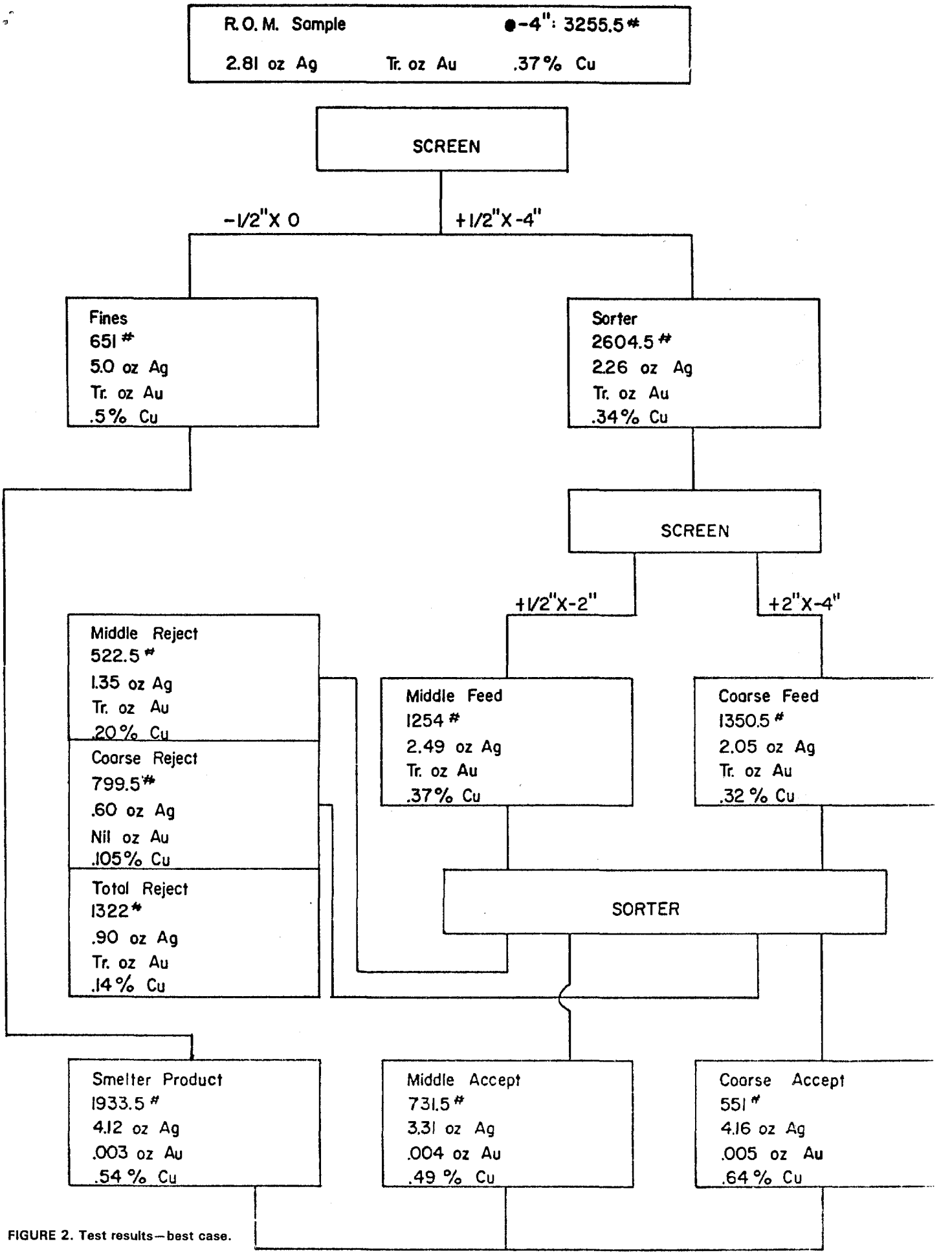

FIGURE 2. Test results—best case.

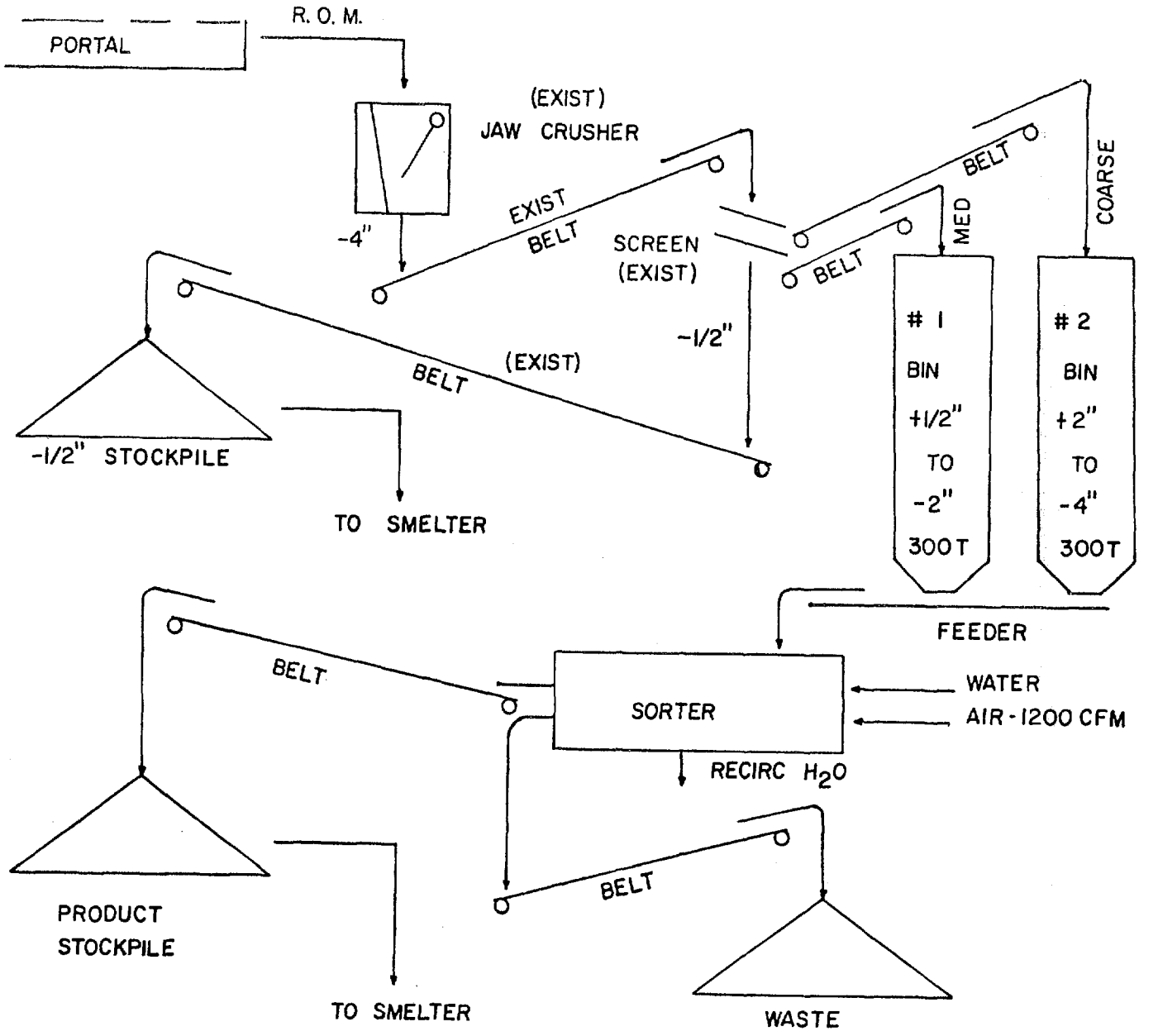

FIGURE 3. Schematic of sorter plant.

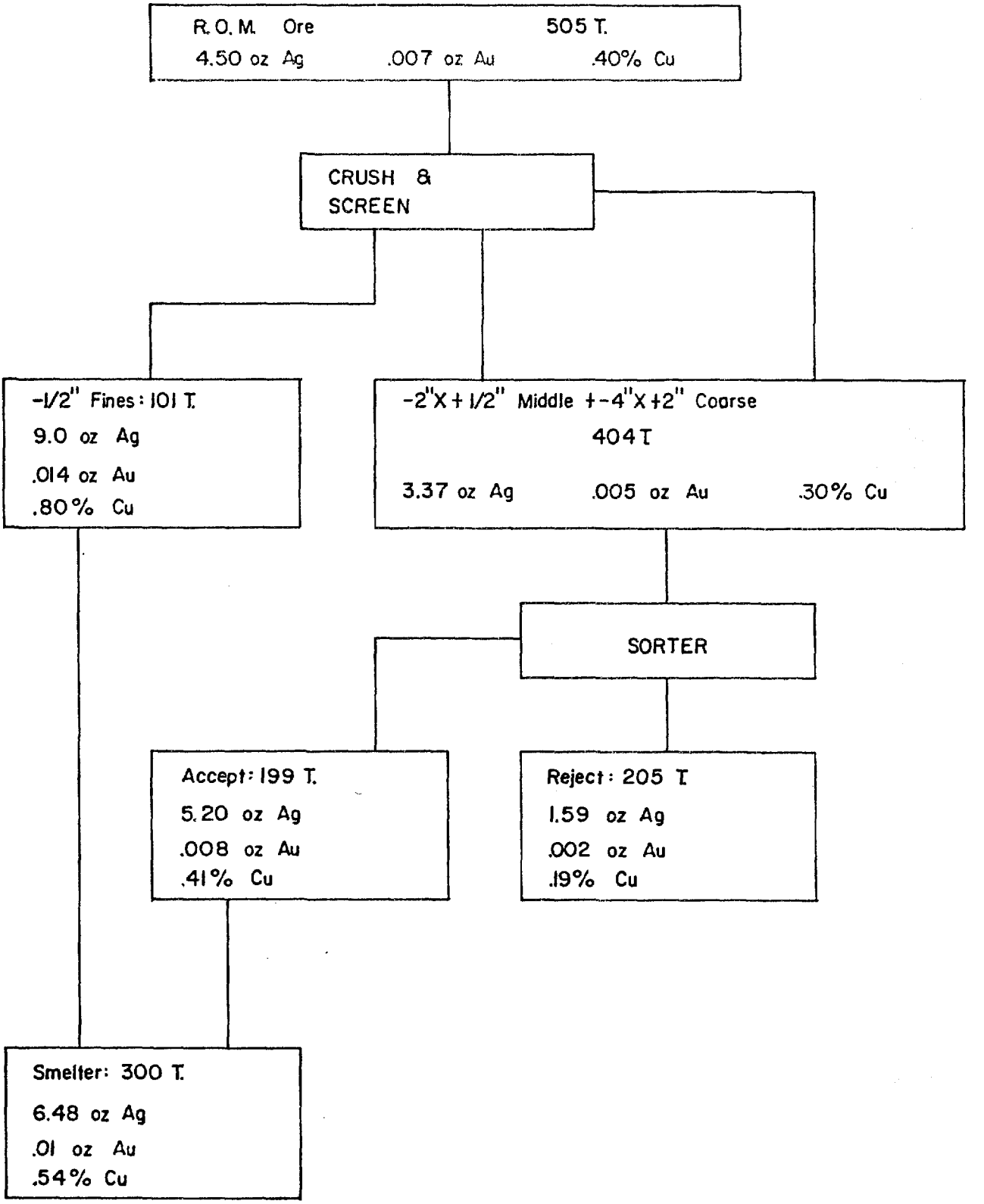

FIGURE 4. Projected production.

to produce two sorter feed sizes, +½" −2" and +2" −4", on a standard 4' x 8' two deck vibrating screen. The two products are conveyed to separate 150 ton storage bins, and the screened −½" material is conveyed out of the building for mixing with the final accept product.

A simple procedure sets the sorting machine for each size product prior to the sorting operation. Each bin feeds separately to a common conveyor by means of a vibrating feeder and single deck screen under each bin. The material is washed and rescreened to +½" prior to being conveyed to the sorter machine feeders. This first washing step is critical to the proper operation of the sorter in that very clean surfaces must be presented to the scanning system.

The material flow to the sorting machine is controlled by a hopper and two variable control vibrating feeders. The material is final-washed before entering the before mentioned sorter machine slide plate. After passing through the sorting machine, accept and reject material are conveyed out of the plant by separate belts. The accept material is stockpiled for shipment and the reject material is put over the dump.

Accept material is shipped to the smelter at Anaconda, Montana, by a contract hauler. It is further crushed to −3/16" and is used as a flux in the smelting process. Payment is made to Black Pine for silver, copper, silica content and crushing.

RESULTS AND CONCLUSIONS

In eight months of plant operation the sorting system has shown the following results:

1. The average split between accept and reject material has been 68% and 32% respectively.
2. The average upgrading of the silver content in the accept material has been 38% with an overall recovery of 93%.
3. The average upgrading of the copper content in the accept material has been 46% with an overall recovery of 95%.
4. Plant availability, after initial de-bugging, has been nearly 85%.
5. Through-put rates have approached 50 tons per hour on combined product feeds with an average air consumption of 900 C.F.M.

Sorting, in the case of the Black Pine operation, has added much flexibility in the mining operation. Mining can now be much less selective because the ore dilution inherent to the operation can be minimized by sorting.

Sorting may be a viable alternative in many operations where ore-dilution problems can not be alleviated with the mining method, whether for upgrading mill grades or direct shipment of the final product.

To paraphrase the manufacturers slogan: "If you can see it, *maybe* you can sort it."

REFERENCES

1. DeWilliam, P.P., Unpublished report, "Status Report - Black Pine Mine, Granite County, Montana, "September, 1972.
2. Kuhn, J. G., Unpublished report, "Preliminary Evaluation, Black Pine Mine, Granite County, Montana," October, 1972.
3. Kuhn, J. G., and McLaughlin, D. D., Unpublished report, "Justification For A Sorter Installation At The Black Pine Mine, Granite County, Montana," January, 1978.
4. Schapper, Mark A., "Beneficiation At Large Particle Size Using Photometric Sorting Techniques," Australian Mining, April, 1977.

JDS - WLK - JRS - Labor

April 14, 1984 Vol. 73 No. 15

SKILLINGS' Mining Review

Gold Fields Evaluating Mesquite Gold Property in Southeastern California—Page 4

ASARCO

Southwestern Exploration Division

April 18, 1984

File Note

Kappes, Cassiday & Associates
Heap Leaching Au-Ag

At the AEG Symposium, the attached brochure was available. The brochure contains a squib on the various people and about the company, plus four interesting papers by Dan Kappes.

1. Leaching of Small Gold and Silver Deposits: Small Scale Mining of the World Conference, Mexico, 1978.

2. Precious Metals Heap Leaching, Simple - Why Not Successful? Northwest Mining Association, 1979.

3. Scaleup Experience in Gold-Silver Heap Leaching: AIME National Meeting, 1981.

4. Sterling Mine Joint Venture - Anatomy of a Small Underground Mine and Heap Leach: 1982.

James D. Sell

James D. Sell

JDS/cg

xc: RLB 4/23/84 cy

Kappes, Cassiday & Associates

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11 August 1984

PROJECT EVALUATION FOR HEAP LEACH

GENERAL PROGRAM FOR DEVELOPMENT

FIRST QUICK LOOK

A preliminary test program designed to tell quickly if an ore is heap-leachable or not should be limited to a few column leach tests since these are rather expensive.

For a low-grade hard (not clayey) rock, a good first test would be a column leach test at intermediate crushed size, such as one inch. Probably at least two samples from the orebody should be tested, and maybe more if there is significant variation in rock type. The best samples are bulk samples composed of fairly large rocks.

A head sample of the material in the column, and the test tailings would be screened into five or six size fractions and these would be assayed to determine if there is an approximate relation between recovery and crushed size (typically there will be between 4 and 12 assays on the tailings so that the final percent recovery calculations are accurate).

Sometimes if 30-day percent recoveries are low, it may be advisable to stop the test, air-dry the sample, crush finer and then re-start the test.

The test program would include a cyanide-solubility test on an agitated portion of the pulverized head sample so that total cyanide soluble gold levels can be used to monitor the progress of the column test.

Preliminary data (such as estimated gold recovery based on solution assays) will typically become available from the column test about three weeks after the sample arrives at the laboratory. Tests should normally run 60 days, so that final information will be available three months after the sample arrives at the lab and the final report will be issued a month later.

The sample size placed in the leach column would range from 50 to 150 pounds, and would generally be tested in a column 6 to 12 inches in diameter and five to six feet deep. The total cost of the test program including report should be about $1700.00 per test. Involved sample preparation procedures would increase the test cost.

PROGRAM FOR COMPLETE EVALUATION OF THE OREBODY

The first step in designing a more complete program to test the orebody is to have a conference between the field geologists and the metallurgists who are designing the test program, to determine such things as:

- availability of core samples for testing deep-level ore
- accessibility for near-surface and deep-level bulk samples
- variability of ore and ability to geologically identify tonnage/geologic parameters of the various ore types

A general testing program usually consists of the following basic steps:

1. DETERMINE ORE TYPES. In conjunction with the geologists, review rotary drillhole chip logs and/or core photographs to try to determine important metallurgical ore types.

2. DETERMINE DISTRIBUTION OF CYANIDE SOLUBILITY OF GOLD WITHIN THE OREBODY. Run a series of cyanide-solubility tests (centrifuge tube leach tests) on all 5-ft (or appropriate) intervals of the rotary drillhole samples, to determine uniformity of cyanide response. These tests are usually run on the same pulverized pulps which were used for fire-assay of the drillhole intervals, and cost about $7.50 each.

A typical major orebody might require 500 such tests. The tests show recovery trends throughout the orebody, and can identify zones with special metallurgical problems. They are used as a guide to further testing and to selection of areas for bulk sampling.

3. RUN COLUMN LEACH TESTS. On bulk samples taken from underground and surface exposures, or if necessary from drill cores, a series of laboratory column leach tests at various crushed sizes would be run. The preliminary tests mentioned above would provide one data point; typically, the ore may be tested at three crushed sizes - 3, 1, and one-half inches. The column test program on each ore type would cost $5000.00. Simple small orebodies might end up with two ore types; complicated large orebodies might involve four to six sets of such tests.

This outline of column leach tests assumes the ore is hard and clay-free. The test program can be modified for clayey ores - for instance, testing at various crushed sizes is not so important as testing using agglomerated samples.

The general laboratory small-column test program for a large orebody can therefore range between $10,000.00 and $40,000.00. A few progress meetings and test reviews are normally included as part of the cost; sample procurement costs, feasibility studies or extended scoping meetings, and client oversight costs are not included in that estimate.

4. NEED FOR FIELD LEACH TEST. Proceeding beyond the small-column test phase is a function of both technical and managerial requirements. For hard, chemically-inactive simple ores which are going to be crushed before leaching, it is technically possible to proceed directly from the lab tests to production heaps, and this is sometimes done on small orebodies.

Large projects, especially on marginal-grade ores, usually require scaleup tests to reduce the degree of risk involved. A field test program consisting of one to three 2000 to 3000 ton field test heaps is typically recommended. The cost for an individual test at a remote location with no existing support facilities ranges from $150,000.00 to $250,000.00, inclusive of mining the sample in a small open pit, conducting the test for 60 days, and all final shutdown and report costs. With reasonable advance planning, "final authorization" could be given on 1 May for a test which is placed under leach 1 July and completed 30 August.

5. DESIGN/BASIC COST STUDY. Simultaneous with or subsequent to the field test, a pre-feasibility study can be developed which contains the basic design and cost elements (or alternatives). For orebodies up to 1000 tons per day such studies take about two months and cost about $20,000.00.

6. FINAL ENGINEERING/DESIGN STUDY AND CONSTRUCTION. The pre-feasibility study is generally sufficient to be accepted without a final full-blown feasibility study. Design and construction can then proceed.

Kappes, Cassiday & Associates

RECOVERY PLANTS FOR THE PROCESSING OF GOLD AND SILVER ORES

• 1845 GLENDALE AVENUE • SPARKS, NEVADA 89431 USA • TELEPHONE (702) 356-7107 • TELEX 170049 KCA •

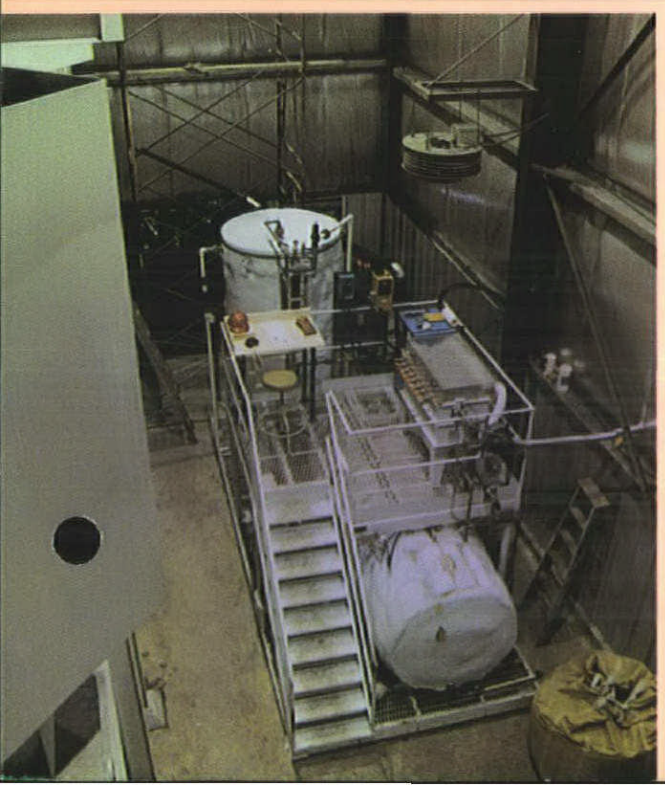
FULL PRODUCTION SYSTEMS

C AND CONVENTIONAL PLANTS

- AVAILABLE ON TURNKEY OR PROJECT MANAGEMENT BASIS
- IN-HOUSE ORE TESTING LABORATORY FOR DESIGN AND START-UP SUPPORT
- PILOT SCALE THROUGH SMALL PRODUCTION SYSTEMS

CARBON AND ZINC RECOVERY SYSTEMS FOR HEAP LEACHES

- TRAILER MOUNTED, SEMI-PORTABLE UNITS WITH COMPLETE ADSORPTION-DESORPTION-RECOVERY CAPABILITIES
- UNITS WITH CAPACITY TO 5000 TONS ORE PER DAY
- ALCOHOL, ATMOSPHERIC, OR PRESSURE DESORPTION
- FREE-STANDING ADSORPTION COLUMNS ALLOW LOW COST SATELLITE LEACHES REMOTE FROM MAIN PLANT



MODULAR PLANTS FOR SPECIAL APPLICATIONS

- MODULAR ADDITIONS TO EXISTING PLANTS
- MODULAR CARBON ADSORPTION CIRCUITS
- MODULAR DESORPTION CIRCUITS UTILIZING ALCOHOL, ATMOSPHERIC OR PRESSURE STRIPPING
- CUSTOM DESIGNS USING PROVEN TECHNOLOGY

Kappes, Cassiday & Associates

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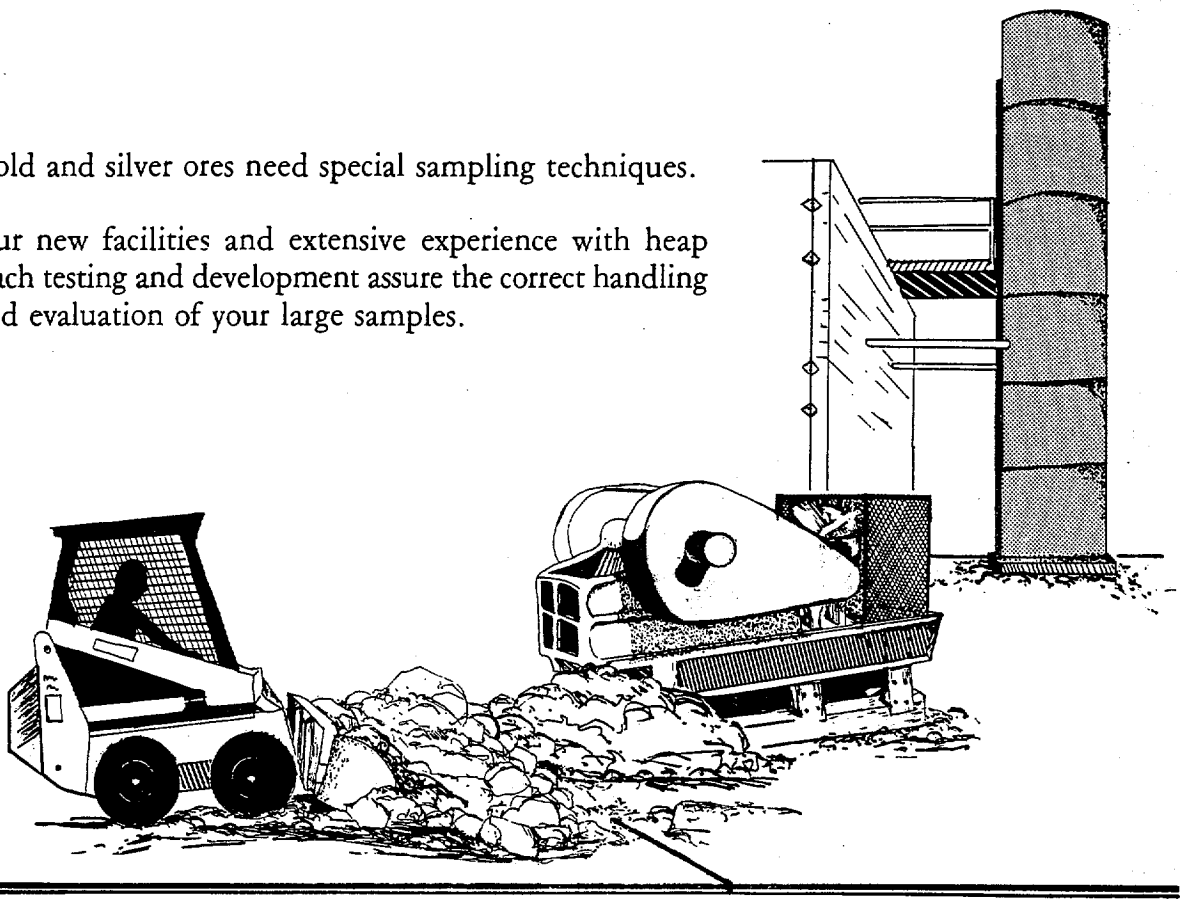
Kappes, Cassiday & Associates

ANNOUNCES

EXPANDED FACILITIES FOR THE HANDLING OF BULK SAMPLES

Gold and silver ores need special sampling techniques.

Our new facilities and extensive experience with heap leach testing and development assure the correct handling and evaluation of your large samples.



NEW CAPABILITIES

- Twenty thousand square feet of fenced yard for receipt of truckload samples
- Crushing facility for run of mine ore
- Mixing and splitting of large bulk samples
- Column tests 4 feet or larger diameter and up to 30 feet high
- Seven thousand square feet of heated building space for sample storage and drying

To discuss your bulk sample application contact: Russell B. Dix, Laboratory Manager

1845 Glendale Avenue • Sparks, Nevada 89431 • USA • Telephone (702) 356-7107 • Telex 170049 MILKAP

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1 August 1986

KAPPES, CASSIDAY & ASSOCIATES
LABORATORY CLIENT REFERENCES

Kennecott Minerals Company

Date of Work: 1984 - 1986

Principals: Wayne Henderson (801) 322-8303
Terry Anderson (801) 322-8360

St. Joe Minerals Corporation

Date of Work: 1984 - 1986

Principals: Harold Ray (314) 244-5261
John Morrisey (314) 244-5261

Noranda Exploration

Date of Work: 1983 - 1986

Principal: Bruce Wallace (416) 867-7236

Ferret Exploration Company

Date of Work: 1983 - 1986

Principal: Ralph Barnard (303) 294-0427

Texasgulf Minerals & Metals Inc.

Date of Work: 1985

Principal: Scott Moser (303) 273-2427

E&B Explorations, Inc.

Date of Work: 1982 - 1983

Principals: Leonard Saleken (604) 689-5453
Brian Kynoch
Paul Saxton

Amax Engineering

Date of Work: 1984

Principal: Brian Ball (303) 234-9020

Cominco Ltd.

Date of Work: 1984

Principal: Mike Fairweather (604) 364-4447

Freeport Gold Company

Date of Work: 1984 - 1986

Principal: Scott Barr (702) 738-9221

FRM Minerals Inc.

Date of Work: 1984 - 1985

Principal: Eric Peterson (303) 989-5800

Northern Dynasty Explorations Ltd.

Date of Work: 1984 - 1985

Principals: Glenn Simpson (604) 682-3727
David Jennings

Imperial Metals Corporation

Date of Work: 1983 - 1986

Principals: Zarko Nickic (604) 669-8959
Ruben Corvalan

Lacana Mining, Inc.

Date of Work: 1983 - 1984

Principals: Mike Easdon (702) 329-5609
Harvey Sobel

Ventures West Minerals, Ltd.

Date of Work: 1983

Principal: Victor Jones (604) 689-3923

Kappes, Cassiday & Associates

1845 Glendale Avenue, Sparks, Nevada 89431
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21 June, 1983

Since 1976, KAPPES, CASSIDAY & ASSOCIATES (formerly the Miller-Kappes Company), has been working exclusively with the technology of cyanide metallurgy as applied to the treatment of gold and slver ores. We maintain laboratory facilities in Reno, and specialize in the evaluation and field management of small projects and the preliminary field test work for major developments.

The company staff includes seven professional personnel who are briefly introduced below. The full-time staff presently includes five other employees who function as lab technicians or as clerical personnel. Field operations, which comprise an important part of the business, raised peak employmennt in the firm to more than 25 persons during the summer 1982 field season.

DANIEL W. KAPPES, President, is a Mining/Metallurgical Engineer, and has been associated with the company full-time since 1972. He is a recognized authority on precious metals heap leaching and has presented several technical papers on that subject. In addition to work on numerous small projects, he has directed laboratory and field testing on five projects which have subsequently become major precious metal mines.

MICHAEL W. CASSIDAY, Vice-President, is a Metallurgical Engineer, and has been associated with the company since 1978. His primary expertise is in the management of field projects. Major projects in which he has been involved include the construction/management of large field leach tests at sites in Nevada, Colorado, and South Dakota; and construction/management of recovery systems for an operating heap leach at Beatty, Nevada. He has recently managed the field construction and start-up of a 100 ton per day carbon-in-pulp plant at Meekatharra, Western Australia.

RUSSELL B. DIX is a Metallurgist and Chemist. His primary function within the company is to oversee the operation of the Reno laboratory and the chemical control of the field test operations. Mr. Dix holds a Master's degree in Metallurgy; in his professional career he has held significant positions as research metallurgist on chloride leaching systems, research chemist on the chemical/ mechanical properties of coal, and Quality Assurance Engineer/Supervisor in a chemical process facility.

RANDALL A. PYPER holds a Master's degree in Metallurgical engineering, and before joining our staff was chief metallurgist of the 2500 ton per day agitated leach plant of Carlin Gold Mines (Newmont Mining Corporation). His earlier professional background includes production and process engineering for PPG Industries, Lake Charles, Louisiana; and mining chemicals technical service and development for the Dow Chemical Company at Walnut Creek, California.

GARY C. COLEMAN is a geologist and mining engineer, and specializes in the operations management of small production facilities. On the 100 ton-per-day Meekatharra, Australia carbon-in-pulp plant, he served as general manager for the company during the final stages of commissioning and the first six months of operation. He is currently general manager of the Santa Clara Mine, a 300 ton per day open pit mine/heap leach in northern Costa Rica, operated by the company for United Hearne Ltd.

TERRY R. HANSON is a chemist with ten years experience working in various capacities for the state of Nevada. His recent experience has included two years with the State Environmental Protection Agency. Within the company, his primary responsibilities include those of field project metallurgist and chemist for special projects.

S. GEOFFREY ALLARD holds a Master's degree in Metallurgical Engineering. His primary position is Research Metallurgist, working on in-house studies of various aspects of cyanide leach chemistry. He also assists in the design and construction management of the leach/recovery equipment for field projects.

The company laboratory in Reno, Nevada, specializes in the evaluation of ores for carbon-in-pulp and heap leach processing, and is available to clients for individual tests, or for the design and execution of a complete test program to determine the process flowsheet for a specific orebody.

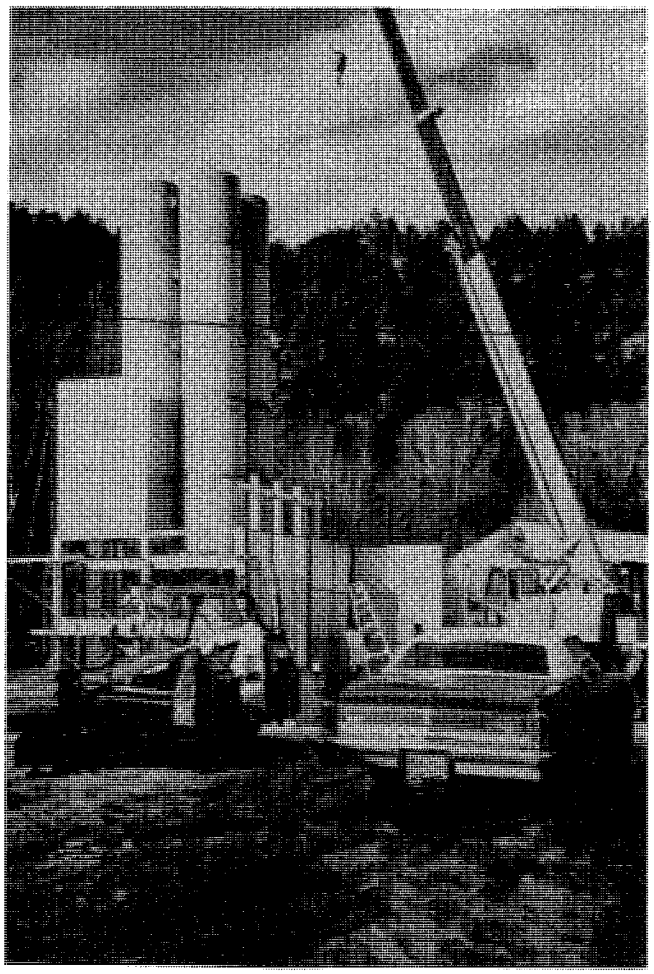
Our staff constitutes a team with varied technical abilities, especially adapted to work on projects requiring a complete range of engineering evaluation. We specialize in being able to handle field test leaching projects and small production plants on a turn-key basis. The top photograph on the final page of this letter shows four 40-foot tall leach colums, holding 25 tons of ore each, which we constructed at a western U.S. location for rather unusual leach tests. The lower photograph shows two 2000 ton field leach tests constructed as part of a typical testing project by Kappes, Cassiday & Associates. This property is now in production at several thousand tons per day.

We specialize in heap leaching; however, recent projects have included a carbon-in-pulp plant in Western Australia, on which we performed all design and construction/management and provided initial break-in operating personnel. We have also managed field projects where, as part of an overall program of heap leach evaluation, we have supervised significant tunnel rehabilitaion and underground sampling programs.

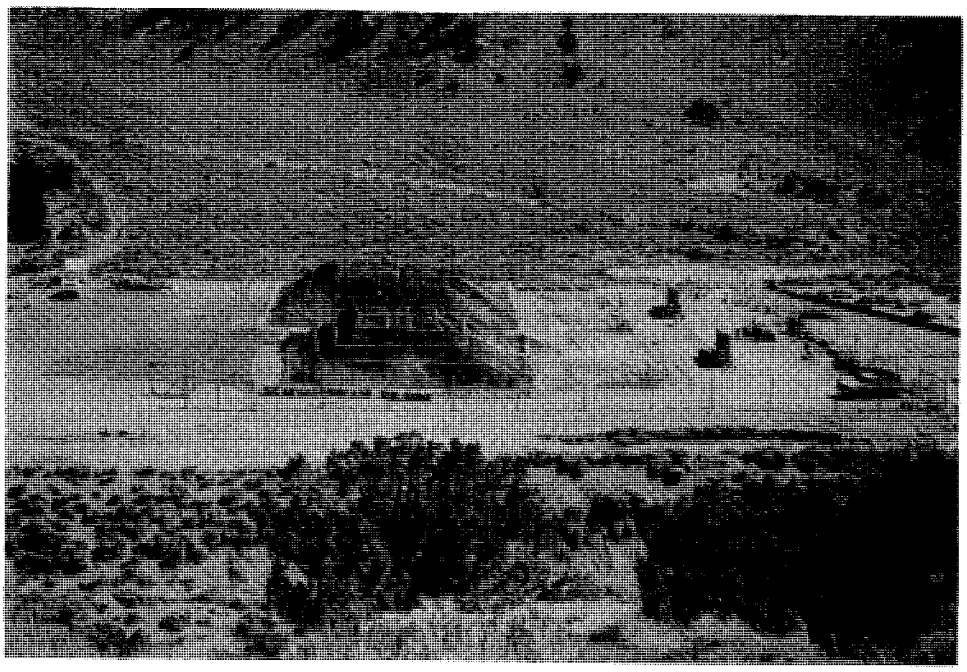
For the $15 million, 3000 ton per day, Northumberland, Nevada project (Cyprus Mines), we did all laboratory and field testing and provided conceptual design (detailed design and construction was by others). For the 200 ton per day Sterling Mine (Beatty, Nevada), we did all testing and recovery systems design. Both of these are currently operating mines. We have performed significant preliminary testing and/or design on five other projects which have become substantial production operations.

We are currently general managers of the Santa Clara Mine of Minera Macacona, S.A. (a subsidiary of United Hearne Ltd.) in northern Costa Rica. This mine employs a total of nearly 60 persons, and in early 1983 was mining 1800 tons of ore plus waste per day, and processing 300 tons per day of this by agglomeration heap leaching. The ore is nearly pure clay, and the operation is in a tropical climate with a very pronounced rainy season.

During 1982, Kappes, Cassiday & Associates provided laboratory or field consulting services to approximately 35 different clients on gold and silver ores with widely varying metallurgical characteristics. We are proud to be specialists in a healthy and challenging segment of the mining industry.



40-Foot Tall Leach Columns



2000 Ton Field Leach Tests

Kappes, Cassiday & Associates

1845 Glendale Avenue, Sparks, Nevada 89431
702-356-7107 - Telex 170049 - Fax 702-356-5609

22 November 1987

SUMMARY OF CAPABILITIES

Since 1976, KAPPES, CASSIDAY & ASSOCIATES has been working exclusively with the technology of cyanide metallurgy as applied to the treatment of gold and silver ores. We handle all aspects of cyanide processing including laboratory testing, engineering design and feasibility studies, construction, and operations management.

The company has extensive field experience with many production and pilot heap leach operations. It has provided testing and conceptual design and consulting services to twelve major projects which have subsequently gone into production, and has provided major field leach testing programs on a turnkey basis at several others.

The "core" staff consists of 25 people, but we have employed temporary staff of up to 50 people during peak periods of field construction projects.

KCA's largest project has been the 4000 ton per day heap leach at Hog Ranch, Nevada. On this project, KCA acted as design engineer/construction manager, and also as general contractor on the recovery plant and support installations. In 1987 KCA has begun field supervision on a 500 tonne per day silver heap leach at Potosi, Bolivia; designed and provided construction supervision on a 2500 tonne/day heap leach at Marvel Loch, Western Australia; and built modular recovery plants for operations at Hassai, Sudan; Gordex (Cape Spencer), New Brunswick;; Aurora, Nevada; and Gilbert, Nevada.

Key professional personnel are briefly introduced below.

DANIEL W. KAPPES, President, is a mining/metallurgical engineer, and has been associated with the company full-time since 1972. He is a recognized authority on precious metals heap leaching and has presented several technical papers on that subject. In addition to work on numerous small projects, he has directed laboratory and field testing on five projects which have subsequently become major precious metal mines.

MICHAEL W. CASSIDAY, Vice-President, is a metallurgical engineer, and has been associated with the company since 1978. His primary expertise is in the management of field projects. Major projects in which he has been involved include the construction/management of large field leach tests at sites in Nevada, Colorado, and South Dakota; and construction/management of recovery systems for an operating heap leach at Beatty, Nevada and Getchell, Nevada. He has managed the field construction and start-up of a 100 tonne per day carbon-in-pulp plant at Meekatharra, Western Australia.

RUSSELL B. DIX is a metallurgist and chemist. He has been with the company since 1980. As Manager of laboratory operations, his primary function within the company is to oversee the operation of the Reno laboratory, to set up field laboratory projects for client companies, and to oversee the process control of field operations. Mr. Dix holds a Master's degree in Metallurgy; in his professional career he has held significant positions as research metallurgist on chloride leaching systems, research chemist on the chemical/mechanical properties of coal, and Quality Assurance Engineer/Supervisor in a chemical process facility.

RANDALL A. PYPER holds a Master's degree in metallurgical engineering, and before joining our staff was Chief Metallurgist of the 2500 ton per day agitated leach plant and 2500 ton per day heap leach facility of the Carlin Gold Mining Company (Newmont Mining Corporation). His earlier professional background includes production and process engineering for PPG Industries, Lake Charles, Louisiana; and mining chemicals technical service and development for the Dow Chemical Company at Walnut Creek, California.

REDFORD McDOUGALL holds a Master's degree in Chemical Engineering. He served as manager of Standard Slag's Atlanta, Nevada gold operation (a 700 ton per day conventional agitated leach plant) and as design engineer for crusher systems and project startup at their Sulfur, Nevada heap leach operation. For eight years he was owner of a company which specialized in supplying scale and corrosion inhibiting chemicals.

W. ROBERT ROSE is a mining engineer with startup and operations experience in heap leach projects for leaching both ore and tailings, and in gold flotation plants and placer mines. He has worked on the Dayton tailings project and on the Olinghouse placer projects, both of which involved processes somewhat unique in western U.S. gold practice.

WILLIAM R. YERNBERG, KCA's assistant lab manager, is a geologist and chemist with extensive hydrometallurgical research experience with a wide variety of minerals including gold, rare earths and unusual metals. Prior to joining KCA he was a research metallurgist with Molycorp, Inc.

CHARLES B. KIMBER is a metallurgical engineer with over 12 years of production operations management in the Aluminum and Copper industries. His copper-industry experience also includes flotation and hydrometallurgical research and development.

Our staff constitutes a team with varied technical abilities, especially adapted to work on projects requiring a complete range of engineering evaluation.

The company laboratory in Sparks (Reno), Nevada, is thoroughly equipped for the evaluation of ores for carbon-in-pulp and heap leach processing, and is available to clients for individual tests or for the design and execution of a complete test program to determine the process flowsheet for a specific orebody.

We specialize in an integrated, practical approach to projects, and have provided the complete range of services from preliminary evaluation through operations management.

In a typical year, we will perform laboratory testing on over 50 different ores, and provide engineering/management services to ten projects.

A list of recent projects which illustrate our range of capabilities is presented below.

1. Hassai, Sudan - BRGM (France)
   Gordex, New Brunswick - Gordex Minerals, Ltd.

   Dates: 1986 - 87

   KCA built, installed and started up portable (modular) gold recovery plants, complete with furnace for producing dore bars, for these two heap leach operations. (250 and 900 tonnes per day plant capacity).

2. Gilbert, Nevada - Kemco, Inc.
   Aurora, Nevada - Minerex/Aurora Partners, Inc.

   Dates: 1987

   Scope of Work: KCA provided testing, project design, and complete portable process plants at both locations (1200 and 1500 tons/day plant capacity). At Aurora, KCA also provided extensive field project construction management, and built and installed a portable laboratory.

3. Marvel Loch, Western Australia

   Dates: 1986-1987

   Location: Marvel Loch, Western Australia

   Scope of Work: KCA provided testing and design services and development of processing techniques, and then provided a project engineer to supervise all aspects of construction and startup, for a 2500 tonne/day heap leach processing a soft white pure clay ore.

   Heaps are currently being constructed over 9 meters (30 feet) high and are performing according to design.

4. Hog Ranch, Nevada - Western Goldfields, Inc.

   Dates: 1981 to 1986

Location: northwestern Nevada

Scope of Work: Beginning in 1981 KCA performed laboratory testing on the project, and in late 1985 prepared a design/cost study. We then were given the contract to perform project engineering and construction management, and we also furnished the field construction crew to install the leach pads, process plant and general site infrastructure. Groundbreaking took place on 28 April, 1986, and the first (700 oz) gold bar was poured on 14 September, on time and within budget.

The project is currently operating as designed.

5. Little Bald Mountain Mine - Northern Dynasty Explorations Ltd.

Dates: June 1985 to present

Location: Little Bald Mountain, Nevada

Scope of Work: KCA conducted laboratory investigations on the LBM deposit and provided conceptual design of heap leach pads, ponds, and operating philosophy. KCA then provided the design, construction, and start-up of a complete gold adsorption-desorption-recovery plant rated at a production capacity of 300 tons ore per day.

The project is now in its third season of production. Direct capital and operating costs were repaid within the first season.

6. Sterling Mine - Saga Exploration Co.

Dates: 1980 to 1983

Location: Southwest Nevada - USA

Scope of Work: Kappes, Cassiday & Associates conducted laboratory investigations on Sterling's underground deposit and provided conceptual design of heap leach pads, ponds, and operating philosophy. KCA then provided the design, construction, and start-up of a complete gold adsorption-desorption-recovery plant.

The Sterling Mine is still successfully producing gold, and still utilizes the portable recovery plant which we built for them in 1980. This was the first of ten such plants which we have built.

7. Minera Macacona, S.A.

Dates: 1983-1984

Location: Costa Rica, Central America

Scope of Work: KCA conducted laboratory and field testing on the high-clay ore of Minera Macacona's Santa Clara mine, and after a four month consulting status assumed all general mine management duties. We then purchased, installed, and started up a 7-foot diameter agglomerating drum and associated equipment.

As part of its general management functions, KCA instituted pit sampling procedures and general mine planning, expansion of the contract equipment fleet, as well as numerous improvements in plant operation and metallurgical accounting. During KCA's management ore production increased from 200 to 650 tons per day, waste production increased from 100 to 1200 tons per day, and attendant gold production increased from 50 to 650 ounces per month.

At present the mine is operating under direct management of the owning company.

8. Meekatharra Operation - Argosy Gold Mines, Inc.

Dates: 1981

Location: Meekatharra, Western Australia

Scope of Work: KCA designed, built, and started up a $750,000.00, 300 tonne per day carbon-in-pulp plant. Limited reserves made the life of the project brief (30,000 tonnes were processed). Operating costs were within target but return-on-investment was poor.

The plant has since been moved to another site in Western Australia.

8. Haveluck Mine - Whim Creek Consolidated N.L.

Dates: 1976

Location: Meekatharra, Western Australia

Scope of Work: KCA developed the concepts for, constructed, and operated the first test heap leach on the Haveluck Deposit, which was the first major heap leach in Australia. This subsequently became a 1000 tonne per day combined agitated leach/heap leach operation owned by Whim Creek Consolidated, N.L.

The mine has been in full production operation since 1979.

Besides these major projects, during the past year Kappes, Cassiday & Associates provided laboratory or field consulting services to approximately 50 different clients on gold and silver ores with widely varying metallurgical characteristics. We are proud to be specialists in a healthy and challenging segment of the mining industry.

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Kappes, Cassiday & Associates

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702-356-7107 - Telex 170049 - Fax 702-356-5609

31 March 1987

We manufacture two basic sizes of portable plants for the recovery of precious metals using activated carbon, however each plant is usually tailored to the specific project so there can be variations on size or process flowsheet. The "small" plant is rated at 300 to 500 tons ore per day placed on a typical heap leach, and the "large" plant is rated at 1000 to 1500 tons per day. Present prices for the plants for US delivery are $140,000.00 for the small plant, and $250,000.00 for the large plant. Prices for overseas delivery are typically 25 percent higher.

The "plant" includes all process items needed to produce dore bullion from process solution, including a small crucible furnace for pouring the dore. Pumps for pumping solution from storage ponds through the plant, and from the barren pond to the heaps, are also included (heap piping is not included). The price of carbon ($8,000.00 for the small plant, $25,000.00 for the large plant) is not included.

The small plant is built within a 12-meter trailer or shipping container. For overseas delivery, an additional 6-meter container is used for shipping of support equipment such as a chemical storage tank and the smelting furnace. The large plant requires an additional 12-meter container.

Delivery FOB Sparks, (Reno) Nevada is 12 weeks after receipt of the order.

We have manufactured several of these plants, including one which was recently shipped to a remote site in The Sudan.

In addition to plant construction, we offer a full range of services to develop your precious metals cyanide leach project, including laboratory testing, design and cost studies, and field supervision through complete turnkey project management. We recently completed the 4000 ton per day Hog Ranch, Nevada heap leach project, on which we provided complete project management and plant construction services.

We have worked exclusively in heap leaching for over ten years, and have a staff of 25 people devoted to this technology.

If you are considering a heap leach project, or the use of other cyanide leach technology, we would appreciate the chance to evaluate your project and show you how we might be of help.

James D. Sell

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LEACHING OF SMALL GOLD AND SILVER DEPOSITS

by

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Introduction and Summary

In spite of a long-term decline in small-scale mining in the United States, there has been a recent dramatic surge in the number of small gold and silver mining operations.

The increased activity can be attributed to the development of a technique called heap leaching. The idea behind the technique is nearly as old as mining - piling up heaps of copper ore and irrigating them with leach solution has been practiced for over 500 years. The "revolution", at least in respect to small-scale mining of precious metal ores, is due to two recent developments:

1. Plastic membranes are now available which can be used to make inexpensive, quickly-installed impervious leach pads.
2. The use of granular activated carbon has been perfected for the recovery of gold and silver. The use of carbon, instead of the more traditional zinc precipitation process, allows the design of low-cost circuits with very flexible operating parameters. Such installations may be operated on a one

shift per day schedule, and may be shut down for weekends or holidays. They can be operated by one or two men with very limited technical training.

Because capital and operating costs are low, heap leaching is applicable to large low-grade deposits. The largest heap leach operation in the U.S. processes 7200 tons per day of ore containing 2.2 ppm gold. One operation in Nevada heap leaches - profitably - 1100 tons per day of ore containing only 1.2 ppm gold; the ore at this mine is ideally suited to the process - it breaks naturally to a fine-grained, permeable sand.

Much smaller tonnages, of correspondingly higher grade, are being mined profitably. At least two operations contain total reserves of less than 150,000 tons, with an ore grade of approximately 5 ppm gold. The life if a small operation, from the original decision to proceed until completion of all activity, can be as short as 12 months.

A typical small operation might leach 200 tons of uncrushed ore per day, with the ore stacked in heaps containing 10,000 tons of ore each. The total capital cost to begin the initial production from a system this size, including the cost of the chemical charge for the first 10,000 ton heap, would be less than US $200,000. This figure includes all leach costs, and "on-site" facilities such as haul roads, but it does not include mining costs and purchase costs of major pieces of construction machinery. Costs are for the year 1978. In the concluding section of this paper, a much smaller operation is discussed - one producing one ton of ore per day, using only manual labor with no mechanical equipment.

The following sections of this paper attempt to identify the technical problems and to describe the mechanical systems which are encountered in a heap leach operation. All the details cannot be covered in

a paper this size, but it is hoped that enough information is present to furnish the reader with a comprehension of how the process works.

As in any technical undertaking, there are pitfalls. The western U.S. has many examples of "failed" heap leach operations. Nevertheless, the system is basically simple, and can be implemented using equipment, manpower and technical capabilities which are for the most part already available in developing countries.

System Description

Figure 1 is a photograph of a heap containing 300 tons of ore, with the ore stacked 4 meters high on top of a plastic pad. During the operation of this heap, solution was pumped from the pond in the foreground (the barren pond) to sprinklers spaced every 2.5 meters along pipes on the heap. Total flowrate on this heap was 15 liters per minute. The leach solution contained 0.5 - 1.0 grams sodium cyanide per liter, and enough lime to maintain a pH greater than 10. The chemicals were added by simply dumping them into the barren pond. The solution flowed down through the heap, then across the plastic to a drain, and from there into the "pregnant" pond on the left side of the photo.

Figure 2 is a photograph of the same heap from a different angle. The solution was pumped from the pregnant pond, in the foreground of this view, through columns filled with activated carbon (4 columns, 20 cm diameter and 1.5 meters high), and then was discharged into the barren pond. The same solution was used over abd over, since the activated carbon removes only the metals (gold and silver) and does not otherwise affect solution chemistry.

This system was usually operated 8 hours per day, and was occasionally shut down for periods of 2 or 3 days. It should be pointed out that in Figure 2 the entire leach system for treating the 300 ton heap is

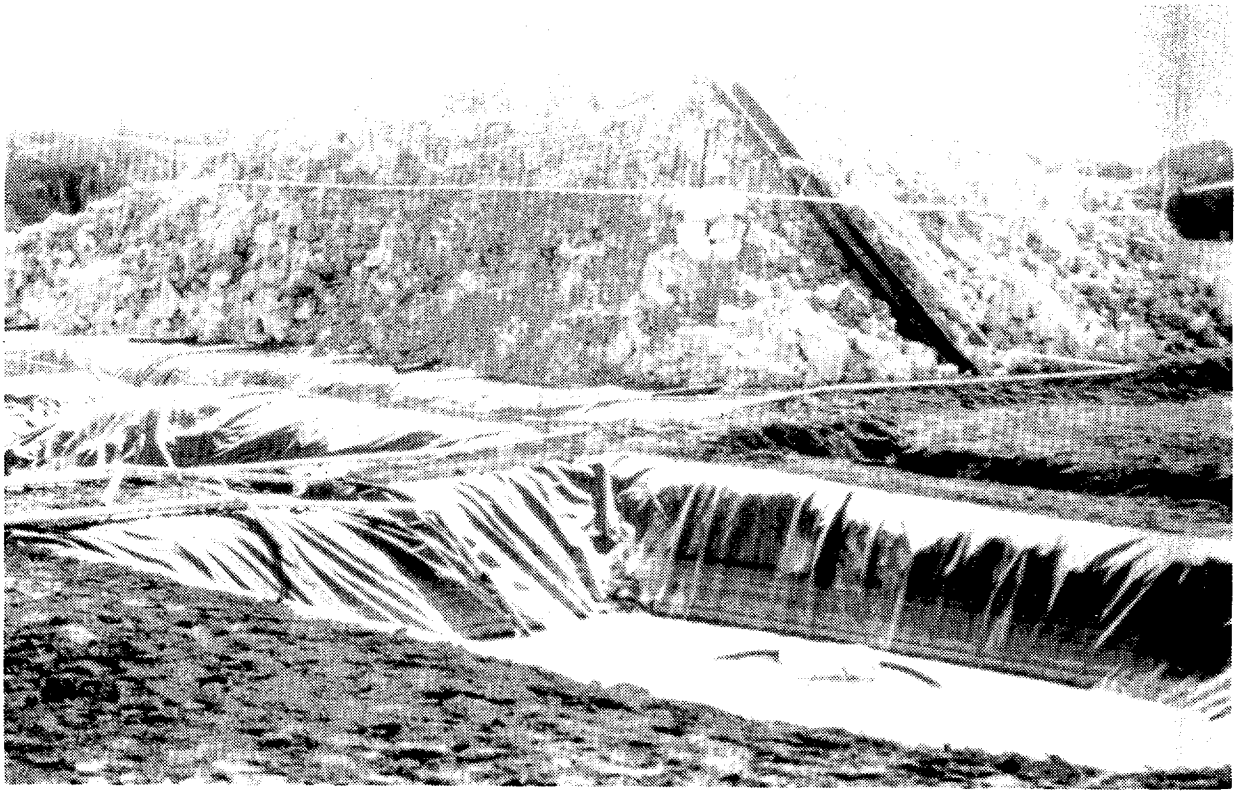

FIGURE 1. Three hundred ton heap leach of mine-run ore from a deposit with a very high clay content.

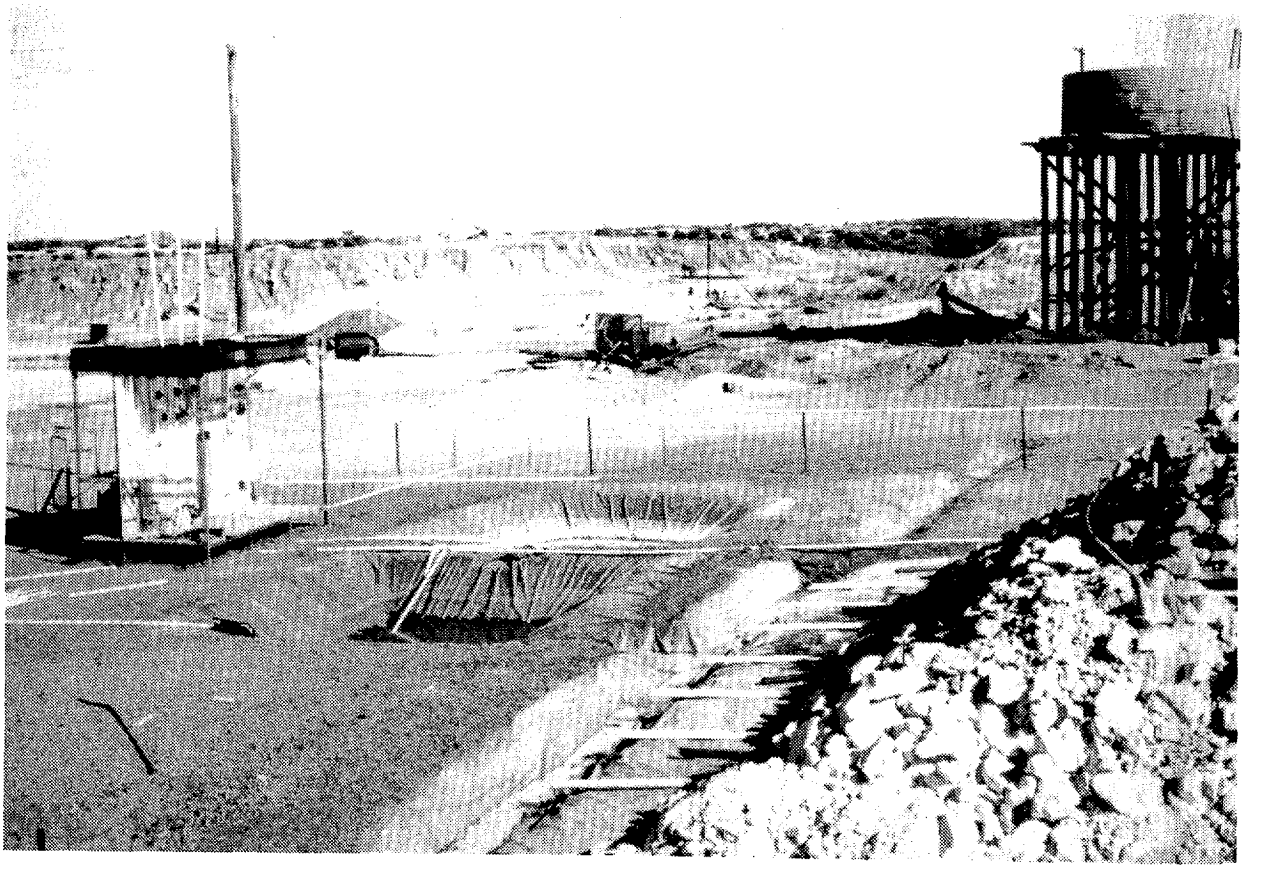

FIGURE 2. Three hundred ton heap leach showing carbon column platform and piping layout.

shown - plastic pads and ponds, 2.5 cm diameter pipes to carry solution, 2 small pumps to pump solution onto the heap and through the columns (the pumps are mounted on the column stand), and the adsorption columns. The only equipment not shown, is the unit which removes the gold from the carbon and converts it to metallic gold. That unit is similar to, and actually of much simpler design than, the adsorption column unit in the photo. In 80 days of operation, 70% of the fire-assayable gold in this heap of 300 tons was removed, and transferred to 100 Kg of activated carbon which was contained in the columns.

At one operation, where the heaps contain coarse gold and large rocks, the heaps were operated intermittently for over two years. Leach operation for at least two or three months is practiced in nearly all cases, because daily operating costs are very low - the only costs are for labor, power, and makeup chemicals. If other "active" heaps are being run in the same area, the labor cost to oversee an old heap is essentially zero. Makeup sodium cyanide need be added only to replace losses due to oxidation of the cyanide, and costs approximately US $0.15 per ton of ore per month of leaching. The power requirement and cost is almost negligible - 0.0006 kilowatts per ton.

Leachability, Permeability and Stacking Methods

The rock in the heap shown in figures 1 and 2 is a soft white Kaolinite clay. The heap is in the shield area of Western Australia, where long-term surface weathering has caused intense lateritization of the top 40 meters of the surface. The primary rock at this locality was a dense, unfractured, metamorphic rock containing coarse gold. Since coarse gold dissolves very slowly, and since the rock was very impermeable, the primary rock could probably not have been heap-leached successfully. Weathering caused the re-deposition of much of the gold as fine particles, and also created an open-structured rock which was quite porous.

Because of the high clay content, however, two specialized procedures had to be used when the heap was constructed. The first of these involved the method of mining. The creation of "fines", or pulverized rock, had to be avoided. Test samples of the ore, ripped with a bulldozer, were found to be impermeable no matter how they were attacked. Successful mining required close-spaced drilling and light blasting, a procedure which resulted in much less fines than ripping while simultaneously limiting the maximu rock dimension to about 0.5 meters.

The second special procedure involved the method of stacking rock onto the heap. A conveyor stacker like that shown in figures 3 and 4 was used. The stacker can be manipulated to create a series of discrete 1 meter high cones of rock, and a large heap can be built as three or more layers of these cones stacked on top of one another. The action of the conveyor discharge throws a portion of the coarse rocks farther than the fines, and creates a highly permeable shell of coarse rock around each cone. By changing the conveyor movement, the permeability of the heap can be "adjusted" to insure good leachability of problem ores. The capital cost of a completely mobile, gasoline-driven conveyor (the tip of one is shown in Figure 4) is US $12,000, and it is capable of stacking 10 to 20 tons of ore per hour.

Suitability of Various Ores

Some other "typical" heap-leachable ores and their special leach problems are discussed in the list below.

Fractured Volcanics and Schists, near High-Grade Veins:

- Gold is often coarse (visible to the unaided eye). This may result in long leach times, up to one or two years for complete recovery.
- Development of at least some ore reserves is sometimes easy, since old mine workings on the veins have often exposed low-grade ore.
- This is the most common ocurrence being heap-leached in the western U.S. Typical example: Round Mountain, Nevada, the largest operating leach at 7200 tons per day.

Gold with Pyrite or Arsenopyrite in Primary Ores or Dense Metamorphic Rocks:

- Successful recovery often requires crushing to 1 cm or even to 6 mesh; therefore, heap leaching can only be justified for very high-grade, small deposits.

Gossans:

- Gold has often been enriched by leaching-removal of sulfur and base metals; this is especially common for gossans on massive-sulfide type orebodies.
- Iron oxides are generally not a problem, but in one case a dense high-iron laterite could not be heap-leached because the iron oxides mechanically covered the gold particles.

Gold in Porous Rocks with Sulfides of Soluble Base Metals:

- Below the oxidixing portions of sulfide veins, gold may be deposited on pyrrhotote or marcasite surfaces. Usually the gold leaches well, but cyanide and lime consumption can be high.
- Sulfide veins often oxidize to a mixture of rock fragments and unstable yellow clay. The clay may prevent successful heap percolation, and in fact this has caused the failure of several leach operations.
- Oxidizing copper minerals or base-metal salts may greatly increase apparent cyanide consumption. Chemical control of heap operation becomes more sophisticated.

Disseminated Gold in Dirty or Hydro-thermally-Altered Limestones:

- Often very large and relatively high-grade deposits. Typical examples: Carlin (2000 tpd, conventional mill); Cortez, Nevada (portion milled and portion heap-leached); Windfall, Nevada (1100 tons/day heap leach on ore containing 1 ppm gold).
- Gold leaches quickly and thoroughly, usually there are few chemical problems and the rock is permeable. Carbon already present in the rock is sometimes activated, and can prevent leaching.
- Since the gold is sub-microscopic and can be found by assay only, there are often no "ancient" mine workings to serve as guides to ore: ore exploration is based on geologic models.

Silver Ores:

- Generally, these can be leached using exactly the same techniques as for gold ores.
- The "loaded" activated carbon contains a low value of silver per unit of carbon, and a large amount of carbon is needed. An on-site carbon stripping and recovery plant is a necessity (for gold leaches, carbon can be shipped elsewhere for stripping or can even be sent

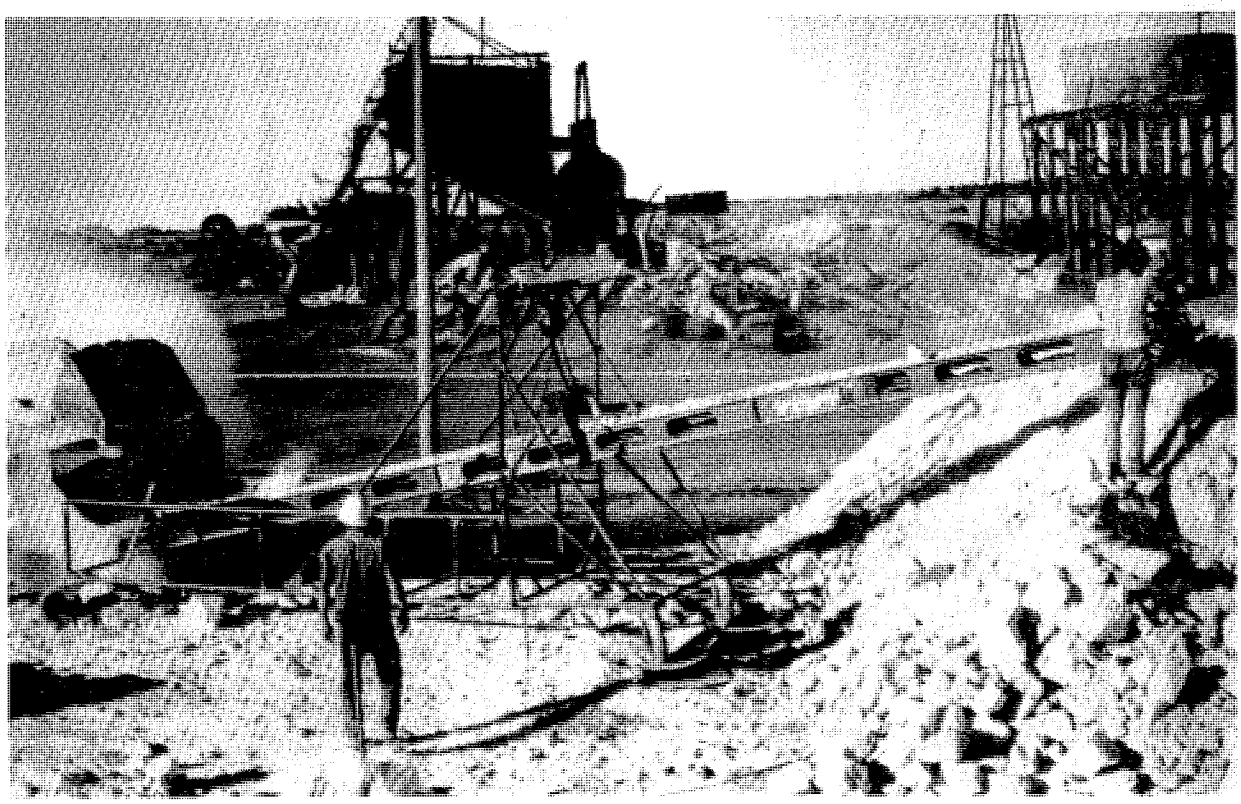

FIGURE 3. Manual conveyor stacker capable of placing 8 tons per hour of mine-run rock.

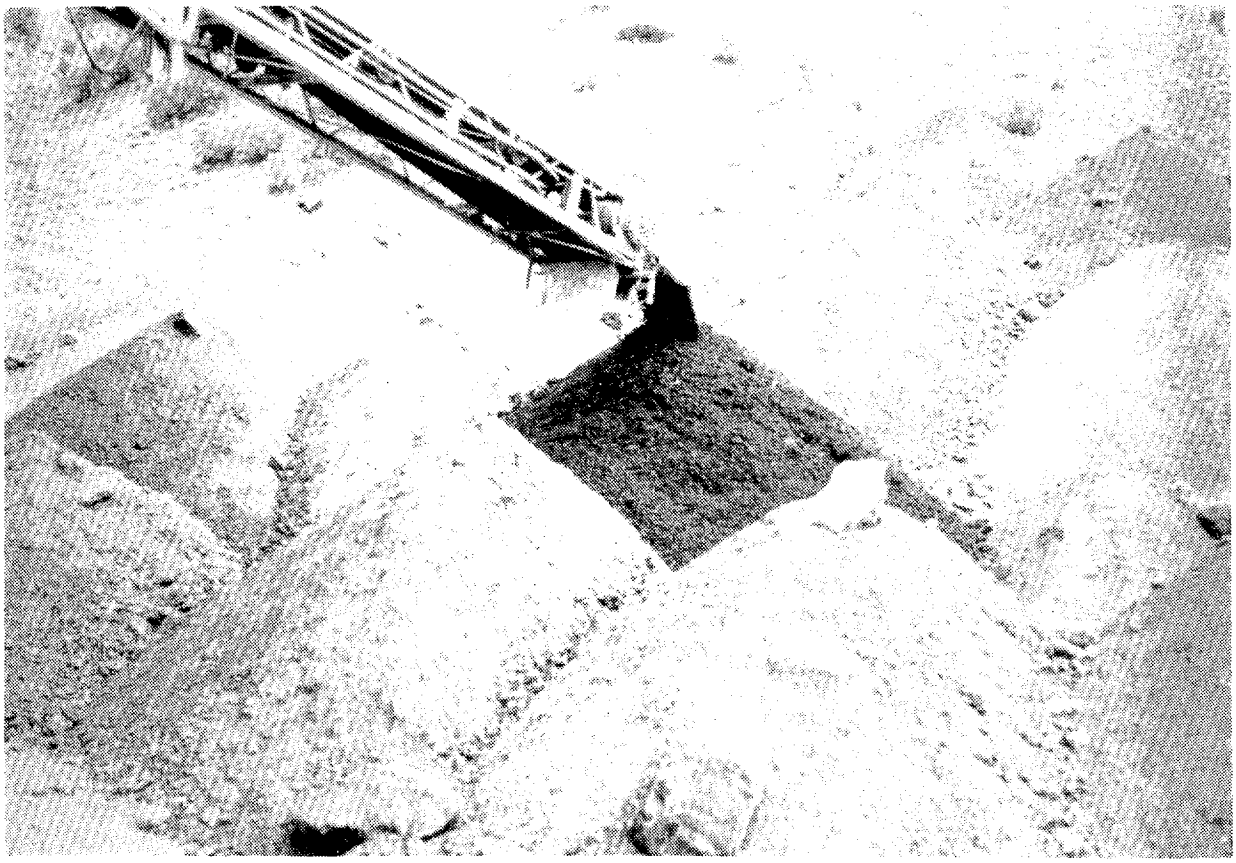

FIGURE 4. Conveying action builds cones of fine material surrounded by a permeable shell of coarse material. The heap is constructed of several layers of these cones. The conveyor stacker shown here is small, but completely mechanized and capable of stacking up to 20 tons per hour.

directly to a smelter).

- Silver recovery is generally in the range 45 - 60%, whereas gold recovery is generally in the range 65 - 80%.

Rock Testing Procedures

Suitability of a deposit for heap leaching depends on the following criteria:

1. Permeability of individual rock fragments - i.e. does the ore have to be crushed to achieve target recovery?
2. Cyanide and lime consumption by chemical reaction over the life of the leach.
3. Total leach time required - a function of the size of gold particles and the size of rock fragments.
4. Permeability of the heap - how high can the rock be stacked? - do special mining or heap loading methods have to be used? - can solution be distributed onto the heap by ponding, or is sprinkling necessary?

The hardest queation to answer is the last. It is not possible to build a laboratory-size heap that is constructed using the same equipment as a much larger field heap, and compaction and pulverization of the rock during construction can greatly affect permeability. The answer to this question can sometimes be arrived at by comparison with heaps built from similar rock types; but if this is not satisfactory, then it is necessary to build large semi-production field test heaps. The importance of using a correct heap loading method cannot be stressed too highly. Many operations, some with very high-grade ores, have failed because this has been ignored.

The first three questions can all be answered in laboratory tests. For a "typical" property containing one or two ore "types", lab testing will require 500 pounds of ore sample, and cost less than US $3000 if done by a consulting laboratory. Laboratory apparatus for the most common type of testing is shown in the photograph of figure 5 and the schematic drawing, figure 6. The apparatus is simple to build and

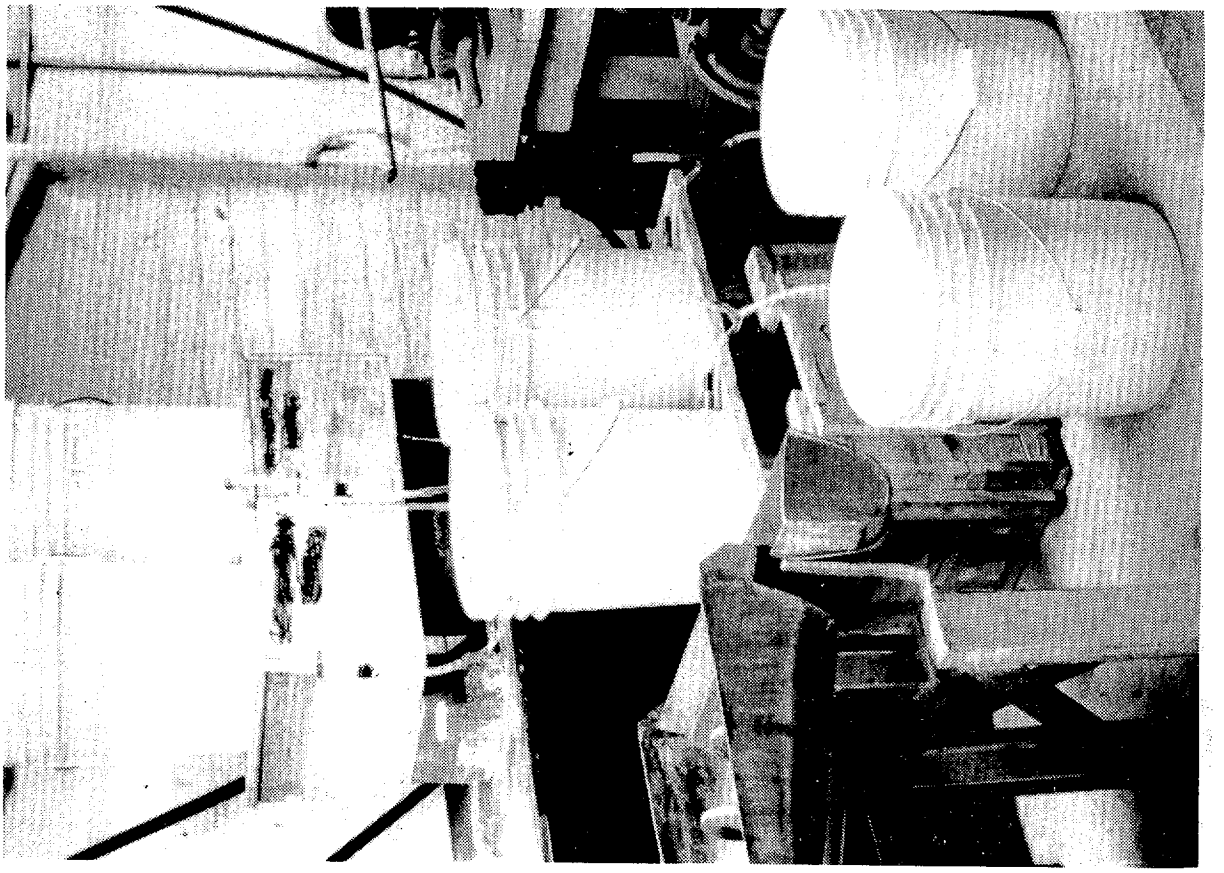

FIGURE 5. Laboratory leach test apparatus.

FIGURE 7. "Wobbler" sprinklers are easily fabricated from plastic fittings.

operate. Generally, the tests are run for two to three months, with the floor tank solution recycled to the head tank once each day. Solutions are assayed for cyanide and lime content, and adjusted, once every few days. The small bottle of activated carbon which is shown in figure 5 absorbs all the gold and silver as soon as it is dissolved from the rock. The carbon is removed and replaced once per week, and is fire-assayed to determine recovery. A test apparatus of this type can be used to examine such parameters as the effect of crushing the rock to various sizes, and the effects of changing cyanide strength, pH values, flowrates, etc. The central leach "column" can be replaced with a column up to 10 feet or more high where chemical reactivity might be a problem, or with wider deeper columns for testing very large rock fragments.

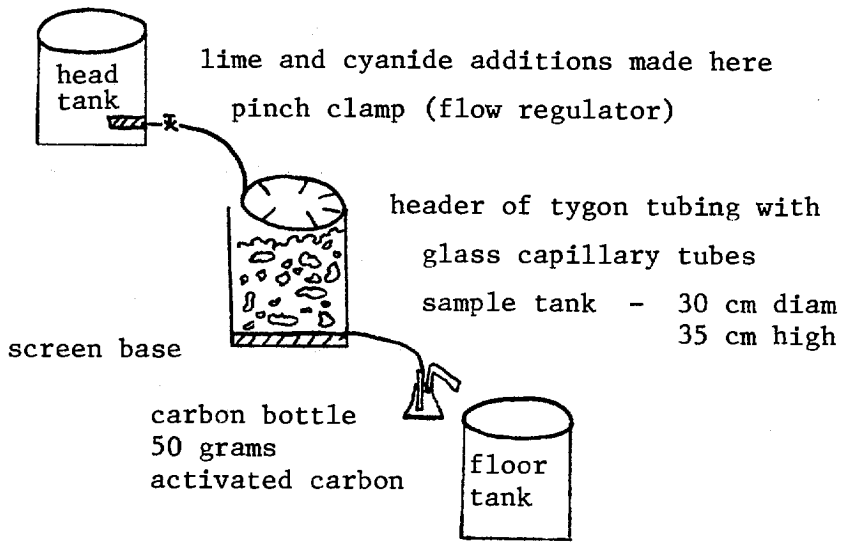

FIGURE 6. Leach Test Apparatus

Pad Materials

The impermeable pad on which the ore is stacked may be made of many materials including clay, asphalt, concrete, and plastic sheeting. Some heaps have even been placed on an unprepared ground surface, though this almost always results in a very low recovery.

In one configuration of the heap leach process, a limited ground area is covered with expensive high-strength pads usually constructed of 20 cm thick asphalt. Ore is stacked onto the pads, leached for a short period (ususally less than 60 days), and then removed with a front end loader. The pads are reused for a new ore charge. The biggest disadvantage of this system is that the pads must be cleanded off at pre-determined intervals to make room for new ore. With such a rigid time restriction, all the inherent flexibility of the heap leaching process is lost.

A much better procedure is to install inexpensive "single-use" pads, and then to leach the heaps as long as they are returning values in excess of operating costs (in some cases, for longer than a year). New heaps are built along side the older heaps. In environmentally sensitive areas, the discontinued heaps can be sealed with a clay cover, or contoured and revegetated.

Pads made of asphalt, concrete, or clay all share the same problem - to be impermeable, they must be well constructed. Specialized compactors and other equipment are needed, and these are often not available to the small miner.

Single-use pads can be made quickly and enexpensively with plastic membranes. There are several different types available, but all have similar properties. The criteria for selection are: 1) highest puncture resistance at lowest cost; and 2) the ease with which pieces of the material can be joined together <u>under field conditions</u> to make impermeable splices. The best materials currently on the market are

hypalon and polyolefins, such as duPont 3110. Hypalon is more puncture resistant than the polyolefins, but it is more expensive and more difficult to field weld.

Typical costs for the pad materials renge from US $3.00 to US $4.00 per square meter. The maximum amount of ore that will fit onto the pad for a typical 10,000 ton heap is approximately 4 tons of ore per square meter of pad material (including the material used to line the ponds). For very large heaps the pad loading factor increases to 8 or more tons per square meter.

Preparation of the ground surface under the pad is done by clearing plant growth and roots from a fairly firm, slightly sloping ground surface. If a small bulldozer is available, a three-man crew can prepare the surface for a 10,000 ton heap, lay the plastic and weld all the seams, and otherwise have the pad ready for loading rock within 4 days. Another 3 days are required to excavate and line the solution holding ponds.

Since all plastic pad materials can be punctured by sharp edges of large rocks, the pad is usually covered with a four to eight inch thick layer of fine gravel. The gravel cover also provides a good drainage base under the heap. For a 10,000 ton heap, 500 tons of gravel, screened to 2.0 cm or smaller (and with all the minus 100 mesh fines removed) would be required. The gravel can usually be produced from the crushed ore.

Solution Distribution

Solution is distributed onto the heap for at least eight hours each day, at the rate of six liters per hour per square meter of heap surface. Distribution is accomplished either by ponds or by sprinklers. Ponding is the simplest method, and allows the heap to be covered for protection against freezing or excess rainfall, however,

it cannot always be used for two reasons:

1. If the heap contains a mixture of coarse and fine rocks with a fairly high percentage of clay, solutions "channel" easily within the heap. The use of ponds results in very inefficient solution flow.
2. Ponds can only be built on the top (flat) surface of the heap. On small heaps, a large portion of the total tonnage lies under the side slopes, and sprinkling is needed to leach this tonnage.

A system of sprinklers is more complicated and requires more maintenance than ponding, but it guarantees good solution distribution. Sprinklers such as those shown in Figure 7 can be made quite simply by inserting a 25 cm length of rubber tubing over a tapered plastic pipe nipple, and forcing this assembly into a small hole drilled into a network of distribution pipes on the heap. Each of the piping systems shown in Figure 8 consists of a network of 2.5 cm diameter PVC pipes for a 2000 ton heap. Each system, including the sprinkler heads, was installed by two men working two days. Only four days would be required to place an identical system on a 10,000 ton heap. Total cost of materials for the sprinkler heads for the 10,000 ton heap would be US $75.00. Other items needed for the sprinkler system, including 700 meters of 3 cm diameter plastic pipe, and a 3/4 horsepower pump, would cost approximately US $1500.00.

Solution Flow and Chemical Control

When solution is first distributed onto a new heap, it may take from three to five days before flow begins to exit from the base. A normal heap will absorb 50 to 80 liters of solution per ton of rock during this period, and thereafter water need be added only to make up for that lost by evaporation.

The leach solution is a dilute solution of sodium cyanide in which the cyanide is made stable by keeping the solution alkaline. The

cyanide strength is 0.5 to 1.0 grams sodium cyanide per liter, and the alkalinity is adjusted to pH 10.0 - 10.5 using calcium hydroxide. For the first several days of operation of a new heap the existing solutions are normally depleted in both lime and cyanide, but can contain from 0.3 to 15 or more ppm gold.

In the operation of a "typical" heap, the gold concentration of the solution exiting the heap reaches a peak after 5 to 10 days, and thereafter slowly declines. Operation of a heap is usually discontinued when gold content drops below 0.1 ppm.

The pregnant solution exiting the heap is collected in a surge pond, and from there pumped through tanks or columns filled with enough granular activated carbon to adsorb the gold and silver. Total flowrate for a 10,000 ton heap is approximately 150 liters per minute. The barren solution from the adsorption columns, and sufficient fresh water to maintain a convenient working volume of solution, flow into the barren solution storage pond.

Reagent additions can be made very simply. Calcium hydroxide and sodium cyanide are usually delivered to the project in steel drums or paper bags, and the solid chemicals are added to the circuit by dumping them directly into the barren pond. For a 10,000 ton heap, chemical addition involves the dumping of a maximum of ten 20-liter buckets of chemicals into the pond each day, an operation which takes less than one hour.

Because the ponds and the heap contain a fairly large reservoir of solution, assaying control for pH and cyanide content is not critical. One or two assays per day is usually sufficient. pH assays can be done quite simpky with a battery-operated pH meter, and assay for cyanide content is performed by a field titration. Total cost of the field assay facilities is US $200.

Assaying for gold and silver content requires more elaborate equipment. An atomic absorption machine, or a good fire assay facility, is required. Minimum cost for this apparatus and a laboratory to house it is US $20,000. While it would be nice to have gold assay capabilities on site, it is not absolutely essential. In regard to the adsorption columns, gold and silver assays are needed only to monitor the rate at which carbon must be removed and replaced in the columns, and for this purpose a delay of three to five days between taking the sample and receiving the assay can be tolerated. Efficient operation of the carbon stripping circuit requires closer assay control, but this circuit can be easily overdesigned so that efficient operation is not required.

Design of the Adsorption Columns

When cyanide solution containing gold is placed into contact with activated carbon, the gold cyanide chemical complex is adsorbed onto the surfaces of the carbon. The reaction is highly efficient - when solution containing 3 ppm gold is pumped continuously through a four-stage column system, five minutes contact time is sufficient to remove 98% of the gold. Simultaneously, the carbon can be "loaded" to 18,000 ppm gold (over 500 ounces per ton). Other than removal of precious metals, the solution is not chemically affected and can be recycled back onto the heap.

For a 10,000 ton heap, the adsorption system is comprised of four columns or tanks 50 cm diameter and two meters high. Auxiliary facilities include an air compressor, and a washing column to remove fines from new carbon. The columns are interconnected with a system of 5 cm diameter pipes with each other and with two pumps. Figure 9 shows a portable unit containing 30 cm diameter columns, which was built for a total cost of US $11,000.

The adsorption system can be built at several levels of sophistication. For a well-engineered portable circuit which is pre-assembled in an industrial center, and which can be operated safely in the field by totally inexperienced personnel, the capital cost for a 10,000 ton unit would be in the range of US $30,000. At the other extreme, a circuit could be assembled on-site using a maximum of locally available materials. Total cost for the "imported" items for such a circuit, including all pumps, would be less than US $5000. Four man-weeks of labor by skilled craftsmen would be needed to construct it. This inexpensive "homemade" circuit would require operators with a fair degree of technical skill, for the following reason: a 200 ton of ore per day operation (10,000 ton heap), treating ore containing 3 ppm gold and 3 ppm silver, would recycle 120 Kg of carbon each day; the chance for carbon spillage and gold loss would be quite high if the system was crudely contructed and the operators were unskilled.

Design of the Stripping Circuit

The opening paragraph in the previous section of this paper described the adsorption of gold onto carbon. The reaction is highly reversible. The loaded carbon can be stripped of nearly all its gold by contacting it with a near-boiling solution of 1% sodium hydroxide containing 20% alcohol. Stripping of heavily loaded carbon can be accomplished within 20 hours. The resulting solution from this operation is so heavily loaded with gold (and silver, if it is present in the ore), that the metals can be recovered from it in a simple electrolytic cell. The electrolytic cell is equipped with a cathode of steel wool, and after a batch of carbon is stripped the steel wool is removed from the cell, placed in a smelting furnace with suitable fluxes, and melted to a dore bullion. The carbon, now stripped of its gold, is ready to adsorb more gold and can be returned to the field columns.

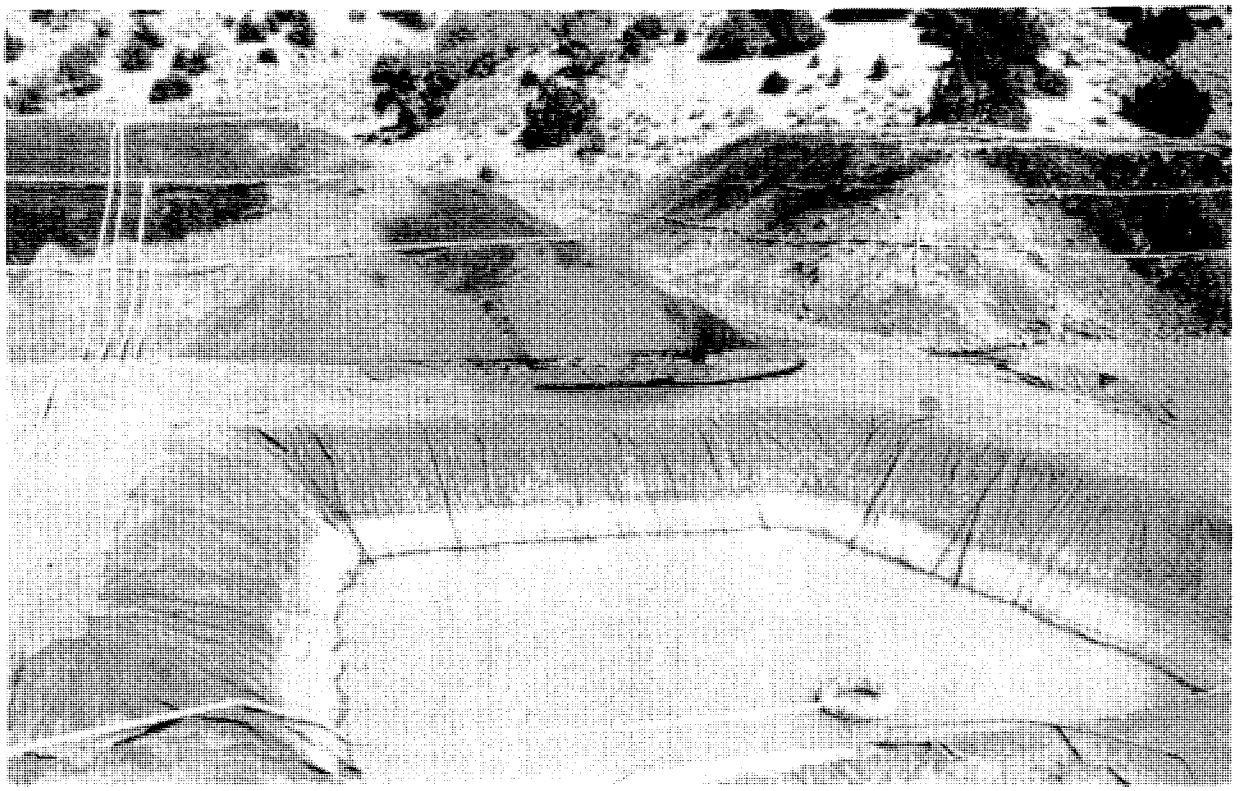

FIGURE 8. Solution distribution piping for two 2000 ton heaps.

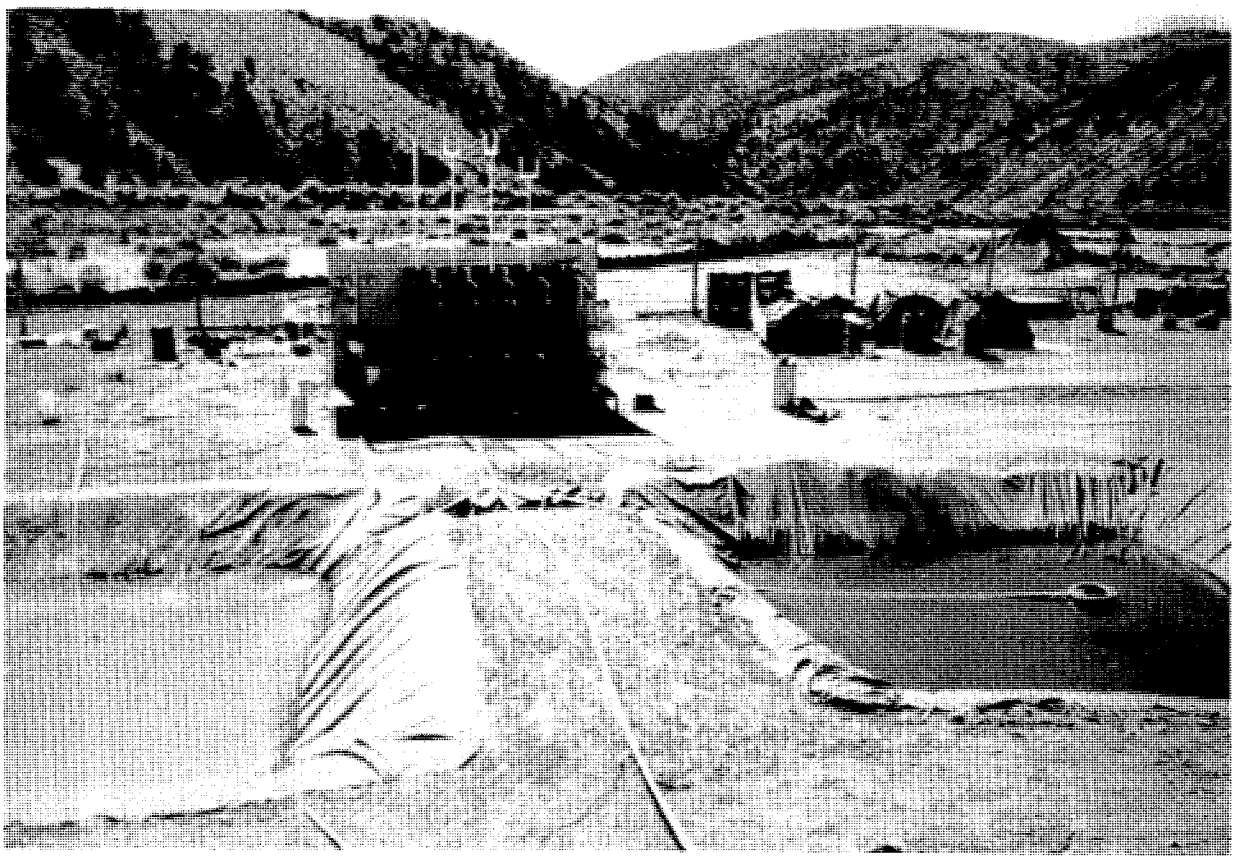

FIGURE 9. Carbon adsorption columns for a 2000 ton heap. The adsorption system for a 10,000 ton heap would be identical, except that the columns would be 50 cm diameter instead of 30 cm.

The stripping circuit is less complicated than the adsorption circuit. It consists of one stripping tank approximately twice as large as the adsorption tanks, a second small solution storage and heating tank with an electric or fuel-fired heater, one small pump, and an electrolytic cell and power supply. Figure 10 shows an extremely crude stripping circuit, fabricated at a cost of less than US $1500, which was capable of recovering 3 ounces of gold per 24 hours.

The stripping solution contains alcohol and is maintained near 90° C, so plastic parts should not be used, but mild steel is quite satisfactory. It is best to fabricate this circuit in an industrial center and transport it to the site. The cost of the stripping circuit for a 200 ton/day operation on a gold ore, including the power supply, is approximately US $10,000. The cost of the stripping circuit for a similar operation treating silver ores would be much higher, since much more carbon must be recycled per unit of value recovered.

The product from the stripping circuit is a mass of steel wool which contains more than its own weight of gold plus silver. It can be melted on site, or packaged and sold directly to a smelter.

Summary of a Total Project

This paper has provided some details and costs of the major technical items needed for a small heap leach installation, but of course there are many additional items needed such as access roads and housing, fencing to protect the ponds from wild animals, office and storage buildings, water well and power supply. As earlier indicated, none of the mining facilities have been included.

For a 200 ton per day operation, the leaching operations would require a well producing 900 liters per hour of water, and a total electric

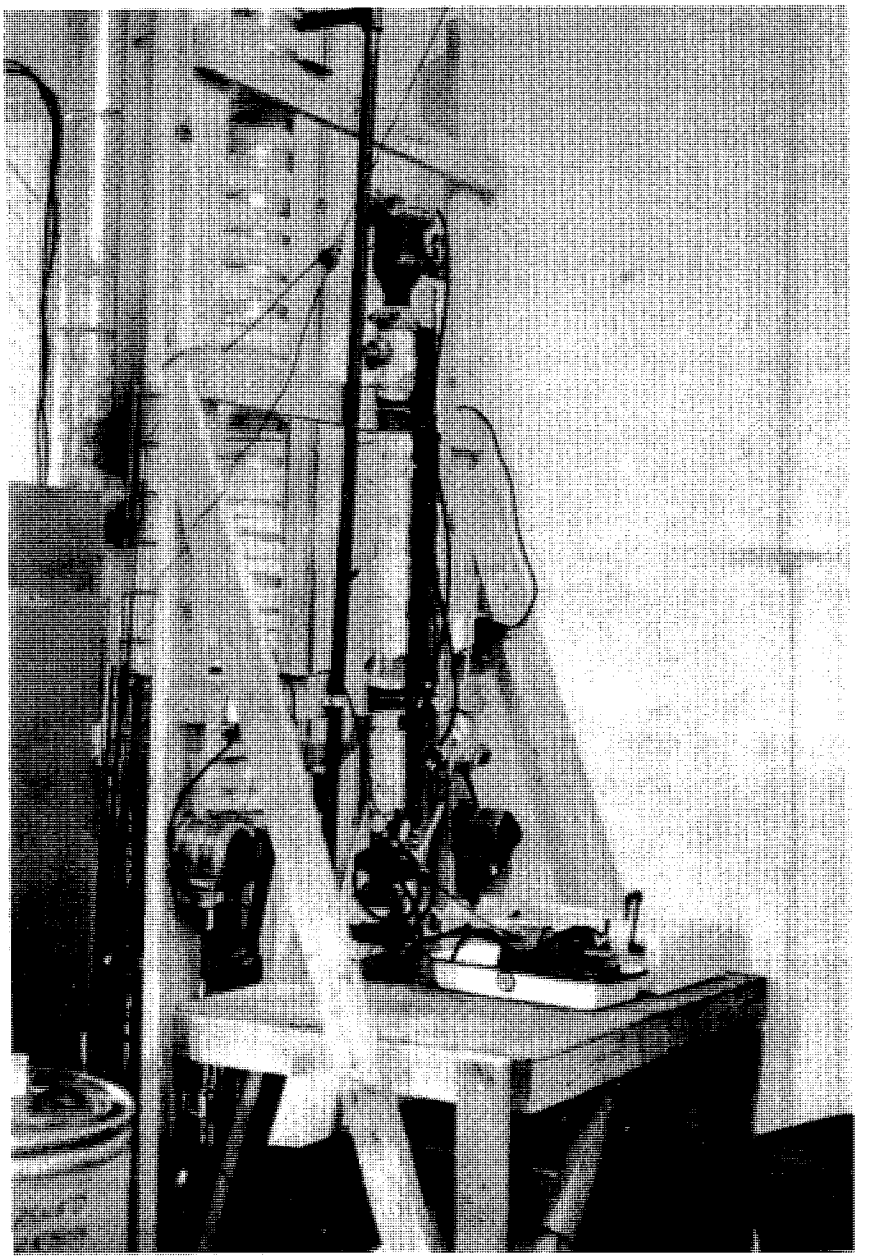

FIGURE 10. Crude stripping circuit capable of recovering over 3 ounces gold per day.

generating capacity of approximately 20 kilowatts (energy consumption for pumping of leach solutions is 6 kilowatts).

Manpower requirements for all of the ore stacking and leaching operations include six laborers, a skilled foreman, and a professional technically-trained manager. Two additional skilled personnel would be needed part-time for assaying and carbon stripping, but this work can sometimes be contracted out to a commercial laboratory.

In the introduction to this paper, a total cost of US $200,000 was presented as the cost of the first 10,000 heap. That figure is probably a little low; estimated values for the individual cost items are shown in the list below.

Pad Material	$	8,000
Gravel Cover		2,000
Heavy Equipment Rental for grading pads and ponds, hauling and stacking ore; bulldozer - 5 days; truck, end-loader, stacker - 50 days		40,000
Columns and Stripping Plant		40,000
Heap Piping		2,000
Chemicals and Carbon		15,000
Well, Pump, Generator, Wires, Pipes		8,000
Misc. support items, pick-up truck, fences, temp. buildings		25,000
Consulting Supervision, Engineering Design Fees and Expenses, Assaying		40,000
Labor, 6 men for 5 months*		30,000
Total	US$	210,000

*1978 labor cost in the western U.S. would be higher, perhaps $60,000.

Addendum

In the spirit of this conference, I tried to visualize the tiniest heap leach operation which might be reasonably undertaken. Where labor costs are very low, the process could be applied on a very tiny scale. It would be possible to build and operate a heap leach using - as the only piece of mechanical equipment - a bicycle-driven pump.

For example, one man could manually "mine" and haul ore from an old, low-grade mine dump to a nearby heap at the rate of one ton per day. The rock would be leached for a total of 60 days, so at any time there would be an active heap of 60 tons. Sometime during each day, a total of 1500 liters of solution would have to be pumped from a storage pond onto the heap, a vertical distance of 10 meters. The solution could be distributed onto the heap by drip pipes, or by another person spreading it form the end of a garden hose. Pumping onto the heap could probably be done in less than two hours each day.

The solution would slowly run out the base of the heap into a collecting pond. During another 2-hour pumping period each day, the solution from the collecting pond would be pumped into the top of the first of a series of three oil drums filled with activated carbon. The drums would be set on a stand, with approximately one-foot elevation differences between them, so the solution would flow by gravity through each of them in turn and then into a second storage pond. New water would have to be supplied to the system at the rate of 100 liters per day.

Using the above system, a one ton per day operation treating ore containing 0.1 ounce gold per ton (3.5 ppm), with a recovery of 70%, would generate US $4900 worth of gold per year (gold at $200/ounce). The gold would be recovered onto a total of less than 80 Kg of

activated carbon (coarse grains of charcoal), which could easily be packaged up and shipped at the rate of 7 Kg per month to a conventional base-metal smelter. Total annual operating costs would be US $900 for carbon, sodium cyanide, lime, and plastic pad material. Initial setup costs would be about US $2000, including the first six month's supply of plastic and chemicals.

Unfortunately, the above scenario leaves out the most important element - the need for some technical knowledge. Even in fairly educated societies such as the western U.S., that knowledge is generally beyond the capability of the individual miner.

To set up the one ton per day operation, two week's technical assistance would have to be provided. Thereafter, one day's assistance every two weeks would have to be available to keep things operating successfully. Even if several properties were being leached in the same geographic locality, it is doubtful that they could carry the cost of the technical assistance and still show a reasonable profit.

Perhaps, in some cases the establishment of a small leaching industry could be justified on government policy grounds, and the technical assistance could be provided free of charge to the small miner. The problem may still not be solved, however: an uneducated gold miner working a small labor-intensive mine is usually accustomed to seeing a small pile of glistening yellow metal as his reward for a day's work. It may be difficult to convince him to continue the routine of mining rock and pumping solution, when all he ever sees is unchanging black "charcoal" for which he gets paid according to some mysterious formula.

James D. Sell

PRECIOUS METALS HEAP LEACHING
SIMPLE - WHY NOT SUCCESSFUL?

A Talk for Presentation to

the

Northwest Mining Association

December 7, 1979

by

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PRECIOUS METALS HEAP LEACHING
SIMPLE - WHY NOT SUCCESSFUL?

by

Daniel W. Kappes

Heap leaching for the recovery of precious metals starts by dumping piles of mined ore on an impervious base, and sprinkling it with cyanide solution to dissolve the gold and silver. Once a heap is started, the leach solutions are merely recycled through the heap and then through a simple metals extraction circuit, for several days or months until the "target" recovery is achieved. For an operating heap of several thousand tons, the only moving parts might be two pumps and an air compressor.

What could be simpler? Surprisingly, of 22 heap leach operations included on a "quick-and-dirty" list made in preparation for this talk, eleven have been failures. Since the list contains several well-publicized, very successful leaches, the failure rate no doubt exceeds 50%.

Maybe some sinister space-warp brings a small corner of the Bermuda Triangle into every heap-leacher's backyard. Or perhaps, appearances are deceptive: Is the process full of unknowables which trap the unwary engineer?

Fortunately, the Bermuda Triangle theory doesn't bear up under close examination. And while there are some unknowables, they appear to account for few if any of the failures.

Much of the problem lies with the inherent nature of the heap leach process. It is ideal for use on the small property, by the small company. Three major problems can face the small company:

1. It can't afford to follow the proper technical path;
2. It can't distinguish between good technical advice and bad; or it just can't find the good;
3. It can't manage the good technical help once it has found it: promoters, managers and technicians are rarely a compatible mix.

In big corporations, the money is usually available, and the personality problems are usually alleviated by inserting one or more layers of technical management between the "technicians" and the "promoters". In small companies, there is often no middle ground: the promoter either makes too many of the technical operating decisions, or else he gives "carte blanche" to the technical operator.

For something specialized like a heap leach, the technical operators are usually drawn from the ranks of consultants, of which there are three types:

1. Those who do a good job;
2. Those with too many irons in the fire;
3. Those with too few marbles in the head.

Perhaps all consultants fall, at one time or another, into every category. What's the answer for the small company? There may not be one, but a few good guidelines to follow are:

1. Especially in the early conceptual stages of heap leach design, get at least two independent studies which include property review, systems design, and overall costs. Accept the one that's most conservative, or form a compromise that doesn't defy logic.
2. Select a technical manager who accepts the compromise design, and who is willing to commit himself exclusively to the project.
3. Don't change technical plans half-way into a project for reasons of management or budget "expediency".

Of course, blaming all the failures on management is a nice way to avoid the technical issues. The rest of this paper presents some of the technical questions to be answered in making the "heap leach" decision.

1. WHAT SHOULD THE OVERALL PLAN BE?

The development of the property should follow a series of well-defined steps, including those listed below:

a. Find some ore. This step should be obvious, but it apparently isn't. Heap leaching may be inexpensive, but it isn't free. At least three operations have failed because the heaps were built with material that wasn't ore even at the projected operating costs!

b. Evaluate the metallurgical geology. Metallurgical testing is only as good as the samples tested. Before testing begins, try to segregate the deposit into blocks according to differences in: mineralogy, extent of oxidation, clay content, and permeability (as measured by water drop absorption). Test each block separately, or at least test a good composite sample.

c. Perform sufficient laboratory tests. These tests may include small bucket leach tests on one or more crushed ore sizes, and high-column leach tests.

d. Perform a field test, if chemical reactivity or permeability problems are expected.

e. Allow adequate time and money to begin production. One medium-sized operation began full production 6 months after making the decision to proceed, but some important decisions were poorly made and costs were high. Generally allow one year from the beginning of the lab test program, and an additional 6 months if a large field test is needed.

2. WHAT CAN SMALL BUCKET TESTS TELL?

The first phase of the testing program usually consists of bucket leach tests on 50-pound samples of ore, crushed to various sizes. The tests are run for 60 days or longer, and they can be used to determine a variety of factors including those in the following list:

a. Recovery as a function of leach time and rock size, which is important in deciding whether or not to crush the ore.

b. The recoverable silver-to-gold ratio, which must be known for the selection and design of recovery systems.

c. Rock acidity and soluble base-matals content, factors which influence the heap height, and the selection and amount of leach chemicals required.

d. Obvious percolation problems; though the lab tests can't identify the seriousness of the problem, they can indicate that careful heap loading methods might be needed.

Laboratory tests have their limitations. They can't, for instance, tell much about the important decision to open-pit mine using ripping instead of drilling and blasting; and they can't say much about how ores with percolation problems should be loaded onto the heap.

3. ARE TALL COLUMN TESTS NECESSARY?

Tests can also be run in narrow laboratory columns up to 20 feet or more high. These tests can be important in determining chemical reactivity problems or giving further information about permeability problems. If the small bucket tests indicate such problems, tall column tests are absolutely necessary; they are a good idea for all ores.

4. HOW MUCH TIME AND MONEY SHOULD BE ALLOWED FOR THE LAB TESTS?

The lab testing program on a normal ore will cost approximately $3500 per sample. Since the bucket tests will usually run for 60 days, at least four months should be allowed for completion of testing. The questions to be answered during the lab test program are important, and it should not be by-passed.

5. IS FIELD TESTING NECESSARY?

Field tests, even small ones, are very expensive and are not always an absolute necessity. For ores with no permeability or chemical reactivity problems, and which are being leached at crushed sizes, the recovery from large field heaps of several hundred or thousand tons is very closely predicted by 50-pound laboratory bucket tests.

Figure 1 shows typical recovery curves, for a soft ore (nearly pure clay), and for a much harder ore. It's interesting to note that ores with heap permeability problems often show the highest laboratory recoveries. The field results can be disappointing unless the heap is built so that it percolates uniformly and at an acceptable rate.

Field tests should always be run where:

a. samples for laboratory tests can't be representative either geologically, or because of the rock size distribution (for instance if the rock is to be leached without crushing);

b. the ore is soft, and permeability might be seriously affected by the method of mining or of loading the field heap;

c. the ore shows chemical reactivity problems by consuming lime or cyanide in the lab tests.

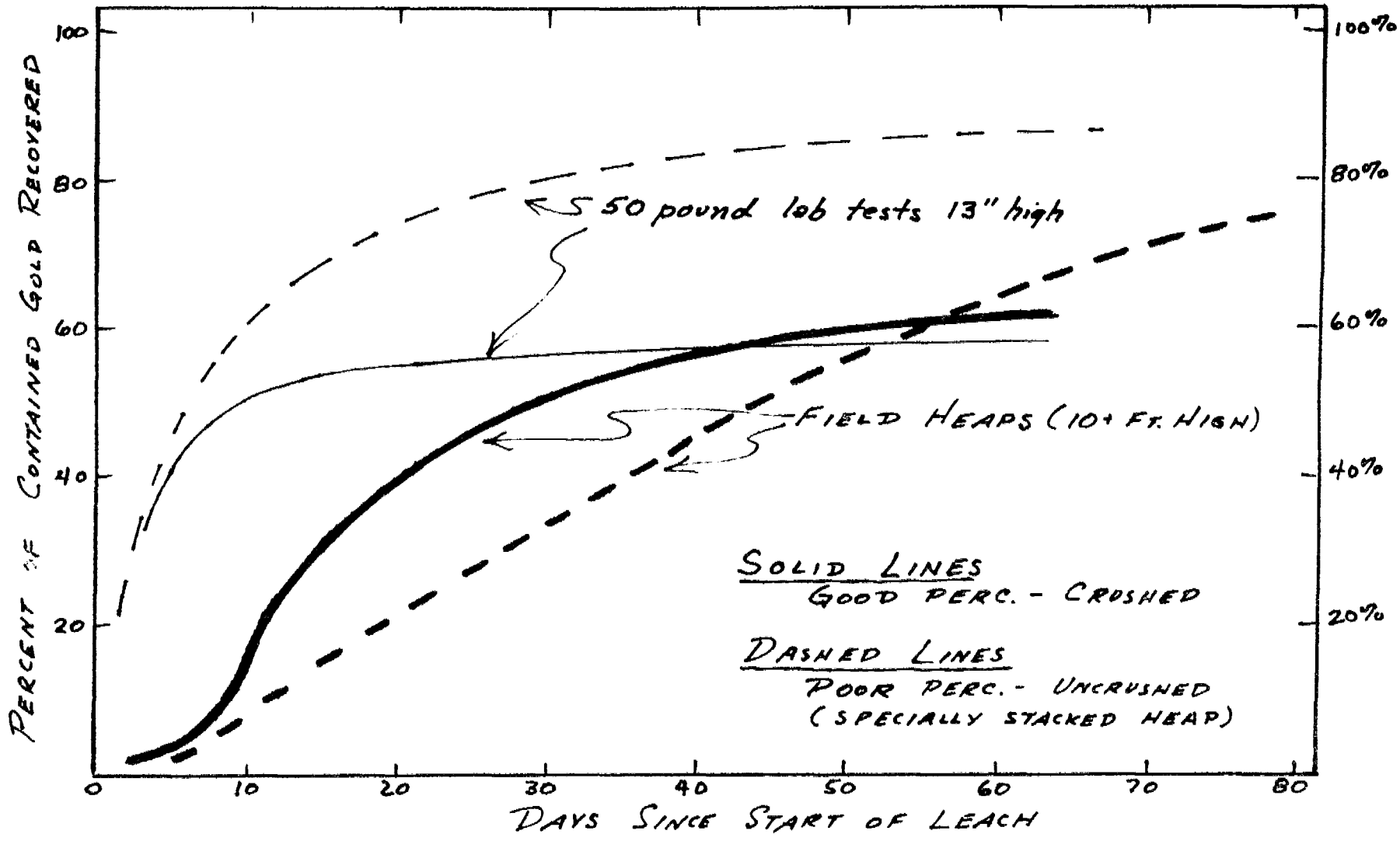

FIGURE 1. TYPICAL LAB AND FIELD TEST RECOVERY CURVES FOR CHEMICALLY UNREACTIVE ORES

If field tests are run, they should be run on heaps large enough to contain at least 50% of the tonnage under the highest part of the heap, and the heap should be about the same height as subsequent production heaps. Thus, for an operation with 12-foot-high heaps, the minimum test size is about 1500 tons.

6. WHAT IF FIELD TESTS ARE FINANCIALLY IMPOSSIBLE?

One of the most common approaches to leaching by the small company is to plan on using the first "production" heap as the test heap. Unfortunately, while lip service is paid to its "test" function, almost everything else about its operation implies that it is hardly a test. It is important that the small company does not fool itself about the test function. The most important achievement is to learn to recognize a test:

- It's a test ONLY if the product isn't needed to pay the bills.
- It's a test ONLY if operations stop until the results can be evaluated (which is usually a minimum of 4 months after solution application begins).

For many small deposits, field tests are very difficult to justify. The tests are very expensive. Costs start at $50,000 and may exceed $200,000 for one test on a few thousand tons of mined, crushed rock.

There are some alternatives if the field tests can't be run:

a. If the lab tests indicate severe percolation problems, find another deposit.

b. For moderate percolation problems, PLAY IT AS SAFE AS POSSIBLE. Stack the first production heaps low, less than 10 feet high, using equipment that doesn't have to be driven on the heap. Low heaps have higher costs for pads and recovery systems, but the extra costs are seldom prohibitive.

c. Mine the ore by drilling and blasting. Ripping with a bulldozer might be less expensive, but it's much more difficult to predict the rock sizes. Also, ripping usually creates a much higher percentage of pulverized rock than blasting.

7. SHOULD THE ROCK BE CRUSHED?

The lab tests will indicate if crushing is economically justifiable, and for some ores may show it to be necessary. For the small-tonnage deposit, portable crushers are usually available on a rental or contract-crushing basis; usually the charge includes a mobilization fee of $10-20,000 plus a per-ton charge.

Proper selection of the crusher is very important. Very often rental crushers aren't capable of the advertised output. Even when they work, adding the crushing "bottleneck" consumes already-short management time and greatly increases the risks for a small operation. It complicates even the very large operations. If the tradeoff is between crushed size and leaching time (for instance, 30 days versus 90 days), it is usually better not to crush. A good rule of thumb for the small operation: Avoid crushing if at all possible; if it's necessary. select a crushing system with PROVEN capabilities.

8. HOW SHOULD GOLD AND SILVER BE RECOVERED FROM SOLUTIONS?

There are three basic recovery systems which have been used on production operations, and all have been developed to the point where there is very little risk of their total failure. For ores which are almost pure silver, sodium sulfide precipitation can be used. The method is inexpensive and functional, but in the startup stages of the field leach there are usually lots of product handling problems to be solved, so extra technical help is required.

For silver, mixed silver-gold, or high-grade gold ores which leach rapidly, the traditional zinc precipitation methods may often be the best choice.

Activated carbon adsorption columns have been widely used on heap leaches, and have the advantages of low initial cost and negligible operating costs and problems. The system is ideally suited to low-grade or slow-leaching ores, especially if silver is absent. If silver constitutes over 20% of the recoverable dollar value (that is, if the silver to gold ratio exceeds 5 to 1), it is usually necessary to include a carbon stripping circuit as part of the field circuit.

8. HOW CRITICAL IS GOOD CONTROL OF THE MINING OPERATION?

It is NOT the goal of a heap leach operation to move large tonnages of rock inexpensively. Like any mining operation, grade control is important. Equally important, is the need to avoid mixing clay or carbonaceous waste rock with otherwise good ore on the heap.

Use of contractors for mining and hauling can have significant cost benefits for heap leach operations. The economic prospectuses of small operations often include a passage that reads something like "The contractor has provided a firm bid to mine and haul 20,000 tons of ore for $3.00 per ton".

Unfortunately, all too often the bid price does not take into account the difference between a mining operation and a dirt-moving operation. Bids are meaningless unless the contractor knows about and agrees to the restrictions on what and how to mine, sample and segregate ore from waste.

A few good guidelines to follow are:

a. Hold extensive discussions about the mining details BEFORE trying to establish a cost figure; make sure the details are at least in writing, if not in the formal contract.

b. Always design the leach operation to accomodate more than the planned-for tonnage. It is much nicer to be leaching waste along side of ore, than to be leaching waste while the ore is sitting in a stockpile.

9. HOW IMPORTANT IS THE METHOD OF LOADING THE HEAP?

Nearly all heap leach operations have experienced percolation problems at some time or another. At least five have failed because the heaps didn't percolate (for most of these, the problems would have been obvious if lab testing had been done).

ALWAYS give extremely careful thought to the choice of heap height, and the design of heap loading methods.

Like every endeavor in the mining industry, each heap leach operation has its own unique challenges. The list above offers simplistic guides to meeting some of them, but it's only an adjunct to good common sense. In the last analysis, the failures - and successes - of heap leaching result from a complicated interplay between the natural properties of ore deposits, the ever-changing economic climate, and the personalities of the people involved.

James D. Sell

SCALEUP EXPERIENCE IN GOLD-SILVER HEAP LEACHING

A Talk for Presentation to

the

AIME Annual Meeting
Chicago, Illinois

February, 1981

by

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SCALEUP EXPERIENCE IN GOLD-SILVER HEAP LEACHING

By
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A precious metals heap leach is a fairly straightforward mechanism. Ore is stacked on an impervious pad and is sprinkled with a pH-controlled cyanide solution. The solution dissolves gold (and/or silver) from the rock and carries it down and out of the heap. Gold is removed from the solution and the solution is recycled.

The prediction of field heap behavior from laboratory tests should be relatively straightforward, provided that the laboratory test procedures are faithful mini-heap leaches. Detailed comparisons of field heap behavior versus lab behavior now exist for several ores. They indicate that field heap performance differs significantly, but predictably, from the laboratory tests. Specifically, field heaps reach the same level of recovery as the lab tests, but they take from three to six times as long to get there.

The data suggests that all production heap leach systems should be designed with flexibility to extend leach times for as long as the ore requires. Fortunately, Aesop's lesson that slow is not necessarily bad, is one of the guiding principles of heap leaching. The capability for unlimited leach times is inherent in the heap leaching process, both technically and economically. Surprisingly, this capability has been "designed out" of several operations in which the ore is stacked, leached, and removed on regular intervals. The data presented here,

and evidence in the literature from other producing heap leaches such as Cortez and Round Mountain,[1] suggest that eliminating this flexibility greatly increases the investment risk.

Example 1. - The Extremely Long Delay: A Clayey Ore

Figure 1 shows recovery curves in lab and field tests for two ores. Both ores leach quickly in the lab; but in the field they are at the extremes of heap behavior - one leaches very fast, the other very slow.

The laboratory test curves in Figure 1 were developed in typical small laboratory bucket leach tests, containing 20 Kg of ore in a 34 cm deep and 28 cm diameter column. A drawing of the test layout and details of the test procedure have been published previously.[2]

Ore A is a soft, almost pure kaolinite clay containing fine gold. Ore "lumps" are stable and permeable, but ore "fines" can form clay layers that are almost impermeable. To minimize the fines problem, the ore for the field heap was carefully blasted and was not crushed before leaching. The field heap was intentionally built in a manner that would result in delayed, steady gold recovery by using a conveyor stacker to construct the heap as a series of 1 meter high cones. Within each cone, the ore fines created an almost stagnant "pocket" about 0.6 meters in diameter. Around each cone the "throwing" action of the conveyor stacker created a layer of coarse rock which allowed free-flow of solutions downward through the heap.

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1. For a good general article and bibliography on heap leaching, see "Design Factors for Heap Leaching Operation" by George M. Potter, Mining Engineering, March 1981, pp 277 - 281.

2. "Leaching of Small Gold and Silver Deposits" by D.W. Kappes, pp 381 - 388, in THE FUTURE OF SMALL SCALE MINING, edited by R.F. Meyer and J.F. Carmen, McGraw-Hill Publishing Co., 1981, 50 pp. Copies available from the author.

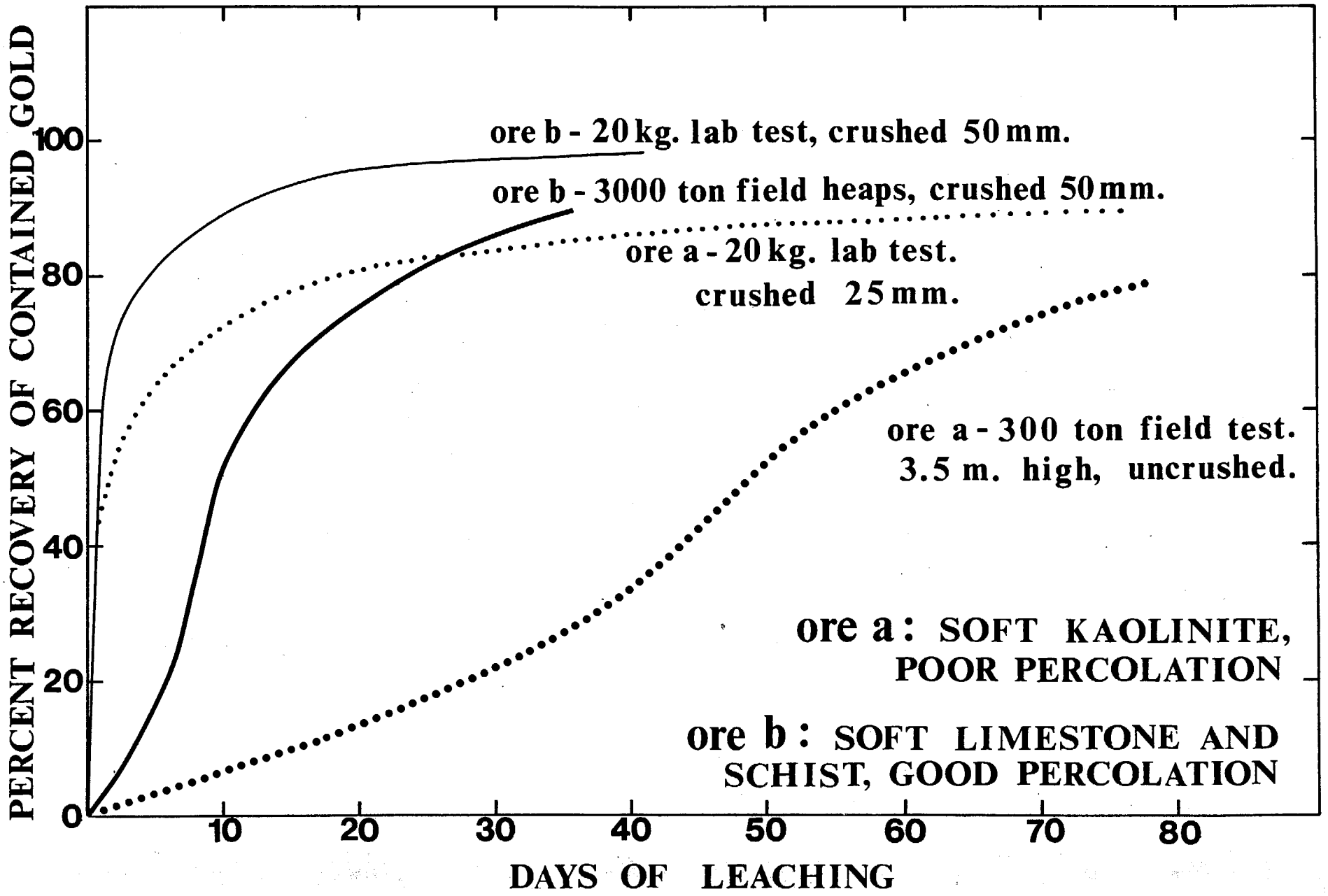

Figure 1: Comparison between lab and field leaches (two ores).

As the curves for ore A in Figure 1 show, 60 percent recovery was achieved in lab tests in ten days, whereas the same recovery took 55 days in the field. It is not too surprising that the field and lab recovery curves for this ore are different since the tests differed in two significant ways:

1. There is no way to take an accurate sample of run-of-mine sized rock for small lab tests (these tests were run on ore crushed to 50 mm).

2. The throwing and segregating actions of the conveyor could not be duplicated in the lab.

The first factor appears to have been not too significant for this ore. Large production heaps have since been run on crushed ore with recovery curves that duplicate that from the uncrushed field test heap. The controlling factor appears to be heap permeability, not rock size.

The second factor is part of a problem common to all laboratory test columns, which is, that there is no way to duplicate the "structure" that is built into the field heap. Ore cannot be compacted the same way it is in the field, and ore segregation, which results from rocks rolling down the heap slopes, cannot be duplicated.

Example 2. - The Short Delay

Ore B in Figure 1 is a mixture of permeable limestone and shale. Both the laboratory tests, and the field heap, were run on coarse-crushed ore.

The ore is hard enough so that ore fines do not clog the heap. As a result, it was possible to build the field heap as a vertical "stack"

of 1.9 meter lifts, to a final height of 3.9 meters, using trucks which drove directly on top of each ore lift.

Laboratory tests were run in short bucket tests and in 2.5 meter high columns, and no difference in recovery times was noted. As in all other tests reported here, flowrates of leach solution in the lab and field tests were roughly identical.

Like ore A, the laboratory tests on ore B appeared to be faithful "mini-heaps", except for one aspect: the important, unduplicatable factor of heap structure. As Figure 1 shows, the laboratory tests achieved 85 percent recovery in eight days. The field heap reached this level of recovery in thirty days. Subsequent production heaps have all behaved identically to the first heap.

Example 3. Ores That Require Crushing

The ore presented in Figure 2 is physically very different from either of the ores in Figure 1. This ore is a hard chert or jasperoid, in which very fine gold is apparently well disseminated. Recovery of the gold in laboratory tests is a straightforward function of the size to which the rock is crushed.

The field test curve shown in Figure 2 was generated from a 3.5 meter high field heap, with rock crushed to 8 mm. The ore in this heap was stacked using a conveyor stacker. There was no clay in the ore and no percolation problems could be detected in the field or in tall-column laboratory percolation tests.

As Figure 2 shows, the ultimate field recovery exactly matched the recoveries predicted from 13 cm high laboratory leach columns. However, as with ore B in Figure 1, there was a significant delay in field recoveries. Fifty percent recovery was achieved in eight days in the lab, versus twenty-seven days in the field.

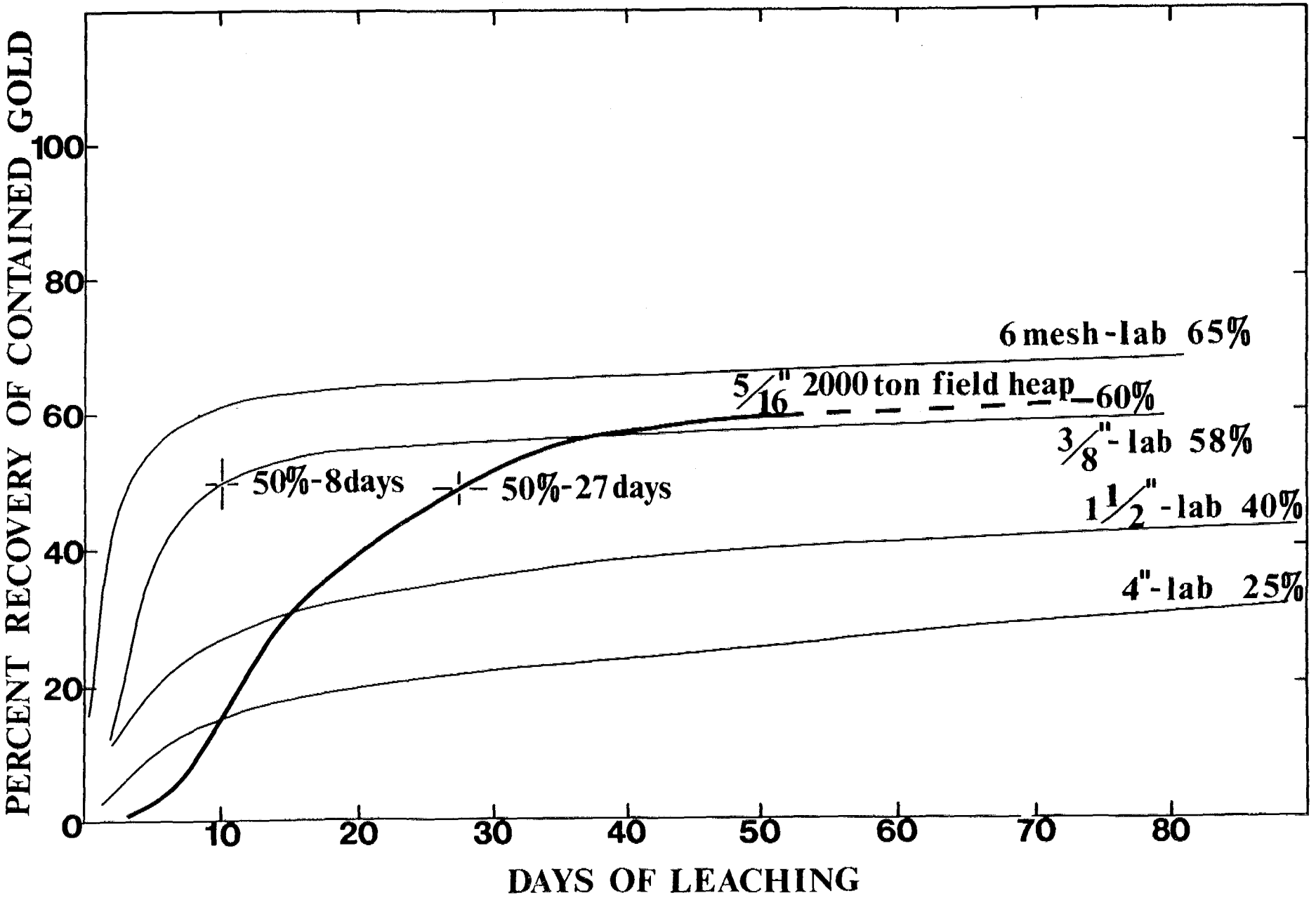

Figure 2: Comparison between lab and field leaches, crushed ore - no percolation problems.

The field delay appears to be a function of heap structure, not of heap height. This same ore was leached in a 6 meter high, 0.6 meter diameter laboratory column located at the USBM Reno Metallurgical Research Center. The recovery curves closely matched the small bucket tests rather than the field leaches.

Figure 3 shows a set of four gigantic columns, measuring 12.3 meters high, 1.25 meters diameter. The ore in these tests is "run-of-mine" size volcanic rock, with gold on fracture surfaces. The correlation for both ultimate recoveries and recovery times, between 13 cm high lab tests and these tall columns, has been very close.

Example 4. - Flowrate and Rock Size Variations

It is easy, in a paper such as this, to present only the data which clearly makes the author's point. The ore in Figure 4 tends to muddle things a bit.

This ore is a soft porous limestone. It is physically similar to ore B in Figure 1, but it breaks differently. Simple bulldozer ripping results in "run-of-mine" ore which is 90 percent smaller than 1-inch. Because of its fine, earthy appearance, the heap was stacked (with ripped rock) using a conveyor stacker, in 1 meter high cones, to a final depth of 3.5 meters. Subsequent laboratory and field percolation tests showed that percolation rates were very satisfactory.

Three laboratory curves are shown on Figure 4. The effect of drastically lowering the flowrate had only a minor effect on recovery. Selected large rocks leached slowly and not very well, but the ore in the field heap was mostly small. The "prediction" from the lab tests would be for a 60 percent recovery after approximately 12 days of leaching. It took 50 days of leaching to achieve the same recovery from the field heap.

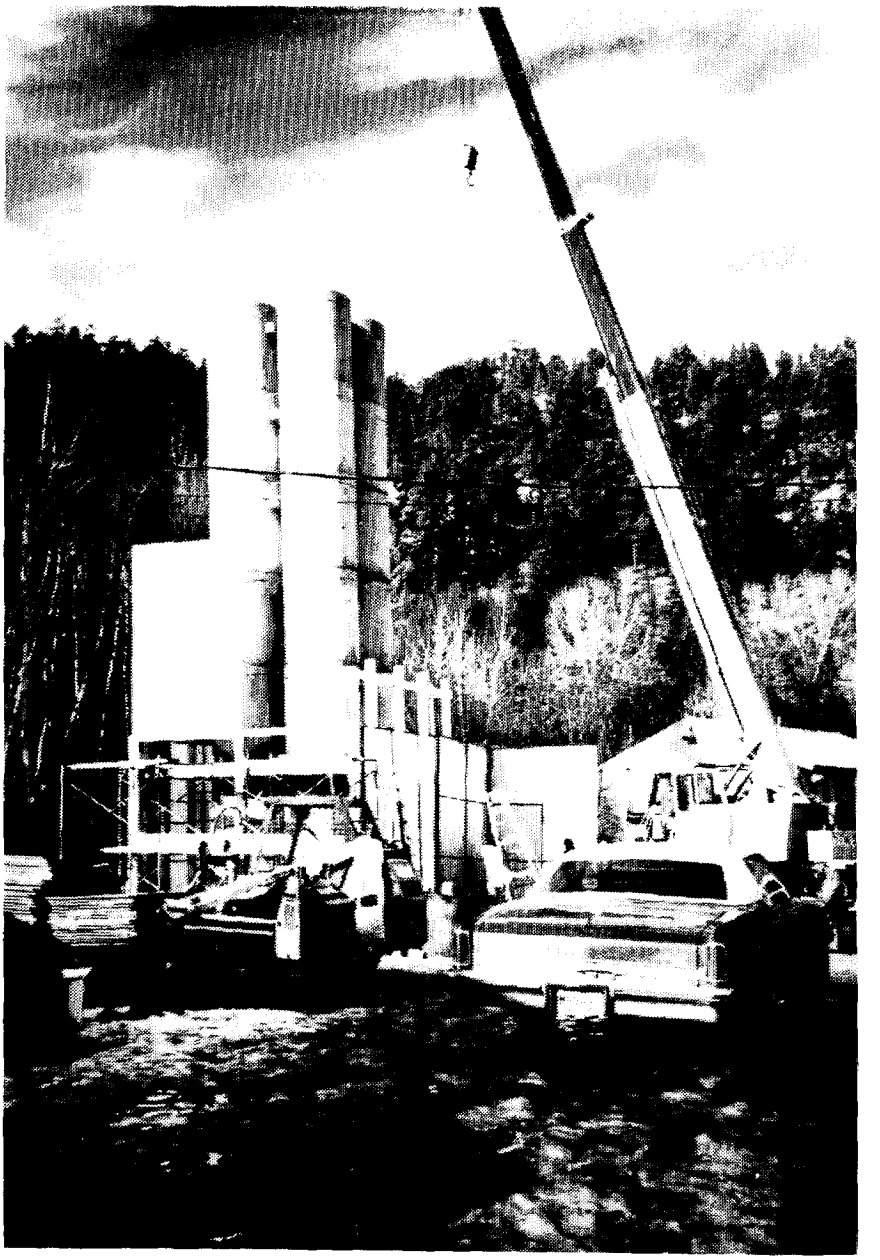

FIGURE 3. Tall leach columns can provide data on scale-up behavior of chemically-reactive ores. These four leach columns, four stories high, hold 25 tons of ore each.

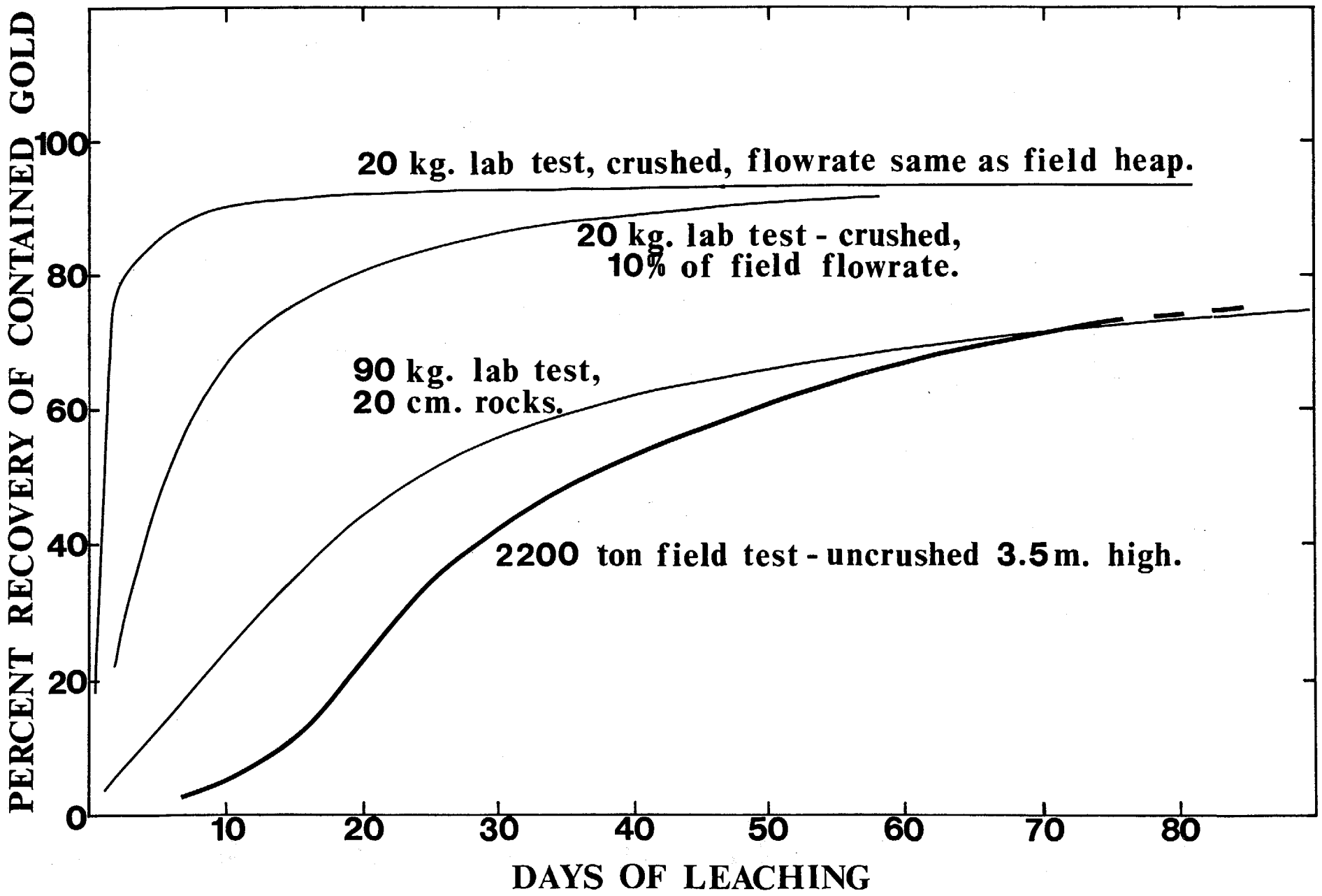

Figure 4: Effect of lab test variations.

Example 5. - Time Factors in Testing

Figure 5 makes a point somewhat removed from the general scheme of this paper. The figure shows laboratory test curves for three different ores. All three ores have approximately the same ultimate recoveries, near 60 percent. Had only short-term laboratory tests been run on these ores, however, the apparent picture would have been radically different. Recoveries after ten days ranged from 14 percent to 50 percent. A problem commonly faced by every testing laboratory is the "when can I have the answer?" syndrome, and a common response is to shorten the laboratory testing time.

Conclusion

In production heaps, allowing extra leaching time does more than insure against scaleup errors. At an ongoing project where the heaps are run counter-currently, the cost to leach "old" heaps is roughly $0.07 per ton per month. Significant additional cash flow can be generated from heaps producing only 0.5 percent additional recovery per month.

It appears that "time" must be added to a heap leach like "reagents" to a conventional mill; the approximate levels can be determined in the lab, but field operators must be given the flexibility to make final adjustments.

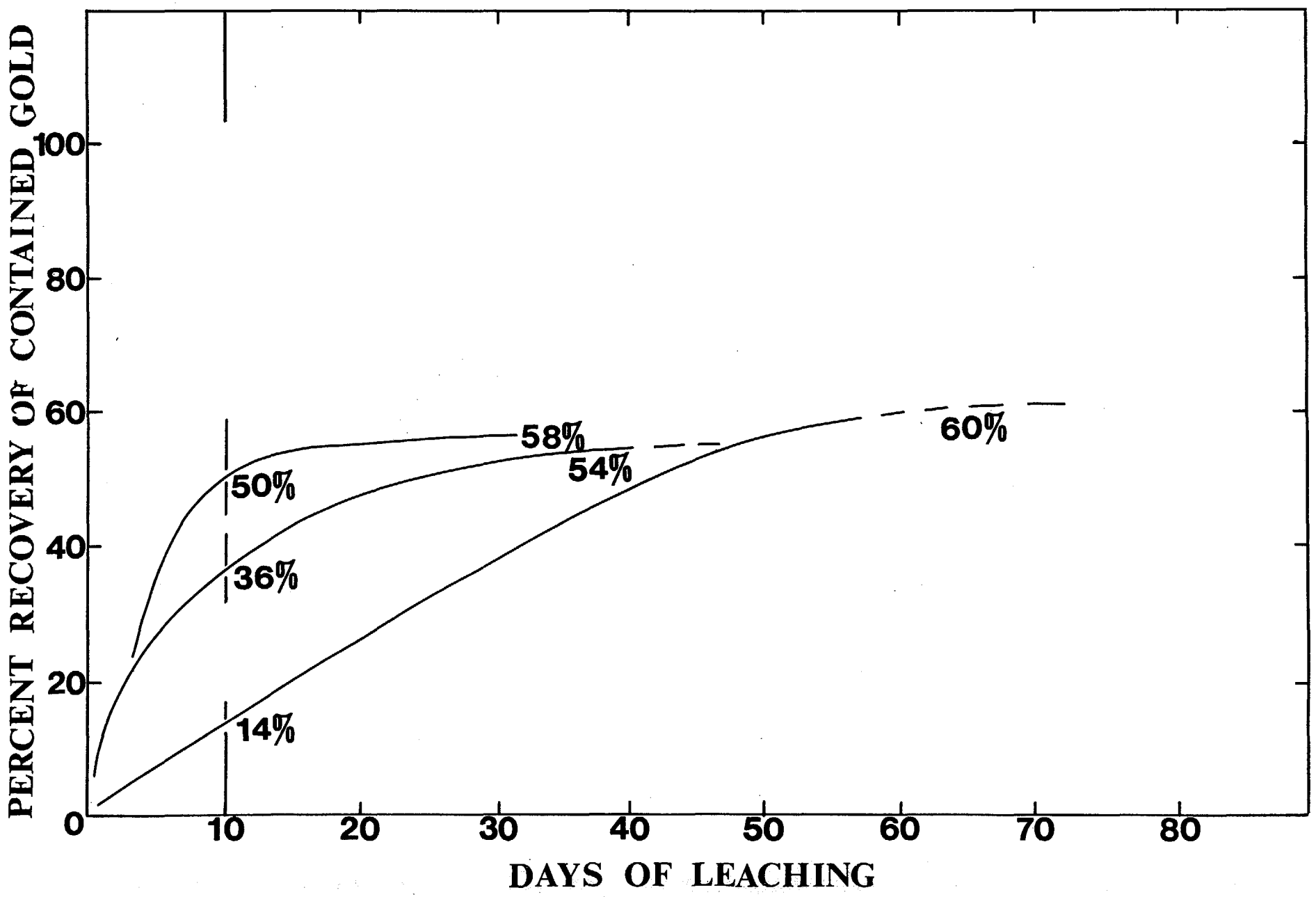

Figure 5: Leach time comparison laboratory small bucket tests, three ores, same conditions.

James D. Sell

STERLING MINE JOINT VENTURE

ANATOMY OF A SMALL UNDERGROUND MINE AND HEAP LEACH

February 1982

by

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Introduction

The Sterling gold mine is typical of the many small heap leach operations active in Nevada. The ore is composed of micron-sized gold particles unevenly disseminated in iron-stained sedimentary rocks overlying a major thrust plate. The gold is recovered from a heap leached by a cyanide solution which is then passed through activated carbon. The carbon is later stripped by a caustic medium. The Sterling mine merits study as its history illustrates that small tonnage, moderate grade underground mines can be profitably worked.

Location

As illustrated on Figure 1, the Sterling mine is located in western Nye County some 10 miles (16 km) southeast of Beatty, Nevada. Physiographically it is on the eastern flank of Bare Mountain adjacent to Crater Flat and the Nevada Test Site. Topography of the mountain is rugged with relief of as much as 2,000 feet (600 m). The adjacent valley is gravel filled, broad and flat. The ore body is within the mountain, allowing access via the deeply eroded canyons. The main recovery plant is located on the relatively flat main valley floor.

The climate is arid and summer temperatures exceed 110°F (45°C).

History

Beatty has been an active mining area since the turn of the century. Most production has come from the Bullfrog district west of Beatty. Prospectors of the time combed the Bare Mountain area and the so-called Panama shaft on the Sterling property dates from the 1910-1920 period. Production here was negligible. Some ore was found in siliceous fillings of limestone caverns and in narrow quartz veins in dolomite. Gold has also been reported from other mines on Bare Mountain, such as the Gold Ace and the Telluride.

Of more economic importance was the fluorspar production from the Diamond Queen, or Goldspar pit, adjacent to the Sterling mine. Most of the production here occurred between 1958 and 1967.

In early 1973, John Livermore and Peter Galli of the Cordex Syndicate (Cordilleran Exploration) staked the Stirling claims to cover the Panama shaft and all the land extending from it north to the Goldspar group. After initial mapping and geochemistry, they drilled holes numbered 1 through 37, some of which encountered gold mineralization. The claims were optioned by one of the present authors, Greg Austin, for SAGA Explorations. Austin drilled 14 more holes establishing an apparent ore trend and drove a short adit to expose the ore. In 1979, Dan Kappes began to study the possibility of heap leaching the Stirling ore. Derry, Michener & Booth, Inc. and E & B Explorations, Inc. of Vancouver joined with SAGA in January of 1980 to provide financing and technical assistance via the Sterling Mines Joint Venture for development of the possible ore zone.

The adit was rehabilitated and underground advance began in April of 1980. Ore was stacked on a pilot pad and leaching began on July 1, 1980. The first gold was recovered in September of 1980. During the Fall of 1981, payback of all capital expenses was achieved. Drilling has extended the ore zone so that several years of additional production and profits are anticipated.

The success of the Sterling Joint Venture is, in part, due to the division of responsibilities amongst qualified professionals. The mining and every day operation of the Sterling is the responsibility of SAGA Explorations of Winnemucca, Nevada. They maintain a work crew of about 30 people under the direction of Jim Bosley, Mine Manager. The geology and exploration is directed by Derry, Michener & Booth, Inc. of Golden, Colorado. DMB has one full-time geologist at the site and provides additional geologists on an as-needed basis. DMB also provides long range engineering input. Miller-Kappes, Inc. of Reno, Nevada designed and carries responsibility for the recovery system. A representative spends about one week a month on the property and is on call at all times. E & B Explorations, Inc. of Vancouver, in addition to providing the bulk of the financing, lends overall direction to the Joint Venture.

Geology

General

Bare Mountain is underlain by sedimentary rocks that range in age from Late Precambrian to Upper Mississippian. Paleozoic and later Cretaceous deformations have folded and faulted the sequence so that stragigraphic correlation is difficult. In addition, Tertiary volcanic activity and associated faulting and subsidence complicate the geology.

The Bare Mountain area lies with the Walker "Lane" or lineament which is 30 miles (48 km) wide and strikes northwest across western Nevada (Figure 2).

The geology of the area is best described by Cornwall and Kleinhampl (1961) on the 15 minute Bare Mountain Quadrangle map and in the Nevada Bureau of Mines & Geology Bulletin No. 77, on Geology and Mineral Deposits of Southern Nye County (Cornwall, 1972).

Local

The rocks in the area of the Sterling mine are a series of light colored (gray, tan, orange) dolomites, siltstones and quartzites. The ore occurs principally in red siltstones of the Wood Canyon formation of lower Cambrian age overlying gray dolomite of the Bonanza King formation. Earlier maps indicate the host unit to be the Johnnie formation, however, the most recent studies (Cornwall, 1972) show the unit as Wood Canyon formation.

It should be noted that very small, but important gold concentrations are found in the Bonanza King dolomite and also that narrow dolomitic beds are present in the Wood Canyon

formation. Also, although ore is predominantly found in red siltstones, gold has been mined from tan, gray, green and even white siltstones. The ore lies above the contact at a thickness of up to 35 feet (10.5 m). Locally, gold mineralization extends as much as 15 feet (5 m) below the contact into the dolomite of the footwall. In many instances a massive quartzite bed is present a meter or so above the ore.

The Wood Canyon/Bonanza King contact is thought to be a thrust contact because of the stratigraphic relation of the younger Bonanza King unit beneath the older Wood Canyon. The thrust relation is not at all certain as the stratigraphic correlation is tenuous and although in some places the contact itself appears sheared and broken, in others it is normal and undisturbed.

In the vicinity of the mine the rocks are folded into a syncline. This fold can be traced in the mine workings and from drill hole data. It has a northeast trend and plunges gently to the south.

Ore Geology

Although several ore pockets have been located, only two have been worked. One, the main zone, has been the focus of activity, the second, near-surface enrichment, has only received cursory attention up to this time.

The main zone is a shoestring body (Figure 3) that ranges from 50 feet (15.5 m) to 150 feet (46 m) in width, up to 35 feet (10.5 m) in height and is known to extend at least 2,000 feet (600 m) in length. In longitudinal section, the zone plunges gently (10°) to the southwest, but rolls and pitches so that the development decline is flat in places and locally rises. In cross section the ore body is roughly canoe or keel shaped being thickest near or at its middle and thinning to both sides (Figure 4). The top of the body is generally flat. Note, however, that the cross section in any one area may be tabular, tilted or have abrupt faulted(?) edges.

The grade of the ore ranges up to several ounces over five feet (1.5 m), but averages 0.25 oz. Au/ton. Silver content is negligible with a gold:silver ratio of about 30:1. There is little or no copper, lead or zinc. A very minor amount of stibnite and galena have been found. The alteration minerals of volumetric consequence are the iron secondaries (hematite, limonite and goethite), the clays and sericite. Pyrite is also present in small amounts and it is assumed that pyrite is the source of the iron.

The gold is present as flakes and particles to 4 microns in size, found with hematite and rimming calcite grains.

Although little work has been done on gold found outside of the main zone, the occurrences suggest similar controls and habits.

Operation

SAGA Explorations operates the Sterling mine. Jim Bosley is Mine Manager. He has a crew of about 30 people including 8 miners, 1 surveyor, 1 foreman, 2 assayers, 2 truck drivers, 1 clerk, 5 laborers, 2 samplers, 2 shift bosses and 5 mechanics.

The surface plant, excluding the recovery plant which will be discussed in detail later on, consists of an office trailer, an assay trailer, warehouse and machine shop trailers, a Deutz 50 KVA generator and a 600 cfm Compressor. Rolling stock, excluding pickups, includes a 22 ton dump truck, a 7,200 gallon (27,250 liter) water tanker, two 4 yd (3 m^3) front-end loaders (a Cat and a Terex), a Case 1 1/8 crawler loader and a small 8 foot (2.4 m) Basic grader.

Water is a major concern. The Sterling at present buys its water from the town of Beatty. An average of 9,000 gal/day (34,065 liters/day) in the winter and 15,000 gal/day (56,775 liters/day) in the summer is trucked to the mine site. About 20% is stored in a tank above the workings for use underground and the remainder is used at the recovery plant.

The mine is worked on a 2-shift basis. Underground equipment includes a Secoma hydraulic drill ATHE 12, 2 Elmac 5250 loaders, 3 Elmac trucks (two D10s and one D5), pneumatic drills, etc. The Secoma is an entirely hydraulic unit. Elmac units were chosen because of their parts standardization (interchangeable engines, transmissions and drivetrains). Split-set rock bolts installed on 4 foot (1.2 m) centers have proven to be the most suitable for ground support especially when augmented by wire screening. The rock bolts are installed using jack-legs. Ventilation is provided through two vent raises. Two fans, one 6 foot (1.8 m) diameter 50 hp and one 36 inch (.9 m) diameter 50 hp, have been installed.

The general mining method has been modified room and pillar with an occasional shrinkage stope. The mine was developed via an 8 foot (2.4 m) x 10 foot (3 m) decline. This drive was positioned so as to lie in the lower central part of the ore body. As drill data was sparse the drive has not always been ideally located, nonetheless, it is essentially a central footwall drift. From this drift crosscuts were driven to the edges of the ore body at approximately 75 foot (23 m) intervals. The crosscuts leave the main drift at nearly right angles and were driven so as to keep the footwall ore contact in the lowest part of the face.

Direction and control of the headings was supplied by visual and sample data, by previous drill hole data and in some cases, by longhole information. Samples were taken of every face. Vertical rib samples were also taken of both walls at 5 foot (1.5 m) intervals.

The samples were analyzed on site at a company owned and operated atomic absorption unit.

The ore zone plunges steeply south from a point midway along its length. At this point the central drift was halted and east and west fringe drifts were driven at the edges of the ore body. These drifts connected the crosscuts and divided the body into "rooms". Stoping of these "rooms" by slashing has begun. The first stope recovered a higher grade section between crosscuts 3 and 4 west

(Figure 5). A 25 hp Joy slusher was required to bring the ore down to the central haulage drift.

A second stope was started between crosscuts 8 and 11 east in low grade ore (Figure 6). Slashing here has shown that the wall rocks can be supported by rock bolts and screening and that 60% of the ore can be retrieved on a first pass with rubber tired LHD equipment. Further stoping is underway with present plans calling for making the western fringe drift the main haulage. Mining will proceed from south to north and from east to west. Future work will include driving below the ore pillars and eventual recovery of the pillars through finger raises.

The broken and sheared siltstone ore drills and breaks easily. The Secoma drill can complete 1.5 rounds per shift, with about 28 holes in each face. Break is about 9 feet (2.7 m) per round. The broken ore is transferred into an Elmac truck for haulage to surface. During the development phase, rate of production was 100 tons/day. This will be increased to 250 t.p.d. during the stoping phase.

On surface the ore is crushed to 3" (7.6 cm) size in a 24 x 36" (6.1 cm x 91.4 cm) jaw crusher prior to being trucked to the pads for leaching.

Heap Leach Design

Initial test work on fine-crushed ore indicated that there would be no percolation or chemical problems. The ore was competent enough that the heaps could be driven on by trucks and endloaders while being constructed, with only minor effects on solution flow. Very soft unsilicified ore yields 90 percent recovery in two weeks of laboratory leach testing, while the bulk of the ore is moderately silicified and yields 70-75 percent of its fire-assayable gold in six weeks of field leaching, when crushed to two inches (5 cm).

To confirm the lab test data, and to provide early reassurance to the investors, a mini-production heap of the first 3,000 tons of ore was stacked ten feet (3 m) high in two five foot (1.5 m) lifts, and leached using a temporary carbon column recovery circuit. Recoveries from this heap verified lab tests and heap size has gradually been increased to the present 15 foot (4.6 m) to 17 foot (1.5 m) height.

The current production heaps are located on a large alluvial fan about three-fourths (1.2 km) of a mile below the mine. To prepare a leach area, vegetation is removed and contours are smoothed by cutting and minor filling using a bulldozer. A thin layer of gravel is spread to cover rough areas of the prepared surface, and then a 30 mil thick PVC pad is installed by the mine crew.

This pad is covered with six inches (15 cm) of gravel screened to 3/4 inch (2 cm), which is made in an on-site screening plant from alluvial fan material. To insure good drainage, 4 inch (10 cm) diameter perforated polyethylene pipe is installed parallel to drainage, on 8 foot (2.4 m) centers within the gravel cover. This pipe is manufactured in 200 foot (61 m) long rolls, making installation very simple.

Heap Construction

The ore heaps are built on the pads in modules of about 12,000 tons each, and are permanently left on the pads. New heaps abut directly against older heaps so as to cover the old heap sideslopes. Eventually, the leaching area will appear to be one large continuous heap.

The ore is stacked on the 15 foot (4.6 m) high heaps in three five foot (1.5 m) high layers, to minimize segregation. A ramp of waste rock is built behind the heap up to the height of the layer to be constructed. Trucks drive up onto the top of the layer and dump their loads and a bulldozer pushes the material over the leading edge of the heap. When a layer is completed it is thoroughly ripped, and thereafter no truck traffic is allowed upon it. Once the third layer is completed, leaching is begun.

Heap Leach Design

Solutions are sprayed onto the heaps ten to sixteen hours per day (during dayshift and evenings) at a rate of 70 gpm (265 liters/m), or approximately 0.003 gallons per minute per square foot (.0209 ml/s per square meter). Because of the high assay grade of the ore and the hot, sunny climate, care is taken to insure good sideslope irrigation. Solution is applied using Bagdad wigglers with 3/16 inch (5 mm) I.D. tygon tubing, spaced on 9 foot (2.7 m) centers. Senninger wobbler No. 8 sprinklers, spaced on 27 foot (8.2 m) centers, are now being tried successfully on the top surfaces, though Bagdad wigglers will be maintained on the sideslopes. Addition of 500 ml per day of Barochem S-35 to the barren pond has almost eliminated scaling of the sprinkler heads.

The leach solution contains 0.5 to 1.0 grams sodium cyanide per liter. Sodium hydroxide is available for pH control, but the rock appears naturally alkaline and it is seldom added.

Water for the operation is hauled 23 miles (37 km) in a 7,200 gallon (27,250 liter) tank truck from the town of Beatty. Evaporation rate on hot summer days is 30 percent of solution sprayed, or about 10,000 gallons (37,850 liters). An additional 5,000 gallons (18,925 liters) per day is required to saturate new ore. The water is high in fluorides and chlorides. These are not affecting the leaching or carbon adsorption behavior, but they are the apparent cause of very high corrosion rates on anodes within the stripping plant.

Pregnant solutions exiting a new heap run 60 mg per liter (ppm) gold or higher, and remain above 3 ppm during most of the time it takes to reach 70 percent recovery. Each of the 12,000 ton heap modules drains to a separate pond and countercurrent leaching of older heaps is planned to begin in late 1981. This will permit very high recovery while maintaining low flowrates through the recovery plant.

Gold Recovery

The gold recovery plant encompasses several unique features, including the following:

- upflow, non-fluidized columns
- permanent residence of the carbon in the columns throughout adsorption, desorption, and acid washing cycles
- plastic (CPVC) pipe construction in the gold stripping circuit
- priming of all centrifugal pumps with a Moyno positive displacement pump

The complete plant is housed in a 40 foot (12.1 m) semi-trailer. Included within the trailer are four 36 inch (.9 m) diameter insulated carbon columns, a power supply and electrolytic cell, two pregnant solution pumps (one for normal operation, and a second for occasional fluidization of the carbon), a totally enclosed acid wash system, and a sink with countertop for handling the electrolytic cell cathodes. Immediately adjacent to the trailer is an 800 gallon (3,028 liter) insulated tank for strip solution storage, heated with 21 kilowatts of electric immersion heaters. The cost of the system was $50,000.00.

Column Operation

Each of the four columns is 36 inches (.9 m) in diameter, 7 1/2 feet (2.3 m) high, and contains 1,000 pounds (455 kg) of 6 x 12 mesh Calgon type PCB carbon. The four columns are arranged in two parallel sets of two columns each. Sixty to eighty gpm (275-365 liters/m) flows upward through the first column in the set at the rate of 8-12 gpm per square foot (335-505 liters/min per square m). A special fluidization pump is used to raise the flowrate to 17 gpm per square foot (867 liters/m per square meter) once every few days. Inflow solutions commonly contain in excess of 20 ppm gold and exit the second column to the barren pond with levels of 0.2 - 0.5 ppm gold.

Gold Stripping

Once gold levels in barren solution climb above 0.5 ppm a column set is taken off-line by switching valves and the alternate set is placed on-line. Gold is then stripped from the carbon in the off-line set using hot alkaline alcohol solution (83°C, 20 percent ethanol, 1 percent NaOH) at a flowrate of 8 gallons per minute (30 liters/m). Addition of sodium carbonate to the strip solution appears to greatly speed up the process and to significantly reduce corrosion. It is also important to calculate alcohol additions on the assumption that the carbon will absorb 12 percent of its weight in alcohol. The strip cycle is about 12 hours, and is preceded by a 2-hour carbon preheating period in which solution is pumped through the carbon then back into the heating tank. Strip solutions sometimes attain peak gold contents exceeding 5,000 ppm.

The electrolytic cell is a rectangular 30 gallon (114 liter) polypropylene tank containing eight stainless steel anodes on 3.5 inch (9 cm) centers and seven steel wool cathodes measuring 18 x 18 inches (46 x 46 cm). Solution flows up and then down between alternate anodes, and the cathodes are made to provide 1/4 inch (0.6 cm) of free solution flowspace between the cathode and the anode.

The cell is run at 400 - 600 amps, and up to 9 volts (with no anode corrosion, voltage is usually about 4). Current efficiency for the first several hours of the strip is 40 - 70 percent, and gold recovery per pass is 75 percent.

Regardless of the current density, anode corrosion is severe. This is attributed to buildup of chlorides and fluorides on the carbon from the field heap solutions, and could possibly be alleviated by a deionized water wash prior to stripping. Mild steel anodes are unusable after one strip cycle. type 316 stainless anodes have to be cleaned after each cycle and replaced every four cycles. A titanium anode is being tried.

Melting

The wet cathodes from the electrolytic cell are immediately transferred to a Denver No. 4 tilting hearth furnace. The iron is fluxed off using sodium nitrate. The product of this single stage melt is a dore button assaying 950 fine or better.

Acid Washing

Following stripping, the carbon in the columns is fluidized with heap solution, then acid-washed in a closed-pumping cycle with dilute HCL until the pH stabilizes below 2.0. The acid wash solution is then neutralized with NaOH prior to being discharged to the barren pond. The columns are again fluidized until the effluent is clear, and then they are ready to be placed back on line. After eight months of continuous operation during which the carbon has never been removed from the columns, they are showing only minor signs of channeling and activity loss.

Conclusion

The Sterling ore body was not exposed on surface. It is too small and linear to be amenable to open cast mining. Yet, as a result of prudent exploration, efficient underground development and the application of low cost heap leaching technology, the ore body has begun to yield a profit to its owners. The Sterling story is a testimony to the fact that small scale mining is alive and well in these United States.

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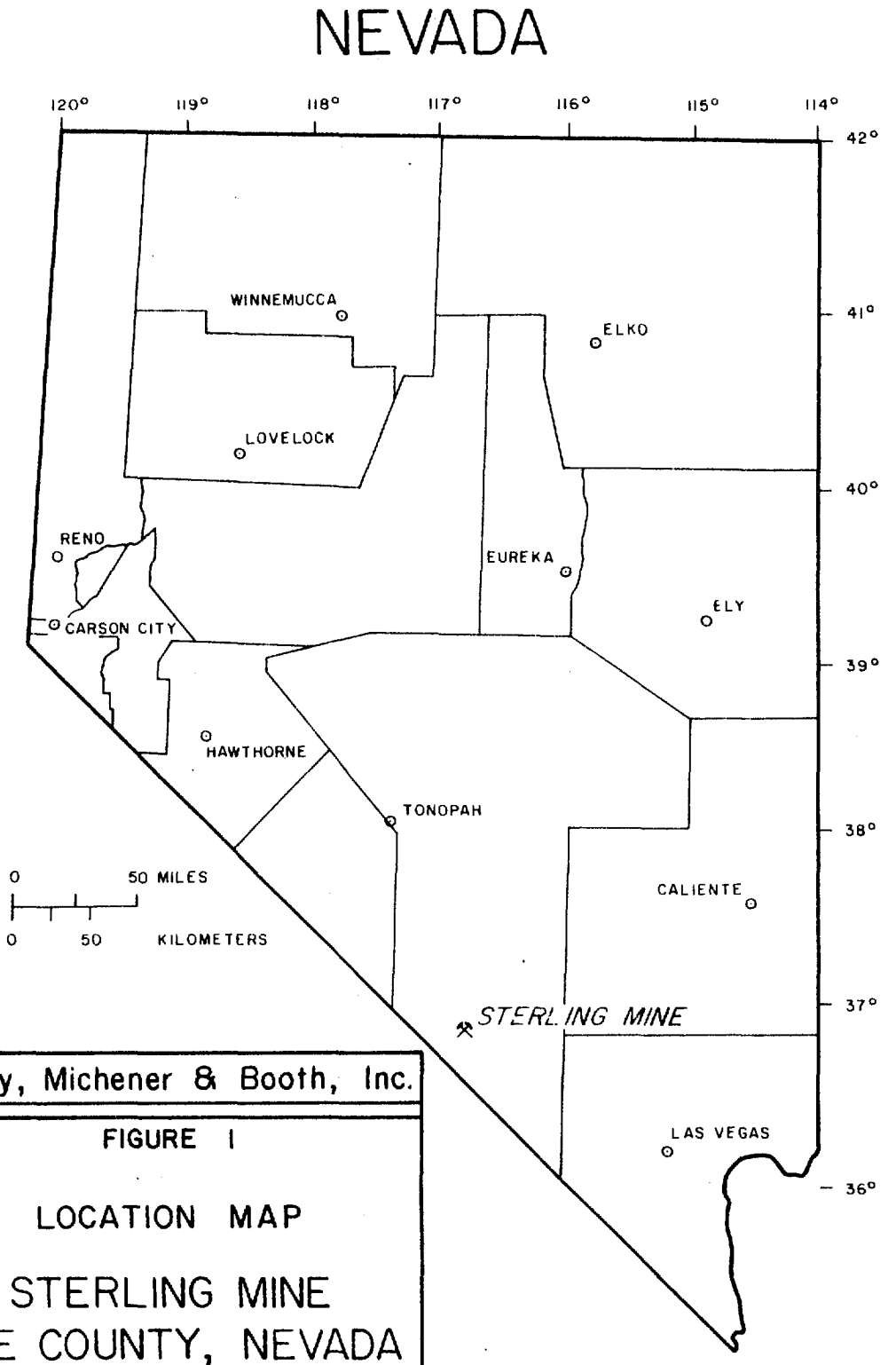

Derry, Michener & Booth, Inc.

FIGURE 1

LOCATION MAP

STERLING MINE
NYE COUNTY, NEVADA

DRAFTING BY: A.M. BUTLER DATE: NOV 1981

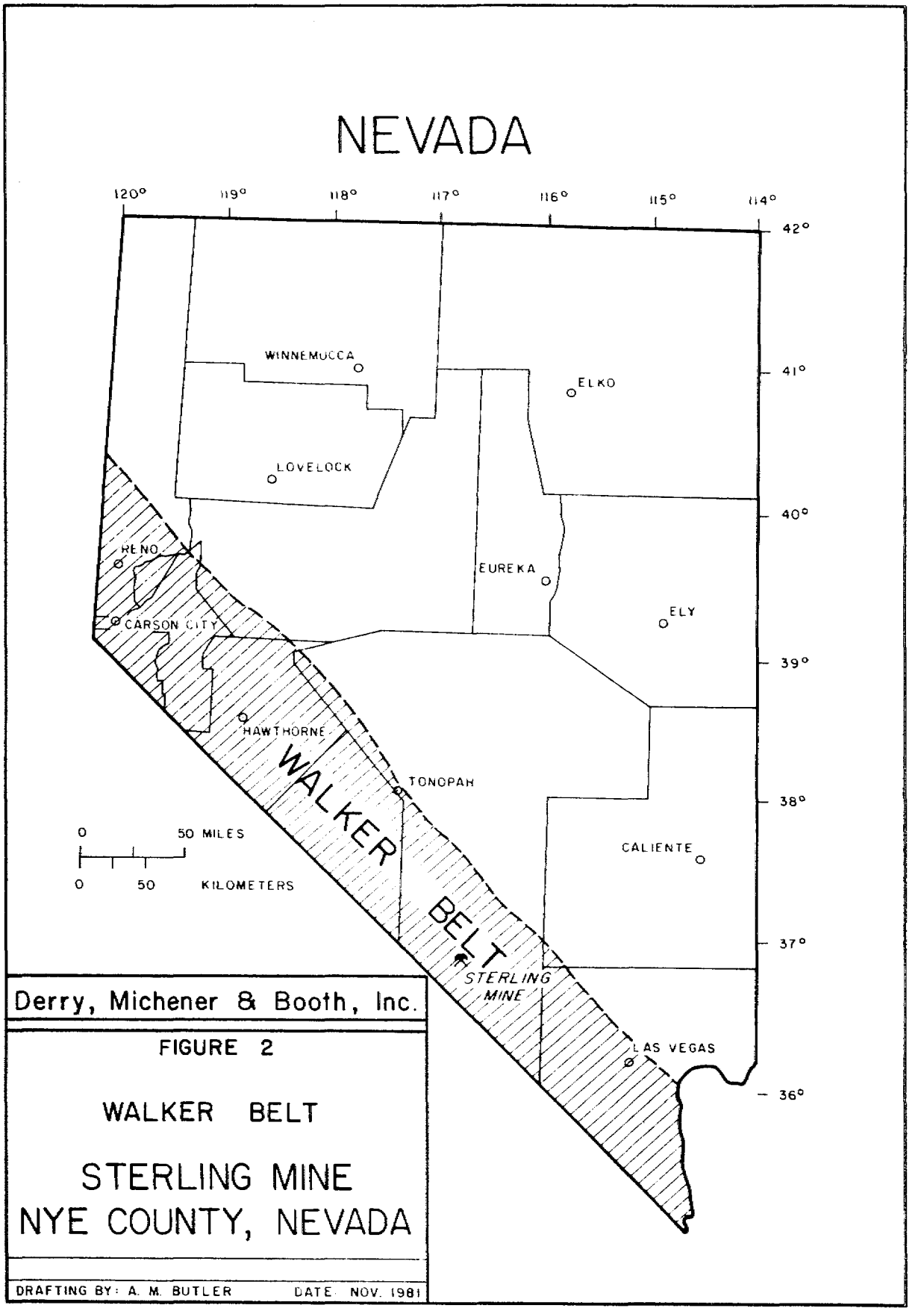

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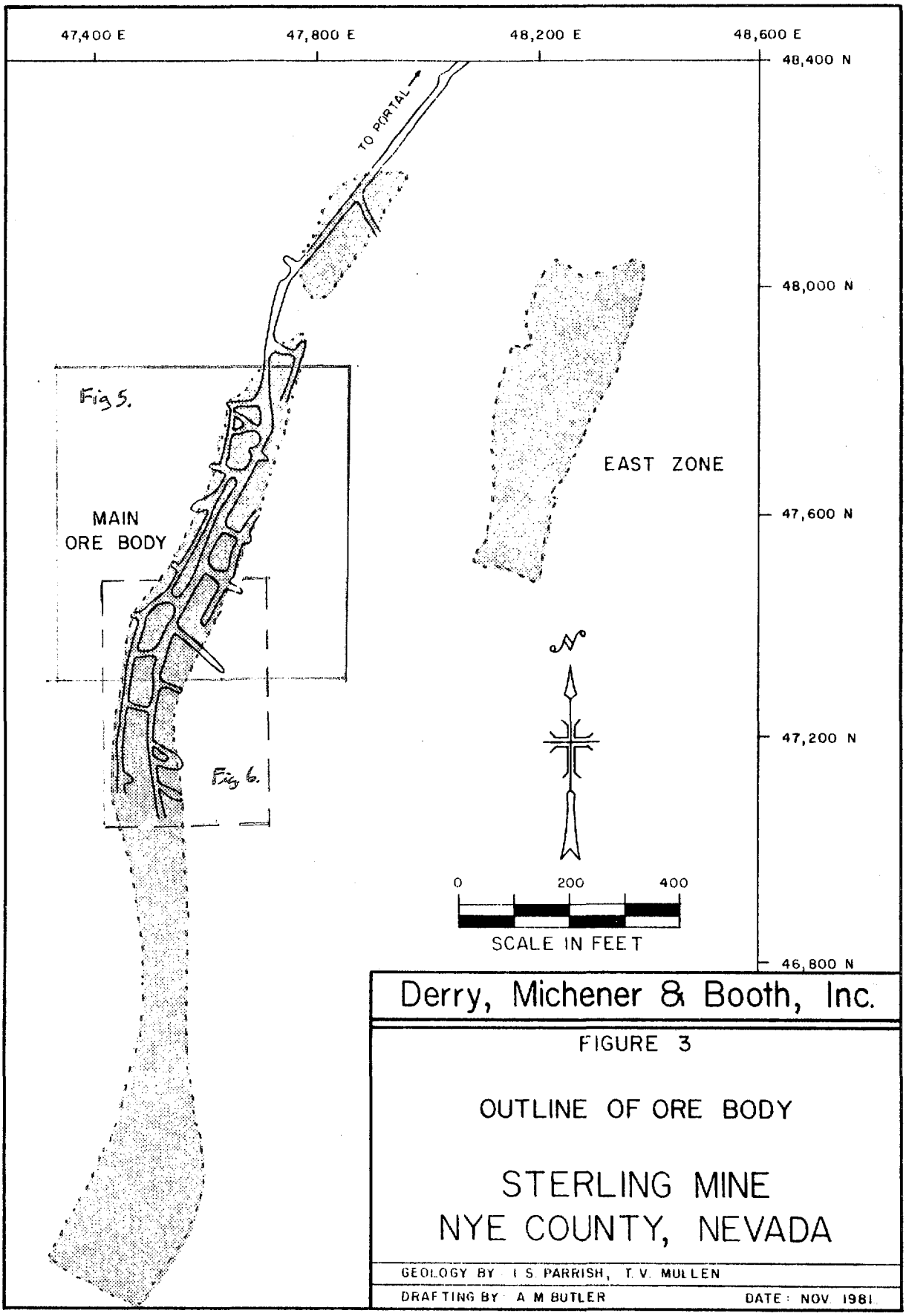
FIGURE 2

WALKER BELT

STERLING MINE
NYE COUNTY, NEVADA

DRAFTING BY: A. M. BUTLER DATE: NOV. 1981



Derry, Michener & Booth, Inc.

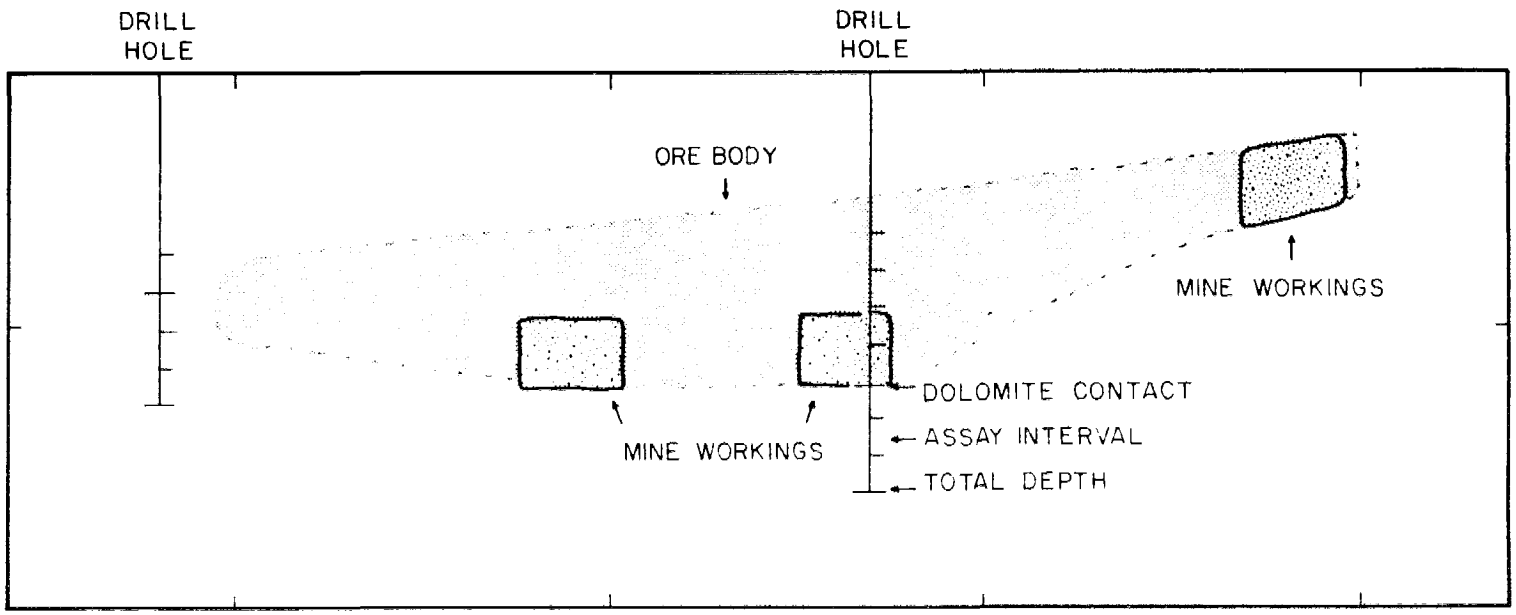
FIGURE 3

OUTLINE OF ORE BODY

STERLING MINE
NYE COUNTY, NEVADA

GEOLOGY BY I. S. PARRISH, T. V. MULLEN

DRAFTING BY A. M. BUTLER DATE: NOV. 1981



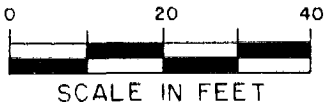


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FIGURE 4

IDEALIZED SECTION THROUGH ORE BODY

STERLING MINE
NYE COUNTY, NEVADA

GEOLOGY BY I.S. PARRISH, T.V. MULLEN

DRAFTING BY A.M. BUTLER DATE NOV 1981

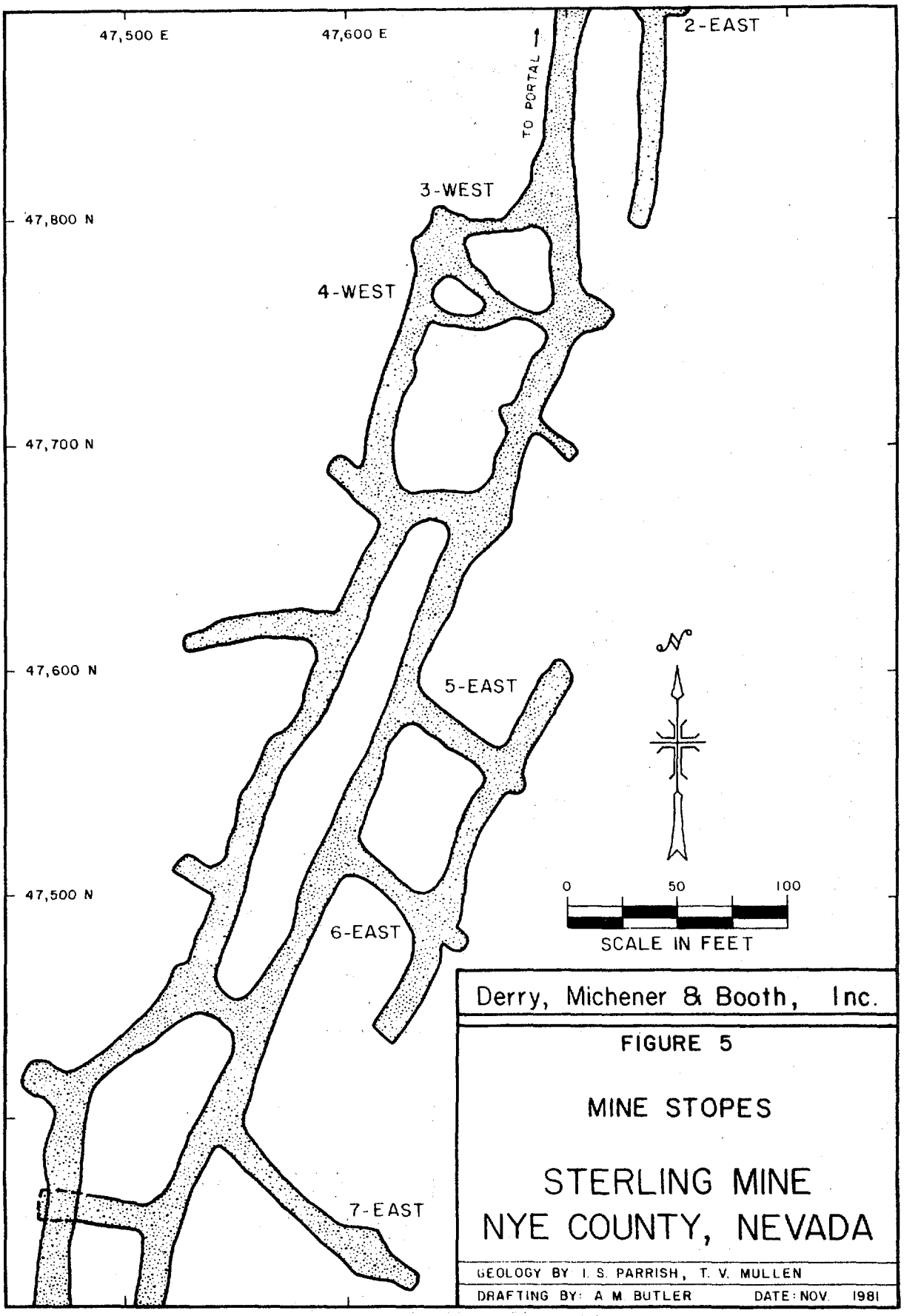

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FIGURE 5

MINE STOPES

STERLING MINE
NYE COUNTY, NEVADA

GEOLOGY BY I. S. PARRISH, T. V. MULLEN

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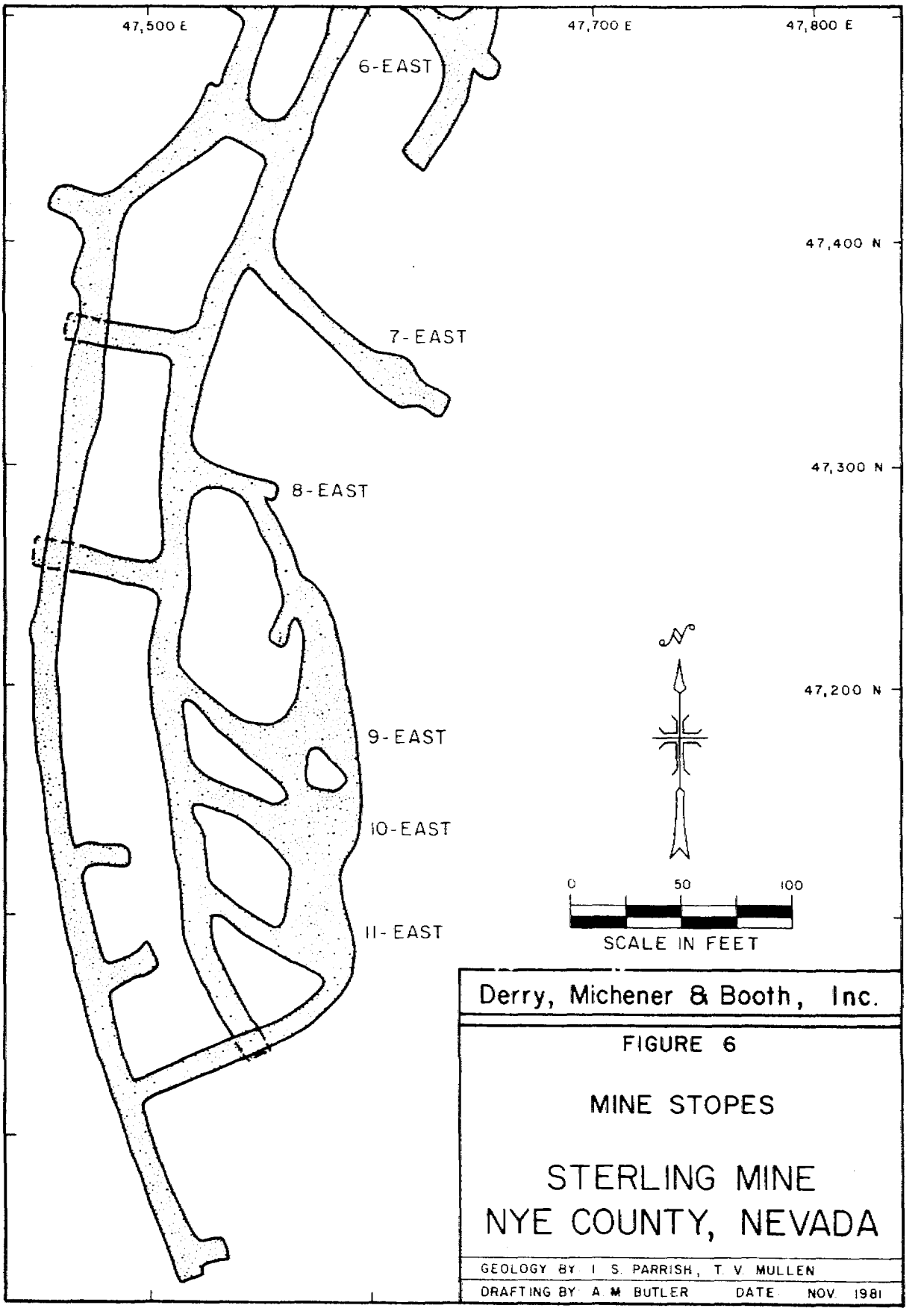
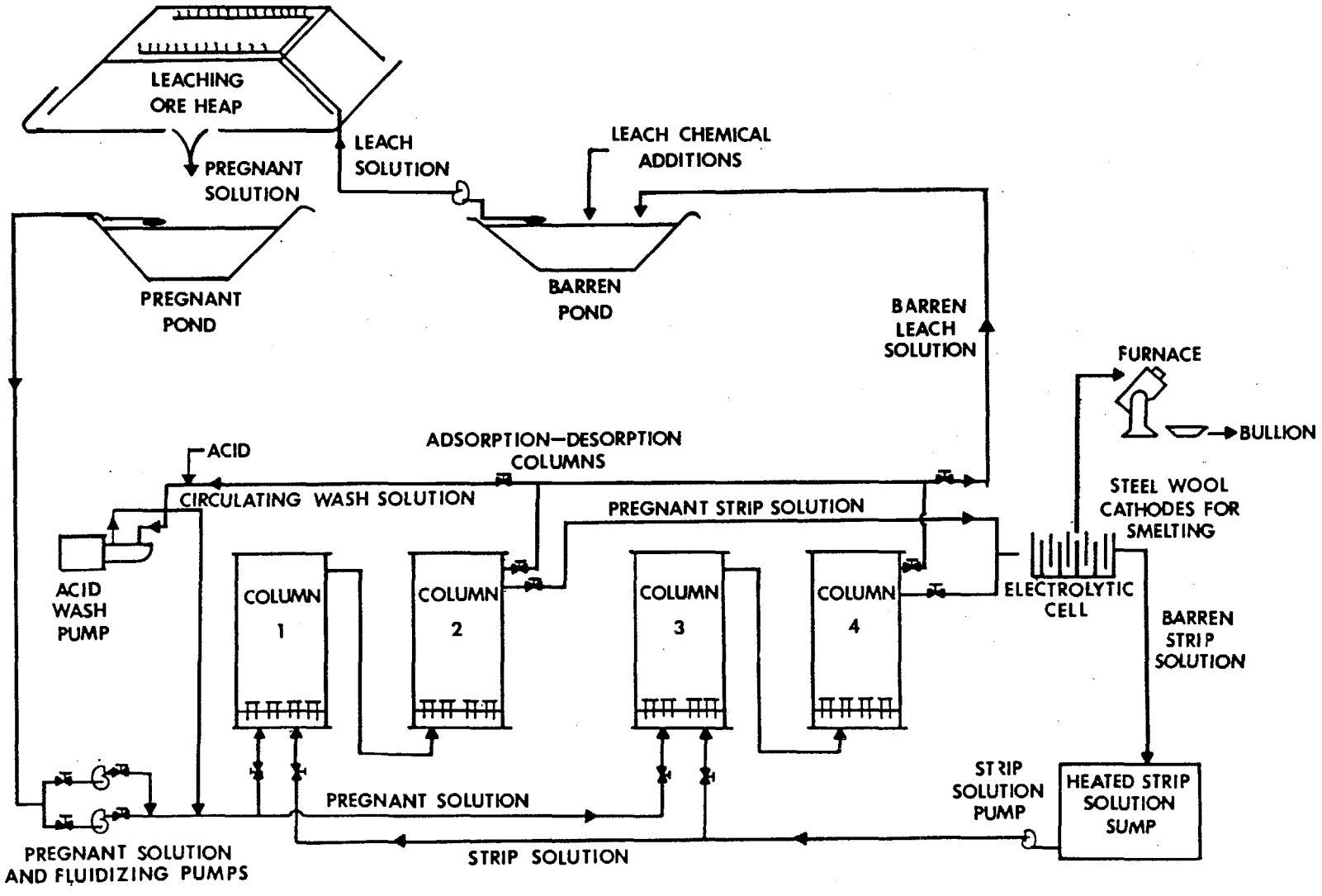

47,500 E
47,700 E
47,800 E
47,400 N
47,300 N
47,200 N
6-EAST
7-EAST
8-EAST
9-EAST
10-EAST
11-EAST
0
50
100
SCALE IN FEET
Derry, Michener & Booth, Inc.
FIGURE 6
MINE STOPES
STERLING MINE
NYE COUNTY, NEVADA
GEOLOGY BY I. S. PARRISH, T. V. MULLEN
DRAFTING BY A. M. BUTLER
DATE NOV. 1981



STERLING MINE RECOVERY SYSTEM

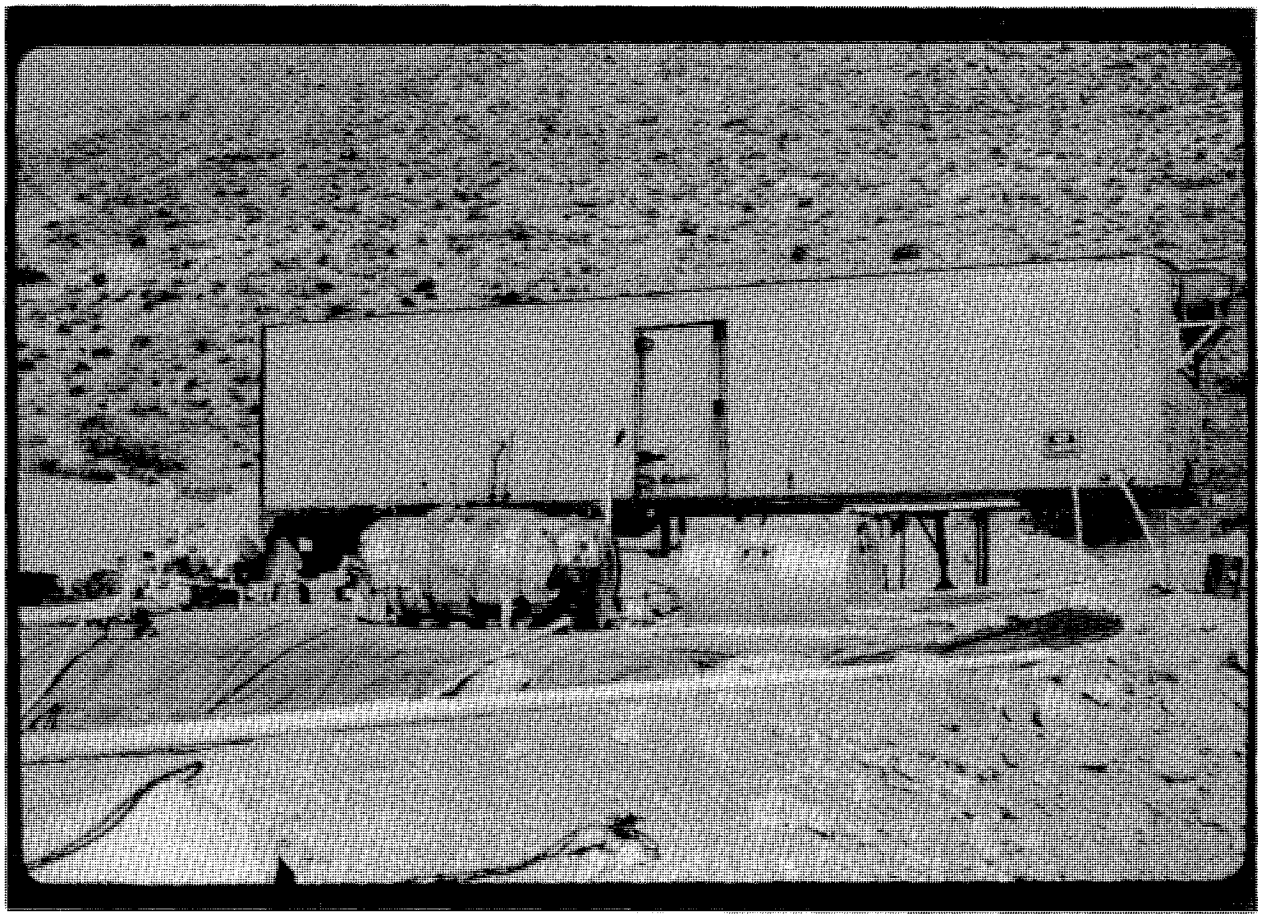


FIGURE 1. The heap leach system is designed for quick-leaching, high-grade gold ores, at the rate of 300 tons ore per day. Each 12,000 ton heap drains to a separate pregnant pond, which allows countercurrent leaching to extract the last traces of gold from the ore.

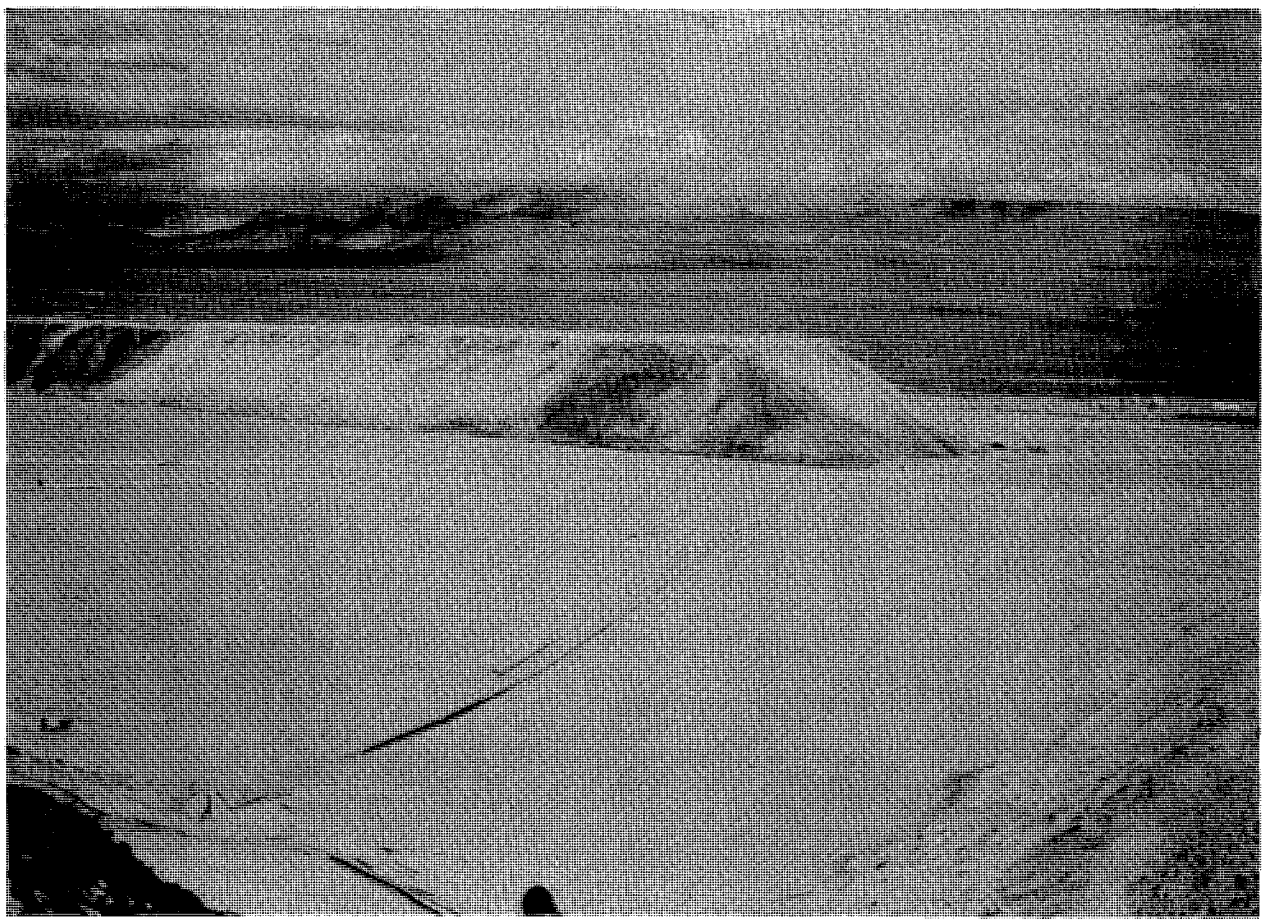


FIGURE 2. Heaps of 12,000 tons each are built, 15 feet high, atop gravel-covered PVC pads. Lots of land is available on the alluvial fan below the mine.

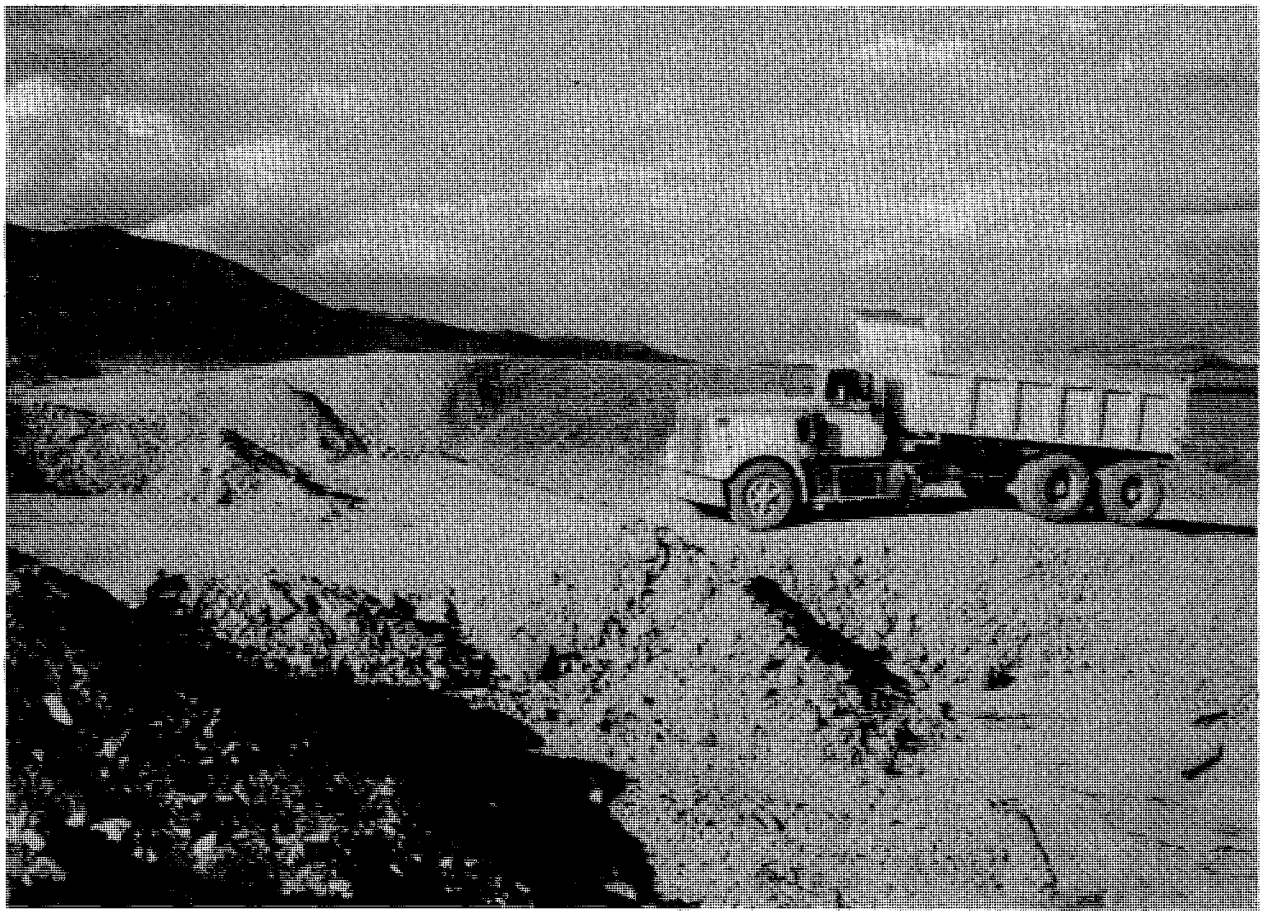


FIGURE 3. Heaps are built in three five-foot lifts. The heap in the background will remain under leach for at least two months. The first layer of the adjacent heap is under construction in the foreground.

FIGURE 4. Individual pregnant ponds are below the heaps. From there, solution is pumped through the recovery trailer, and is then discharged to the barren pond in the right foreground. Solid caustic soda and cyanide are manually added to the barren pond before solution is recycled to the heaps.

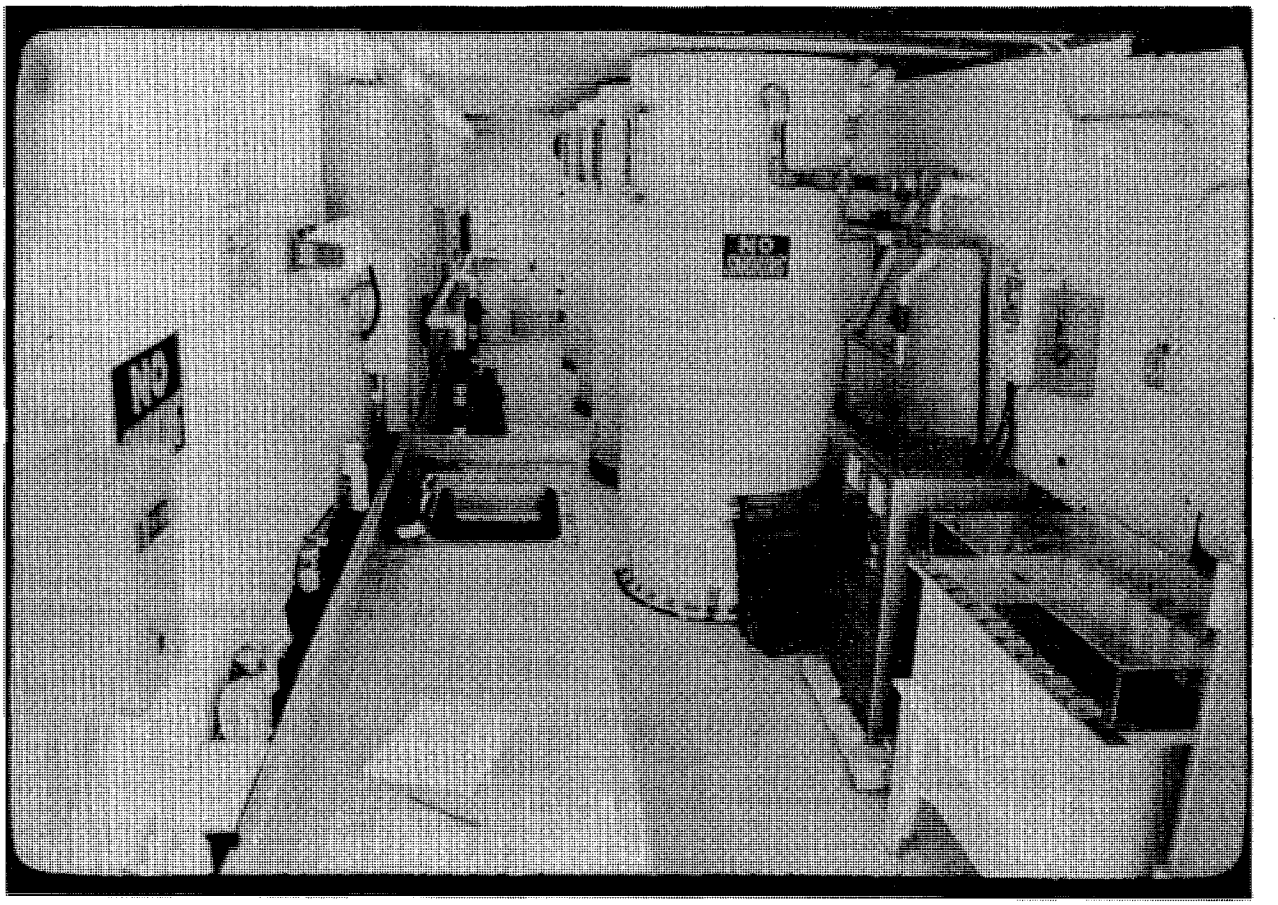


FIGURE 5. The recovery trailer contains four recovery columns holding 1000 pounds carbon each. All construction was done in Reno, then the system was transported 700 miles to the project site. Key features include insulated adsorption/stripping columns and a rectifier/electrolytic cell system for in-place recovery of gold.

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NEW DEVELOPMENTS IN HEAP LEACHING

A Paper for Presentation
at

GOLD & SILVER RECOVERY FORUM 85
Randoll International, Inc.

Santa Fe, New Mexico
14 October 1985

In the 1550's, Agricola wrote a convincing description of heap leaching for (strangely enough) Aluminum. (The ancients did not like aluminum cookware, but they used a lot of alum).

In its current reincarnation heap leach technology is now 20 years old, and it is showing encouraging signs of coming into adulthood.

The simplistic approaches of the early years are now being supplemented by new sophistication in areas such as the control of scale formation, systems for water balance control, conveyor stacking, and countercurrent leach techniques.

Heap leaching is alive and well in the western U.S., and is beginning to be applied elsewhere. Significant studies or actual gold leaching operations have been undertaken on every continent except Asia (maybe there also) and in most significant gold producing nations. Leaching of copper and uranium ores has developed in parallel, if to a lesser extent.

Where do we go now? The ultimate goal is low price per ounce - which requires a combination of low operating costs and high gold recoveries. The table below shows typical operating costs of a 2000 ton per day heap leach.

TYPICAL OPERATING COSTS
2000 TON PER DAY HEAP LEACH

MINING	$ 4.00	/ TON
CRUSHING & STACKING	1.00	
LEACH PADS	0.80	
PONDS & PIPES	0.20	
OPERATING LABOR	0.70	
GOLD RECOVERY	0.40	
CYANIDE/LIME	0.60	
G & A	1.00	
TOTAL OPERATIONS	$ 8.70	/ TON
CAPITAL	2.70	
GOLD NOT RECOVERED	6.70	(35 percent loss of 0.06 oz/ton in ore)

Emphasis is often placed on lowering operating cost factors by building high heaps. While this is admirable, the primary emphasis must be maintained on that final factor, RECOVERY: a small percentage change there more than offsets a change in any other cost category.

The factors which affect recovery are still not well understood. Major developments are still needed in several areas:

Understanding the Nature of Chemical Reactions and Flow Mechanisms:

- why do tall heaps behave slower than short heaps?
- are there chemical additives that can enhance recoveries?

Management of Water Balance:

- solution application systems to conserve water
- solution application systems to conserve heat
- systems to minimize rainfall intake

Methods of Handling and Preparing Difficult Ores:

- crushing and agglomeration methods for damp ores

- development of efficient stacking systems

THICKNESS OF HEAPS: FALSE ECONOMICS?

A major question which always affects costs is the heap height. Higher heaps require less management and are much less of a problem than shallow heaps.

Operations like Landusky-Zortman and Candelaria provide models of heaps 60 and 100 feet high which are tempting to duplicate. Many operations are stacking new heaps atop old ones.

The cost tradeoffs are not very straightforward, however. Lab column data indicates that increases in column height can lower recovery rate or total recovery percentage. Were does this stop? There is some recent data which indicates the development of "roll fronts" - gold moving slowly downward through the heap. Some multi-lift heaps show definite signs of delayed or lower recovery.

Raising heap height to 60 from 20 feet decreases costs by $0.75 per ton. A recovery drop of five percent wipes out the advantage.

CAN SOLUTION MOVEMENT WITHIN ROCKS BE MADE MORE EFFECTIVE?

An interesting question is: How does solution move through the grain boundaries of the rock to get the gold?

In a lot of ores, the gold is on fractures that appear too fine to permit solution movement.

The near-surface ore at the Sterling Mine, Beatty, Nevada, provides an interesting example. Here, ore microscopy established that gold occurs as 1 micron particles which form "necklaces" around calcite grains in a calcite-rich shale. The grain boundaries are tight, yet recovery from ore containing 0.30 ounces gold per ton - at 3-inch crushed size - was 90 percent in three weeks.

Evidence for grain-boundary gold which can be recovered by heap leaching exists in other deposits.

Interesting work is being done on chemical additives to enhance leach rates and recovery. Faster-leaching heaps, or high heaps with low-heap performance characteristics, should result.

NEW GEOGRAPHIC FRONTIERS REQUIRE NOVEL SYSTEMS FOR SOLUTION APPLICATION.

The western U.S. is an ideal location for leaching: moderate rainfall, moderate temperatures, remote but still accessible.

In some areas of the world, improvement in techniques is necessary for heap leaching to be undertaken.

Heap leach field tests have proceeded in the high desert areas of South America at altitudes of 14,000 feet (4300 meters). Ice forms on the heaps every night during the SUMMER. Modified solution distribution methods are key to the operation:

> For seasonal ice in the western U.S., buried pipes and drip points are used, but this is an ineffective method of solution distribution. Tests conducted several years ago at Windfall, Nevada indicate that the angle of the wetting cone is only 10 degrees. In a 30-ft high heap with drip points on 8 foot centers, only 45 percent of the heap is effectively leached.

Alternatives for conventional sprinkling are needed for conditions exactly opposite to the those in freezing environments. Sprinkling in hot desert conditions uses up scarce water resources.

PERMANENT VAT LEACHING MAY BE A USEFUL TECHNIQUE

We have recently evaluated heap leaching in extreme desert environments in North Africa. A modification - leaching in permanent vats - shows promise of eliminating evaporation problems.

The use of Permanent Vat Leaching has also been proposed for high mountain areas. Permanent vats - large, plastic-lined heap leaches with plastic-lined sidewalls - offer the advantage of self containment for winter shutdown and the ability to leach below an ice cover.

I think we may see several installations of the Permanent Vat Leaching technique in the near future. The main roadblock to Permanent Vat Leaching has been that small vats incur high containment costs - the containment for a 50,000 ton vat costs $1.50 per ton - but this can be overcome by the use of ore to build temporary internal dikes. Evaluation indicates that a 2000 ton per day Permanent Vat Leach will incur containment costs only $0.40 per ton higher than heap leaches.

RAINFALL CYCLE IS AS IMPORTANT AS RAINFALL AMOUNT

For heap leaching in high rainfall areas, the annual cycle of rainfall can be more important than the total rainfall. In Costa Rica the annual rainfall is 5 meters - 200 inches - but for six months of the year the country is dry.

Here, high clay ores have to be dried and then agglomerated. Fortunately, during the dry season the ores can be dried by solar heat and stockpiled for treatment during the rainy season.

In tropical climates during the dry season or in northern high-desert climates, the solar drying area required to dry wet clays is roughly 10 square meters - 100 square feet - per ton per day. Capital and operating costs of a solar drying system are comparable to crushing costs.

Different conditions apply on the northern coast of South America. Rainfall is less than in Costa Rica but is spread uniformly throughout the year. Drying in the open air is not possible, and field drying tests indicate that a 1000 ton per day operation will require a rather impractical drying area - an area under canopies of ten acres (4 hectares).

Investigations here are centering on the use of log washers to mix and agglomerate the high clay ore in its wet condition. Pinson Mining Company has applied similar equipment for agglomeration at the Prebble orebody in Nevada.

SOLUTION DISCHARGE IS USUALLY NOT NECESSARY

Water imbalance - the need to discharge excess water - is usually not a problem in tropical climates. Tropical jungle areas show high humidity near ground levels under forest canopy, but cleared areas show low humidity during most days. Under such conditions, evaporation of sprinkled leach solution is rapid. Typical daily average evaporation levels are shown below.

	24-Hour Average Evaporation Percent of Solution Applied through Sprinklers
High Desert, Summer	20
High Desert, Winter	5
Tropical Jungle, Dry Season	20
Tropical Jungle, Wet Season	10
Cool, Rainy Climates	0

Because of the high rates at which solutions are applied to heap leaches, it doesn't take much evaporation to keep the systems in water balance.

Our experience at a heap leach operation in Costa Rica was that the systems are in balance during months with 15 inches of rain or less. The table below shows the typical relationship between sprinkling rate and rainfall.

Sprinkling rate:	5	gallons/ day/ sq ft
	0.0035	gallons/minute/ sq ft
	10	liters / hour/ sq meter
equals	7-1/2	inches of rain per day
	19	feet of rain per month
	6.0	meters of rain per month
10 % Evaporation:	22	inches of rain per month
	0.6	meters of rain per month

The worst weather conditions in the world for heap leaching are cold, damp climates. We've just completed engineering for a heap leach in western France which will employ covered storage ponds, solution neutralization and discharge. Witteck Development Corporation recently announced that they're installing a neutralization system at the Gordex Mining Company in eastern Canada.

COVERED PONDS OR COVERED HEAPS SHOULD BE CONSIDERED

Where rainfall is heavy, solution balance can be greatly improved by covering the ponds, which may have a surface area equal to half that of the active heap area.

Either for heavy rainfall or for cold weather leaching, roofs have been considered but not yet applied. The cost of inflatable buildings or suspended/supported membrane roofs is $4.00 per square foot. A "modular" roof covering a 2-month active leaching segment, that moves continuously across the top of the heap on a series of steel runners, should cost less than $0.50 per ton of ore leached.

CONVEYOR STACKING SYSTEMS ARE BEING PROVEN IN PRACTICE

Conveyor stacking equipment is now becoming established as an important heap construction method. Conveyor stacking allows complete flexibility to create internal structures (controlled channels) within the heap where this is necessary, or to build the heap as a thoroughly homogenous mass of agglomerated material.

Since no heavy equipment needs to drive on top of the heap, a conveyor stacking system - for any heap of crushed ore - results in lower heap construction costs per ton and improved percolation characteristics.

Ortiz uses a permanently mounted bridge conveyor stacking system. Portable bridge conveyors mounted on tires have been used by the construction industry and appear feasible for heap leaching.

Falcon Exploration Company at their Tonopah Divide mine in Nevada pioneered the use of extension conveyors for building heaps, which permit efficient working surfaces on the heaps.

Sectional conveyor systems have been used at Amboy in California and at Getchell, Little Bald Mountain, and Buckhorn in Nevada. These systems permit direct stacking with almost no manpower needs beyond the crushing plant discharge.

Typical installations involve between three and ten short sectional conveyors between the preparation systems and the heaps, but we've recently evaluated systems with up to 3000 feet of fixed conveyor leading to telescopic conveyors at the heap construction site. These systems appear to be feasible and simple extensions of technology already proven in other industries.

SIMPLE LOW-COST HEAP LEACHING IS ALIVE AND WELL

Conveyor systems and other heap leach systems need not be expensive. The Little Bald Mountain operation of Northern Dynasty Explorations Ltd., used a custom-designed stacking system, contract mining and crushing, and a portable recovery plant in a first-phase leach this past summer. Total capital and operating costs for this 600 ton per day installation were less than $1,000,000.00, and payback occurred in less than five months. Performance like that, of course, is exceptional, and requires a combination of good ore and simple production systems with something that hasn't changed at all since Agricola's time: a team of professional managers with common sense and the dedication to work long and hard hours.

Kappes, Cassiday & Associates

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18 June 1987

Dear Sir:

This letter comprises a bid for one 200 US gpm carbon Adsorption-Desorption-Recovery plant (ADR plant) for a project located in Nevada or nearby states.

The plant price is $178,000.00 FOB Sparks, Nevada. Nevada sales taxes (approximately 6 percent) are not included, and will be charged if applicable.

Installation

Installation costs include shipping to site, complete setup and startup costs. It is assumed that the operator will have two semi-skilled personnel (the future plant operators) available for two weeks during setup to assist with setup, and to learn pre-startup procedures. (If these personnel are not available, the costs shown below will increase $150.00 per man-shift not furnished). Startup assistance and training will be furnished one shift per day for two weeks following setup (Training may be delayed until sufficient process solution is available).

Total cost of Installation, Startup
and Training: $ 26,500.00

Our rate for startup assistance beyond the above "installation" cost, for a KCA engineer familiar with carbon ADR plant practice, is $8,800.00/ month, plus expenses, for time planned and committed on long-term basis, or $440.00 to $580.00 per day for daily services, plus expenses.

Optional extras are discussed in a later section.

General Plant Description

The proposed plant is a complete system for adsorbing gold and silver from cyanide solutions through upflow, fluidized carbon beds, followed by non-pressurized desorption using caustic, caustic-alcohol (preferred method) or caustic-glycol strip systems. The Plant is built conservatively oversize to permit operation in excess of design capacity.

Gold and silver are recovered onto steel wool cathodes. Cathodes are direct-smelted in a 225-pound propane-fired tilting crucible furnace, which is mounted on an external concrete slab (slab and enclosure not furnished).

Piping and pump systems are included for pumping solution from the pregnant pond through the columns and discharging to the barren pond; and for pumping from the barren pond to an adjacent point leading to the heaps. The pumps will be "Tsurumi" submersible pumps mounted in floats or on slides on the pond berm. Pumps will be furnished complete with motor controllers and dual controls inside the plant and at the pond edge.

The plant is designed to permit loading of carbon to 150 ounces gold per ton or better (depending on the system chemistry; similar plants KCA has built achieve loading approaching 1000 ounces per ton), and stripping of two column loads simultaneously. When utilizing a 48-hour strip cycle (and 150 oz/ton loading) the plant is capable of recovering 3700 ounces gold per month. A 24-hour strip cycle can often be attained, which doubles gold production capacity.

Description of Included Items, Alternative 1.

The plant consists of the following items of equipment:

one - column module container consisting of a 45-ft. semi-trailer with high quality floor, ceiling and wall insulation, internal fluorescent lighting, forced air (propane) heating/ventilation, good tires and brakes, external paint with client logo, internal fire extinguishers, one window, one window/escapeway

one - recovery module container consisting of a 28-ft. semi-trailer with high quality floor, ceiling and wall insulation, internal fluorescent lighting, forced air (propane) heating/ventilation, good tires and brakes, external paint

two - steel stairs for plant access when mounted at trailer height above ground

six - adsorption/desorption/acid wash columns, 10 sq. ft. cross-sectional area, 5.5 ft. settled carbon bed depth (8 ft. overall height), arranged as three sets of two columns each; mild steel columns with epoxy lining

one - DSM-type screen and sump for catching and recycling floating carbon

one - pregnant solution pump

one - barren solution pump

one - shedding vortex flowmeter with totalizer

two - floating pond intakes

one - storage tank for strip solution

one - 30 kW electric immersion heater system for heating of strip solution (propane heating may be optionally furnished)

one - sink and countertop

one - work table for cathode manufacture

one - electrolytic cell and variable-output power supply, including cell hood and exhaust fan

one - propane-fired tilting pot furnace with 225-pound crucible

one - main electrical switchbox for external electrical connection (only one connection is required, with other voltages derived internally)

- valves: butterfly and ball valves, steel body with stainless steel working parts

- piping: PVC and CPVC, some steel, depending on duty requirements

- connections: flanged fitting will be used preferentially (if appropriate)

- heat tracing and insulation: all external pipelines

- freeze protection: complete drain capabilities.

Options and Additions

The plant as quoted above is a complete plant capable of producing dore bullion. However, the following options and additions may be desired or required in specific instances. Prices quoted below are installed prices unless otherwise indicated; they do not include Nevada Sales Tax, approximately 6 percent additional.

Concrete Slab Below Trailer Modules. It is normally a good idea to mount the trailer modules on a 6-inch thick concrete slab with a curb and central drain to the pregnant pond.

This slab should have dimensions 32 by 60 feet. KCA will install this slab as part of the plant setup for a cost of $7200.00.

Pressure Strip Option. The quoted plants are designed for non-pressurized stripping of carbon, at an operating temperature of 83°C. Most KCA plants are run using an ethanol strip system, although they can be operated using ethylene glycol or non-organic strips.

We can provide the plants with the capability to strip using 260°F, 25 psi pressurized strip circuit for an additional cost of $120,000.00.

Strip Solution Heating Using Propane. The quoted plants required 30 kW of electric power for heating of strip solution during the preliminary phase of the strip cycle. If this power is not readily available, it is possible to heat the solution using a propane-fired heater. The additional cost to substitute propane heating for electric heating is $5000.00.

Carbon Regeneration Furnace. For an operation using a 200 gpm plant (approximately 1200 tons ore per day), we do not recommend the installation of a carbon regeneration furnace.

Normally, the activated carbon used in the system will maintain its activity sufficient to meet design conditions for slightly in excess of one year. The Sterling, Nevada, plant, which is a similar design to the plant proposed here, has been running since 1981 with average carbon replacement every 14 months. Other KCA plants have similar operating experience.

Once activity is diminished, the carbon can then be sent for custom regeneration in batches, or sold to a large operator. The cost to regenerate carbon will be approximately $12,000.00 per regeneration cycle (probably, $12,000.00 per year).

KCA will furnish an Allis-Chalmers rotary kiln designed for 50 pounds carbon per hour regeneration capacity, complete with screens, tanks, propane hookup, and carbon transfer facilities, all installed and operable, at a cost of $105,000.00 in client's building or $160,000.00 in a concrete-slab, steel-frame building.

Smelting Furnace Slab, Building and Scrubber. The smelting furnace which is furnished with the plant may be operated by placing it on a gravel base or, preferably, a concrete slab measuring 10 x 20 feet. Open air installation and operation is sufficient, although a simple open-walled structure with corrugated steel roof is preferred.

The quoted price for the plant does not include a slab or housing for the furnace. The installed price includes installation of the furnace in client's housing.

If mercury is present in the ore, or if the furnace is to be housed inside a building, it may also be desirable to install a furnace hood, exhaust fan, and wet exhaust scrubber.

KCA will provide a simple slab for a cost of $900.00.

KCA will provide a complete installation including steel pre-fab building, exhaust system and wet scrubber (designed for low-efficiency particulate removal) at an additional cost of $55,000.00.

Mercury Retort. If sufficient mercury is present in the ore, a retort should be installed prior to smelting. KCA can provide a retort sufficient to retort a complete set of electrolytic cell cathodes at a cost of $55,000.00. This retort could be housed within the same building as the smelting furnace at an additional cost for that building of $16,000.00.

Activated Carbon. The quoted prices do not include operating supplies or activated carbon. We currently recommend Pica G210AS carbon, but suitable carbon may be available from Westates Carbon Corporation or Calgon (Calgon type GRC-22 may be acceptable, but lot samples should be examined). Price of carbon is approximately $1.50 per pound.

The plant will require an initial carbon purchase of approximately 12,000 pounds (sufficient for startup plus makeup required during first six months of operation).

Schedule of Construction and Installation.

The plant can be ready for shipping 13 weeks after bid confirmation and receipt of down payment. Shipping will take one week. The plant can be ready to pump and accept process solution within two weeks after arrival on site. The desorption cycle of the plant will be ready one week thereafter.

Operating Manual.

The bid price includes a Basic Operating Manual which includes a valve list, layout drawing, flowsheet and a general discussion of operating procedures.

One section of the operating manual contains all purchase order information and maintenance literature which applies to the commercially-supplied components of the plant.

This Basic Manual has been sufficient for other plants which we have furnished. A Detailed Manual can be prepared, which lists operating plant philosophy and step-by-step operations for all logical operating conditions, at a cost of approximately $8,000.00.

Terms and Conditions.

We will require a payment of 30 percent of the total plant price with bid confirmation ($53,400.00), a 20 percent progress payment six weeks thereafter ($35,600.00), a payment of 25 percent upon shipping ($44,500.00), and the balance of 25 percent ($44,500.00) six weeks thereafter.

With the execution of suitable documents giving KCA the right to repossess on demand and sell the plant, extended payment terms will be provided by KCA at an additional cost calculated at 18 percent annual interest compounded daily on the net balance "overdue" based on the payment schedule outlined above. Late payments will be charged at the same interest rate.

For suitable projects, lease terms are possible.

General Technical Parameters.

1. Other than carbon, basic reagents and supplies needed for plant operation include those shown below. The consumption figures are approximately monthly levels at a full production rate of ten strip cycles per month:

muriatic acid (acid washing of carbon)	100 gal/mo.
steel wool and misc. cathode supplies	$150.00/mo.
ethanol	1000 gal/mo.
sodium hydroxide	800 lbs/mo.
sodium carbonate	800 lbs/mo.
furnace fluxes	400 lbs/mo.
furnace crucibles	2/mo.

2. The electrical loads for the plant consist of the following items (the power shown is installed power).

process pumps, two @ 15 hp:	25 kW
misc pumps	3 kW

strip solution heaters	30 kW
electrolytic cell power supply	12 kW
lights, ventilation, instruments	5 kW
Maximum expected power draw	75 kW
Average expected power draw	30 kW

3. Major categories of spare parts consist of:

rupture discs
pump seals and impellers
spare diode for electrolytic cell rectifier
minor stock of replacement pipe fittings
miscellaneous supplies and hardware items

For a plant operating in Nevada, we recommend a minimum amount of spare parts. Spare parts investment will be less than $3000.00.

4. The plant will be built to withstand operation and periodic shutdowns during freezing weather. One advantage of this type of plant is that it can be shut down and started up relatively easily, several times a shift if necessary, or at nights or on weekends. (Shutdown during freezing conditions requires that correct drain procedures be followed.)

5. The barren solution process pump shall be furnished to provide a pressure at the pump of 100 feet of head at the design flowrate, unless otherwise agreed to in writing.

We appreciate the chance to make this bid, and look forward to working with you on the project.

Sincerely,

Daniel W. Kappes

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1 OCTOBER 1986

SMALL AND LARGE HEAP LEACHES - DIFFERENCES IN APPROACH

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by

Randall A. Pyper and Daniel W. Kappes

for presentation at

CIM District 6 Meeting
Victoria, B.C.
3 October 1986

Heap leaching has had a reputation as a bit of an underdog - a low-cost, low-technology approach to properties that couldn't be treated any other way. The recent spate of very large heap leaches, however, has served to dispel that concept.

For example, the Mesquite operation of Consolidated Goldfields in Southern California is a $70 million dollar investment with a production rate of 12,500 tons per day.

The reasons to use heap leaching instead of conventional milling are illustrated by two recent installations in Nevada - Paradise Peak in central Nevada (FMC) and Hog Ranch in the northwest corner of Nevada (Western Goldfields/Ferret Exploration).

As Figure 1 shows, the capital costs of a conventional mill are very significant. Paradise Peak cost $20,000.00 per daily ton of capacity, whereas Hog Ranch - a low-key heap leach installation - cost less than $2,500.00 per daily ton.

At a gold price of $400.00 per ounce and an 11 percent cost of capital, an orebody with a gold content of 0.08 ounces per ton would have to show a very high recovery differential - 40 percent - to justify the mill. This assumes that milling and heap leach operating costs are identical, but of course the mill operating cost is significantly higher than the heap leach.

In the case of Paradise Peak, the mill could be justified because the ore is high grade and contains a large amount of silver which is not readily heap leachable. But for many large orebodies, heap leaching is the choice from a financial standpoint.

Thus the choice of treatment - conventional mill or heap leach - is between two equally valid methods that have both "come of age".

Technology is like the U.S. auto industry, however. As the technology matures, it tends to get larger and to develop a lot of complications.

Has heap leaching grown out of its original role as a "Model T" method for treating small deposits?

The answer should be "no" - Large and small deposits may require technically different approaches, but each should be valid in the appropriate situation.

Why is it that properties show this reverse economy of scale? i.e., why do they get more expensive per ton of throughput as they get larger? There are at least three reasons -

1) Small operations are more closely controlled by people who care - sophistication in design can easily give way to personal "hands-on" involvement in daily operations.

For instance, Little Bald Mountain in northeastern Nevada, near Placer's Bald Mountain operation, was able to successfully run a conveyor stacking system directly on leach pads because the managers are there to supervise each conveyor movement; Hog Ranch requires an 18 inch gravel cover.

2) Small operations can afford to be more labor intensive - they generally have a shorter life and are higher grade than the large ones.

FIGURE 1. CAPITAL COST COMPARISONS - MILL AND HEAP LEACH

	PARADISE PEAK	HOG RANCH
DAILY ORE TONNAGE	4000	4000
TYPE OF MINING	OPEN PIT	OPEN PIT
TYPE OF PROCESSING	CYANIDE MILL	HEAP LEACH
CAPITAL COST	$ 80 MILLION	< $10 MILLION
CAPITAL + FINANCE COST/TON	$ 15.00	$ 1.90
CAPITAL COST, OZ GOLD/TON	0.0375	0.005
PERCENT ADDED RECOVERY NEEDED TO JUSTIFY THE MILL - ORE GRADE 0.08 OZ GOLD/TON	41 %	

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FIGURE 2. CAPITAL COST COMPARISON - LARGE AND SMALL HEAP LEACHES

	MESQUITE	HOG RANCH	LITTLE BALD MTN
DAILY TONNAGE	12,500	4,000	400
CAPITAL COST - (INSTALLATIONS)	$ 50 MM	$ 7.0 MM	$ 0.5 MM
(TOTAL)	$ 70 MM	$ 10.0 MM	$ 1.0 MM
INSTALLATIONS, $/DAILY TON	$ 4,000	$ 1,700	$ 1,000

3) It's easier to modify a small operation to correct or work around initial design difficulties. LBM avoided any agglomeration in the first year even though the ore had a relatively high clay content; by selective mining and careful stacking, production losses were minor. Agglomeration was installed in year two, where it did not affect the initial capital requirement.

Another example of the "reverse economics" of size involves the amount of money spent initially on orebody development prior to the decision to proceed with production.

At what point should a company suspend exploration drilling and begin operations? Should it design operations only for proven tonnage, or for "expected" tonnage?

The small company might stop when it has enough tonnage to breakeven, particularly if it has good chances for developing additional tonnage later on. Usually, the sources of capital available to the small company are very expensive - stock offerings may appear to provide "free" money, but in terms of management time and future flexibility, this is very costly capital. Generally, the sooner a small company can begin developing a cash flow the better.

The large company, on the other hand, has more resources available to it - especially in the critical area of management talent needed to development or "massage" investment capital - its cost of capital is lower. At the same time, it generally places more importance on long-term stability and on financial measures of performance (i.e. return on investment becomes more important in relation to promotional value).

To accomplish its goals, the large company will need to show a decent return on investment and a decent life - and to do this, it will have to prove up at least one and a half times the amount needed for simple recovery of investment. In fact, to meet long-term stability goals, it may find it desirable to develop two or three times as much reserves as it needs.

The goals of a company do and should change. We have dealt with companies that - correctly - placed very small properties into production. Within a year or so, their minimum target for new properties often becomes much larger than the property they just developed. The first property was quite successful in both return on investment, and in allowing the company to out-grow it.

FIGURE 3. APPROACH TO ORE RESERVE DEVELOPMENT

LARGE COMPANY GOAL: TO CREATE LONG-TERM STABILITY
(DRILL FOR PLANNING PURPOSES)

SMALL COMPANY GOAL: TO DECREASE COST OF FUTURE CAPITAL
(DRILL ONLY ENOUGH TO JUSTIFY DEVELOPMENT)

RECOVERABLE GOLD CONTENT (OZ/TON)	TYPICAL HEAP GRADE (OZ/TON)	BREAK-EVEN TONNAGE	20% DCFROI TONNAGE
0.03	0.045	3,000,000	5,000,000
0.05	0.077	1,000,000	2,000,000
0.06	0.092	460,000	800,000
0.08	0.12	100,000	160,000
0.10	0.15	50,000	85,000
0.15	0.23	20,000	35,000

GOLD PRICE US $ 400/oz

CONCLUSION

Orebodies aren't usually flexible. The small orebodies need the small-company approach. This aspect of the business is generally acknowledged though not always appreciated.

If both approaches are valid, can they both be done by the same company? Probably not.

The concept of a small-company is not very well understood in business. A "small-company approach" is not created merely by lowering the financial goals of the big company. Small companies have more personal involvement by the managers. For a big company to get small successfully, it has to find a way to shift personal involvement from upper management to lower management.

A recent article in one of the business magazines made the point that - fortunately - companies aren't biological organisms. They don't merely grow up, grow old, wither away and die. Good companies may go through many cycles of being large or small, but it's probably difficult to be both sizes at once.

In conclusion, it should be said that most companies which have built expensive "Cadillac" operations - the Mesquite heap leach, the Paradise Peak mill - seem to have made valid decisions. These companies have relatively low costs of capital and very long-term goals of maintaining corporate stability.

The other end of the spectrum - low-cost, rapid development - is equally valid and important. It can be the road to success for small companies with big future plans.

XXIII

Consideration in the Choice of Carbon Adsorption Systems for Gold Heap Leaching

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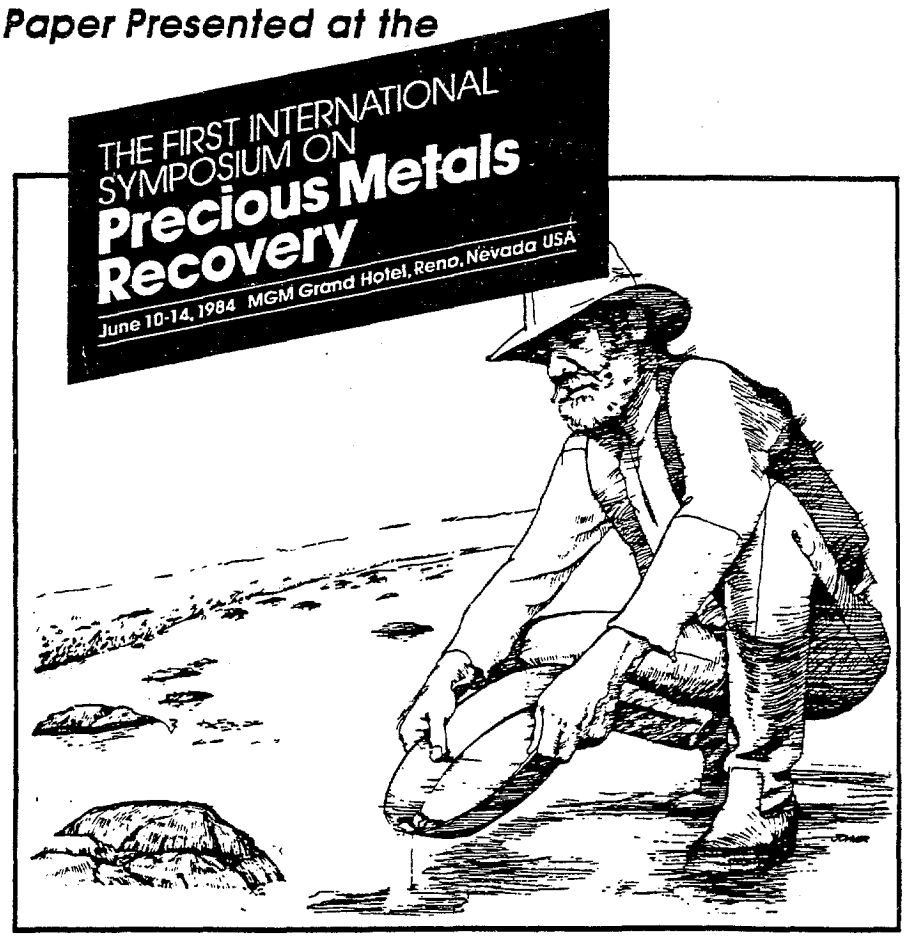
Paper Presented at the



CONSIDERATIONS IN THE CHOICE OF CARBON ADSORPTION SYSTEMS FOR GOLD HEAP LEACHING

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INTRODUCTION

Carbon adsorption has become the most common approach for recovering gold (and silver, if present) from heap leach liquors and other precious metals-bearing solutions. The majority of gold/carbon adsorption plants use fluidized or fixed carbon beds contained in multiple tanks or stages for obtaining intimate carbon/solution contact. These systems operate in the counter-current "continuous/batch" mode, although truly continuous systems are presently being developed and improved. The continuous/batch mode means that the pregnant solution continuously passes through a series of carbon batches or stages. As the solution contacts the carbon, the precious metal values are adsorbed onto the carbon. Solution exiting the last stage is very low in gold and silver content and is termed "barren solution." As each batch of carbon becomes loaded with gold and silver, it is sequentially moved to the immediate upstream stage. The carbon from the leading stage is removed from the adsorption circuit, stripped of gold and silver values (desorption), possibly acid-washed and/or heat regenerated, and is then returned to the last (downstream) stage in the adsorption circuit.

Several designs exist for providing continuous solution flow and separation of the carbon batches. These include: a) a multiple-tank cascading system utilizing gravity flow between stages; b) a multiple-tank system using pump pressure to force solution through closed-top tanks; and, c) a multi-stage tower with continuous upflow of solution through all stages. This paper will describe the three systems in some detail and will explore the advantages and disadvantages of each one in relation to the others.

CASCADING SYSTEM

One of the first carbon adsorption circuits used in a major gold heap leach operation was at the Cortez Gold Mine in east-central Nevada. This system successfully employed a

five-stage cascading system and was the model for many of the existing operating plants (1). Examples of major heap leach operations in the western United States using this type of system include Carlin Gold Mining Company's Maggie Creek operation (2500 tons per day), Amselco Mineral's Alligator Ridge Project (3000 tons per day), and Goldfield Mining Corporation's Ortiz Mine (2000 tons per day as of 1980) (2). The Pinson Mine in Nevada incorporates a series of carbon stages for recovery of gold from the thickener overflow in their 1200 ton per day carbon-in-pulp plant (3). There are many other gold/silver operations world-wide employing this type of design, such as the 600 ton per day heap leaching operation of Minera Macacona in Costa Rica. This indicates the degree of acceptance that cascading-type systems enjoy in the industry. Some cascading-type systems are pictured in Figures 1, 2, and 3.

Diagrams of a typical cascading-type system and an individual tank are presented in Figures 4 and 5. Solution enters the bottom of each tank and flows through a distribution plate of some type and up through the carbon bed. The overflow from each tank then flows by gravity to the bottom of the next tank, etc. Several methods are employed to feed solution to each tank. The design in Figure 5 shows the feed solution entering the tank through a down-comer in the center of the tank and discharging below the distribution plate. The down-comer is fed by a launder with a dart valve to control flow so as to eliminate air entrainment in the solution as it flows down into the tank. Air entrainment in entering feed solutions can cause channeling of solutions and carry-over of carbon out of the system. This factor must be seriously considered in the design of a cascading system. The feed launder contains another dart valve to allow an operator to by-pass the tank by directing solution to the overflow launder and then to the next tank in the series. Other designs pipe the feed solution through the side of each tank below the distribution plate. The advantage of the down-comer system is the ease of by-passing an individual tank without extensive piping as is required in the side-entry design.

Another advantage of the cascading-type system is the relatively easy access to the inside of the tanks. Often it is necessary to check each tank for proper carbon level, extent of fluidization, or cementing of carbon to the bottom or sides of the tank. It may be necessary to replace or repair the bubble caps or distribution system. This can be accomplished quite easily through the open top tank. Carbon and solution sampling are also accomplished quite easily with this system.

On the other hand, easy access to an open-top tank can be a security risk in that carbon theft is a possibility. Some cascading systems have a lockable hinged screen over the top of each tank to reduce this risk.

A major disadvantage of the cascading system is the structural requirements for raising the tanks to the elevation necessary to obtain the pressure head for proper flow through the system.

Walkways and working platforms must be constructed at each level to provide for ease of operation and maintenance. This increases capital and installation costs for the series of tanks and also increases building costs if the tanks are to be enclosed. Portability of the adsorption system also becomes quite expensive, although this may not be a factor in the conception of the project.

The large structures involved with cascading systems increase the operating costs associated with heating a large building during colder months. The possible exposure of operators to hydrogen cyanide vapors and solutions in the tanks and launders along with the high moisture content of the air when processing cold solutions in an enclosed building lead to a poor working environment and possible building maintenance problems.

There are several other minor disadvantages or operating problems associated with a cascading system. One is the need for operators to climb up and down stairs or ladders in order to properly attend to their duties. Minor problems can develop into major ones if operators are too lazy to perform regular checks, and stairs contribute to this situation. It should also be noted that access to the drain valves on the bottoms of the upper tanks can become difficult. Also, the use of windows or viewports in the sides of the tanks is pre-empted unless further walkways are installed. A final problem related to this system is that carbon movement between cascading stages requires the use of eductors. It can be argued that the use of eductors contributes to carbon attrition, although the severity of this attrition has not been quantified.

CLOSED-TOP TANK SYSTEM

Figures 6 and 7 picture actual closed-top tank systems utilizing relatively low pump pressure to force the gold/silver-bearing solutions through a series of tanks with closed and sealed tops at the required flowrate. With these systems all tanks can be at ground level since gravity is not required as a driving force. Examples of operating closed-top systems include Whim Creek Consolidated's Haveluck Mine (500 ton per day heap leach) in Meekatharra, Western Australia (4); Saga Exploration's Sterling Operation in southern Nevada (250 tons per day) (5); and E & B Exploration's El Plomo Project in southern Colorado (300 tons per day). Closed-top type systems have been used at several other mining operations, and this type of processing is typical in the chemical industry.

Figures 8 and 9 show diagrams of a typical four-stage closed-top system and an individual stage, respectively. Solution enters the bottom of each tank below a baffle or distribution plate, much the same as in a cascade system. General operation is also quite similar during the adsorption phase since flowrate control, degree of fluidization, carbon levels, etc. are all critical. However, the basic characteristics of the closed-top

system allow much more flexibility in the amount and type of process variations available when designing the overall adsorption-desorption-recovery (ADR) circuit.

An example of this flexibility is the fact that a simple feed-pump change can allow the use of a complete new range of process flowrates. This may be desireable to compensate for design errors or different carbon size selection. Cascading systems do not allow this capability since the driving force (pressure drop) for flow through the system is limited by the elevational difference between each tank.

Another example of this flexibility is that a properly designed adsorption tank can double as a stripping vessel and can also be used for acid-washing the carbon. In addition, a four or six tank system can be operated as either a single series of tanks or two parallel sets. The Whim Creek tanks (Figure 6) were originally designed to operate as six tanks in series, but were later set up as two sets of three tanks with the circuits operating in parallel. As shown in Figure 6, all the Whim Creek tanks are mounted on a carousel stand which is rotated to place each tank in the required position. The Sterling tanks (Figure 7) operate as fixed-bed units with two sets of two tanks operated in parallel. Stripping and acid-washing are conducted in the same tanks and the carbon is continuously cycled through these various steps for over a year before it is transferred out of its respective tank and replaced. This flexibility can serve to reduce the amount of tanks required for the various operations in an ADR circuit as well as reduce the degree of carbon handling.

Along these lines, another factor to consider in favor of closed-top systems is potential portability. Costs and difficulty for moving ground-level tanks are much reduced compared to tanks mounted two or three stories high. For smaller production facilities, an entire ADR plant can be installed in a 40-foot semi trailer for ultimate portability. Larger plants can be constructed by joining two trailers together. The Sterling plant shown in Figure 7 is an example of a trailer-mounted ADR plant.

In comparison with the cascading tank system, other advantages of the closed-top system include major capital cost savings in support structures and building size along with savings in heating costs during winter months. Operators are not exposed to cyanide-bearing solutions since the leach solutions are not exposed. Also, the possibility of air entrainment in the solution fed to each tank is eliminated. Because the tanks can be pressurized, carbon can be transferred out of a tank using direct water pressure rather than sucking the carbon through an eductor. This can cut down on carbon losses due to attrition and can simplify the transfer system.

Windows mounted in the side of the tanks at a convenient viewing height allow operators to view the behavior and level of carbon inside the tanks. This eliminates some operating headaches in adjusting both column flowrate during adsorption and carbon levels during carbon transfer. Also, since the tanks are all mounted at ground level, operators are not required to climb stairs or ladders to perform their duties.

Access to the inside of the tanks in the closed-top type system is much more difficult than for the open top cascading system. Although this is advantageous from a security standpoint, it can cause problems and delays when performing maintenance on distribution plates, etc. Proper design should include manholes to make access easier. Obtaining carbon samples can also be difficult, especially if the carbon is never transferred out of a given tank. Inter-stage solution samples are easily obtained through a valved sample port.

The previous discussions regarding the inherent flexibility of closed-top tanks do not mention the costs associated with this flexibility. While structural costs are much lower, a properly designed adsorption system for simple series operation must include provisions for by-passing individual tanks for maintenance purposes. This piping would be somewhat more extensive than the simple launders used when by-passing a cascade-type tank.

If a closed-top system is constructed to incorporate stripping and acid-washing in the adsorption system design, piping and installation costs can become high and operation of the plant can become complicated and sophisticated. However, if this is the case, direct comparisons with an adsorption-only cascading system cannot be made since a cascading tank system does not offer the flexibility for this arrangement.

Although the gold/silver industry is quite familiar with pressurized operations such as pressure filtration and compressors, an important point to remember when considering this type of system is the safety factor encountered when operating closed tanks under pressure. The tanks must be designed to withstand operating pressures which are relatively low (<20 psi), but the possibility always exists for over-pressurization by pumping against a closed tank discharge valve. To avoid unpleasant consequences, safety pressure relief valves and/or rupture discs should be employed and operators must be well trained in performing the various tasks requiring pressurization. The tank depicted in Figure 9 is designed with both relief valves and rupture discs.

TOWER-TYPE ADSORPTION COLUMNS

The final style of continuous/batch adsorption system to consider is the multi-stage adsorption tower or column. This system incorporates several stacked sections each containing a batch of carbon and separated only by a distribution plate. There

are several operations in the western United States that use this type of adsorption system. The Carlin Gold Mining Company employs a five-stage carbon column for processing tailings overflow water before recycling it to their milling circuit (6). Fischer-Watt's Hayden Hill property in California uses a five-stage tower for processing solutions from their 1000 ton per day heap leach. Western States Mining's Goldstrike Mine in eastern Nevada uses two sets of two four-stage columns to process their heap leach liquors and they are presently building another heap leach plant at their Drum Mountain property in western Utah using tower-type adsorption columns.

A diagram of a typical five-stage adsorption column is presented in Figure 10, and the Hayden Hill and Goldstrike columns are pictured in Figures 11, 12, 13, and 14. Operation of these columns is straight-forward. As Figure 10 shows, the pregnant solution is pumped into the bottom of the tower and overflows from the top over a carbon catch screen before flowing to the barren pond. Eductors are located between each stage to transfer carbon down the column as it gradually becomes loaded (for smaller columns, simple gravity-flow carbon transfer is possible). Windows are located in each stage for observation of carbon behavior. Valved sample ports are properly located to obtain both solution and carbon samples.

A major selling point of this type of system is the low capital cost. Structural and piping costs are reduced significantly, although a solid concrete foundation is required. Floor space requirements are very small, but height requirements usually result in outdoor operations. Portability is somewhat simplified since only a single piece of equipment must be moved for each multi-stage adsorption system. Semi-portable ADR plants capable of processing 1000 tons of ore per day have been built.

As mentioned previously, the height of the tower usually means that the tower will be installed out of doors. During periods of colder weather, a certain amount of freeze protection is required to protect eductor and water supply lines. Insulation and heat tape can help here, especially in conjunction with a positive draining eductor system. However, it should be pointed out that this type of system lends itself much more easily to outdoor processing since inter-stage piping or launders are not involved.

There are several inherent disadvantages to this type of system. Foremost is the lack of flexibility. If maintenance of any kind is required on one of the stages, the entire adsorption operation must be shut down. If the required repair work is major, this could result in a very substantial loss of production, although good heap leach system design includes enough pond volume flexibility to handle a certain amount of downtime. Also, access to the inside of the individual stages is very difficult and dangerous, even with properly designed manholes in each section.

Operating an adsorption tower presents certain headaches. A major problem is the necessity of the operator to climb up and down the ladder to perform sampling and carbon transfer duties. Some operations have employed remote-operating valves so that inter-stage carbon transfer can be initiated from ground level. However, it is still necessary to verify positive carbon transfer and to determine when correct carbon levels are attained in each stage. If an operator becomes lazy, it is possible to end up with a double batch of carbon in one stage while another stage is completely empty. Also, since eductors tend to become plugged on occasion, valves are required to isolate the eductor for removal and repair without draining the entire column. These valves are also necessary to prevent short-circuiting of a portion of the leach solution through the eductor lines.

DISCUSSION OF COMPARISONS

The preceding descriptions contained some comparisons between and among the three systems. In general, the cascading system would be the most expensive but would be preferred from a tank maintenance point of view. Operating problems are not excessive and based upon the number of these systems presently in use, the design is quite successful. It is noted that the cascading-type design is used in virtually all of the larger processing plants (greater than 1000 tons per day), probably because the designers wanted to draw on the experience of the earlier models and operators. However, the traditional hesitation by the mining industry to accept new or unfamiliar technology is evident.

The closed-top tank system would be in the middle of a cost comparison and has the advantage of extreme flexibility regarding flow path, multi-function potential of the tanks, and method of carbon transfer. Operating problems are minimal, but maintenance on the inside of the tanks would be somewhat difficult. This system can also be made much more portable than the cascade system.

The tower adsorption column is by far the cheapest of the systems for flowrates above 100 gallons per minute and is the most straight-forward to operate during the adsorption cycle. However, flexibility is extremely limited and operating problems exist during carbon transfer. Maintenance on the inside of the tower is very difficult and potentially dangerous. Cold-weather operation of an outdoor adsorption system would present less problems with this system. Portability of this system is relatively good.

It is difficult to recommend any one system since so many factors must be included in the decision-making process. Certainly, operator preference must be considered since people tend to have more faith in a familiar system. However, proper design of any of the systems will yield a functional and cost-effective plant. If flexibility is considered important, the closed-top system cannot be surpassed. This is especially true if the adsorption system were designed integrally with the stripping and

electrowinning systems. As far as maintenance and operating problems are concerned, none of the systems stand out as clear winners. Cost tends to be a major factor in choosing equipment designs and styles, but in this case the most expensive system is the industry favorite, especially in the larger plants. It is felt that this will change as more of the other types of designs are put into production on larger-scale projects.

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(2) McQuiston, F. W., Jr. and Shoemaker, R. S., Gold and Silver Cyanidation Plant Practice, Vol. II., SME of AIME, 1981, New York, New York.

(3) Randol International LTD, loc. cit., p. 202.

(4) Whim Creek Consolidated N.L. announcement: "Haveluck Mine, Meekatharra, Western Australia - Recent History," 1981.

(5) Parrish, I. S., Austin, G. G., and Kappes, D. W., "Sterling Mine Joint Venture: Anatomy of a Small Underground Mine and Heap Leach," AIME annual meeting, Dallas, TX, 1982.

(6) Randol International LTD, loc. cit., p. 417.

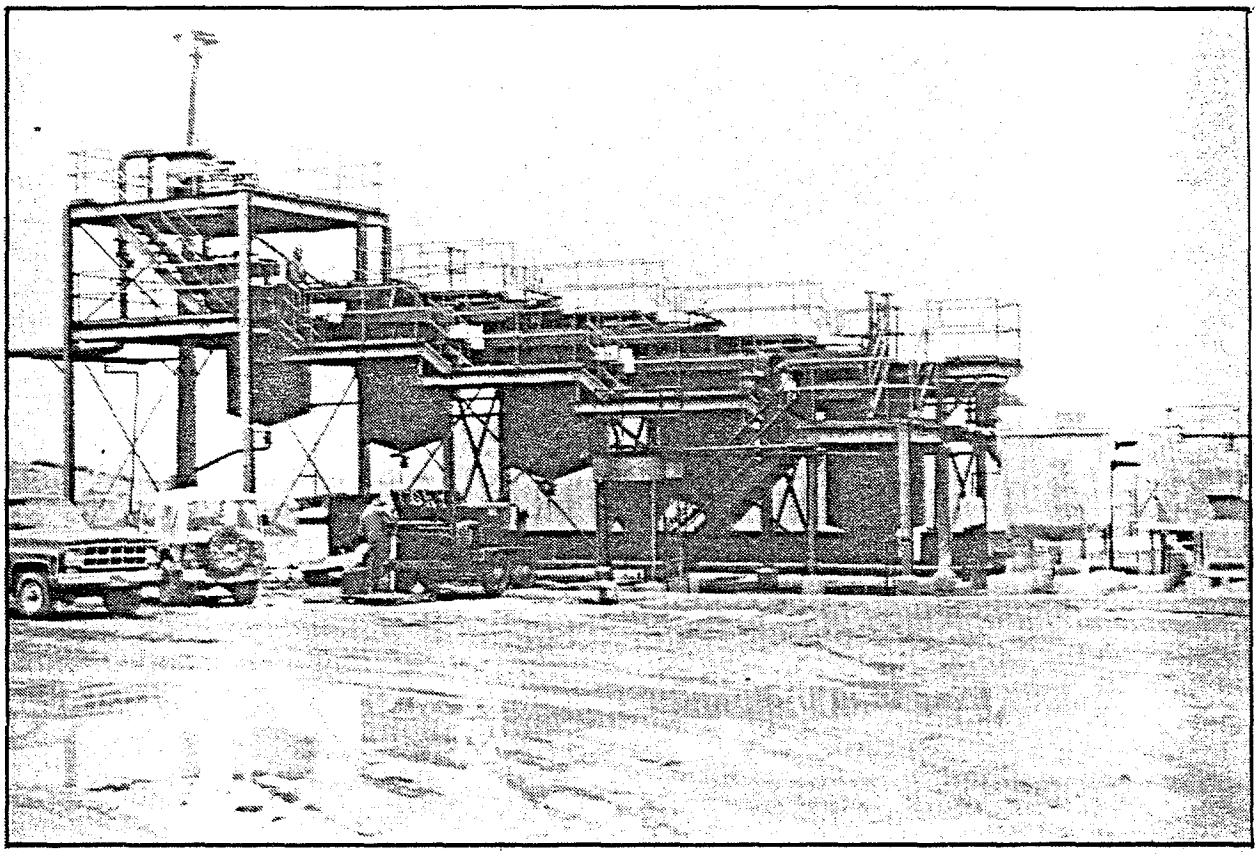

Figure 1. Cascading System - Newmont's Maggie Creek Operation

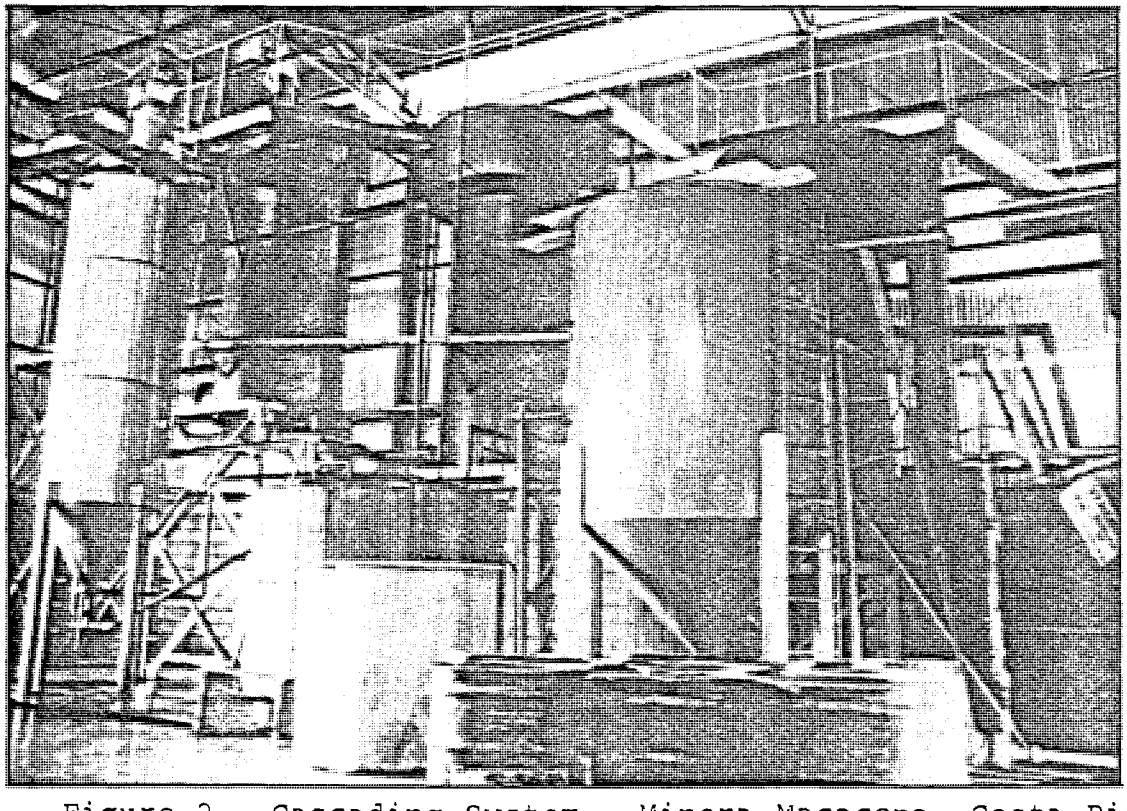

Figure 2. Cascading System - Minera Macacona, Costa Rica

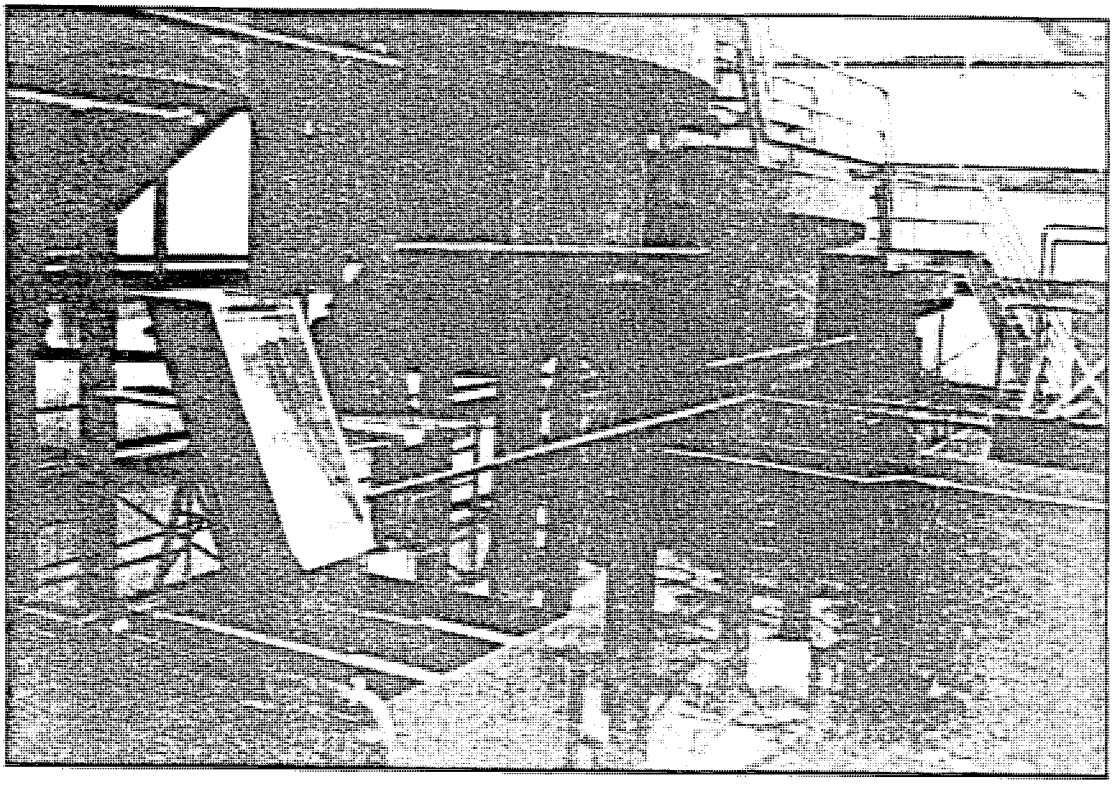

Figure 3. Cascading System - Goldfields' Ortiz Operation

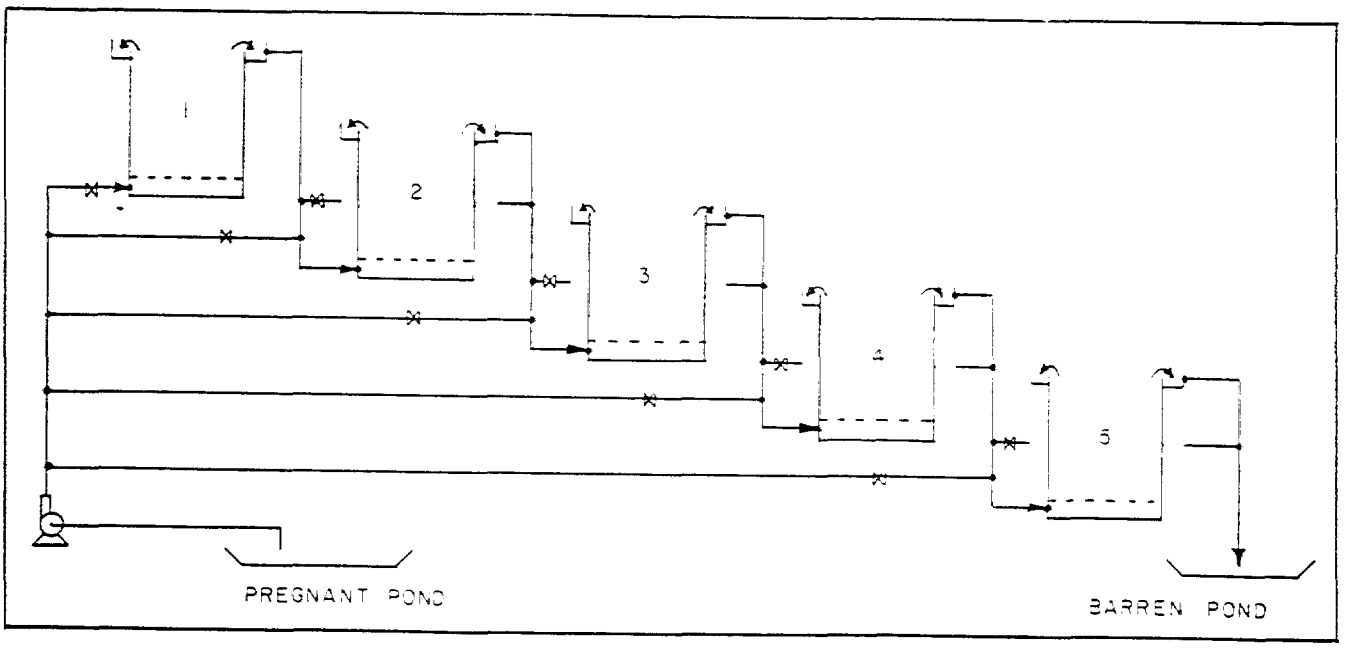

Figure 4. Typical 5-Stage Cascading System

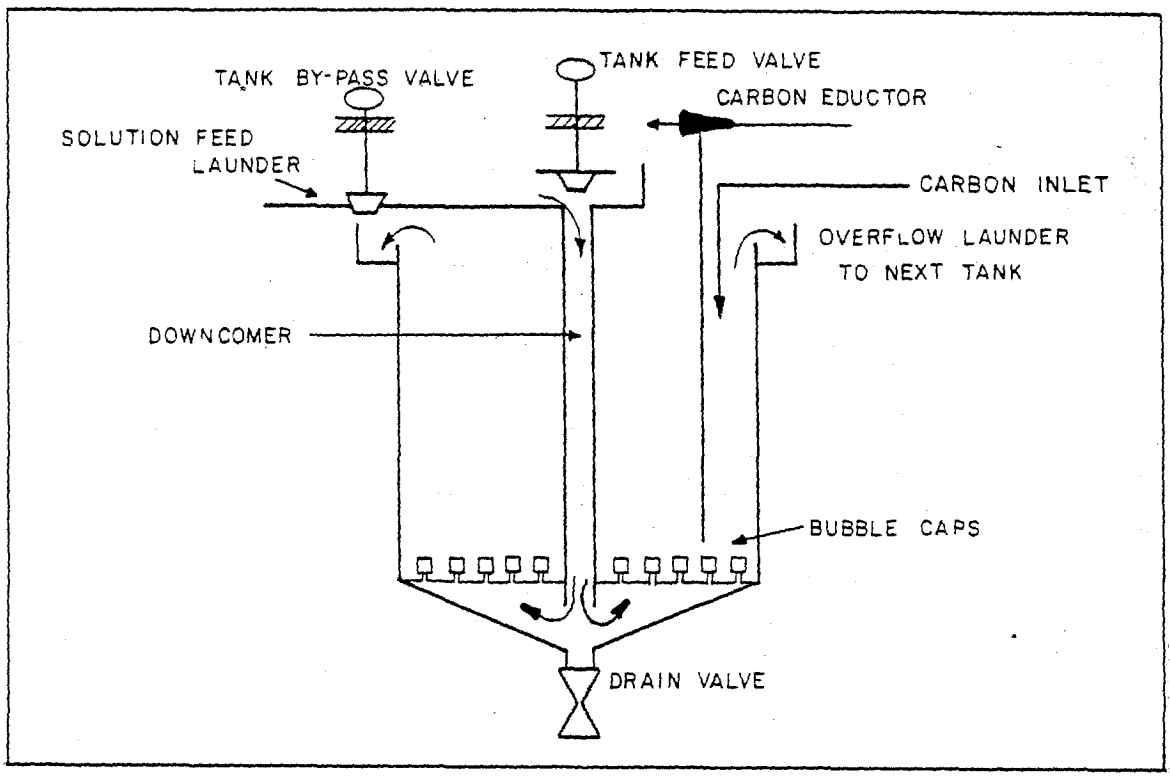

Figure 5. Typical Cascade-Type Tank

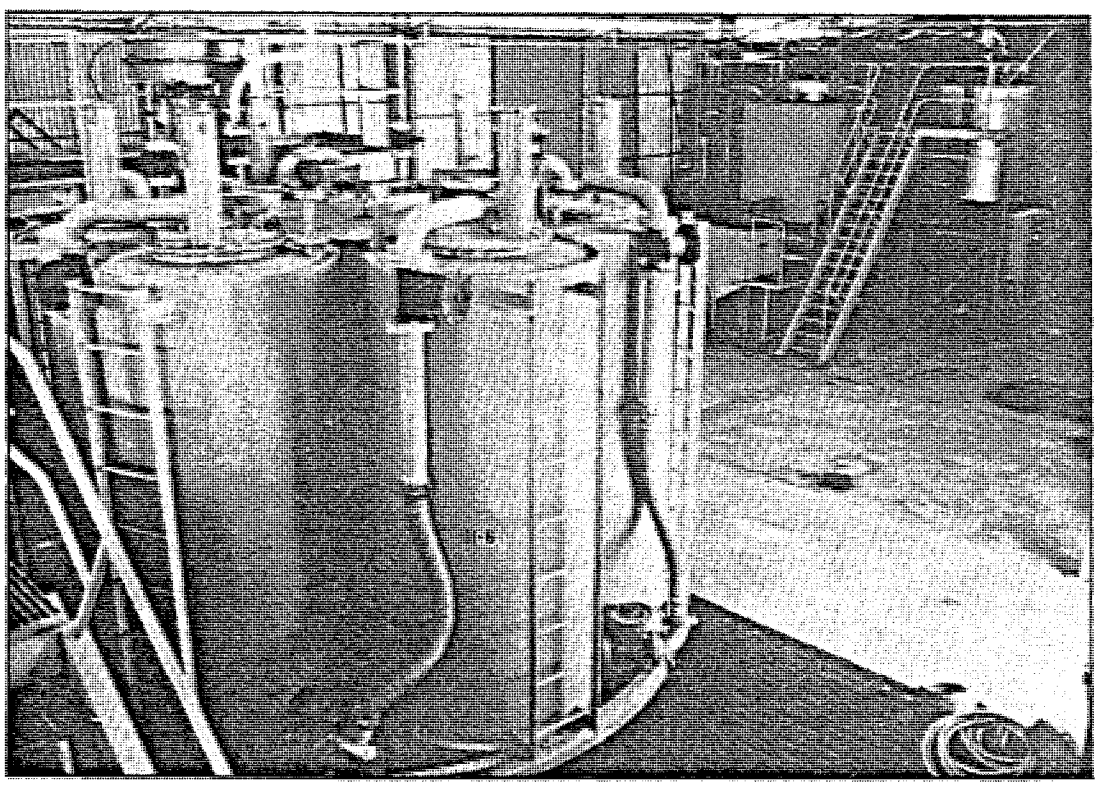

Figure 6. Closed-Top System - Whim Creek

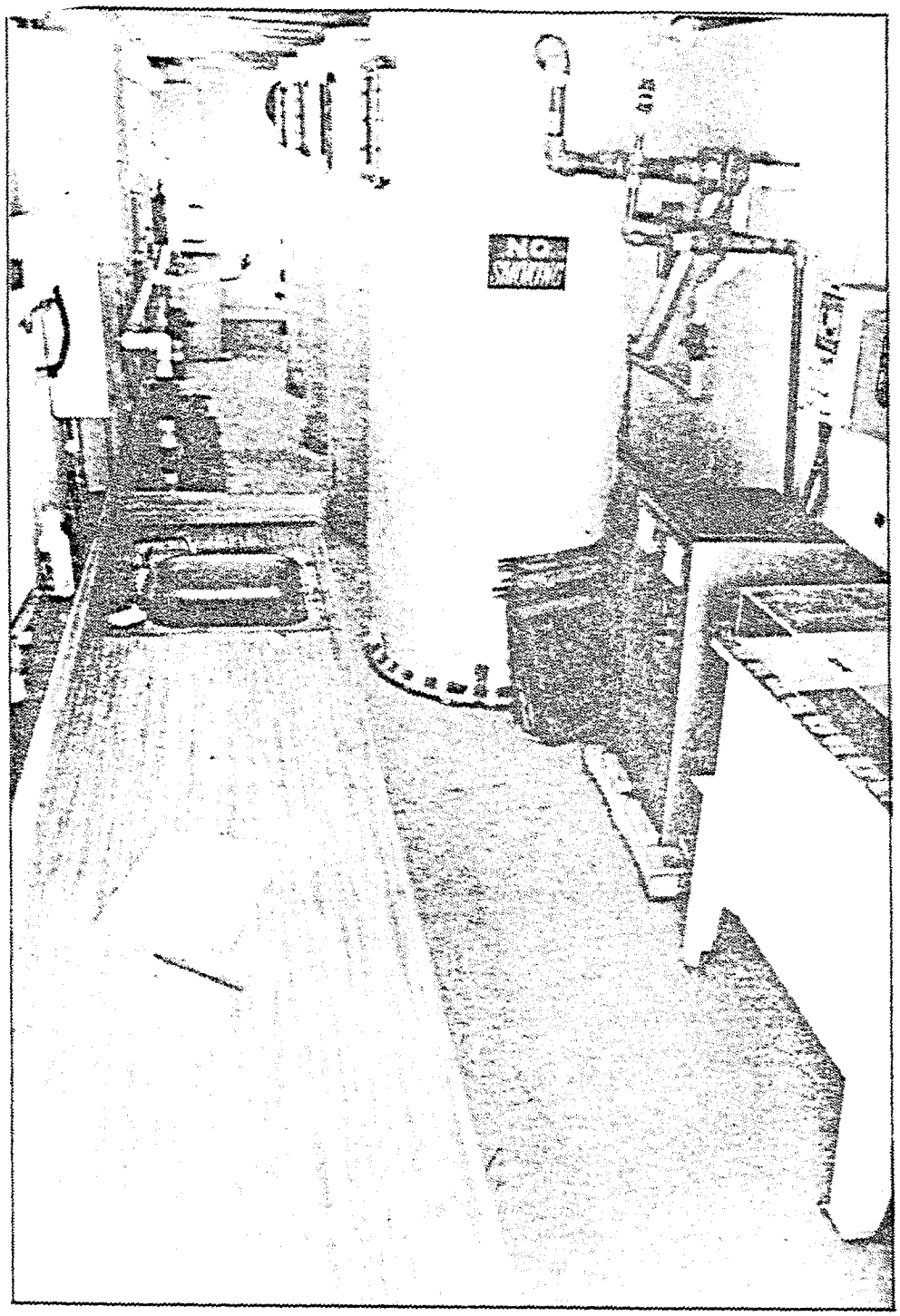

Figure 7. Closed-Top System - Sterling Mine

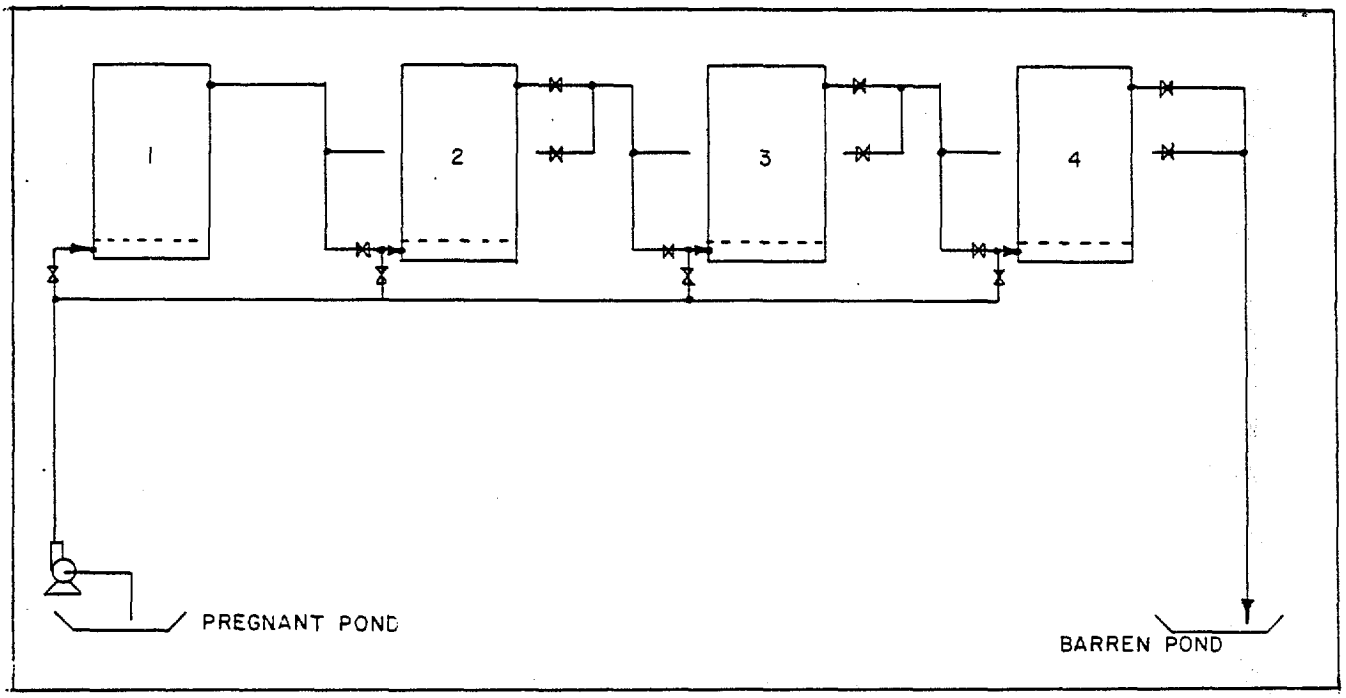

Figure 8. Typical 4-Stage Closed-Top System

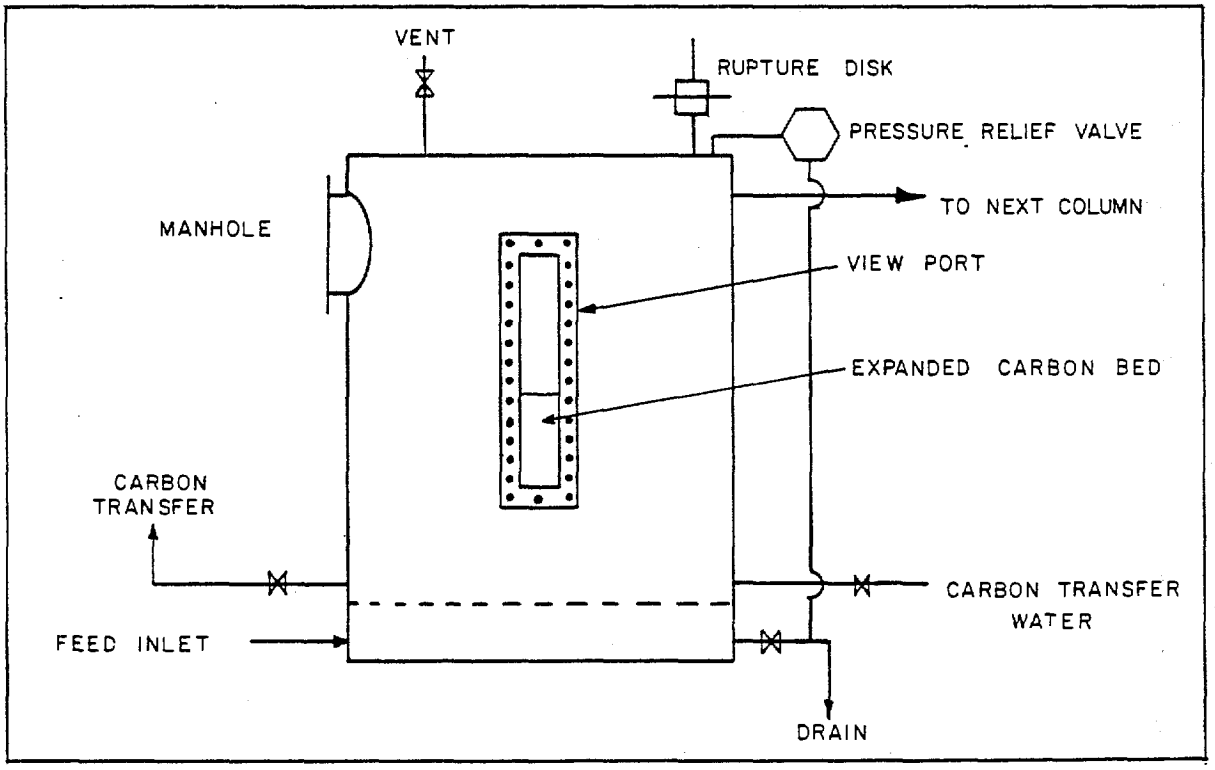

Figure 9. Typical Closed-Top Tank

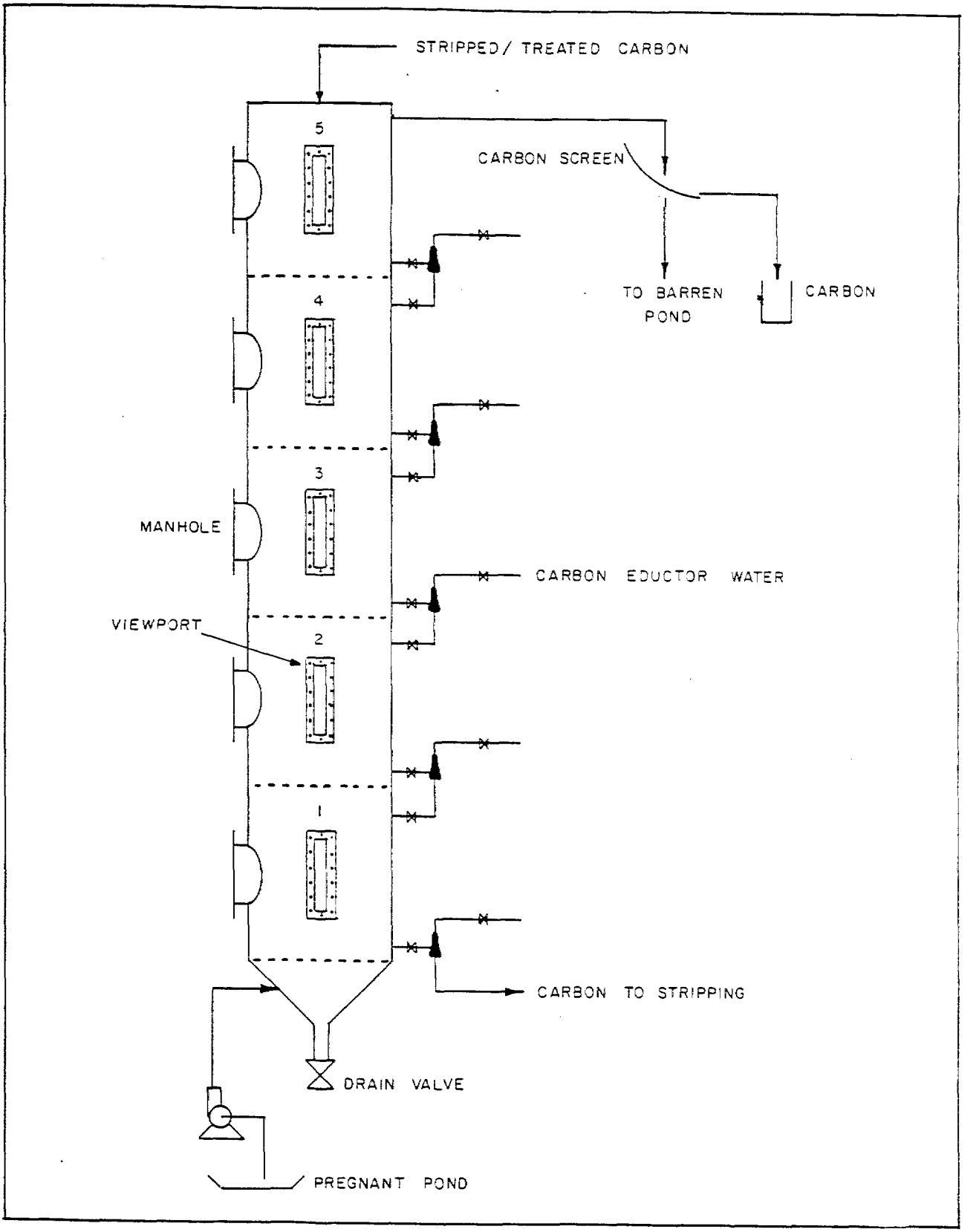

Figure 10. Typical 5-Stage Adsorption Column

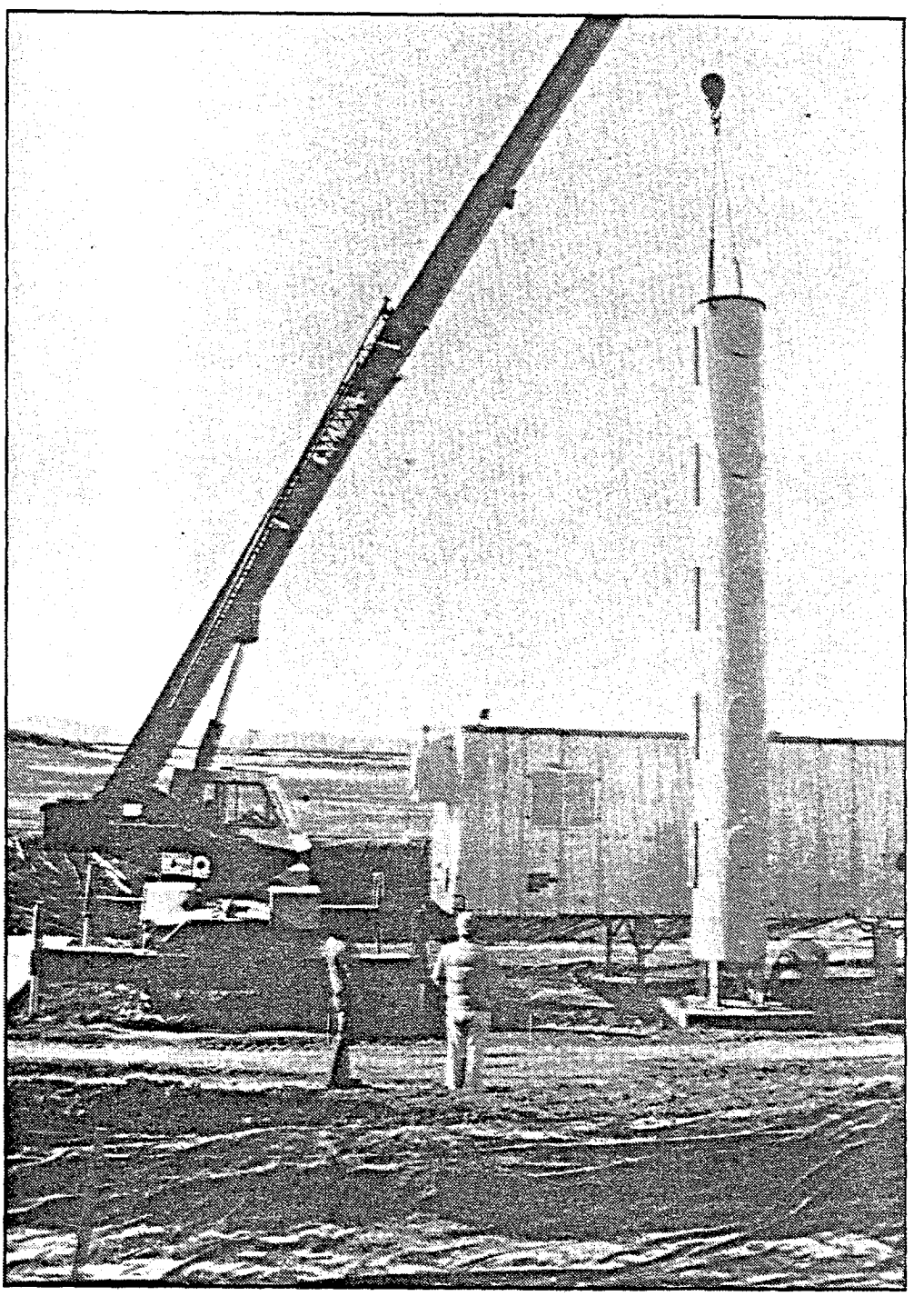

Figure 11. Installation of Adsorption Column - Hayden Hill

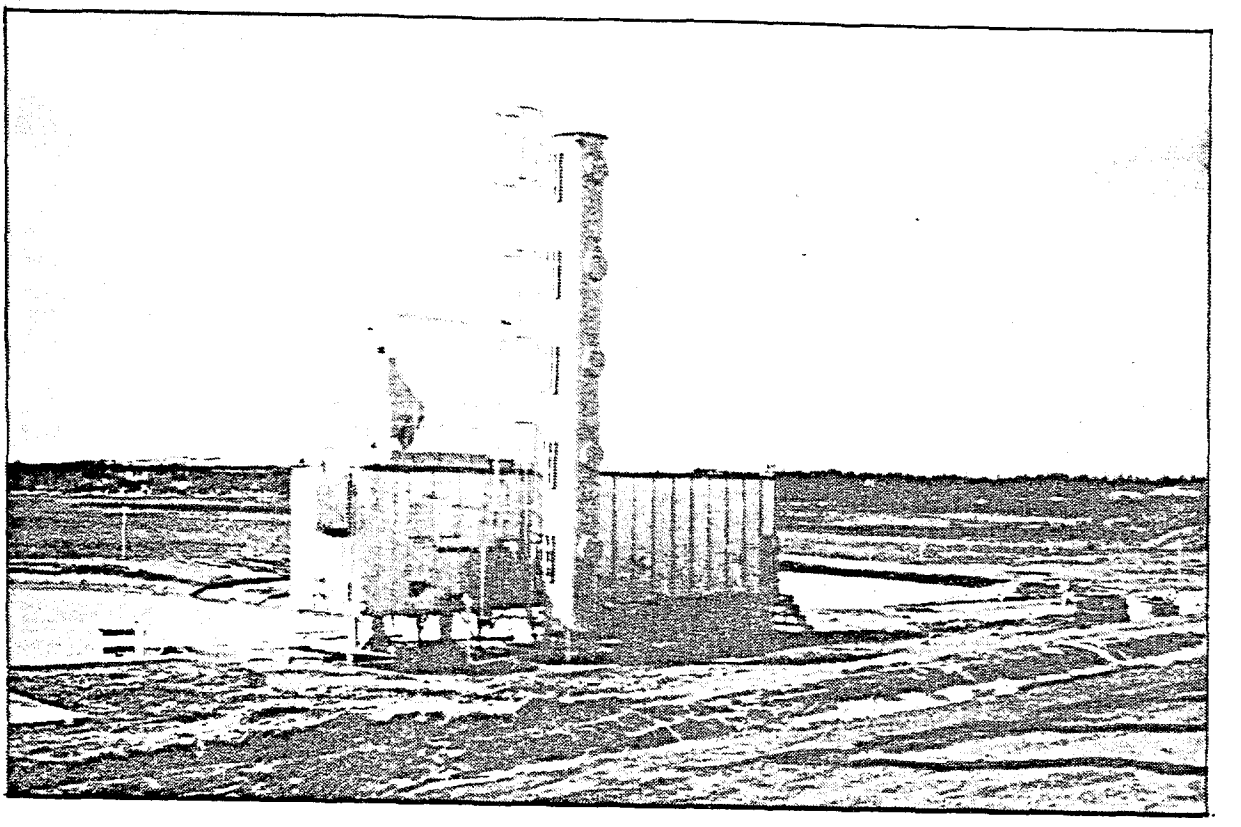

Figure 12. Adsorption Column with Stripping and Recovery Plant - Hayden Hill

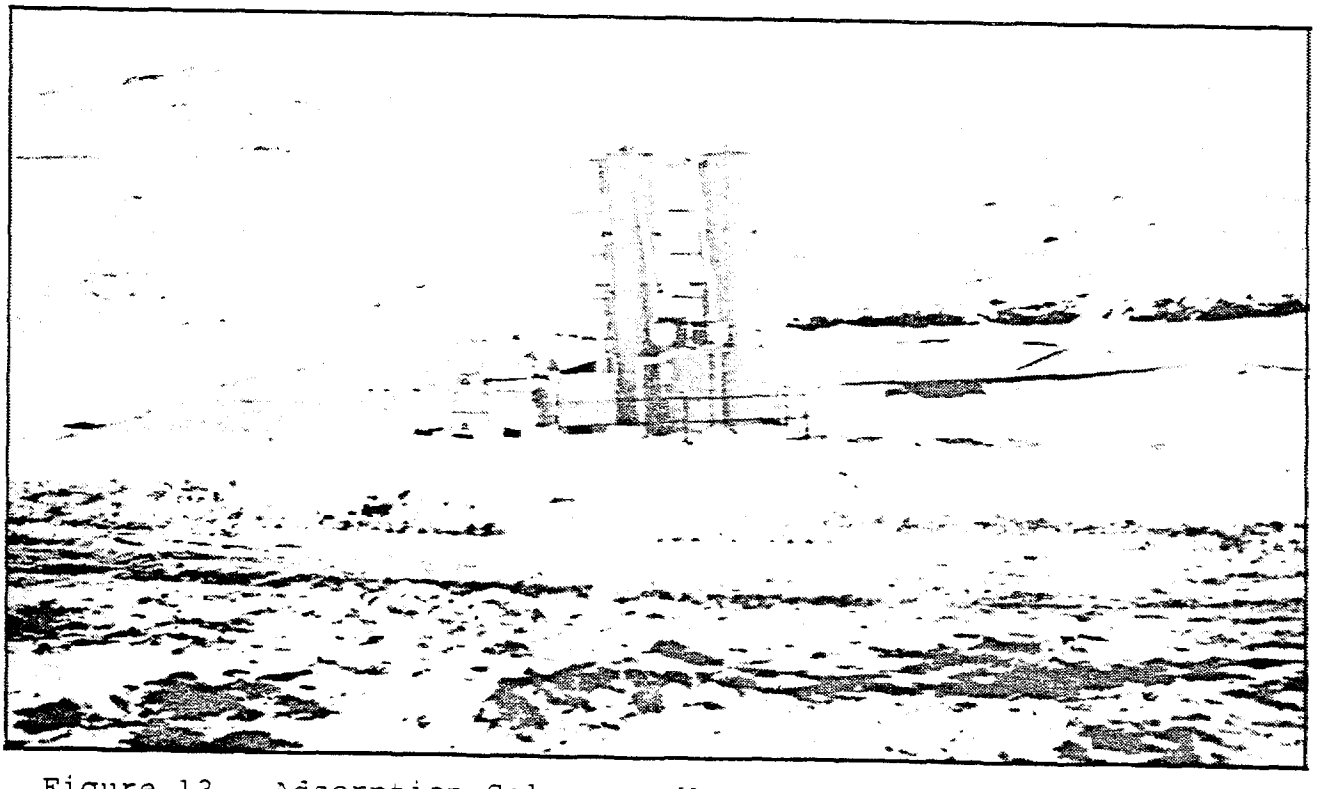

Figure 13. Adsorption Columns - Western States' Goldstrike Mine

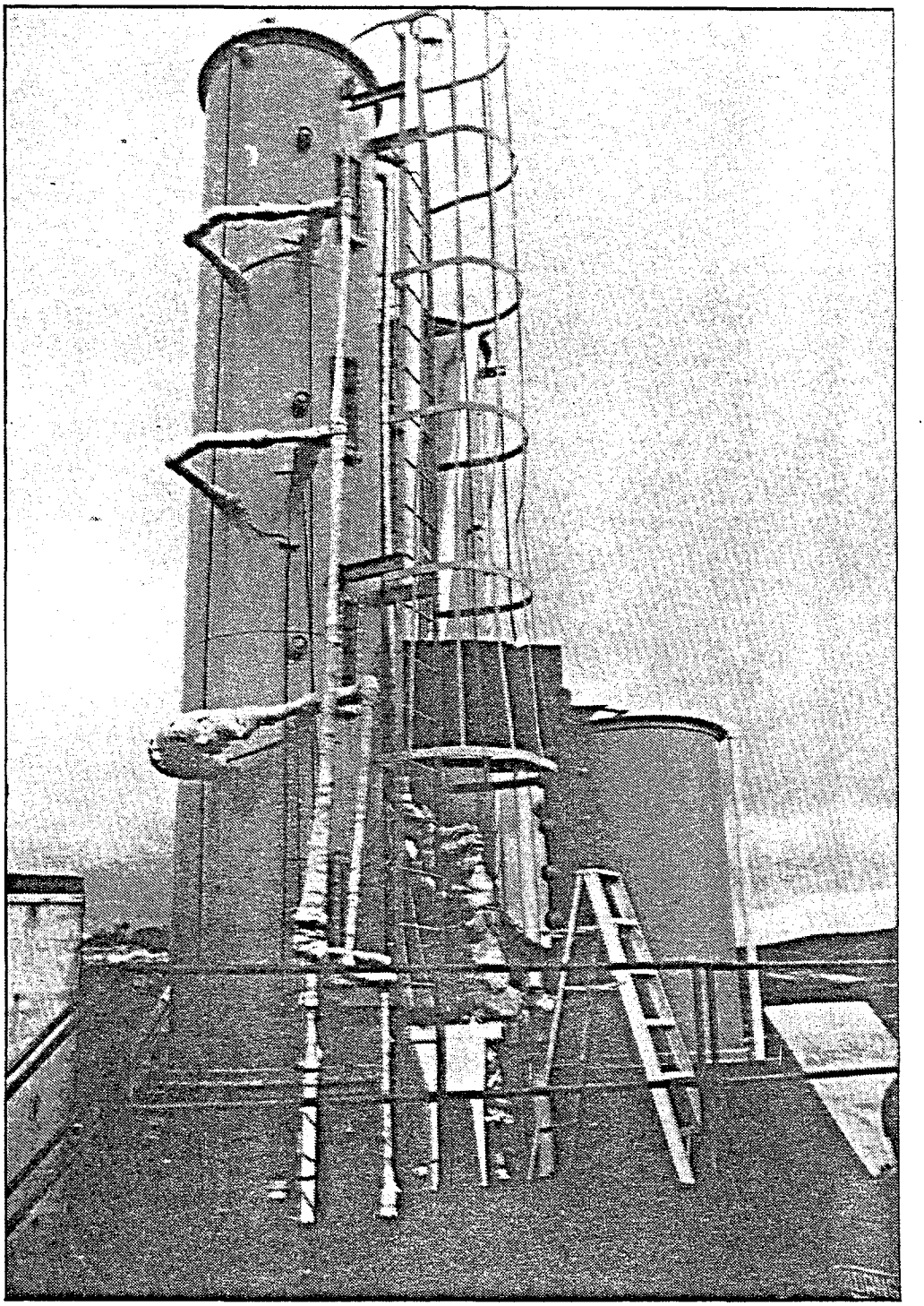

Figure 14. Adsorption Columns - Western States' Goldstrike Mine

MINING ENGINEERING

JUNE 1984

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ASARCO

JDS

Exploration Department
Frederick T. Graybeal
Chief Geologist

July 2, 1984

Dr. Peter G. Vikre, Manager
Great Basin Exploration Division
Reno, Nevada

Carlin Exploration Program
Nevada

Dear Mr. Vikre:

Thank you for forwarding Mr. Maher's report of March 13, 1984 on Stream Sediment Sampling for Carlin Gold Deposits. I agree that the report is well written and well organized. More importantly, I understand that in every case where an anomalous value was encountered during Mr. Maher's 1983 field work, those values were immediately followed up with additional sampling and staking where necessary and that this report was prepared later as time allowed. I presume Mr. Maher will approach the 1984 program with the same instinct for instant action following consultation with yourself and that formal reports will be forthcoming as time permits.

I have a number of comments regarding the statistical analysis used by Mr. Maher to generate threshold values. These threshold values were derived by collating all data from the 1983 and several earlier field seasons. I am not convinced that the data collected from these areas can be treated as part of a single population on a quantitative basis. There is no question that the elements subjected to statistical treatment are significant with regard to Carlin exploration (although I will discuss silver presently), but rather I am not sure that threshold values are similar in different mountain ranges. Mr. Maher appears to recognize this problem and has derived threshold values by taking an arbitrary value of two standard deviations above a mean generated by plotting only background populations (pages 5 & 6). However, it is not entirely clear why the 2x standard deviation parameter was selected. This technique (advocated in the first Hawkes & Webb volume) always seemed arbitrary to me because it eliminates threshold variations which occur when certain elements are disbursed through a greater volume of rock than others, leading to a lower

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threshold value because a greater percentage of the total will be anomalous. In addition, there is also the problem of decay of values down-drainage from the source causing an additional complication which results when threshold values are a function of sample density or position in the drainage basin.

Mr. Maher's probability diagrams for Au, Hg, and As were obviously constructed with a great deal of thought. Threshold values are clearly defined by breaks in curves for both Hg and As, but I am not at all convinced that the probability diagram for Au defines a clear threshold value. Given the axes used in the probability diagrams a threshold value is marked by an upward break in the curve which represents the point where a greater number of samples with high values becomes present than would normally be expected from the probability distribution of the background population. There is no such break on the probability diagram for gold. Therefore, it is not clear to me that we have recognized a threshold value for gold.

To carry this further, I have tabulated below for selected rock types ratios of threshold/background values for the various elements considered important by Mr. Maher. This tabulation is similar to his Table 2, except that I have added Au, As, and Hg, and have included a category for limestone. Some of this additional data comes from Hawkes and Webb.

Threshold/background values

Shale	Limestone
As 9	Hg 5
Ag 7-10	Ag 2.5
Sb 3-5	Au 2
Tl 1.5	
Au 1	
Hg 0.4	

Note that in both the shale and limestone categories the ratio of threshold to background for Au is close to unity. This indicates that Mr. Maher's threshold values for gold are at or very near the average background gold contents for those rocks and further contributes to my suspicion that the threshold selected for gold by Mr. Maher is simply too low. I'm not going to argue this point further and neither you nor Maher should feel compelled to reply in print. My suggestions would be to proceed as recommended by Mr. Maher and we can discuss this further in the field.

I agree with all of the principal characteristics of Carlin-type systems listed on Table 3 and would add the following comments. Mr. Maher's text emphasizes the Paleozoic age of Carlin-type host rock. I recall that in the initial stages of Carlin exploration, roughly 15-20 years ago, dogma held that Roberts Mountain Formation was the only favorable host rock. Gradually exploration groups came to realize that not only is any silty limestone an acceptable host rock, but that the Roberts Mountain Formation grades laterally into relatively unfavorable lithologies. At the present time, we look for any favorable lithology regardless of the lateral correlations. I think it might also be reasonable to forget about the age of the host rock as long as the lithology is acceptable. Unless Mr. Maher can show that Paleozoic silty limestones had an unusually high Au content which could be concentrated by hydrothermal fluids I suggest that any silty limestone might do as an initial target for stream sediment sampling. A second comment concerns the presence of high silica alteration zones which, although present, are in my experience relatively minor portions of the system volumetrically. Concerning elemental associations I note that Mr. Maher correctly omits silver. Carlin deposits are characterized by the lowest Ag/Au of any type of gold deposit presently known. I would add 3 additional characteristics to his Table 3.

First would be that the total size of a Carlin system may be huge, even by comparison to porphyry copper systems. Within this system mineralization may be erratically distributed and zoning patterns may be poorly defined or absent, but the systems appear to be among the largest hydrothermal systems in the world. A second characteristic I would add to Mr. Maher's table is that the Carlin-type systems often contain widespread occurrences of small (generally insignificant) mineral deposits of diverse type which may appear to be unrelated to the typical Carlin metal suite. Either of these two additional characteristics may serve to encourage further work or attract work to the area. The third characteristic is the unbelievably obscure nature of Carlin ore in outcrop.

Regardless of the foregoing, it would appear that your Carlin program is on a firm conceptual base. I have no objection to any of the areas selected by Mr. Maher, but in view of the extremely large size of some of the Carlin systems, and I think specifically of Carlin, Jerritt Canyon, and Getchell-Pinson-Preble, I wonder whether additional work within a five mile radius some of the lesser deposits such as Tonkin Spring, Northumberland, Gold Acres, Mercur, Windfall, Manhattan, and perhaps others might not also be productive. These deposits

have been known for a long period of time and the initial surge of exploration around them has likely died or been refocused as discoveries in other areas have been made. Ore deposits , including Carlin deposits, do tend to occur in clusters and the tendency for some of these ores to be discovered long after the initial deposit and by a different company is well illustrated in the Arizona porphyry copper province by the Kalamazoo discovery adjacent to San Manuel, the Superior East discovery adjacent to Magma, Van Dyke discovery adjacent to Inspiration, the Pumpkin Hollow discovery adjacent to Yerrington, Sunnyside discovery adjacent to Red Mountain, Mission discovery adjacent to Pima, the Casa Grande discovery adjacent to Sacaton, and there are others. The geological reasons for these discoveries are different, but all were made long after the initial discovery and by a different company. It seems that a company's efforts became focussed on their discoveries and a certain complacency developed with respect to the possibility of additional discoveries in the same area. Freeport has apparently covered their flanks well at Jeritt Canyon, but a review of county claim maps would show if there is open ground near some of the "lesser" Carlin-type deposits and if some existing "close-in" blocks are held by someone other than the mine operator. The large size of Carlin systems and the extremely obscure appearance of Carlin ore in outcrop justify further work within these districts.

I sense at times the desire of some of our geologists to explore and make a discovery in "virgin territory". They forget that virgin territory does not always have the best potential even though they all accept the axiom that the best place to find a new mine is adjacent to an existing mine. So, I offer the suggestion that some of the known Carlin-type environments with single producing mines might yield to additional careful geological and geochemical analysis, particularly those districts where only single small mineral deposits are known to occur. Search of claim and assessment records would be a good winter project as you plan for the 1985 field season.

Very truly yours,

F. T. Graybeal

F. T. Graybeal

290
324
333
338

ABSTRACTS

OF PAPERS TO BE PRESENTED AT THE MEETING
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AND ASSOCIATED SOCIETIES,
ANCHORAGE, ALASKA
MAY 30-31, JUNE 1, 1984

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Mineral Beneficiation Department

August 20, 1984

Memorandum to: Mr. H. G. Kreis

Subject: Your memorandum of July 27th to Mr. Rideout - Assay Values at the Aquarius Mine, Ontario, Canada

Your memorandum discusses the problem of obtaining accurate ore grade assays at Aquarius and proposes a solution to this problem by treating an entire 50 pound muck sample by gravity means to recover the coarse gold and then fire assaying the gravity tailing by conventional methods. The purpose would be to reduce the "nugget effect".

I think we all agree that individual muck samples at Aquarius are somewhat less than perfect and I agree with you that your proposed procedure would give much more accurate assays of the muck samples. However, there are other considerations as discussed below.

1. The only feasible way to gravity concentrate a 50 pound sample is with a jig and, unfortunately, a jig does not lend itself to short operating periods giving total metal accountability. This is because a significant portion of the collected gold will remain in the jig bed rather than pass through the bed into the hutch where it can be easily collected. This means that each 50 pound sample would require removal of the jig bed and hand sorting gold particles from the bed material consisting of steel shot, magnetite and pyrite. This is a very long laborius procedure (± 2 hours) and not practical for each sample. In addition, the jig product would still have to be tabled to obtain a concentrate suitable for complete fusion.

2. Given sufficient manpower for the added labor and assaying involved, the proposed procedure could be accomplished and would surely give more accurate assays of the samples. However, the samples themselves are nothing more than grab samples and, as such, do not represent the round, shift, or stope production very accurately until, as you pointed out, a sufficient number are averaged together. I seriously question whether the added cost and time involved with the proposed sample treatment are warrented given the inaccuracy of the sample in the first place.

current assay method is not accurate enough to know if the muck samples are any good.

3. We gave considerable thought to this problem in 1982 during the original sample plant operation. In addition to gravity concentration we considered grinding to flatten large gold particles and then screen them out before sampling the screen undersize. However, the same problem of inaccurate original samples caused us to abandon the plan. The final decision was to rely on large tonnages treated in a mill with accurate tailing assays and actual concentrate sales figures used for metal accounting.

implies samples are worthless

D. E. Crowell

DEC/ab

cc: M Rideout
C Johnson
R Mielke
J Sample

JDSell
SAAnzalone
ARRaihl/TDHenderson/File AP 2.6

JDS

ASARCO

Southwestern Exploration Division

October 4, 1984

File Memorandum

Noranda's Tecoma Deposit
Tecoma District
Elko County, NV

During the UGA Field Trip a few copies of Ian Douglas' paper was passed out. I attach a copy.

The property is in the very low foothills some 3½ miles east-southeast of the Jackson Mine and part may actually be in Box Elder County, UT.

Ian was part of a Noranda team working this part of Nevada. They had sampled the Jackson area and found some values and were working outward checking all the jasperoids. It was actually Ian's wife who found the outcrop--she was not part of the team and was away from the group so as not to get "underfoot" or bother Ian!--and mentioned that she saw this interesting jasperoid zone with an old pit nearby. The first sample yielded 6.5 ppm gold and the area was quickly staked. Phelps Dodge picked up the adjacent school section in Utah.

Noranda has sold the drilled-out deposit, while retaining a royalty, to Western States Machinery who is presently doing some large scale tank leach tests. Inc., a wholly owned subsidiary of S. J. Grove and Son.

The deposit dips steeply to the east and though PD may be near to the edge of the proposed open-pit, their depth at the sideline and a rising hill on their property probably precludes any mining by them. Apparently most of the ore is on the Noranda claims as "barren" holes were said to be drilled eastward toward PD's ground. PD has drilled several holes on their block.

Ian's Master's thesis at Stanford has been accepted and should be on their library shelf soon.

James D. Sell

James D. Sell

JDS/cg

Attachment

cc: FTGraybeal (w/attach)
PGVikre "

Tecoma District
Elko Co., NV

GEOLOGY OF THE TECOMA DISSEMINATED GOLD AND SILVER DEPOSIT

IAN H. DOUGLAS

WILLIAM M. ORIEL

Paper slightly corrected and added to, then presented at the December 1984, NW Mining Conference, Spokane, WA. Paper No 25.

with Maps & Sections

See

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ABSTRACT

The Tecoma deposit is in a mineralized low angle, younger-over-older fault zone between the Devonian Guilmette Fm. and the Mississippian Chainman Fm. Ore is predominantly in silicified carbonate and shale along the fault contact, with lesser amounts of dolomite and shale ore above and below the fault, respectively. The Tecoma deposit contains geologic reserves of approximately 1.5 million tons of .05 OPT gold and 3 OPT silver. Ore contains barite, pyrite, hematite, acanthite, native gold, arsenic oxides, and mercury.

The deposit is in the eastern fringe of the Tecoma mining district. Historical production was from high-grade silver and lead replacement bodies along the axis of a northwest-trending anticline at the Jackson Mine. The district contains miogeoclinal rocks of Devonian through Permian age that are complexly faulted and folded, and intruded by altered quartz monzonite porphyry dikes and Tertiary rhyolite domes. Silicified bodies replacing carbonate and shale (jasperoid) are common throughout the district but are concentrated near the porphyry dikes.

INTRODUCTION AND PREVIOUS WORK

The Tecoma deposit, located in the Tecoma district, is an

epithermal carbonate-hosted disseminated gold and silver deposit discovered in late 1980 by Noranda Exploration Inc. It is geologically similar to the Rain and Alligator Ridge gold deposits and the Taylor silver deposit. It is unusual in containing approximately equal dollar value of gold and silver at an approximate ratio of 1:60 Au:Ag.

The district was studied in detail by Ian H. Douglas as part of a Master's thesis at Stanford University (Douglas, 1984). The district is mentioned in Regional studies by Hill (1916), Granger et al (1957), Lovering (1972), and Smith (1976). County maps by Hope and Coats (1970) and Doelling (1980) cover the Elko and Box Elder County sides of the district, respectively.

LOCATION

The Tecoma district is between latitudes 41 ,25' and 41 ,30' north and longitudes 114 and 114 ,5' west in northeastern Elko County, Nevada and western Box Elder County, Utah (Figure 1). It lies within the Jackson Spring 7.5 minute quadrangle, in T41N, R70E (Nevada) and T8N and T9N, R19W (Utah). The district is 15 mi northeast of Montello, Nevada, a town situated along the Southern Pacific Railroad, and 5 mi north of Nevada State Route 233-Utah State Route 30. The area lies at the extreme south end of the Goose Creek Mountains. The topography consists of gentle foothills intertonguing with flat alluvial pediments of Tecoma Valley. The elevation ranges from

5,100 to 6,567 ft.

HISTORY AND PRODUCTION

The Jackson Mines, discovered in 1906 (Hill, 1916), were the principal producers in the Tecoma district. Three thousand tons of argentiferous cerussite ore were mined between 1906 and 1947, and 1,814 tons were mined between 1947 and 1951 (Granger et al, 1957). Total metal content was 72 oz gold, 46,697 oz silver, 5,917 lbs copper and 2,243,512 lbs lead (Granger et al, 1957). The Queen of the West Mines, located 1 1/2 mi east-southeast of the Jackson Mines were the only other producers in the district.

Noranda Exploration Inc. aquired its initial land position in the district in 1980. The Tecoma deposit was initially sampled in November, 1980 and the first ore hole drilled in September, 1981. A 6 ft shaft of unknown vintage was dug in the footwall of the deposit, 20 ft from the discovery outcrop. Three rotary holes were drilled by and unknown party, one of which was collared in ore.

DISTRICT GEOLOGY

Miogeoclinal rocks of Devonian through Permian age are exposed in the Tecoma district. The units consist of the Devonian Guilmette Formation, Mississipian Joana Limestone and Chainman-Diamond Peak Formation, Pennsylvanian Ely Limestone, and Permian Buckskin Mountain Formation and Pequop Formation. These units generally strike north-northwest and dip 50 to 80 degrees northeastward. They are intruded by Tertiary(?) Quartz

monzonite porphyry dikes and Miocene Rhyolite domes. Miocene Rhyolite flows and tuffs lie disconformably on, and are faulted against the Paleozoic section. The units are complexly faulted by high- and low-angle faults. Figure 2 depicts the simplified geology of the Tecoma district.

Six Paleozoic units are exposed in the Tecoma district. The Devonian Guilmette Formation is a thick-bedded, fossil-poor limestone. There is an arenaceous limestone near the top of the unit and quartzite near the base. At all exposures, the top of the formatin is a low-angle fault contact with Chainman Formation shales (generally silicified) overlying the limestone. These shales were previously intrerpreted as Pilot Shale by Douglas (1984). The Mississippian Joana Limestone is a medium-bedded, cherty, fossiliferous limestone. The formation is over 800 ft thick at the Queen of the West mies (Figure 2). The base is not exposed and the top contains interbedded chert-pebble conglomerate of the Diamond Peak Formation. The Diamond Peak Formation and Chainman Formation are time equivalent facies that are interbedded in the Tecoma district and mapped as Chainman-Diamond Peak Formation. The Diamond Peak Fm. is chert-pebble conglomerate interbedded with chert sandstone and siltstone. The Chainman Fm. is mudstone and shale interbedded with chert sandstone and siltstone with a thin silty limestone unit near the base. A recent conodont study from a sample of this limestone yielded an age of Late Kinderhookian (D. Miller and A. Harris, personal communication). The Chainman-Diamond Peak Formation is

overlain by the Pennsylvanian Ely Limestone (?). The Ely Ls. is a thick-bedded bioclastic limestone that crops out in the far western part of the Tecoma district. The Ely Ls. is absent in nearby ranges and may be preserved in the Tecoma district due to local non-erosion (Stevens, 1981). The top of the Ely Ls. is not exposed in the Tecoma district. Overlying(?) the Ely Ls. is the Permian Buckskin Mountain Formation. The Buckskin Mtn. Fm. appears to be in excess of 3,400 ft thick in the Tecoma district and forms a large part of the exposed Paleozoic section. The formation is composed primarily of calcareous siltstone with interbedded calcareous sandstone and subordinate bioclastic and cherty limestone and weathers a characteristic purplish color. The Buckskin Mtn. Fm. is in turn overlain by the Permian Pequop Formation. The Pequop Fm. is in excess of 5,000 ft thick in the Tecoma district and is the most commonly exposed unit. It is composed primarily of silty, thin-bedded limestone with subordinate sandstone, siltstone, and bioclastic limestone. The unit weathers into characteristic light grey plates.

Igneous rocks are divided into three units: Quartz monzonite porphyry dikes, Rhyolite domes, and Rhyolite flows. Tertiary(?) Quartz monzonite porphyry dikes intrude the paleozoic section. They contain quartz, K-spar, plagioclase and biotite phenocrysts in a fine grained quartz-feldspar matrix. The dikes, intensely argillized throughout the district, are composed of kaolinite and quartz and are too altered to date. The dikes are spatially associated with

numerous jasperoid bodies throughout the district. They intrude northwest-, east-, and northeast-trending high angle faults. Miocene Rhyolite domes also intrude the Paleozoic section, along north-trending normal faults. They are flow-foliated, crystal-rich and composed of sanidine and quartz phenocrysts in a partially devitrified aphanitic matrix. Miocene Rhyolite flows lie disconformably on and are faulted against the Paleozoic section. The flows are rhyolitic in composition and contain abundant quartz and sanidine phenocrysts.

The structure of the Tecoma district is complex. Four sets of high-angle faults and one set of low-angle faults have been mapped. The oldest appear to be low-angle faults which place Chaninman Fm. shales on Guilmette Fm. This same relationship has been observed in the Pilot Mountains to the south by Miller (1982, 1983). These faults are either younger-over-older thrusts or listric-normal faults. Other faults in the district are, from oldest to youngest, northeast-trending normal (?), east-trending reverse, northwest-trending normal, and north-trending normal. The north-trending normal faults appear to represent the present cycle of Basin and Range extension.

Broad, northwest-trending anticlines are exposed at two places in the district. At the Jackson mines, the anticline is faulted along its axis near the workings. To the northwest of the Jackson mines it plunges northwest, and to the southeast of the mines it plunges southeast. A poorly defined,

northeast-plunging anticline is exposed south of the Tecoma deposit and may be covered under alluvium immediately west of the deposit.

GEOLOGY OF THE TECOMA DEPOSIT

INTRODUCTION

The Tecoma deposit is in a mineralized, low-angle fault zone between the Devonian Guilmette Formation and the Mississippian Chainman Formation. These units are displaced by a series of high-angle faults and intruded by quartz monzonite porphyry dikes. Ore is predominantly in silicified carbonate and shale along the fault contact, with lesser amounts of dolomite and shale ore above and below the fault, respectively. The Tecoma deposit contains geologic reserves of 1.5 million tons grading .05 OPT Au and 3 OPT Ag.

GEOLOGIC UNITS

Two formations host ore at Tecoma: the Devonian Guilmette Formation and the Mississippian Chainman Formation. Other units in close proximity to the ore are Permian Buckskin Mountain Formation and Tertiary quartz monzonite porphyry dikes (Figure 3).

The Guilmette Formation is a coarse grained,

recrystallized dolomite that was original a limestone. The dolomite contains few fossils and because of recrystallization, is poorly- to non-bedded. An arenite near the top of the unit is composed of well-rounded and well sorted quartz grains in a carbonate matrix. It is grain supported, 25-40 ft thick, and occaisionally has clean carbonate lenses within it.

The Chainman Formation is composed of mudstone, shale, and siltstone with thin sandstone stringers, rare chert-pebble conglomerate beds (Diamond Peak Fm.) and a limestone bed near the base. Mudstone and shale are clayey, soft, silty, limonite-stained and rarely carbonaceous where silcification has prevented complete oxidation. Sandstone and siltstone are composed of varicolored, poorly sorted, grain supported, angular chert grains and well-rounded quartz grains in a siliceous or calcareous matrix. Conglomerate is composed of angular to rounded, varicolored chert and quartzite pebbles in a sandy matrix. The limestone is thin-bedded, laminated, silty, has shaley partings and is varicolored. When silicified, the limestone is incistinguisable from silicified shale and mudstone.

The Buckskin Mountain Formation, in normal fault contact with the Chainman-Diamond Peak Formation, is composed of red, calcareous, quartz-rich siltstone. The unit forms a gentle hill northeast of the deposit with no outcrops.

Quartz monzonite porphyry dikes are composed of quartz, feldspar, and biotite phenocrysts in a very fine grained groundmass of quartz and feldspar. Quartz eyes are usually

partially resorbed and less than 1/4" in diameter though the dike has quartz eyes up to 1" in diameter. The dikes do not crop out, but can be traced easily in float and appear to intrude the PN Fault (Figure 3). The dikes appear to be pre-ore and possibly the heat source for the hydrothermal system that formed the numerous jasperoid bodies and the Tecoma deposit. The dikes do not form ore but can be enriched in Au, Ag, and trace elements.

STRUCTURE

The Tecoma deposit contains at least 5 different faults juxtaposing units against each other. A low-angle, northwest-trending younger-over-older fault is the dominant, and oldest structure in the deposit. The youngest structure is a north-trending normal fault. Because of lack of crosscutting relationships, the relative ages of the northwest-trending normal faults, east-trending reverse faults, and northeast-trending normal(?) faults are uncertain, though on a district-wide scale, the northeast-trending faults are the oldest and the northwest-trending faults are the youngest.

The low-angle fault juxtaposes Chainman Formation onto Guilmette Formation. An unfaulted section for the district includes at least 800 ft of Joana Limestone between the Guilmette and Chainman. Examination of the cross-section on Figure 3 shows that the fault is cutting downsection in the Guilmette and there is a suggestion that is is also cutting

downsection in the Chainman. Displacements on all other faults are small in comparison to the low-angle fault. They range between 30 and 50 ft. The northwest-trending Min and PN Faults (Figure 3) may have channelled hydrothermal fluids into the low angle fault zone and thus would be pre-ore. The north-trending Valley Fault displaces ore and is post-ore.

ALTERATION

Three types of alteration are present in the Tecoma deposit: silicification, dolomitization, and argillization. Silicification is concentrated in a zone up to 55 ft thick that parallels the low-angle fault. Intense silicification is shown in the stipled pattern on Figure 3. Outward from the silicifed rock, partial silicification and quartz-barite veinlets are common especially in the dolomite. Silicic alteration decreases away from the fault contact in both directions.

The dolomite in the Tecoma deposit is interpreted to be hydrothermally dolomitized limestone of the Guilmette Formation. The same unit 2.5 mi to the northwest is composed of thick-bedded limestone. The dolomite is coarsely recrystallized, veined by quartz and barite and hydrothermally brecciated. Iron and arsenic oxides are concentrated in the carbonate matrix of the breccia. One core hole drilled to 902 ft shows dolomite breccia dominant to 540 ft, and dolomite present to the bottom of the hole.

Argillic alteration is most intense in quartz monzonite

porphyry dikes. The dikes are completely altered to kaolinite and quartz. The shales and mudstones overlying the ore may also be argillized and decarbonized. Regionally, the Chainman is very carbonaceous, however, only scarce carbonaceous zones have been seen in drilling at Tecoma.

MINERALIZATION

Most of the ore at Tecoma occurs in silicifed rock above and below the low-angle fault. Approximately 70% of the ore is siliceous and the rest occurs in shale and dolomite. In addition to enrichment in gold and silver, the ore is strongly enriched in arsenic, mercury and barite. The ore contains approximately 5% barite, 5,000 to 10,000 ppm Arsenic and 1 to 10 ppm Mercury.

Siliceous ore is typically brecciated; multiple episodes of brecciation are common. Barite is very abundant as are As and Fe oxides. Arsenic occurs as a pistachio green supergene mineral, dussertite ($BaFe_3(AsO_4)_2(OH)5H_2O$), generally coating fractures. In very high grade portions, copper oxides are present in trace amounts. Siliceous ore is slightly higher than average grade in both gold and silver.

Shale ore looks similar to unmineralized altered shale except that it can contain quartz veinlets, thin silicified zones, and disseminated oxidized sulfides. The oxidized sulfides often have hematite cores rimmed by green (As?) oxides. Shale ore is usually barite-poor and is slightly lower

than average grade in gold and considerably lower than average grade in silver.

Dolomite ore is the most variable of the ore types. Brecciation is ubiquitous, though much non-ore is also brecciated. Dolomite ore ranges from brick-red, vuggy, gossany, baritic and clayey to black-brown gouge to light grey and coarse grained. Thin zones of silicification are common as are quartz-barite veinlets and disseminated and vein barite. Dolomite ore is lower than average grade in gold but average grade in silver.

The low-angle fault is the primary control on ore deposition. Movement along this fault probably increased the permeability of the units around the fault. Hydrotheral fluids flowed up along the northwest-trending normal faults (PN and MIN Faults) and then flowed out along the permeable fault zone.

Native gold is the only identified gold-bearing phase. Visible gold has been seen only once in an outcrop sample of silicified shale containing barite and dussertite. Acanthite or argentite is the primary silver-bearing mineral. Trace amounts of aguilarite (Ag_4SeS), native silver, and electrum have been identified in S.E.M. studies.

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apparently this Tecoma area listed in the PP!

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JRS

ASARCO

Southwestern Exploration Division

October 5, 1984

File Memorandum

Osceola Placer
Continental Gold, Inc.
Continental Materials Corp.
White Pine County, NV

The placer gold operation at the Osceola Placer, White Pine County, Nevada, being operated by Continental Gold, Inc. (a unit of Continental Materials Corporation) was visited on September 12, 1984. Mr. Hank Eyrich, VP of Mining and Geology, Mr. Wayne Bartlett, exploration geologist, and Mr. Greg Kosanke, assistant plant manager, conducted various parts of the tour.

The Osceola Placer has been operated since 1872 by various groups.

A series of pictures are included to show various aspects of the present operation. Continental Gold operates with two 35-ton trucks, a 10-cubic-yard front-end loader, a smaller loader for tailings distribution, a D-9 cat with rippers to clear overburden, a large trenching back-hoe to test the pay zone, plus several other support trucks. Approximately a dozen men are employed.

The stripped pay dirt gravels are loaded into trucks for transport to the mill. The large front-end loader can dig some 3-5 feet of bedrock which contains the bulk of the gold (Photo 1). The trucks transport the gravel-clay-bedrock-gold to the dump hopper (Photo 2) where the muck falls onto an inclined slotted scalper which scalps off the plus ten-inch boulders, while the minus ten-inch material goes onto a conveyor belt (Photo 3). No water is used at this point and some gold could be lost by being in the clay-sand that adheres to parts of the reject boulders, although Continental Gold believes little gold is lost at this point.

The minus ten-inch material moves by a conveyor (Phots 3 and 4) to feed a four-inch vibrating screen with water sprays. The washed and cleaned plus four-inch cobbles are rejected. The minus four-inch material, partially pulped, is fed into a 4' x 20' prewasher (Photo 4) rotating cylinder, where additional water is added to completely pulp the material.

The pulped clay, silt, sand, pebbles and cobbles of the minus four-inch material is then fed into a rotating screen trommel, 6' x 20' (Photo 4, lower right). The first half or so of the trommel is slotted one-inch and 80-85% of the pay values pass through this one-inch and are fed directly to the gold saving system. The plus one-inch through-flow pebbles and cobbles continue down the trommel where the last portion is slotted one and one-half inches where an additional 10-15% of the material passes through and is sent to a saving system.

The reject minus 4" - plus 1½" material is fed onto a long vibrating 2" screen trough where the plus 2" material is sent directly to reject (blue conveyor, Photo 5). The minus 2" is fed directly to a short, broad riffle where some coarse gold nuggets are recovered before that reject material is stacked by conveyor (Photo 5, left side).

Meanwhile, back at the large rotating slotted screen trommel, the minus 1" and minus 1½" material which passed through the slots is collected and sent to the main gold saving system (Photo 6). This material passes through 30' long sluice boxes which have a base of corduroy matting, with ¼" screen on top to help trap the gold. Holding the screen and matting in place are 1" angle irons (Hungarian riffles) spaced on 6" centers down the sluice. Approximately 95% of all the gold recovered is at this section of the sluice.

After passing through the sluice, the material is fed to a 10-mesh vibrating screen (yellow section on Photo 6) and the plus 10-mesh overflow is discharged by conveyor to a reject pile. The underflow of minus 10-mesh is run over a riffle length which has only the corduroy mat base and the mesh screen. About 5% of the total gold recovered is from this slime recovery riffle sluice. The slime reject goes to a slime pond where the material is settled and the water is decanted, pumped back up and used as the precondition wash water.

Photo 7 is an overall view of the entire mill-gold recovery system.

The recovery riffle gold-black sand is cleaned each day.

On September 12, 1984 some 800 cubic yards of material was fed in at the upper hopper and approximately 20 ounces of gold recovered. This included one nugget of 1½ ounces and a dozen or so 1/2-3/4 ounce pieces. The bulk of the remaining gold is in 2-5 gram sizes and in flour gold recovered from the black sand by use of mercury. Some amalgam from the previous operations is also recovered.

Photo 8 is a picture of a group of nuggets, actual size, with their weight in grams (31.1 grams = 1 Troy oz). The fineness is 850 Au - 50 Ag.

The plant handles 200 cubic yards per hour; however, in general, on the 12 hour operating day only 800-1000 cubic yards are run through.

Last year they processed 13,000 cubic years and have slightly doubled that amount this year. Their known drilled/backhoed reserves will be gone by the end of this season.

The average grade of determined reserves was 0.02 opt gold recovered. To date, after two years of operation, the actual recovery has been 0.018 opt gold recovered.

Exploration is mainly in the area of the deeper channels where the previous operators had not been able to reach except by some bedrock drift mine openings. A rotary rig was employed to secure a general indication of bedrock, the pay interval, and overburden configuration. After sufficient values were found and a probable economic area outlined, this portion was stripped by the bulldozer to near the suggested top of paydirt. The backhoe was then used to cut a 25' deep slot and the entire material from this interval was run through a trommel (1") and the underflow tabled to secure the available gold. Knowing the volume of the slot and the recovered gold, the ounces per cubic yard were assigned. Several such segments would constitute the entire trench length across the indicated channel. The dozer would then restrip the trench down to the bottom of the slot and the backhoe would repeat the process. Generally only 50 to 75 feet cummulative depth could be evaluated in this manner or to bedrock.

The slots were spaced 50 or more feet apart, depending on initial area plans. The various slot trenches were then recalculated into a mining area.

At the 0.02 opt value, a stripping ratio of 5 to 1 can be mined, while some of the higher grade areas actually supported a 10 to 1 stripping ratio.

Faces exposed by mining are also sampled for some control, especially between the outside "ore" trench and non-ore trench. The actual mined reserve yardage has been less than originally planned, mainly due to the lack of values carrying as far as projected toward the adjacent non-ore trench. Overall grade though is quite close to that calculated.

Continental Gold has done some exploration for bedrock gold values in the veins and has outlined some ten- to twenty-thousand tons of plus 0.2 opt gold material. This has been subleased to the plant manager, Mr. Kosanke (Greg Kosanke's father--both local Osceola residents). Mr. Kosanke has a 3-5 tpd mill which was used for recovery of gold in the initial rotary and backhoe operations.

As noted above, the known reserves as outlined by Continental Gold has been essentially mined. It was stated that with the sale of the plant, trucks, etc. the two-year recovery operations will about break even over exploration-development-mining activities with the low price of dore and the backlog of nuggets to sell.

James D. Sell

James D. Sell

JDS/cg

Attachments (Photos)

cc: FTG/RLB (w/xerox of photos)

Photo 1

Photo 2

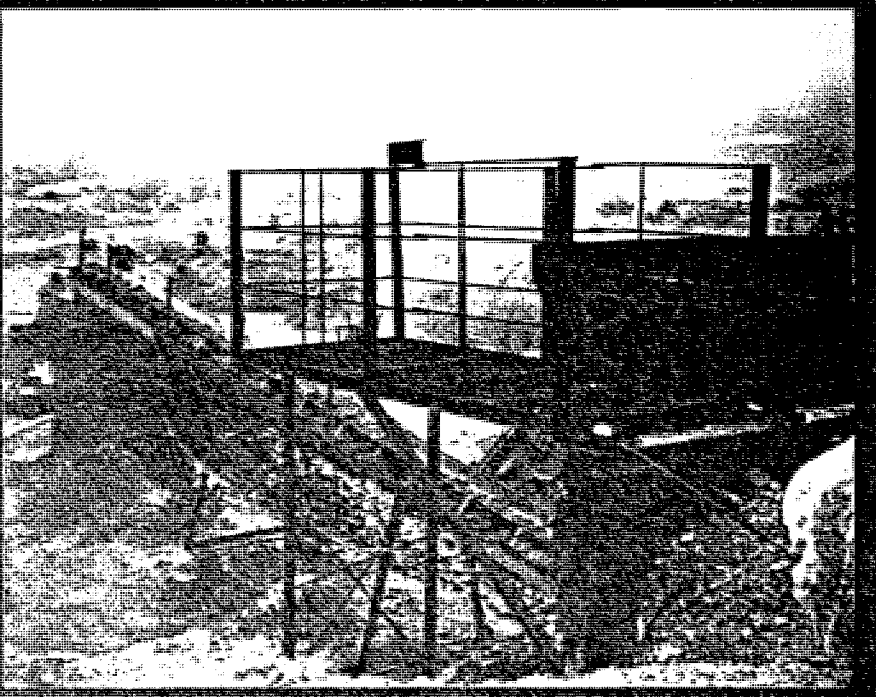
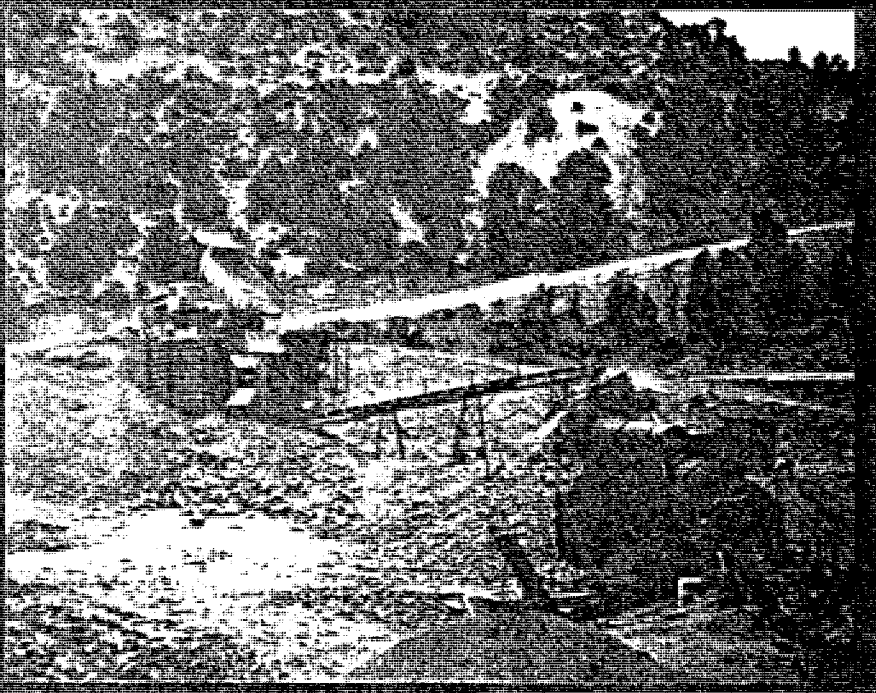


Photo 3



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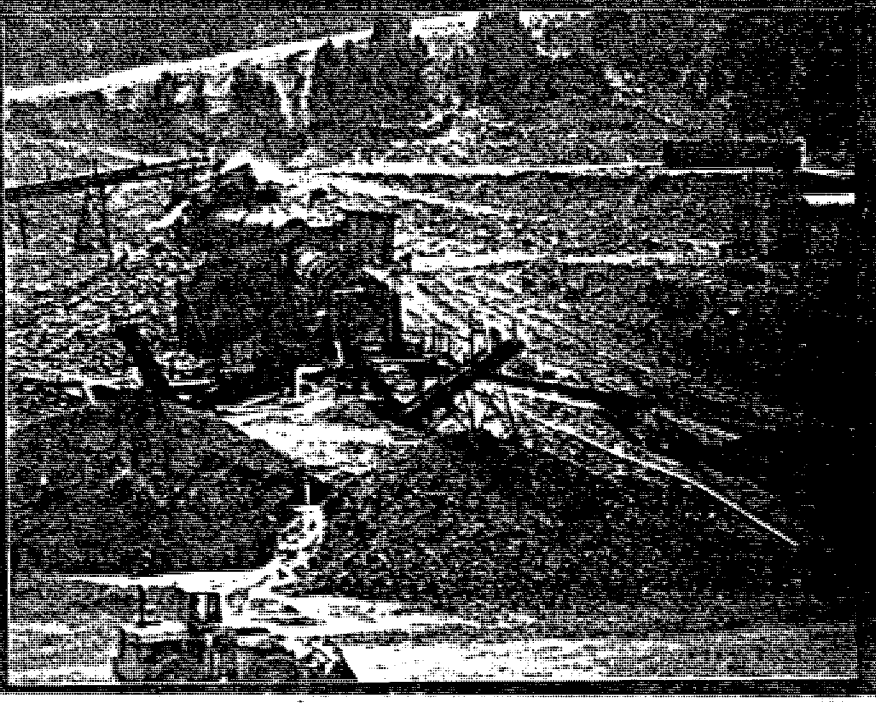

Photo 5

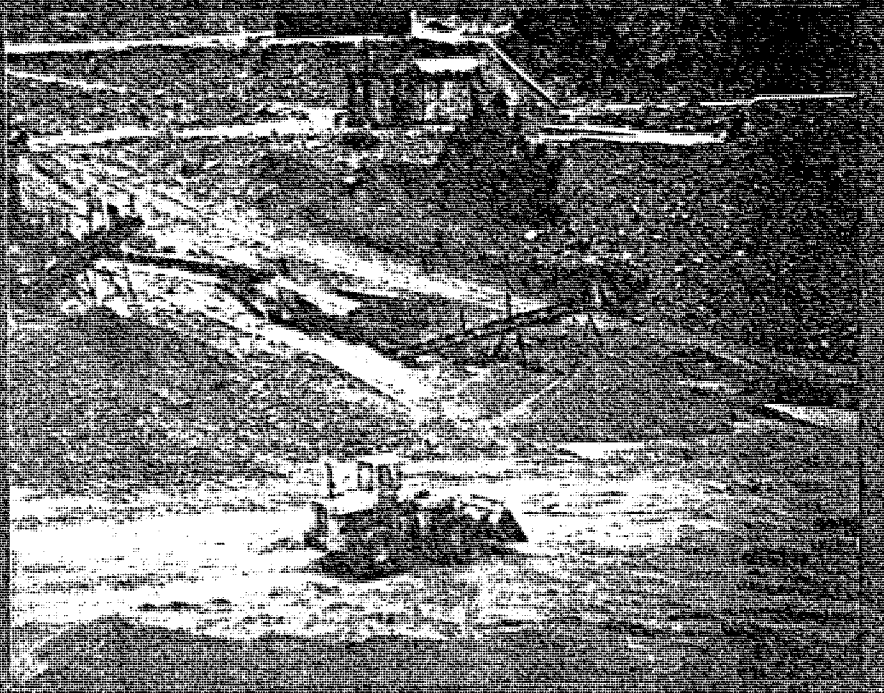

Photo 6

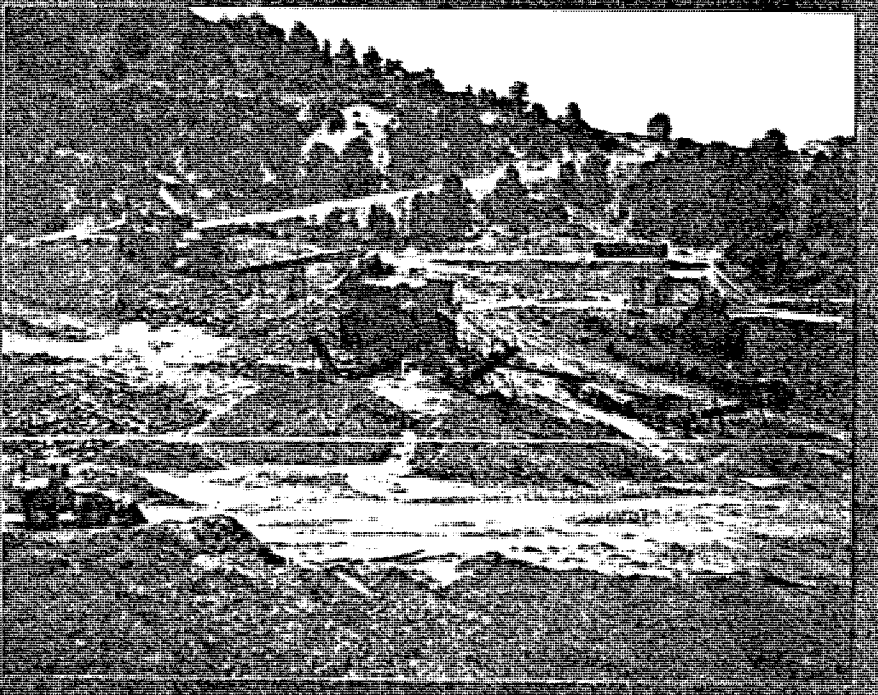

Photo 7

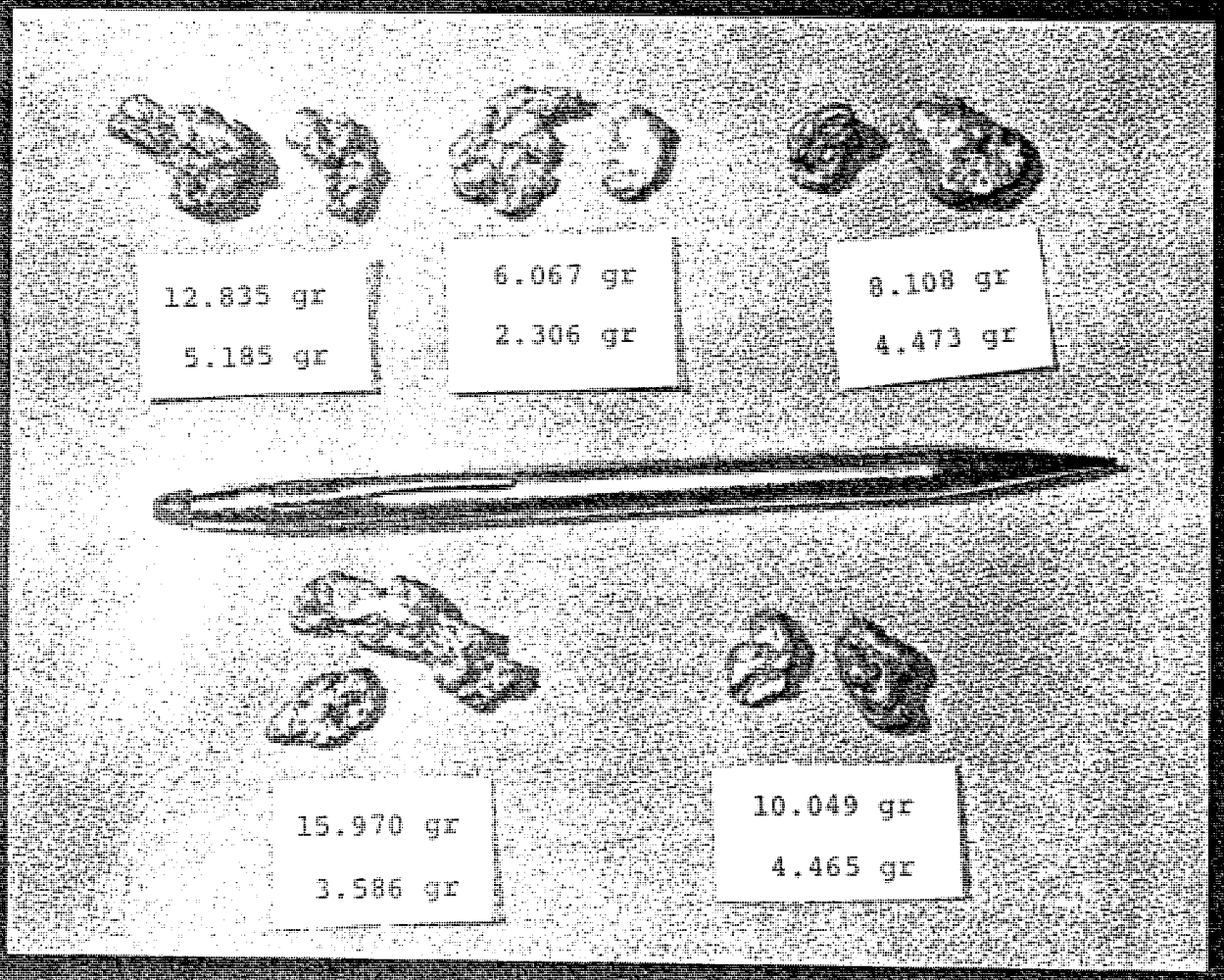
12.835 gr
5.185 gr
6.067 gr
2.306 gr
8.108 gr
4.473 gr
15.970 gr
3.586 gr
10.049 gr
4.465 gr

Photo 8

Oct. 6, 1984 Vol. 73 No. 40

SKILLINGS' Mining Review

JDS

ASARCO

Southwestern Exploration Division

November 5, 1984

Memorandum to Staff

Some Gold Notes

Attached for your review on discovery and operating costs and cash flow potential are four articles:

1) North American Hardrock Gold Deposits - E&MJ October 1984.

2) Dean Witter Reynolds (Canada) article which was distributed at the Denver SME meeting.

3) SME/AIME Preprint #84-312 - The North American Gold Mines: Operating Costs, by A. W. Garson (gives added information on the Dean Witter Reynolds' diagrams).

4) SME/AIME Preprint #84-355 - Optimum Production Rate for High-Grade/Low Tonnage Mines, by R. Glanville.

James D. Sell

JDS/cg

Attachments

cc: FTGraybeal (w/attach)

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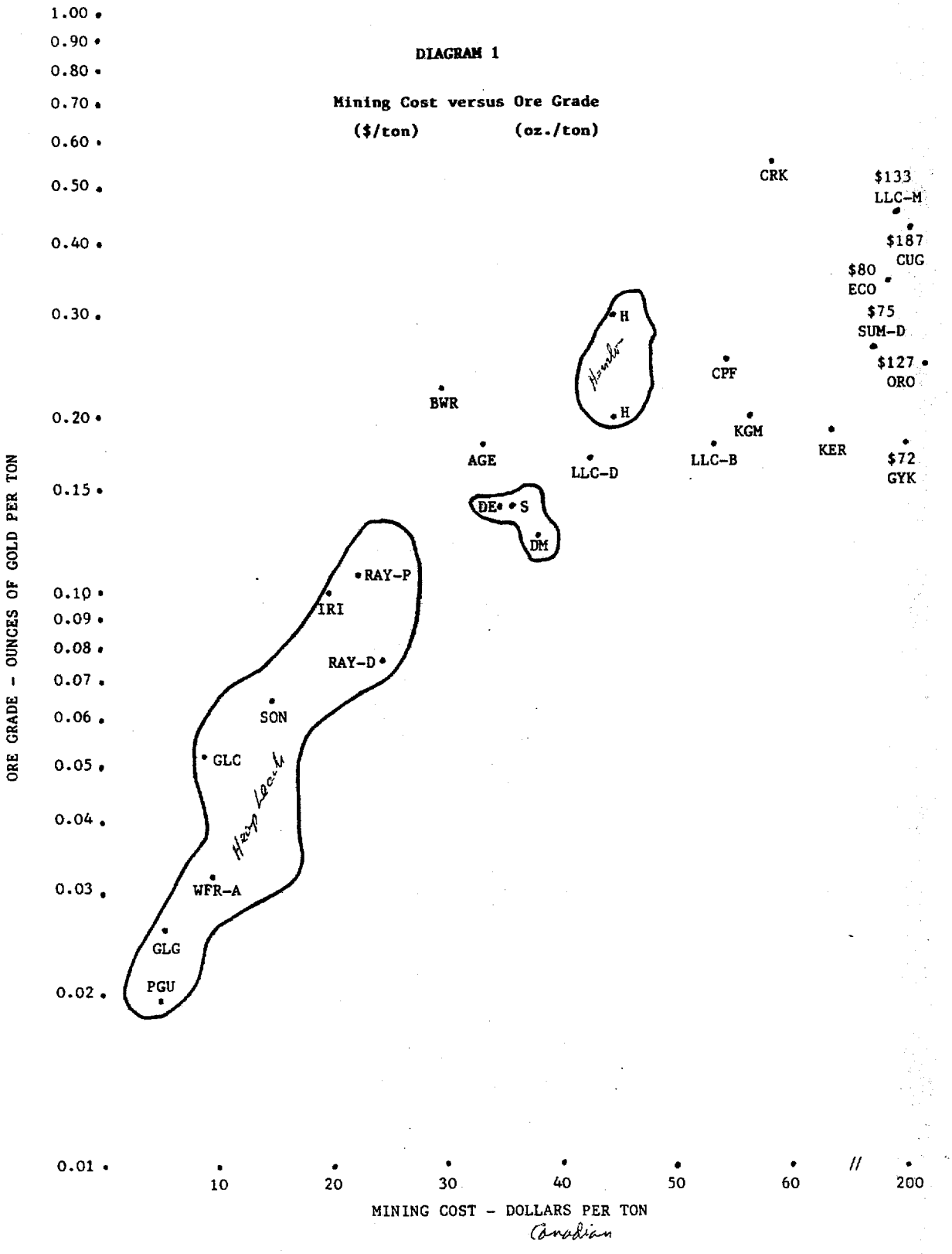
DIAGRAM 1

Mining Cost versus Ore Grade

($/ton) (oz./ton)



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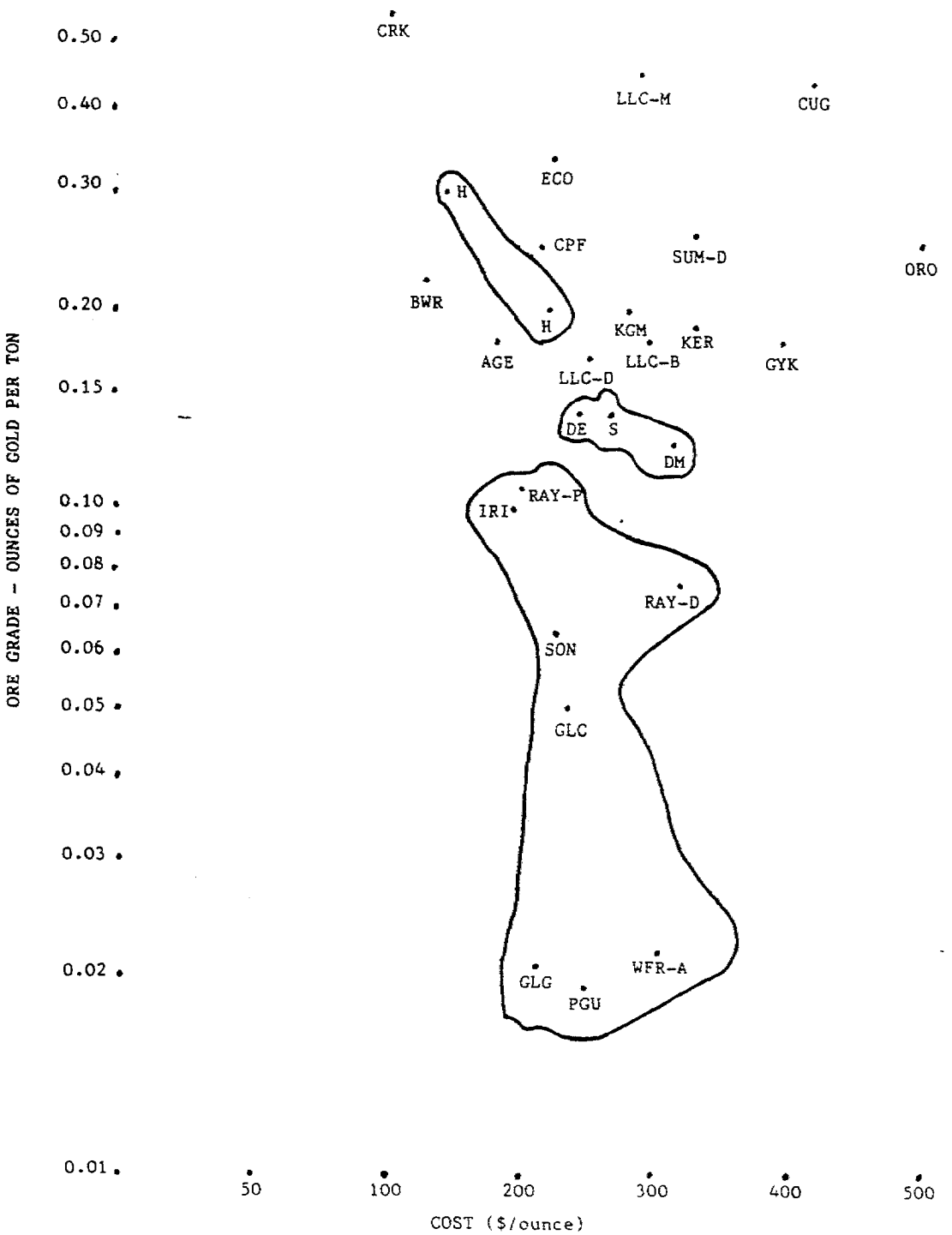
Directors, shareholders or employees of this Company may be beneficial owners of the securities referred to herein. The information contained herein was obtained from sources which we believe to be reliable, however such information is provided on the understanding that this Company is not responsible for financial losses or damage of any kind whatsoever resulting from action taken in reliance thereon or from inaccuracies therein. Neither the information nor any opinion expressed herein constitutes a solicitation by this Company or any of its employees of the purchase or sale of any security.

DIAGRAM 2

Mining Cost versus Ore Grade

($/oz.) (oz./ton)



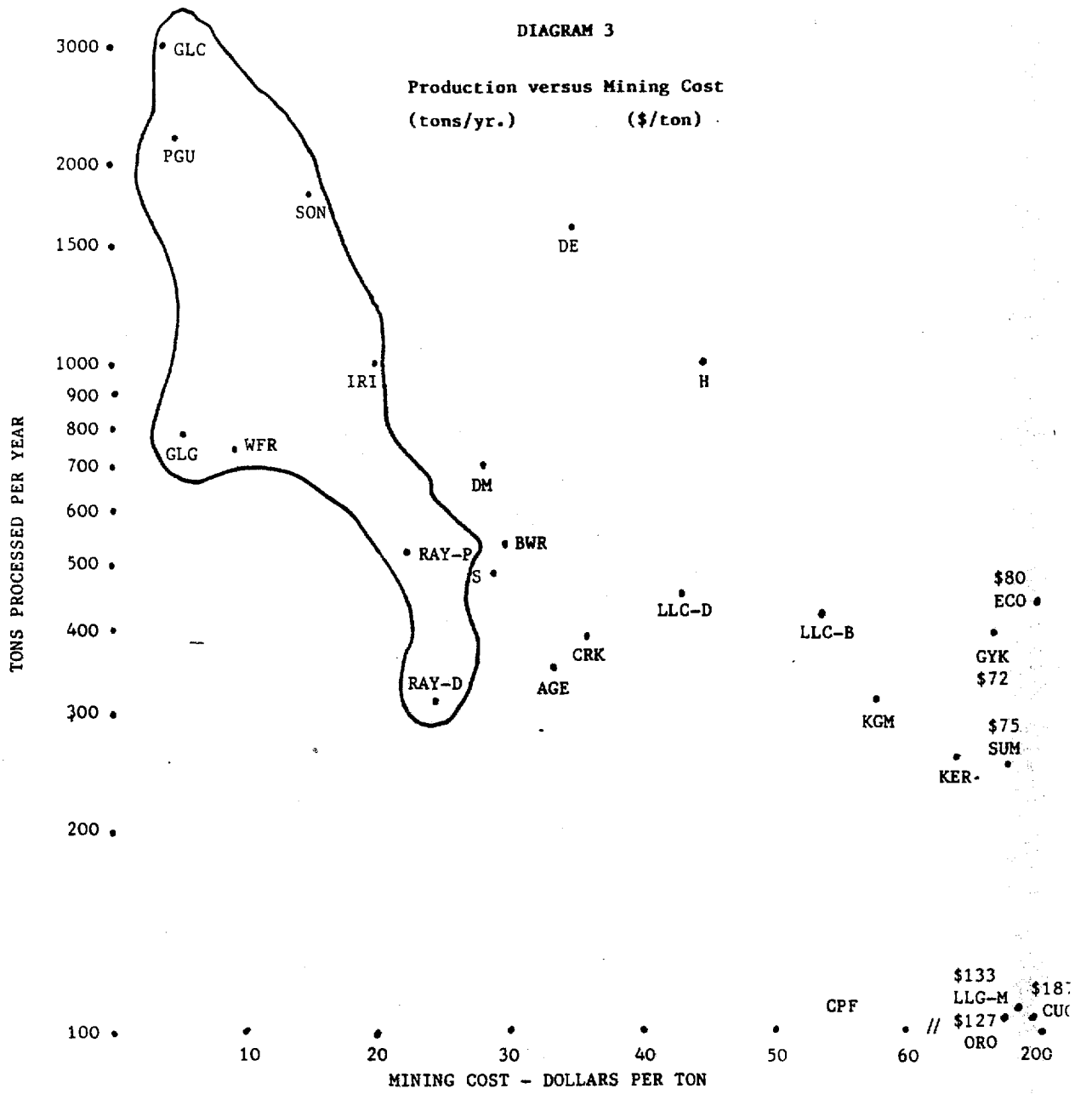


DIAGRAM 3
Production versus Mining Cost
(tons/yr.)
($/ton)
TONS PROCESSED PER YEAR
3000
2000
1500
1000
900
800
700
600
500
400
300
200
100
GLC
PGU
SON
DE
IRI
H
GLG
WFR
DM
BWR
RAY-P
S
LLC-D
LLC-B
$80
ECO
CRK
GYK
$72
RAY-D
AGE
KGM
$75
SUM
KER-
CPF
$133
LLG-M
$127
ORO
$18
CU
10
20
30
40
50
60
200
MINING COST - DOLLARS PER TON

LEGEND

Low Grade Open-pit Producers	Stock Symbol
Pegasus Gold Limited	PGU - T
Glamis Gold Limited	GLG - V
Wharf Resources Limited	WFR-A - T, V (Annie Creek)
Sonora Gold Corporation*	SON - T
Rayrock Resources Limited	RAY-D - T (Dee Deposit)
Inca Resources Limited*	IRI - T, V

* Impending production

Senior Producers	
Dome Mines Limited	DM - T
Campbell Red Lake Limited	CRK - T
Sigma Mines Limited	S - T
Detour Lake	- (part of Campbell Red Lake)
Lac Minerals Limited	LLC-D - T (Doyon)
	LLC-M - T (Macassa)
	LLC-B - T (Bousquet)
Echo Bay Mines Limited	ECO - T

Other Producers	
Sullivan Mines Incorporated	SUM - T
Kerr Addison Mines Limited	KER - T
Giant Yellowknife Mines Limited	GYK - T
Cullaton Lake Gold Mines Limited	CUG - T
Agnico-Eagle Mines Limited	AGE - T
Kiena Gold Mines Limited	KGM - T

Impending Producers	
Hemlo - Participants	H
Orofino Mines Limited	ORO - T
Breakwater Resources Limited	BWR - T
Consolidated Professor Limited	CPF - T
Galactic Resources Limited	GLC - T

Anthony W. Garson, B.Sc., M.B.A.
Mines & Metals Analyst
(416) 868-0303

ECONOMIC GEOLOGY

and the

Bulletin of the Society of Economic Geologists

(ISSN 0361-0128)

Volume 79 SEPTEMBER–OCTOBER 1984 Number 6

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JDS

ASARCO

Southwestern Exploration Division

November 14, 1984

File Memorandum

Sediment Hosted Precious
Metal Deposits
Northern Nevada
SEG, Oct. 31-Nov. 3, 1984

The Society of Economic Geologists Field Trip Road Log for the Sediment Hosted Precious Metal Deposits, October 31-November 3, 1984 held in conjunction with the GSA Reno Meeting, contains the following articles of interest:

1. Guidebook Geology of the Pinson Mine, Humboldt County, NV by E. L. Kretschmer (8 p.)

2. The Geology of the Enfield Bell Mine and the Jerritt Canyon District, Elko County, NV by D. J. Birak and R. B. Hawkins (31 p.)

3. General Geology of the Carlin Gold Belt by H. F. Bonham, Jr. (4 p.)

4. Geology of the Windfall - Rustler Ore Deposits by W. B. Wilson (9 p.)

5. History and Geology of the Alligator Ridge Gold Mine, White Pine County, NV by P. J. Klessig (9 p.)

6. Geology and Ore Deposits of the Taylor Mining District, White Pine County, NV by S. R. Havenstrite (24 p.)

7. Selected References from Road Log Section (p. 87-89).

These articles have been filed in the Library.

James D. Sell

James D. Sell

JDS/cg

cc: WLKurtz
AEGiesecke
FRKoutz
GJStathis
HGKreis

LACANA GOLD INC.
RELIEF CANYON GOLD MINE
PAUL SONERHOLM, CHIEF ENGINEER
DECEMBER 1984

Relief Canyon gold mine is a 100 % Lacana owned property located 32 km (20 miles) east of Lovelock, NV and lies between 1700 m and 1400 m (5800-4700 ft) above sea level.

Discovery consisted of a single stream sediment sample taken by a Duval exploration team in 1979, who later drilled 4000 m (13,000 ft) of exploration holes and developed 5.4 to 9.1 Mt (6-10 MMT) at a grade of 1.37 to 2.06 g Au/t (.04-.06 oz Au/T). Lacana optioned the property from Duval in 1982, drilled 15000 m (50,000 ft) of exploration/ development holes and delineated an ore body of 8.3 Mt (9.2 MMT) at a grade of 1.10 g Au/t (.032 oz Au/T) at a 2:1 stripping ratio at a cut-off grade of .051 g Au/t (.015 oz Au/T). Run of mine pilot heap leaching tests indicated 70 % recovery in 60 days.

Epithermal gold mineralization occurs within a breccia unit developed between an overlying shale and an underlying massive limestone, both of Triassic age. The breccia attains a thickness of 61 m (200 ft) and has an antiform plunging to the southwest. The orebody consists of the daylighted and shallow portions of the northwest flank of the antiform.

Mineralization consists of micron (< 3 u) electrum (Ag 10:Au 1) with accessory Hg, F, Fe, Si and CaC03.

Mining is by contractor using 4.7 m (15 ft) lifts. Blasthole drilling is on 3.7 m by 3.7 m (12 ft by 12 ft) to 3 m by 3 m (10 ft by 10 ft) centers with 0.6 m (2 ft) to 0.9 m (3 ft) subs in 11.4 cm (4.5 in) holes, at a powder factor of 0.47 kg/m3 (0.8 lbs/yd3). Muck is sorted by grade into ore (>0.51 g Au/t)(.015 oz Au/T), lean ore (0.34 to 0.51 g Au/t)(.010 to .015 oz Au/T) and waste. Lean ore is stacked on a separate permanent leachable dump. Grade control is by leach assay on every blast hole and 1 A.T. fire of every 10th sample.

Loading of the heap is by 9.8 m3 (12 yd3) wheeled tractor (992-C Cat), hauling is by 54 t (60 T)(773-B Cat) and 77 t (85 T)(KW Dart) end dump trucks. Current mining rate is 4,535 t (5,000 T) per day of ore and 9,070 t (10,000 T) per day of waste and lean ore.

Permanent mine run heaps are on 366 m by 140 m (1200 ft by 460 ft) pads which are covered with 1.5 mm (60 mil) HDPE which is protected by 30 cm (1 ft) of compacted -19 mm (3/4 in) ore. Grade on pad bottom is 4 % on width and 1 % on length. Heap height is 4.7 m (15 ft) with plans of an ultimate height of 14 m to 18 m (45 ft to 60 ft). Loading is by end dump on top of the pad. The loaded pad is ripped on 2 m (6 ft) centers to a depth of 1.5 m (4 ft) before laying down the sprinkler lines.

NaCN concentration in the leach solution is 500 gm/t (1 lb/T) and consumption is 250 g/t (0.5 lbs/ton) of ore. The pH is maintained at 10.5 to 11. Caustic (NaOH) consumption is 150 g/t (0.3 lbs/ton) of ore. Leach solution flow is 3.3 ml/s/m2 (0.004 gpm/ft2) of heap; evaporative losses have not been established. Current application rate is 91 L /s (1200 gpm) to the heap. The volume of the pregnant and barren solution ponds are 32.3 M L (7.1 MM gal) each. Leach solution flows by gravity from the pads to the pregnant pond, through the carbon columns and into the barren pond. Reagent make-up is in the mill building. Regenerated barren solution is pumped 18 m (60 ft) vertically and 549 m (1800 ft) laterally to the pads through 25.4 cm (10 in) HDPE pipe then to the sprinklers which operate at 137.8 kPa (20 psi).

Adsorbtion is in 5 columns, each of which is 2.1 m by 2.4 m (7 ft by 8 ft) and holds 0.9 t (1 T) of 12 by 30 mesh carbon. The columns are designed to handle a flow rate of 19 L/s/m2 (23 gpm/ft2) of column area. It is anticipated that gold loading on carbon will be in the order of 5.1 kg Au/t (150 oz Au/ton) of carbon, which is advanced as needed. Carbon stripping is in 0.9 t (1 T) batches for 6 hours at a temperature of 115 degrees C (240 degrees F) and at a pressure of 275.6 kPa (40 psi). The plant is set up with 1 carbon stripping vessels and 1 electrowinning cell which contains 6 cathodes, each of which holds 1.8 kg to 2.7 kg (4 to 6 lbs) of steel wool. The cathodes are expected to load to 10.3 g Au/g (150 oz Au/lb) of steel wool.

Carbon is reactivated in a kiln at a rate of 0.9 t/day (1 T/day) after an HN03 acid wash.

Fineness of the dore produced is 750 and approximatly 10 % of the cathode material is Hg, which is recovered by retorting the cathode material.

The time from option to production has been 24 months with 4 months for construction. Costs of construction and preproduction have been $ 6.7 MM U.S. of which the plant has been $ 4.2 MM U.S. The total work force is approximatly 40. Production is estimated at 762,000 g Au (24,500 oz Au) per year.

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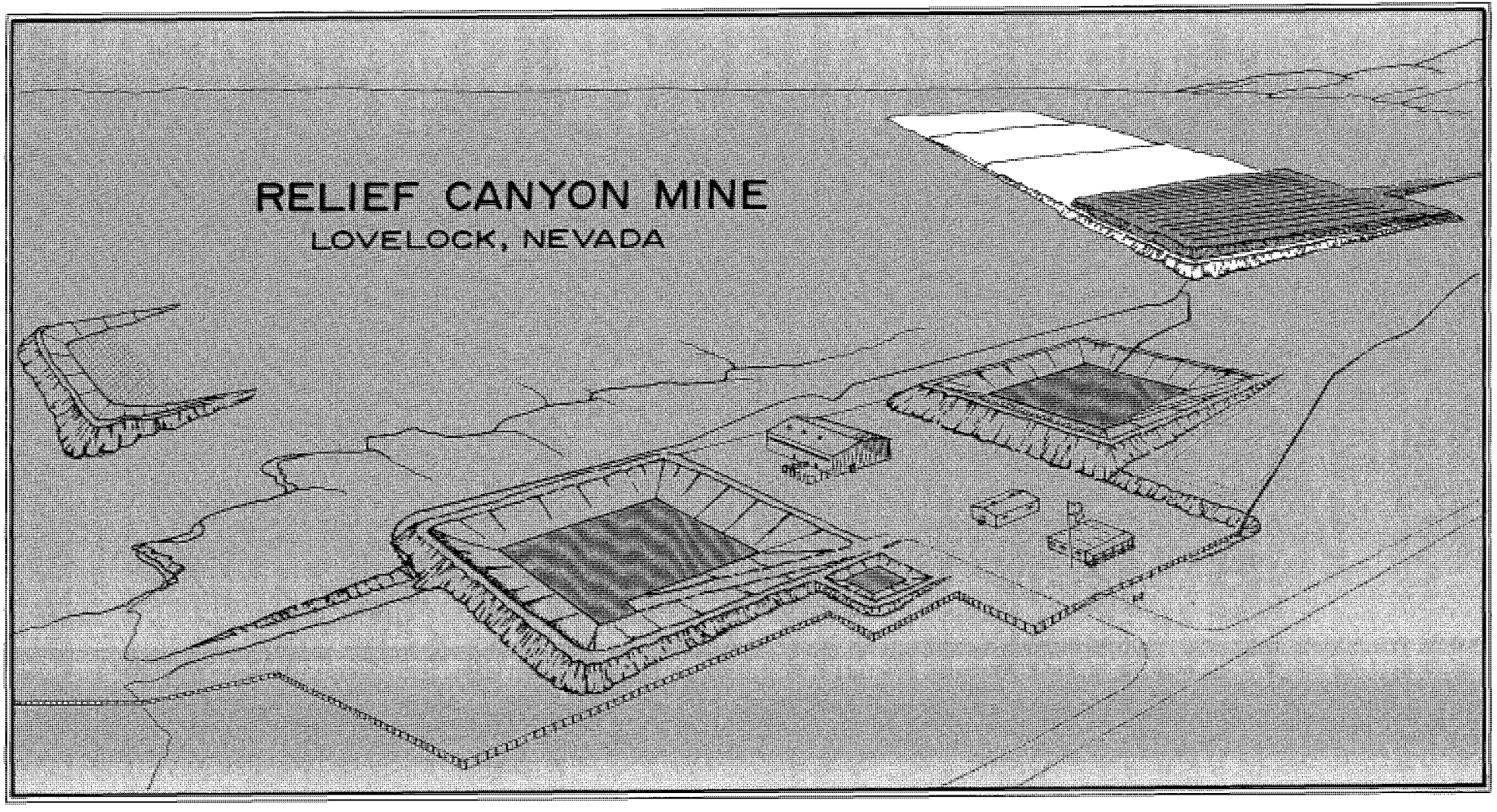
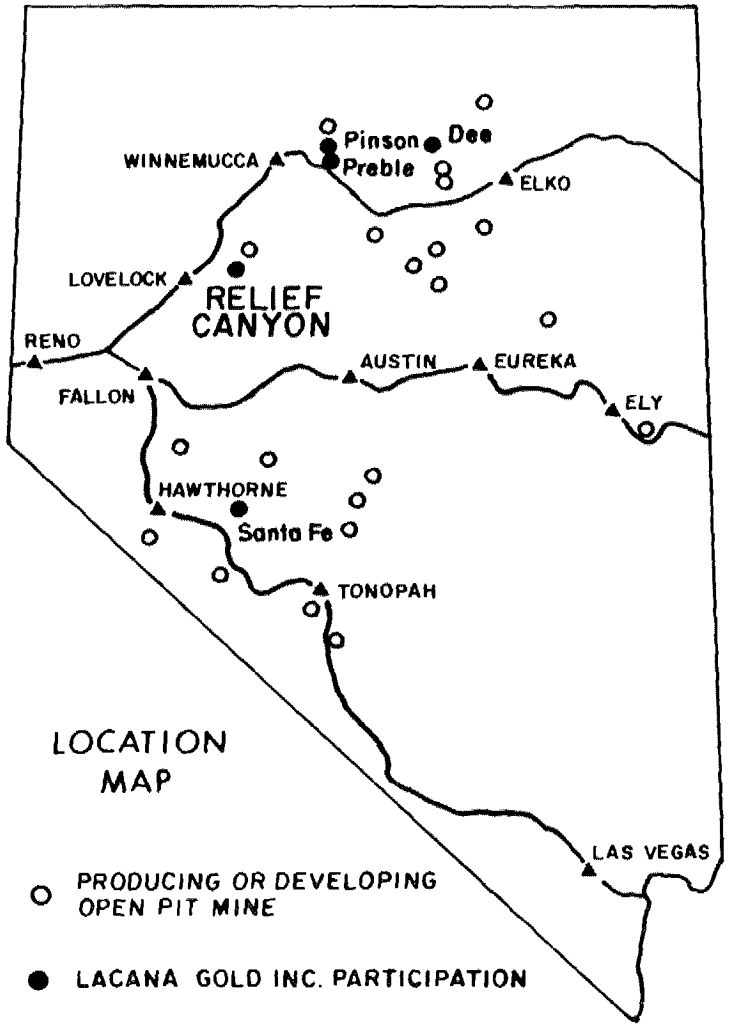
LACANA GOLD INC.




Pinson
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WINNEMUCCA
ELKO
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HAWTHORNE
Santa Fe
TONOPAH
LOCATION MAP
LAS VEGAS
PRODUCING OR DEVELOPING OPEN PIT MINE
LACANA GOLD INC. PARTICIPATION

SUMMARY

Project
Minimum 1,000,000 tons ore per year open pit gold mine using heap leach extractive technology. The property is 100% owned by Lacana Gold Inc.

Schedule
On schedule; under budget; 23 months from acquisition to gold production; only 4 months construction.

Capital Cost
Construction: $4,800,000; preproduction: $1,900,000; total project cost: $6,700,000; $1,600,000 is available as working capital.

Employment
Approximately 40 new jobs at the mine and additional ancillary jobs created in the area.

Ore Reserves
Sufficient for a minimum 8 year life, with possibilities for extension.

Production
Approximately 24,500 ounces of gold annually.

Official Opening
October 19, 1984

MILESTONES

1978 — Falconi and Associates located claims on approximately 2,000 acres over the long inactive Emerald Spar Fluorite prospect.

1979 — Duval Corporation initiated a precious metals exploration program in the Humboldt Range, and discovered gold on the Falconi ground.

1981-1982 — Duval acquired an option from Falconi and staked an additional 2,300 acres. Widely spaced drilling indicated the presence of low grade gold mineralization.

October, 1982 — Lacana optioned the property from Duval. Check assaying, metallurgical tests, and geological studies indicated the potential for an economic, heap-leachable gold deposit.

1983 — Lacana completed a 50,000-foot drilling program which "proved up" the deposit; conducted bulk heap leach tests on two 4,300-ton samples which confirmed the leachability of the ores; and initiated economic studies.

April, 1984 — Final feasibility studies were positive; production decision was approved by the Board of Directors of Lacana Gold.

June, 1984 — Construction of the gold recovery plant and leach facilities started. Ore development commenced.

September, 1984 — Sprinkling of cyanide solution onto the first 100,000 ton ore heap commenced. A total of 400,000 tons will be processed in 1984.

October, 1984 — The first bullion was poured.

MINE DESCRIPTION

Location
Relief Canyon is located 20 miles east of the town of Lovelock, Pershing County, Nevada. The mine-plant complex is situated on Packard Flat at the southern end of the Humboldt Range. Elevation of the plantsite is 4,720 feet.

Access from Lovelock is by Interstate Highway 80, and paved State Highway, the last 2 miles being all weather gravel road. Employees commute daily.

Property
The Relief Canyon property comprises 3,560 acres of unpatented lode claims and 800 acres of unpatented mill-site claims.

Development
The ore body has been sampled by 63,653 feet of reverse circulation percussion drilling, of which Lacana drilled 50,063 feet. Comprehensive metallurgical and bulk pilot scale tests confirmed that a commercial leaching operation was feasible. A minimum recovery of 70% is projected for run-of-mine ore.

Reserves
A proven and probable diluted mining reserve, totalling 8,000,000 tons grading 0.035 ounce of gold per ton is defined. A cut-off of 0.02 ounce of gold per ton was used. A total of 271,352 ounces of gold is contained in this tonnage. The ore contains recoverable amounts of silver.

Geology
Low grade, disseminated, epithermal gold mineralization occurs within a breccia unit developed between the Triassic Cane Spring Limestone and the overlying Grass Valley Shale.

The breccia comprises variably silicified and sericitized fragments of shale and limestone. This unit attains thicknesses of 200 feet and generally dips gently to the west, is terminated to the east by a diorite dike, and to the west by range front faults. The bulk of the gold occurs near the shale contact, associated with increased silicification and fluorite. The ore is strongly oxidized.

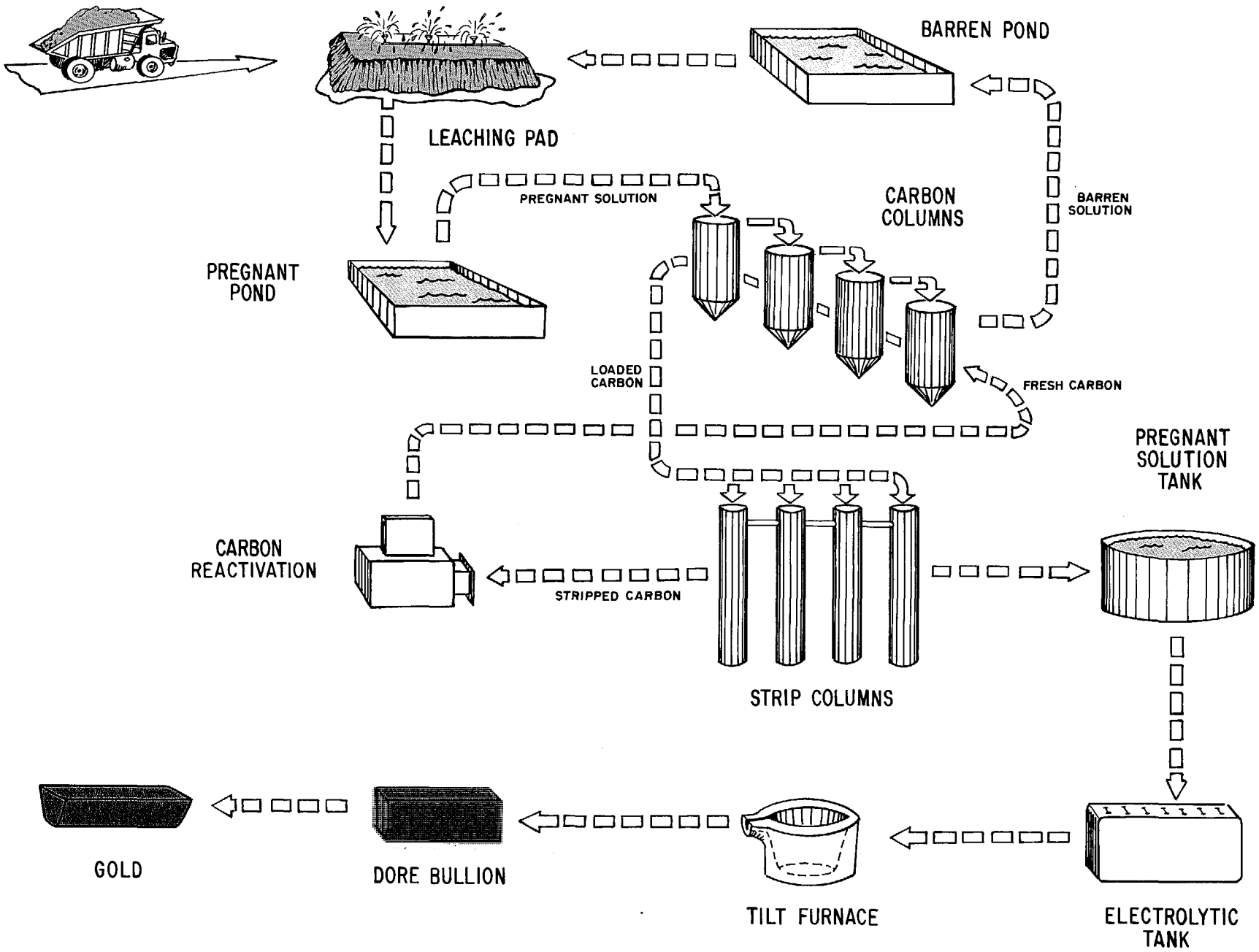
Mining
Mining at Relief Canyon is by standard open pit techniques and utilizes 15-foot high benches. Blast holes are drilled to a depth of 17 feet on a 10 ft. by 10 ft. square pattern. Holes are loaded with an ammonium nitrate and diesel fuel mixture. The mining is by contractor.

Blast hole cuttings are assayed for grade control, and assigned to three categories based on grade. The average waste to ore ratio is 2 to 1.

The broken rock is loaded with 13-yard front-end loaders onto 65-ton trucks and transported to the heap-leach pad, low grade ore stockpile, or waste dump.

The present mining involves moving approximately 15,000 tons of ore and waste daily on a 5 day, 10 hour per day basis. Topsoil from the leach pad and mining sites has been stockpiled for future restoration.

FLOW SHEET


BARREN POND
LEACHING PAD
PREGNANT SOLUTION
CARBON COLUMNS
BARREN SOLUTION
PREGNANT POND
LOADED CARBON
FRESH CARBON
PREGNANT SOLUTION TANK
CARBON REACTIVATION
STRIPPED CARBON
STRIP COLUMNS
GOLD
DORE BULLION
TILT FURNACE
ELECTROLYTIC TANK

GOLD RECOVERY PLANT

Uncrushed ore is stacked in 100,000 ton heaps on impervious plastic membrane. Currently, each cell has dimensions of 400 feet x 300 feet x 13 feet. It is anticipated that this height will be doubled or tripled in time.

Weak cyanide solution is sprinkled on the heap at a rate of approximately 5 gal/sq ft/day. A minimum gold recovery of 70% will be achieved in two 30 day sprinkling cycles. After percolating through the heaps, the gold bearing solution flows on a gentle slope beneath the ore to a corner of the leach pad, and is collected and pumped to the treatment plant by way of a storage pond.

The pregnant solution passes by gravity through a series of tanks containing activated carbon granules, upon which the gold is adsorbed. The barren solution is then recharged with cyanide, sent to the barren pond, and back to the heap.

The gold loaded carbon is transferred to the carbon stripping facility where gold is removed from the carbon by caustic cyanide solutions at elevated temperature and pressure. The carbon is acid washed, reactivated by heat, and returned to the circuit. The concentrated gold bearing solution is pumped through an electrolytic cell where the gold is deposited on steel wool. The gold plated steel wool is melted in a furnace and poured into doré (gold-silver) bars. This product is sold to a refinery for final processing.

Initially, the Relief Canyon facility will treat 1 million tons of ore annually to produce 24,500 ounces of gold.

LOCAL BENEFITS

Relief Canyon is making a significant contribution to the economy of Northern Nevada. Approximately 40 new jobs have been created directly, and the economic spinoff will create additional jobs in the service industries. Contributions are being made to the tax bases of Pershing County and the State of Nevada.

Approximately $6,700,000 has been spent on exploration, mine development, and construction.

The mine co exists with cattle ranching, and provides an example of successful multiple use of B.L.M. land.

RELIEF CANYON OWNERSHIP AND MANAGEMENT

Relief Canyon is 100% owned and financed by Lacana Gold Inc., a 75% owned U.S. subsidiary of Toronto-based Lacana Mining Corporation. In addition to Relief Canyon, Lacana Gold Inc. owns a 26.25% interest in the successful Pinson Mine, as well as a 29.3% interest in the recently commissioned Dee Gold Mine, and is completing development of the Santa Fe Gold Project, Nevada and the Gilt Edge Gold Project, South Dakota. Lacana Mining Corporation has been successfully producing silver and gold for many years from its joint venture operations at Guanajuato and Encantada, Mexico.

The Relief Canyon Mine Project Manager is D. Hopkins assisted by W. Coughlin, Plant Superintendent, Paul Sonerholm, Chief Engineer, and M. Shonnard, Mine Accountant.

Lacana Gold Inc. senior management includes E. G. Thompson, President; H. L. Sobel, V.P. Exploration; M. Easdon, General Manager; P. Dircksen, Manager Northwest; M. J. Fiannaca, Manager Project Development and R. Hall, Manager, Special Projects. C. M. Marshall, consultant, provides senior engineering and management direction.

ACKNOWLEDGEMENTS

Duval Senior Geologist, Jim McKee, assisted by Dennis Geasan, discovered the deposit.

Falconi and Associates, Duval Corporation, Southern Pacific Land Company, and Pinson Mining Company have been extremely helpful to Lacana Gold Inc. in developing the property.

The timely cooperation of Federal and State regulatory authorities enabled the project to proceed on schedule.

The principal contractors – Mountain States Engineers, Target Construction Inc., and Western Plant Constructors, all made extensive efforts to meet tight schedules.

Our appreciation to these groups is extended.

CORPORATE

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Ph: (702) 329-5609

H. L. Sobel, Vice President
M. M. Easdon, General Manager

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Dr. W. H. Gross, Chairman
E. G. Thompson, President and C.E.O.

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EXPLORATION DEPT.

November 1, 1984

Mr. Frederick T. Graybeal
Chief Geologist
ASARCO Inc.
180 Maiden Lane
New York, New York 10038

Dear Fred:

Enjoyed meeting you at ASARCO's breakfast meeting. Have you seen the enclosed proposal on a gold exploration program? It came to me from someone I consider to be quite savvy and knowledgeable in the gold mining business. If you have any interest in participating, please let me know.

Wishing you the best of luck in your exploration efforts.

Sincerely,

Paul

Paul C. Lucke

PCL:er

Enclosure

Bill Trebilcock
International Investors 212-687-5200
Bradley Resources 203-226-6836

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W.L. Kurtz via/telecopier
P.G. Vikre via Federal Express

PAUL C. LUCKE
Consultant

P E X

REINHEIMER NORDBERG INC.
641 LEXINGTON AVENUE NEW YORK 10022 212 753-9111

A PROPOSAL TO EXPLORE FOR DISSEMINATED GOLD DEPOSITS IN NEVADA AND ADJACENT STATES

September, 1984

Precambrian Exploration, Inc., 4955 Iris Street, Wheat Ridge, Colorado 80033

Phone: 303-424-9104

PEX EXPLORATION VENTURE

SUMMARY AND TERMS

PEX is organizing a joint venture to carry on a program which has been in operation for the past six years exploring for disseminated gold deposits in Nevada. PEX will operate the venture and contribute to it all of its expertise, information, properties, and twenty unstaked prospects which represent the expenditure of $1,400,000 in past years. The participants will finance $1,500,000 of exploration expense over a three year period to earn a 50% interest in all of the properties acquired by the venture.

The program employs a geochemical procedure referred to as the CARGO system. In simplistic terms, it employs drainage sediment geochemistry refined and tuned to the degree necessary to pick up gold dispersed in the secondary environment during the weathering of a Carlin-type deposit. Over the past six years it has been used to explore 3,500 square miles in Nevada with one discovery and five "exploration successes" to its credit. The discovery is the Tonkin Springs deposit in Eureka County, Nevada, where 250,000 ounces of gold have been defined by drilling thus far. This property will not be included in the joint venture. The "exploration successes" are gold deposits identified by the CARGO system but not acquired by PEX. In two cases, the land had already been claimed, in one case, the prospect was not followed up, and two other existing unmined deposits were identified as a test of the system.

The proposed exploration program will be designed to explore 1,000 square miles per year of prospective territory in Nevada and selected areas of the adjoining states. It will also continue the evaluation of one claim group and twenty unstaked gold prospects which were identified in prior CARGO programs.

PEX will operate the program under the direction of a management committee which will have representation from all of the participants. PEX will provide monthly technical reports and arrange management committee meetings as required. PEX will charge the syndicate for its actual cost for personnel, facilities, supplies and third party services. Mikron Laboratories, a PEX affiliate, will provide assays and laboratory work at its normal rates less its maximum volume discount.

The $1,500,000 budget will be spread approximately equally over three years and it is estimated that 60% of the funds will be spent on reconnaissance and 40% on follow up work including geology, detailed sampling, staking and a modest drilling program on each prospect of merit. The first year of the program will terminate December 31, 1985, and will be followed by two calendar year programs in 1986 and 1987. Funds will be called from the participants as required.

When a prospect has developed to the point that it requires a work program that cannot reasonably be included in the reconnaissance budget, then the prospect will be designated a development property. PEX will send the participants a work program and budget for the ensuing twelve months. If all or any of the participants elect to finance the work program in its entirety, then PEX will carry out the program, and any participants who elect not to participate will have their interest diluted by a formula that gives the property a deemed initial value equal to the amount of the work program - i.e., non-participants are diluted 50% by the end of the work program. In the event the participants do not agree to finance the work program in its entirety, then the operator will be free to negotiate a farm out with third parties on the best terms it can obtain.

It is recognized that certain participants in the venture are likely to be exploring Nevada and adjoining states for their own account. In order to minimize problems arising out of such competitive operations, PEX will designate one regional exploration area at a time of a size that can be explored to the property acquisition stage within a three-month period. Participants will have the opportunity to review the proposed regional exploration area and require the withdrawal of all localities within the area in which the participant is conducting exploration operations and/or have property interests. PEX will carry out exploration within the approved regional exploration area and make the data and findings available to the participants within the three-month period following approval of the regional exploration area. The gold exploration targets found by PEX will be acquired by staking or other means, each surrounded by a one mile wide area of interest and made subject to the terms of the venture agreement. The remainder of the approved regional exploration area will then be released and be immediately open to all participants (except PEX). It is anticipated that such a proposed regional exploration area will be of the order of 300 to 400 square miles in extent.

TECHNICAL REVIEW - CARGO SYSTEM

The procedure referred to as the CARGO system was developed in 1978 and designed specifically to find disseminated gold deposits within the semi-arid western U.S.A. The need for a discriminatory regional exploration approach is directly related to the exceedingly weak surface expression of this class of ore deposit. The ore discovery at Tonkin Springs, Nevada (Table 1) illustrates the point in as much as the deposit was overlooked by six major exploration companies over the past 15 years while drilling an adjacent prospect. The fact that the ore occurrence crops out on a small poorly vegetated hill indicates how easily such ore occurrences can go undetected when relying on conventional exploration procedures. By contrast, the regional reconnaissance program undertaken by PEX in 1979 revealed a gold anomaly in the drainages, 10 square miles in areal extent surrounding this gold deposit (Figure 5). This example would imply that a significant opportunity for ore discovery still exists in Nevada and adjoining states for an organization using the CARGO system.

The exploration success of the CARGO system is remarkable as indicated by the record summarized in Table 1.

TABLE 1: Exploration Success of the CARGO System in Nevada May 1978 to October 1983

Deposit	CARGO Program	Anomaly Characteristics	Action by PEX	Discovery Announcement
Gold Quarry	Orientation Survey 1978	Strong gold anomaly equivalent to Cortez and Gold Acres	Attempted to stake Dec. 1978. Claimed 10 months earlier by Newmont	8.5 MM oz gold Newmont, 1980
Jerritt Canyon	Orientation Survey 1978	Strong gold anomaly in 'A' fraction of stream sediment	Area sampled to obtain data on an occurrence uncontaminated by mining (at time of survey)	3.00 MM oz gold Freeport, 1978
Tonkin Springs	PEX Program 1979	10 sq. mile gold anomaly	Staked by PEX Jan.-May 1980	250,000 oz gold PEX, 1982
Atlas Discovery	PEX Program 1979	1-sample anomaly	No action by PEX	250,000 oz gold Atlas, 1984
Dee Mine (Boulder Creek)	PEX Program 1980	3 sq. mile anomaly One exceptionally high gold value	Land already held; Leased by CORDEX	300,000 oz gold CORDEX, 1982
Preble	At the request of CORDEX, PEX carried out sampling over Preble.			
	1983	0.5 sq. mile anomaly	Test only	140,000 oz gold

See Figures 1 - 7

For the record, it may be noted that the research which resulted in the CARGO system commenced in 1978 while Mehrtens* was Regional Manager of Rio Algom's U. S. exploration operations. Some $100,000 was spent on the research at that time. During this development work, an orientation survey was carried out around the known mines at Cortez, Gold Acres and Carlin. The scope of the survey was expanded to take in Freeport's new discovery at Jerritt Canyon. Other potentially attractive localities were also sampled which led to the first "success" achieved by the CARGO method at Gold Quarry, Nevada, which was not known at that time (Table 1, Figures 1 - 4).

Following Rio Algom's decision to cease its U. S. exploration activities, Mehrtens set up Mineral Ventures, Inc. (MVI) and with the financial support of CARGO PARTNERS, proceeded to routinely apply the exploration method to selected areas within Nevada. The Tonkin Springs gold anomaly was discovered and staked by MVI within the first ten months of the venture (Fig. 5). In all, CARGO PARTNERS - MVI spent one million dollars in regional exploration for gold in Nevada.

CARGO PARTNERS - MVI brought in Precambrian Shield Resources Limited in December 1981 to provide on-going exploration funding and set up Precambrian Exploration, Inc. (PEX) as the operational entity. At the same time, MIKRON Geochemical Laboratory was constructed to provide in-house facilities. Under these new arrangements, PEX spent some $300,000 in regional exploration. The ore discovery at Tonkin Springs in October 1982 diverted PEX resources away from reconnaissance. Now that the tempo has slowed at Tonkin Springs, PEX personnel are ready to resume reconnaissance operations.

On a technical basis, the CARGO system "found" four mines over six years by covering 3,500 square miles in Nevada at an average sample density of one sample per square mile.

It is likely that future ventures employing the CARGO system will enjoy greater success when compared to the record of the past six years for the following reasons:

1. Experience in interpretation of the prospect data has grown substantially. For example, single sample anomalies were routinely discarded. The recent Atlas discovery in the Roberts Creek Mountains, Nevada, was evidenced by a single anomalous gold value picked up by PEX in 1979 but never acted upon. This will not happen again (Fig. 5).

2. The Preble example illustrates the need to increase the sample density from one to two samples per square mile (Fig. 7).

3. Previously, only areas underlain by Paleozoic sedimentary rocks have been explored using the CARGO system. The FMC

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*M. B. Mehrtens has a degree in Geology and a Ph.D. in Applied Geochemistry (Royal School of Mines, London) and 27 years of experience in the resource industry.

discovery in Tertiary volcanic rocks near Gabbs, Nevada (1983) points out the advantages that might be gained by spreading a wider net.

4. During the past five years, the financial resources did not allow property acquisition other than by staking open federal land. This constraint kept us out of the Dee play (Fig. 6).

5. Knowledge gained by PEX geologists in drilling out the Tonkin Springs deposit has provided detailed information valuable in all phases of a gold exploration program including area selection.

KEY TO LOCALITIES

NEVADA.

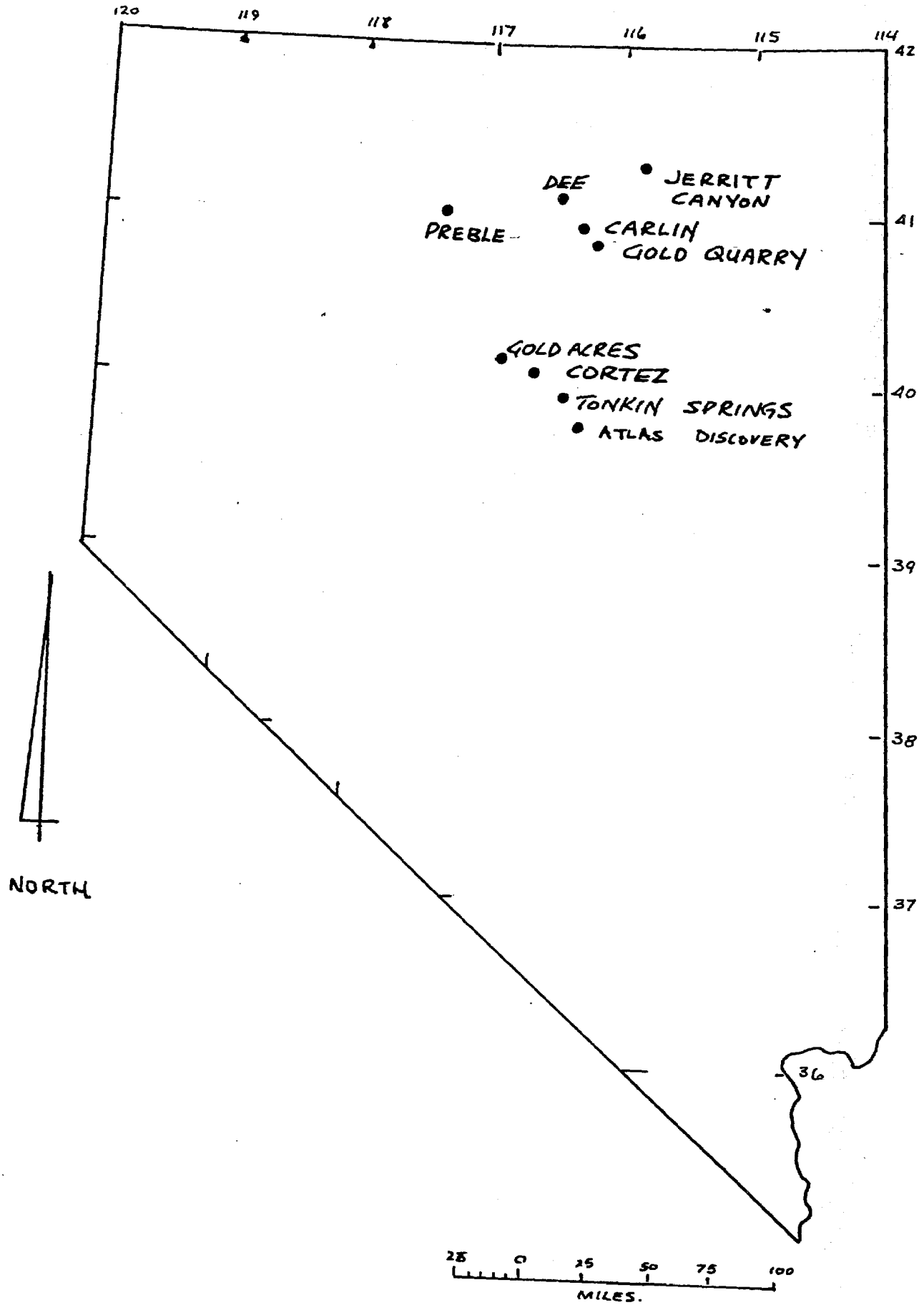


FIGURE 1

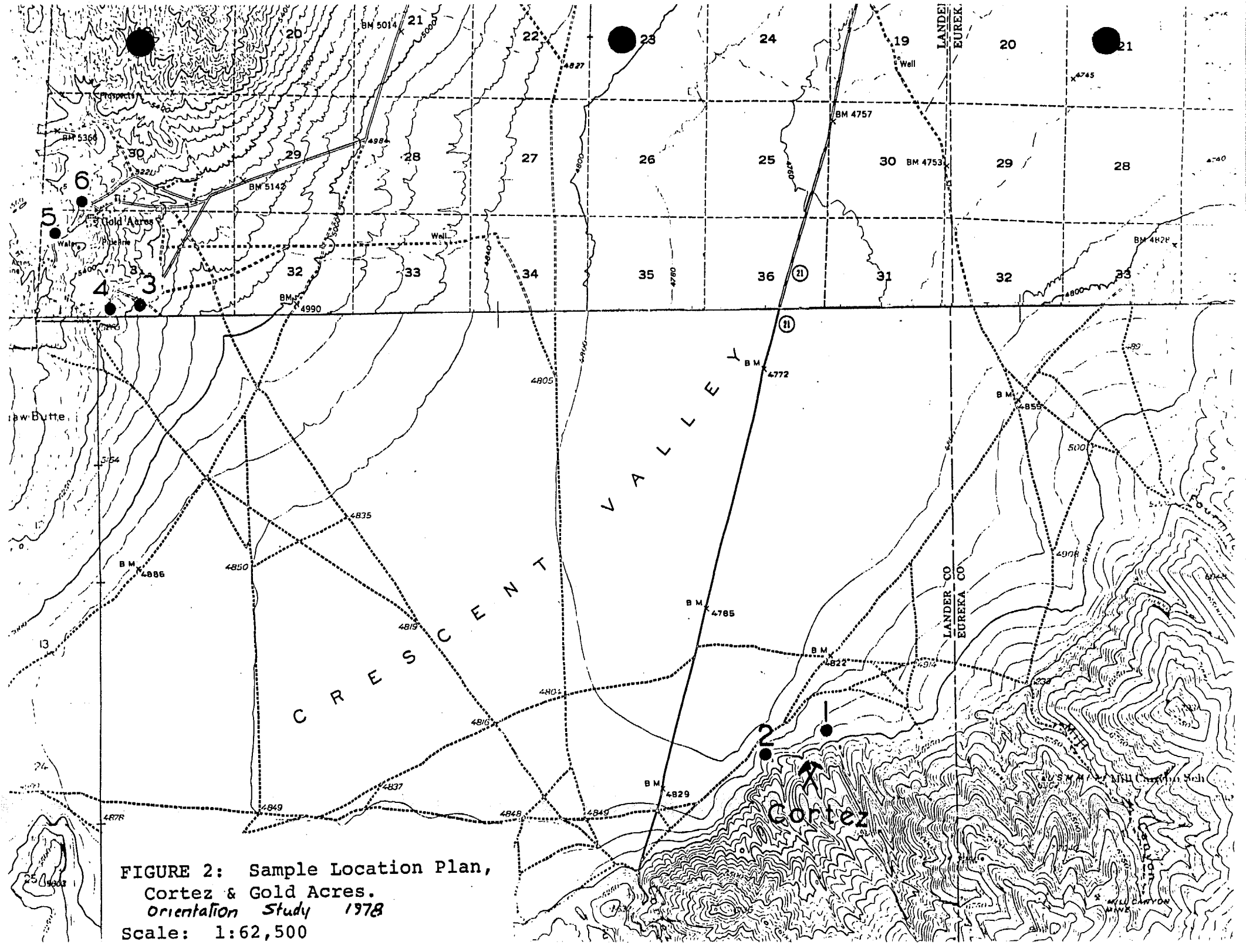


FIGURE 2: Sample Location Plan, Cortez & Gold Acres.
Orientation Study 1978
Scale: 1:62,500

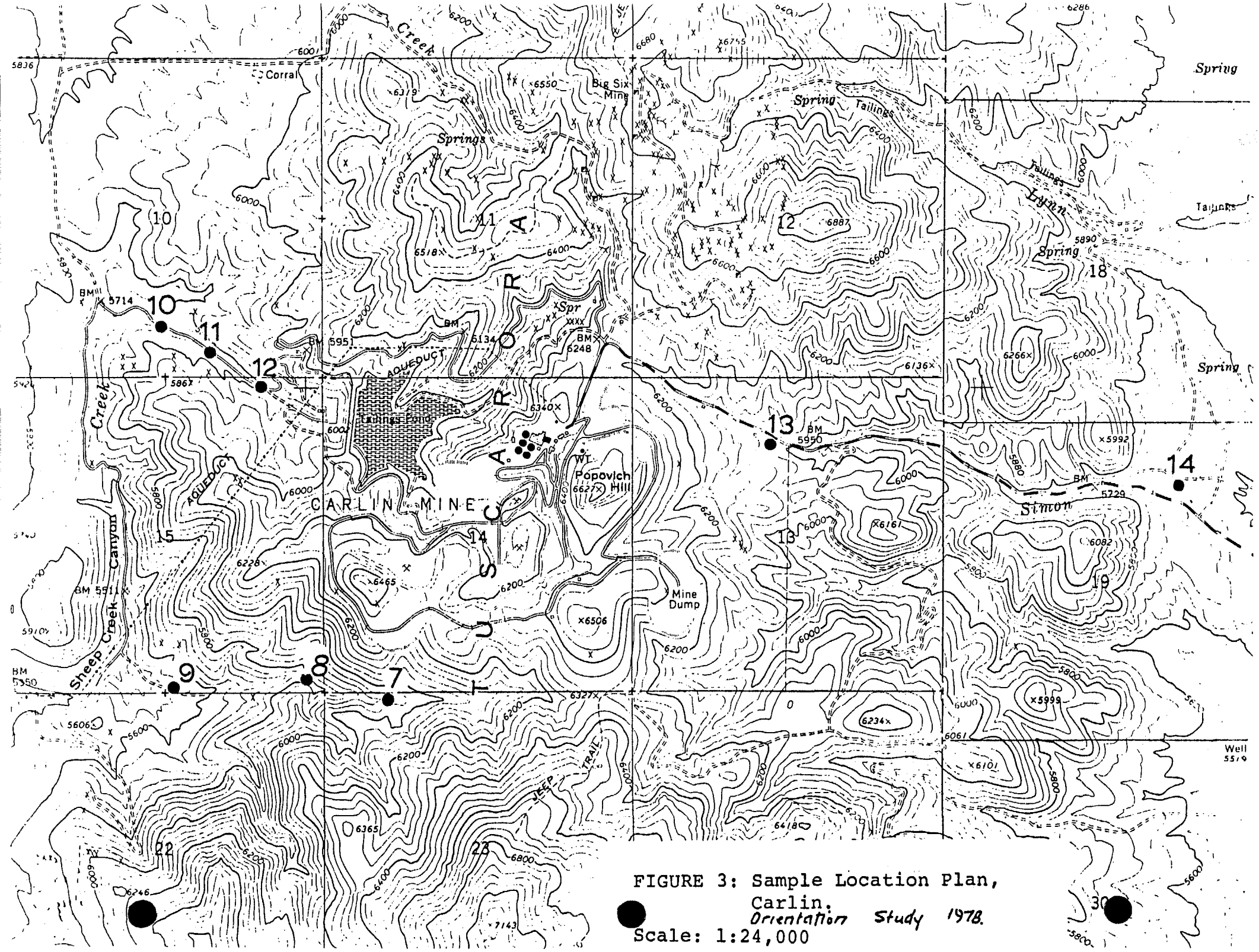
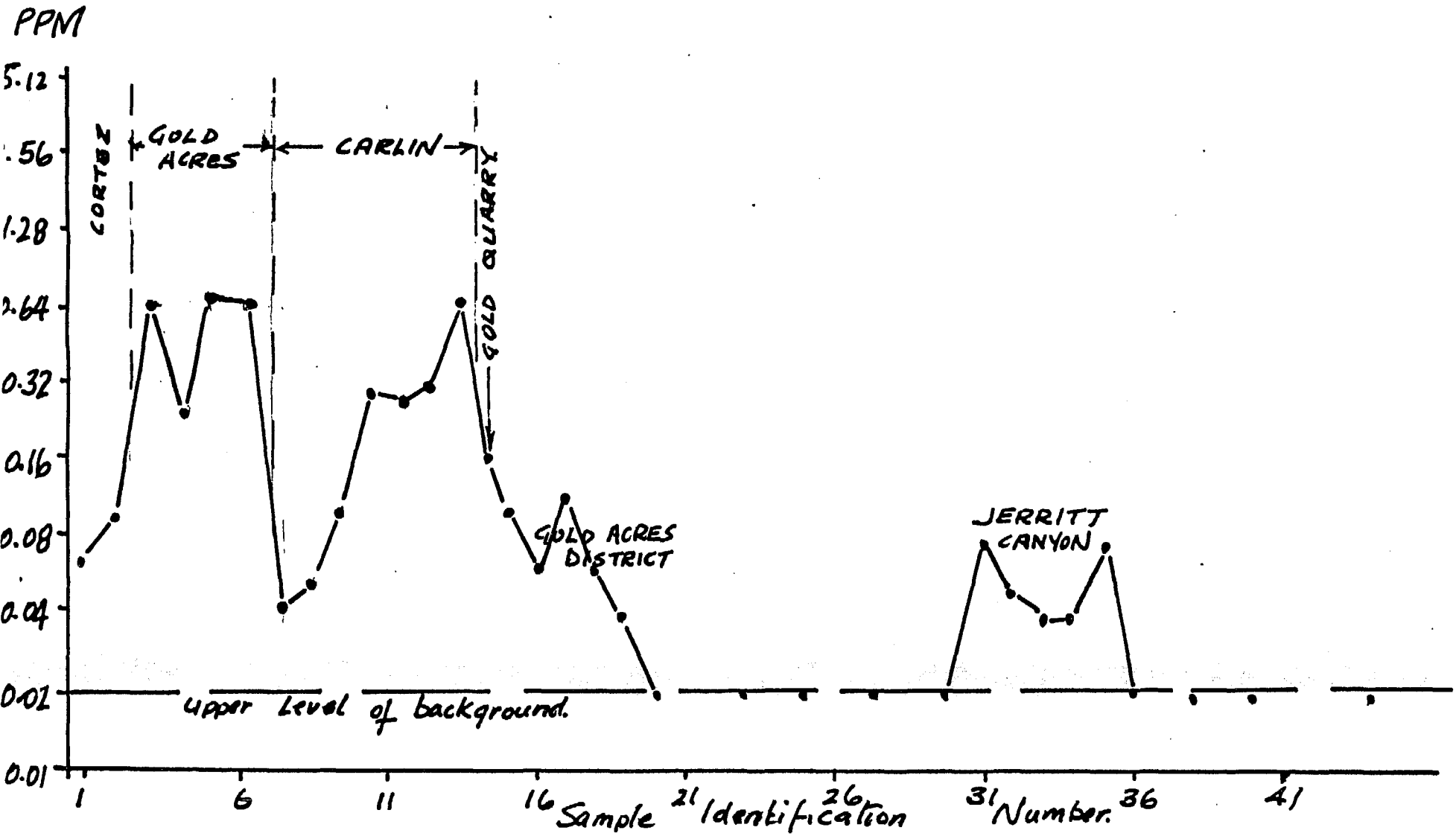
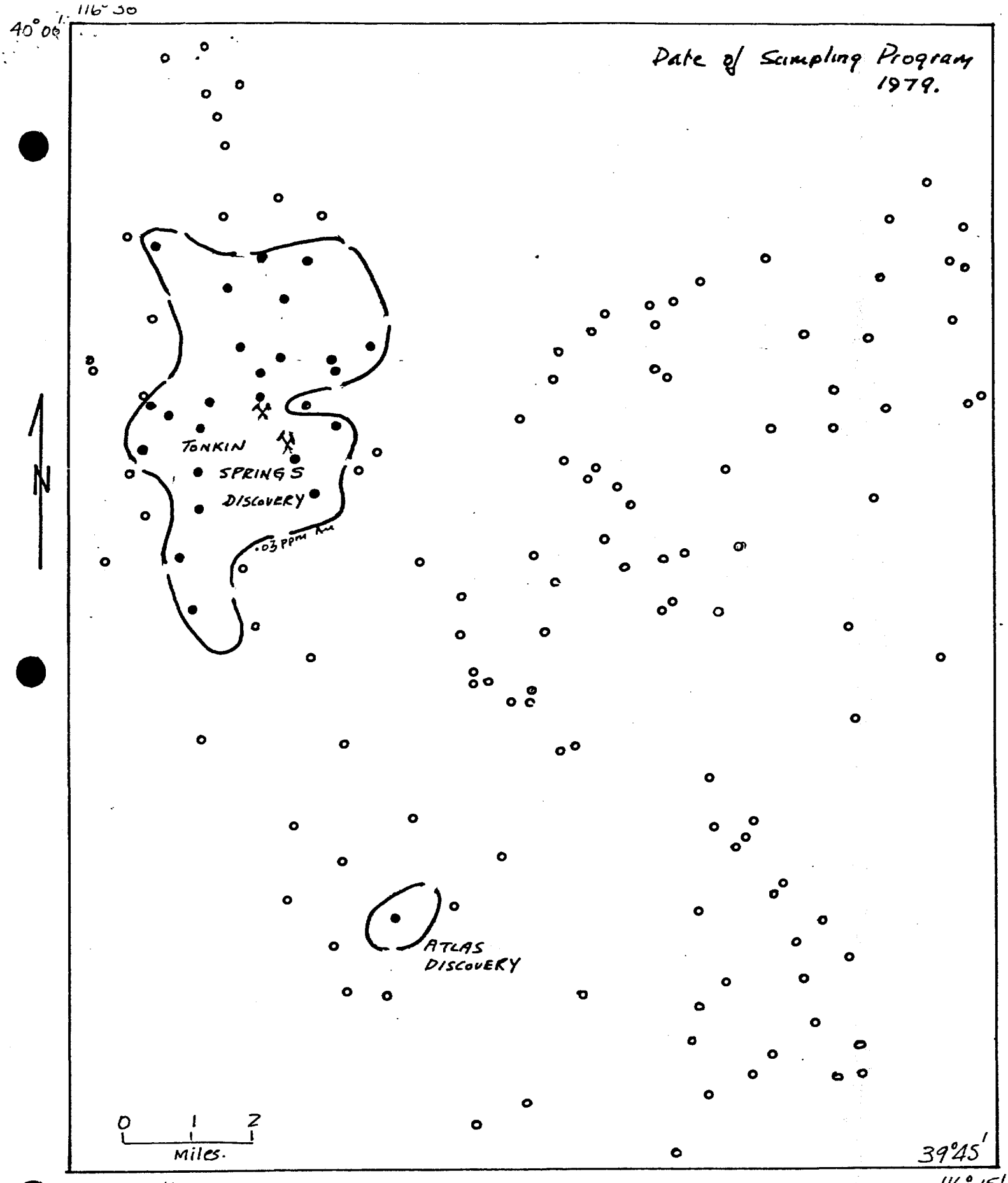


FIGURE 3: Sample Location Plan, Carlin.
Orientation Study 1978.
Scale: 1:24,000

ORIENTATION SURVEY, 1978

GOLD IN STREAM SEDIMENTS
[data on the 'A' Fraction]





KEY

⚒ GOLD DEPOSIT.

o Stream sample Location Less than .02ppm Au

● Stream sediment sample Greater than .02ppm Au.

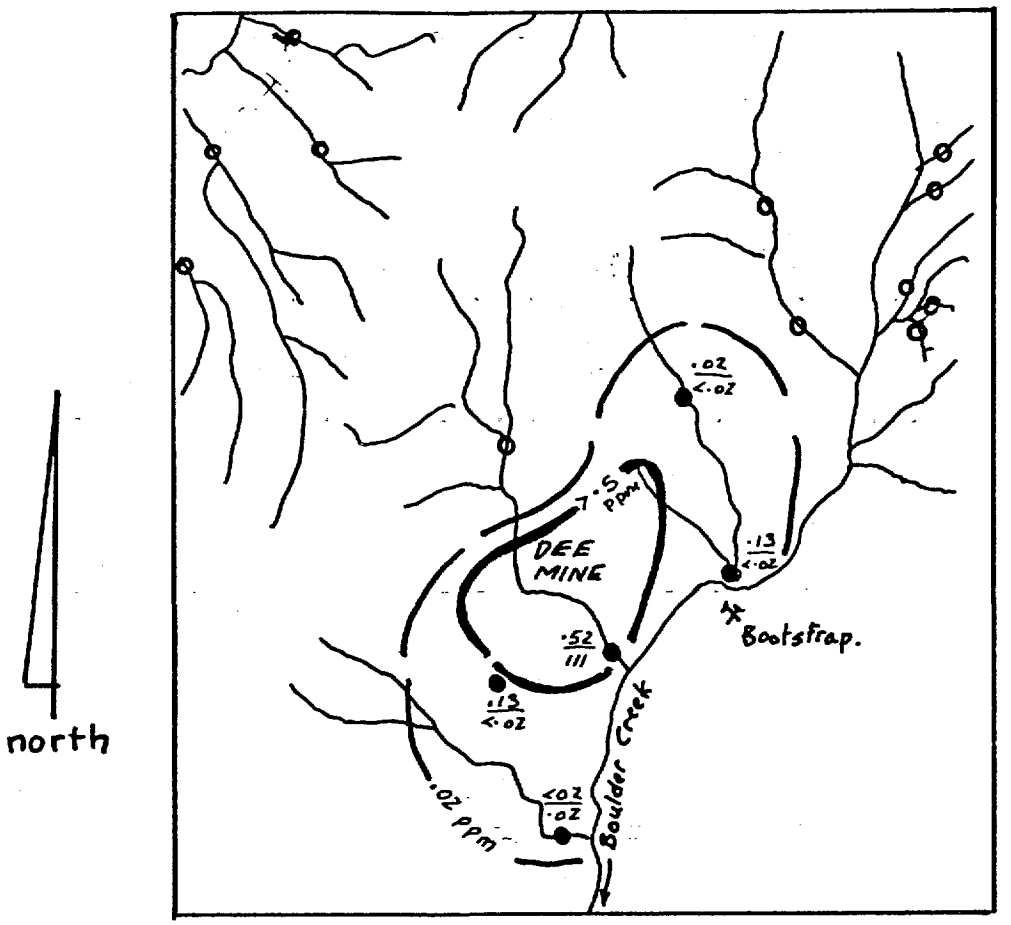
[Data on the "A" Fraction Drainage Sediment]

MAP SHOWING THE DISTRIBUTION OF GOLD IN DRAINAGE SEDIMENT

ROBERTS CREEK Mtn QUAD., NV.

FIGURE 5

Date of Sampling Program - 1980



Key

o Drainage sediment sample Location containing less than .02 ppm Au in both Fractions 'A' and 'B'

● .52 ppm Au in Fraction 'B'
111 ppm Au in Fraction 'A'

⚒ Gold Deposit

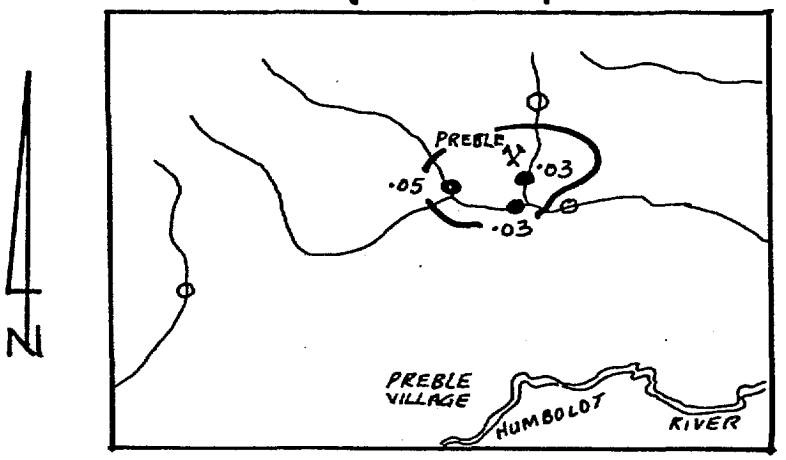
Scale: 0 — 1 MILE

GOLD DISTRIBUTION IN DRAINAGE SEDIMENTS

BOOTSTRAP - BOULDER CREEK AREA
NEVADA

FIGURE 6

Date of Sampling - 1983



Scale:
0 1 MILE

o Stream sediment sample location containing less than ·02ppm Au in Fraction "A"

● ·05 ppm Au in Fraction A of drainage sediment.

X Gold deposit.

GOLD DISTRIBUTION IN DRAINAGE SEDIMENTS
Preble Area, No. Central Nevada.

FIGURE 7

PEX EXPLORATION VENTURE - NEVADA AND ADJOINING STATES

PROGRAM AND BUDGET

The following comments and budget estimate refer to a proposal you received from Precambrian Exploration, Inc, dated September, 1984. This additional information provides a breakdown of the proposed expenditures with comments concerning the staff and facilities of PEX.

Over the course of the next three years, the Joint Venture is expected to explore 3,000 square miles of prospective terrain primarily in Nevada and to delineate 30 to 50 gold prospects. Of these, 2 or 3 may be economically significant. Initially, the PEX inventory of 20 gold prospects identified in previous CARGO programs but neither followed up nor acquired, will be examined. This immediate prospect examination phase may result in the acquisition of properties within the first six months of the Venture.

Any property acquired by the Venture in which limited percussion drilling defines potentially significant precious metal mineralization, will be recommended as a Development Property. Under arrangements that will require Management Committee approval, a Development Property will be funded as a separate project outside the scope of this program and budget.

Each annual program will be composed of three main elements:

1. Reconnaissance sampling of 1,000 square miles (more or less) of prospective terrain.

2. Evaluation of prospects followed by acquisition of highly prospective situations by staking and/or leasing.

3. Exploration of acquired properties up to and including preliminary rotary-percussion drilling.

The reconnaissance proposed in Year I is scheduled to be carried out within a 700 square mile region of the Walker Lane belt in Nevada and within a 300 square mile area in Imperial Co., California. The Californian reconnaissance area would be worked during the winter of 1985-86. As previously mentioned, the winter of 1984-85 should be occupied in following up the 20 prospects in the PEX "data bank" - particularly those in southwest Nevada.

The PEX in-house facilities include a fully equipped sample processing and analytical laboratory to provide high quality geochemical gold data (and other elements). Field operations are conducted out of mobile field camps employing five company-owned 4x4 pick-up trucks, two living trailers and two 400-gallon water carriers. The regular staff consists of one chemist, three geologists, and a bookkeeper/receptionist.

The following budget estimate is for a calendar year. It is assumed for the purposes of this document that exploration costs will be spread evenly over the 3-year period. It will be appreciated that such even expenditure is unlikely.

Fixed Costs	U. S. $
Salaries including employer FICA -- regulars	210,000
Salaries including employer FICA -- temporaries	25,000
Office/lab lease (6,000 square feet)	34,000
Utilities	8,000
Phones	6,000
Supplies - laboratory	15,000
Supplies - office	5,000
Supplies - field	7,000
Insurance: General liability	12,000
Insurance: Medical	12,000
Food (field camps)	8,000
Gas and vehicle maintenance	8,000
	350,000
Property Acquisition (mainly staking)	50,000
Property Exploration (mainly drilling)	100,000
	US $500,000

It should be noted that "Fixed Costs" cover all geological and geochemical activities on reconnaissance, in prospect evaluation and in property investigation. These "Fixed Costs" also include an amount of $80,000 annually that would be spent on sample processing and analysis regardless of whether the work was contracted-out or done in-house. Accordingly, the proportion of the total proposed expenditure that can be properly described as "Overhead" is minor.

The proposed "mix" of activities that make up a calendar year program has been shown by the PEX performance and experience over the past five years to impact substantially on the potential for success. The reasons that would be submitted to support this view are mentioned below:

a. Prospects identified by the PEX reconnaissance programs are generally the product of hydrothermal gold bearing systems that have affected rocks of diverse lithogies. Only specific rock types can act as ore hosts since the ore-forming mechanism frequently involves the addition of silica to the rock at the expense of an original carbonate component. Geochemical anomalies can, therefore, be quite readily assessed on the basis of bedrock geology and geochemistry. There are, however, notable exeptions to the foregoing, for example the Delamar gold deposit, Nevada, which is developed in a sheared quartzite. A second important ore type is that formed in altered silicified volcanics in which an original carbonate component is lacking. A typical example of this deposit type is the FMC discovery near Gabbs, Nevada. The Picacho district at Glamis, S. California, is another variation. Given these variations and complexities, the past five years of relevant

experience gained by PEX permits this important step of prospect evaluation to be accomplished expeditiously. In the final analysis, however, it all comes down to a question of judgment - a component that cannot be quantified.

b. Past reconnaissance exploration by PEX has produced the occasional striking anomalies that have since been found to have reflected important ore deposits, e.g., Gold Quarry and the Dee Mine, both in Nevada. Such high magnitude anomalies are turned up frequently enough to provide the incentive to persist with reconnaissance.

c. Under the terms of the proposed Joint Venture, when a property is found to contain potentially significant mineralization, it will be moved out into a separate and new Joint Venture. The next stage of exploration on such a property will presumably be performed by the Joint Venture participants leaving PEX free to carry out the regional program.

The cost of a 3-year program of the kind described above is, therefore, estimated at U.S. $1.5 million.

M. B. Mehrtens

Oct. 5, 1984.

Mike Mehrtens

F.T.G.

NOV 20 1984

Kirsty/Scott JDS

November 16, 1984

RECEIVED

NOV 20 1984

EXPLORATION DEPT.

F.T.G.

NOV 20 1984

MEMORANDUM TO: F. T. GRAYBEAL

FROM: D. M. FLETCHER

Manhattan Mercury
Homestake McLaughlin
Napa County
California

RECEIVED

NOV 26 1984

EXPLORATION DEPARTMENT

On October 26th this writer in company with the G.A.C. field trip group visited the geological section which encompasses the Homestake McLaughlin gold deposit. Our tour guide was Carl Nelson, formerly a Homestake research geologist but currently a consultant based at Denver. At no time did we enter or were permitted to enter the Manhattan Project area.

As you are aware the Manhattan Project is located about 2 kilometers northwest of Knoxville, Napa County (SW¼ sec. 36, 12N-5W, and NE¼ sec. 1, 11N-5W). This gold deposit was discovered in 1980 as a result of Homestake's search for epithermal microscopic gold associated with mercury hot spring deposits. Since that time Homestake has spent over $40 million dollars in exploration for similar deposits without success other than the "Manhattan" deposit.

Carl Nelson describes the McLaughlin deposit as a mushroom shaped, silicified, transecting, brecciated sinter positioned within Tertiary olivine rich basalt. The mineralized (Au,Hg) vent breccia sinter appears to be fault controlled and it's positioning results from repeated, episodic, explosive refracturing of the silicified sinter. Reported minerals are pyrite, metacinnabar, stibnite and redingtonite. Drilling delineated a 400 meter long open pit 20M ton orebody grading 0.16 OPT gold. Carl Nelson states the gold is encapsulated with silica and extraction requires an autoclave leach. In the leach, mercury competes with gold and tends to hinder recovery. A 1985 start-up is anticipated at a milling rate of 3,000 tpd, to yield a projected 200,000 ounces of gold per year.

Nearby the project on the winding highway adjacent to an incised drainage, "fill" was placed recently to support the road. The "fill" material evidently comes from the pre-mine stripping at McLaughlin

Hydrous chromium sulfate, type locale is Redington mine at Knoxville (Hg mine)

and selected pieces of this float were taken and sent for analysis by this writer (see attached results.) It appears sample #097738, yielding 52.2 grams gold is the ore grade material. Specimens of my samples are described as follows:

#97738 - brecciated sinter, red color, dense tough and silicified with 25% cinnabar and numerous open vugs.

#97739 - brecciated sinter, green color, silicified, patchy network of cinnibar, numerous open cavities with encrusted green clay.

#97740 - layered,rhythmic banded white opaline silica, with abundant vugs, not brecciated, sparse wisps of cinnibar.

#97741 - rubble-breccia, angular volcanic fragments (felsite) up to 10 mm, in dark grey tuffite? matrix.

#97742 - layered, rhythmic banded dark grey opaline silica with occassional brecciated fragments (similar to #97741).

Should you require any specimens from the above sample for possible fluid inclusion work let me know.

D. M. Fletcher

Attachment

ACME ANALYTICAL LABORATORIES LTD. 852 E.HASTINGS ST.VANCOUVER B.C. V6A 1R6 PHONE 253-3158 DATA LINE 251-1011

GEOCHEMICAL ICP ANALYSIS

.500 GRAM SAMPLE IS DIGESTED WITH 3ML 3-1-3 HCL-HNO3-H2O AT 95 DEG. C FOR ONE HOUR AND IS DILUTED TO 10 ML WITH WATER.
THIS LEACH IS PARTIAL FOR Mn.Fe.Ca.P.Cr.Mg.Ba.Ti.B.Al.Na.K.W.Si.Zr.Ce.Sn.Y.Nb and Ta. Au DETECTION LIMIT BY ICP IS 3 ppm.
- SAMPLE TYPE: ROCK CHIPS AU** ANALYSIS BY FA+AA FROM 10 GRAM SAMPLE. HG ANALYSIS BY FLAMLESS AA.

DATE RECEIVED: NOV 6 1984 DATE REPORT MAILED: Nov 13/84 ASSAYER. D. Toye .DEAN TOYE. CERTIFIED B.C. ASSAYER

ASARCO EXPLORATION FILE # 84-3263 PAGE 1

SAMPLE#	Mo ppm	Cu ppm	Pb ppm	Zn ppm	Ag ppm	Ni ppm	Co ppm	Mn ppm	Fe %	As ppm	U ppm	Au ppm	Th ppm	Sr ppm	Cd ppm	Sb ppm	Bi ppm	V ppm	Ca %	P %	La ppm	Cr ppm	Mg %	Ba ppm	Ti %	B ppm	Al %	Na %	K %	W ppm	Au** ppb	Hg ppb
097735	1	13	6	3	4.8	23	4	30	.65	6	5	53	2	30	1	3417	2	2	.01	.01	2	5	.01	256	.01	3	.02	.01	.01	2	52200	2600000
097739	1	4	28	3	.1	19	2	38	.47	7	5	3	2	15	1	57	2	4	.03	.01	2	64	.09	1651	.01	10	.17	.01	.09	2	2300	110000
097740	1	4	3	2	.1	7	1	29	.48	5	5	ND	2	14	1	97	2	5	.01	.01	2	20	.01	218	.01	4	.04	.01	.01	2	755	65000
097741	1	16	14	20	.1	36	3	34	1.34	66	5	ND	2	280	1	169	2	67	.05	.05	16	72	.10	456	.01	7	.56	.02	.24	2	90	44000
097742	1	7	8	11	1.0	29	2	35	.64	70	5	ND	2	11	1	55	2	9	.01	.01	2	38	.04	127	.01	2	.08	.01	.05	2	260	26000

HOMESTAKE

International Mining

September 1984

1/18/85

FROM: W. L. KURTZ

GOLD BAR, NY

TO: HGK
FRK

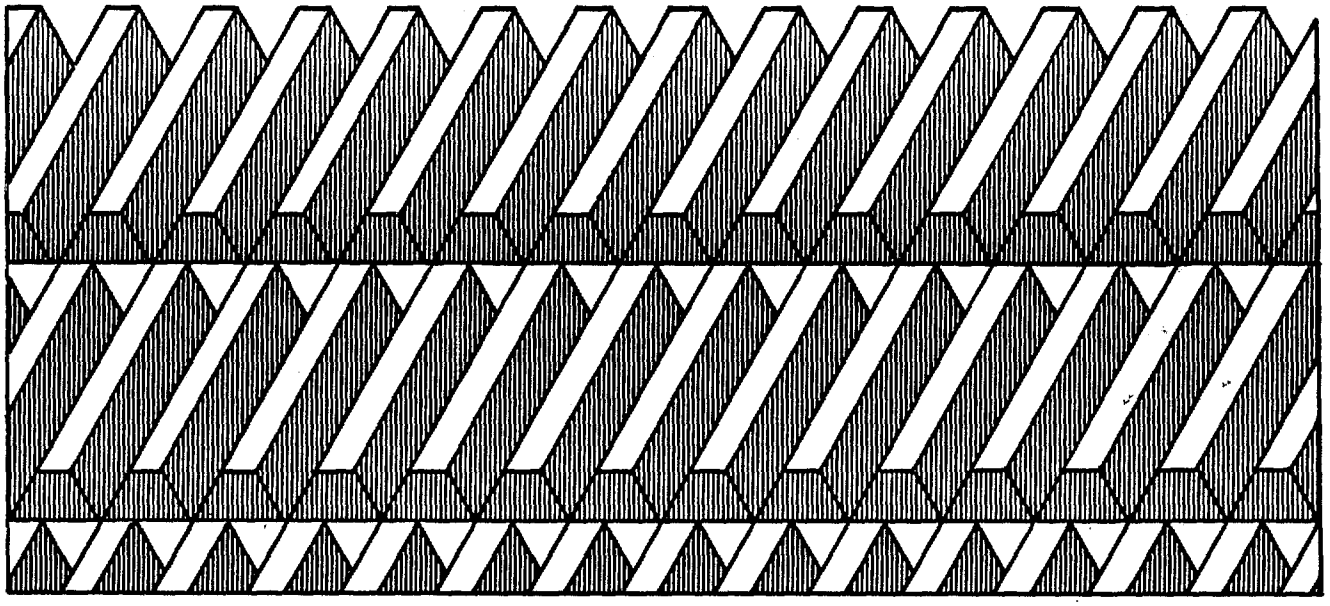
Atlas Corp discovered in NY — reported in press as 7.7 million tons @ 3g/t (0.090z/t)

Starts 1.5m below surface and up to 80m thick

from Mining Journal Sep 14, 1984

cc JMSelf

FINAL ENVIRONMENTAL ASSESSMENT



PARADISE PEAK PROJECT

DEPARTMENT OF THE INTERIOR
BUREAU OF LAND MANAGEMENT

DECEMBER 1984

SUMMARY

Introduction

This Environmental Assessment (EA) presents the conclusions of environmental analyses of FMC Corporation's Paradise Peak Project, a precious metals extraction operation proposed in Nye County, Nevada about 8 miles south of the town of Gabbs. The Paradise Peak Project would involve the construction, operation, and abandonment of facilities for extracting and processing gold/silver ore. The Proposed Project is described in a Plan of Operations submitted to the Bureau of Land Management (BLM), Tonopah Resource Area Office, which is responsible for reviewing the plan to determine compliance with BLM regulations governing surface mining under the general mining laws. As part of BLM's review process, the EA describes the projected impacts of the proposed mining operation on the natural and human environment.

FMC proposes to mine approximately 1 million tons of ore per year and process the ore by a cyanide agitation leach method. The plant would produce 40,000 to 115,000 troy ounces of gold and 850,000 to 3,950,000 troy ounces of silver per year for 12 years of operations, based on current estimates of mineral reserves. Up to 200,000 pounds of mercury would be produced annually as a secondary by-product.

Major components of the project would include an open pit mine, mine waste dumps, process plant, tailings dam and impoundment, and access roads which would directly disturb 354 acres on FMC's claim block area. Additional project components, located offsite, would include a temporary 1.5-mile 60 kV electric transmission line, a permanent 20-mile 120 kV electric transmission line, a water supply system, and a buried telephone cable. Primary access to the mine site would be provided from Nevada State Highway 361 via a local county road known as the Poleline Road. An asphalt access road would be constructed from the Poleline Road to the plant Administration Building.

Alternatives to the proposed action are analyzed in the EA. Alternatives considered in detail in the analysis include: the No Action alternative; water supply alternatives; reclamation alternatives; and a land sale alternative. The EA also briefly documents several alternatives that were considered during early project design, or during

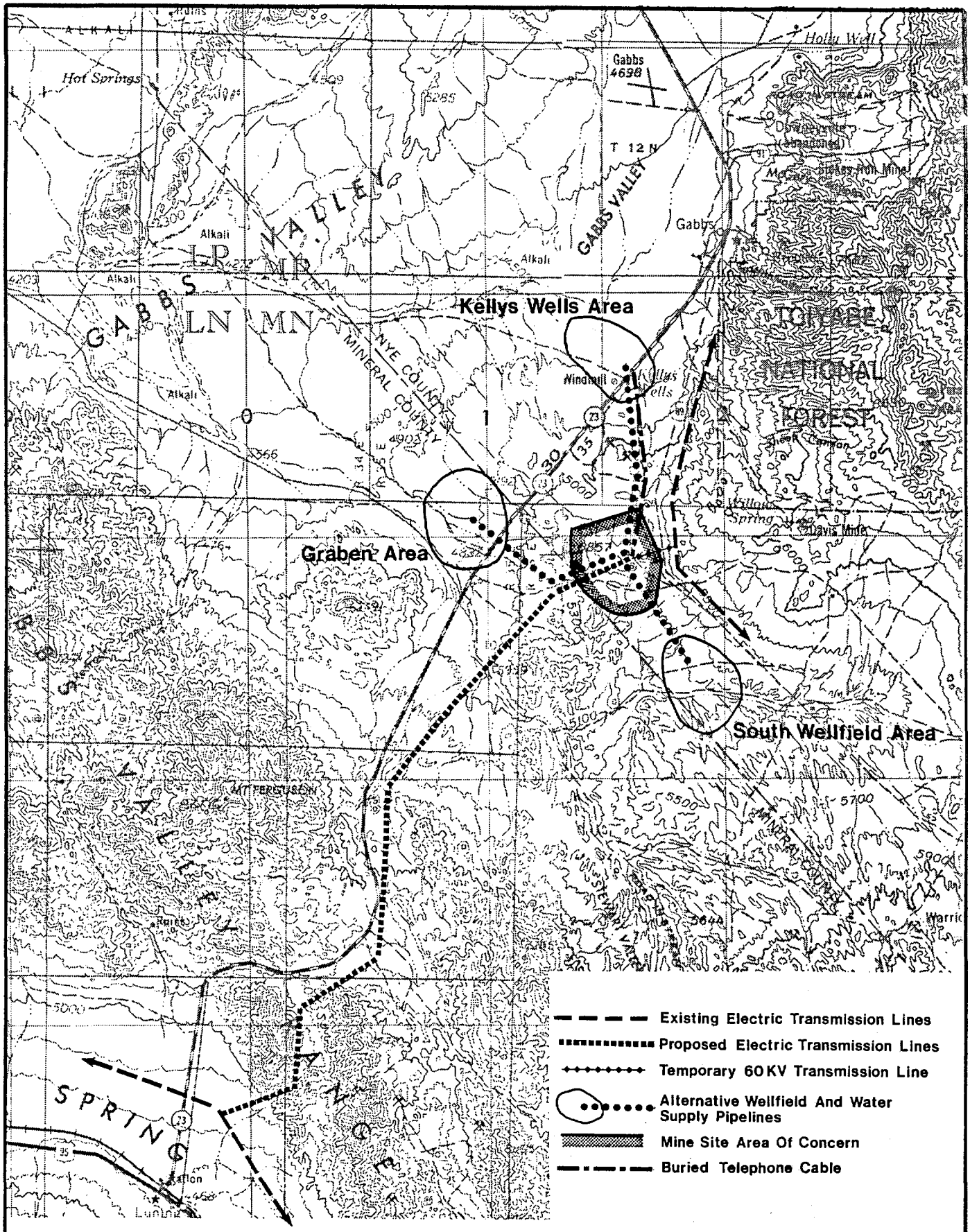


Figure 1-2
Location Of Proposed Facilities For The Paradise Peak Project

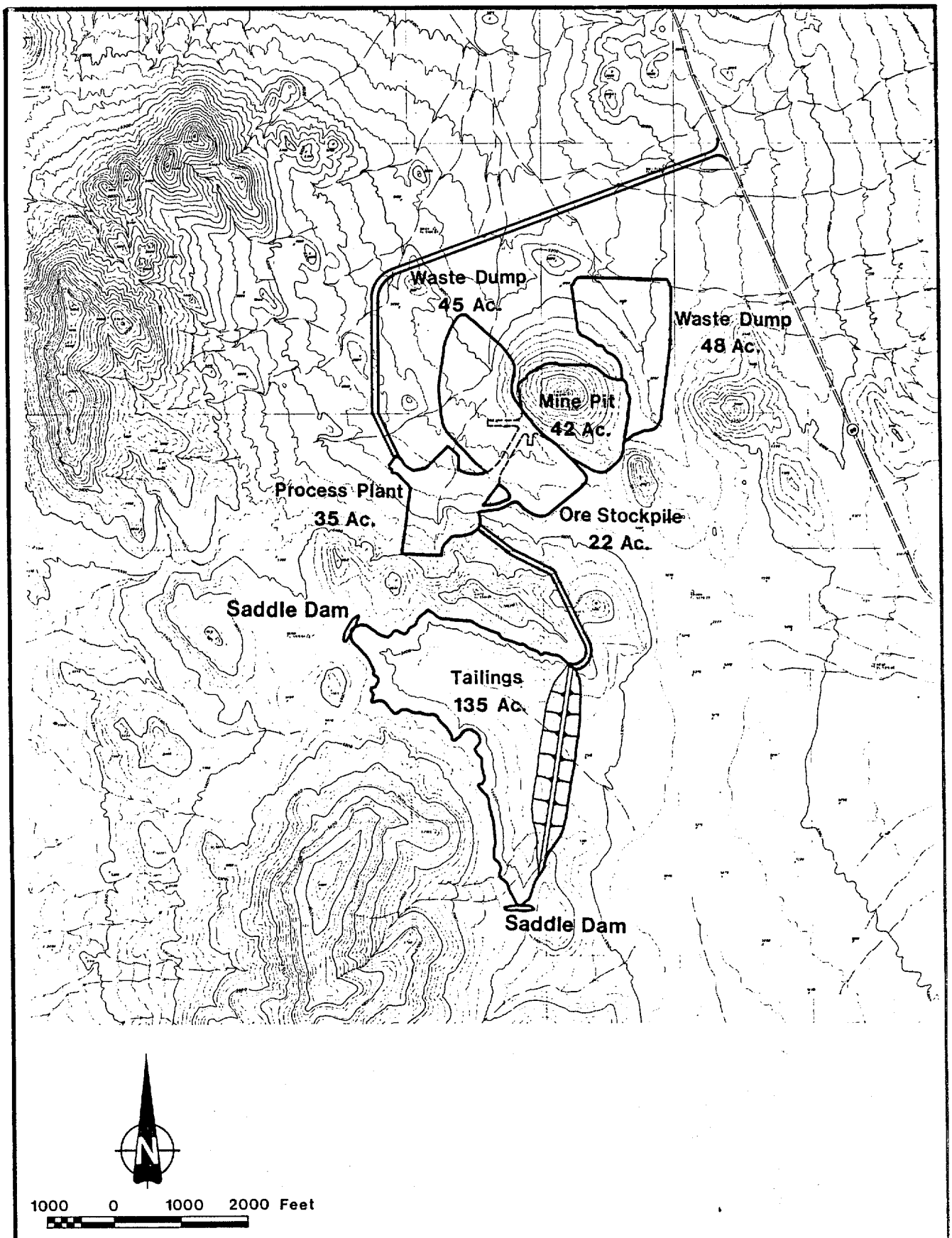


Figure 1-3.
Facilities Layout for Paradise Peak Project

Source: ERT 1984

Employment requirements would vary with the project phase. Peak employment during construction (Figure 1-4) would be 300 to 400 workers. The operations workforce would be approximately 200 employees.

The following sections provide detailed descriptions of proposed development and operations plans.

1.2.2 Mining Facilities

The proposed surface mine (Figure 1-5) would eventually disturb 42 acres of land located on a conical hill that rises to an elevation of 5,500 feet. Initially, a short pre-production phase would remove the top 40 feet of the hill; about 45,000 tons of waste material would be generated and used as fill in haul road construction. Production mining would then be initiated, resulting in the removal of approximately 12 million tons of ore and 17 million tons of waste rock over the 12-year project life. Mining would proceed through a series of mining benches at 20-foot intervals until the entire hill was mined. Mining would continue by drop-cutting to develop the open pit. Ramp systems and haul roads would be constructed as required and would be 50 feet wide and at an 8 percent maximum grade.

$\frac{17}{12} = 1.42$

The mine would operate 2 shifts per day, 5 days per week, and 50 weeks per year. Waste rock and ore would be drilled by crawler-mounted drills, blasted using a mixture of ammonium nitrate and fuel oil, and removed by 4-cubic yard hydraulic front-shovels and 35-ton capacity haul trucks. Crawler dozers and a motor grader would be used for road building, shovel area cleanup, and road and dump maintenance duties associated with the mining operation.

Approximately 17 million tons of waste rock must be mined and disposed of over the life of the project. A low grade ore stockpile and two waste rock disposal areas would be located adjacent to the mine. The low grade ore stockpile would cover 22 acres when at capacity. It would be located immediately south of the ore body. A waste rock disposal area, covering 45 acres, would be located immediately west of the ore body. A second dump, covering 46 acres, would be located immediately northeast of the ore body. The waste dumps would be designed to provide minimum interference with the proposed mill facilities. Consequently, the dumps would be elevated, with coincident uphill truck haulage, rather than expanded to occupy a larger land area.

Geotechnical investigations for facility design at the site have identified isolated occurrences of expansive clay soils. These are residual soils of weathered volcanic bedrock and are typically overlain by a variable thickness of alluvial soils (Harding Lawson Associates 1984c).

Hydrocompaction of loess soils is a potential hazard in the region. The problem would be localized, occurring only where there is a relatively thick deposit of susceptible soil and water is added by some artificial means. Loess deposits commonly occur northeast (leeward) of playas and alluvial fans in the area (Dohrenwend 1982b). At the site, a 5-foot thick loess soil was identified east of the proposed tailings dam location. There are no loess deposits identified at the proposed mill site or the preferred tailings dam site (Harding Lawson Associates 1984a).

Landslides on natural slopes are uncommon in the project area. Only one landslide was identified in surficial geologic mapping of the 475-square mile Gabbs-Luning area (Dohrenwend 1981). The limiting factors are the dry climate, thin soil cover in upland areas, and general lack of clay soils. No active or potential landslides or rockfall hazards have been identified at the project site.

Erosion, deposition, rapid changes in stream course, and flash flooding on young, active alluvial fans may occasionally cause damage to roads and structures in the area. Most of the project site and facilities are located in the upper portions of drainage basins where the quantity of runoff and thus potential hazards from active fans are limited (see Surface Water Section 2.4.1). The exception is at the northeast corner of the waste rock dump area, where the drainage from a relatively large basin crosses the site.

2.2.3 Mineral Resources

Recovery of economic minerals has been an active process in the project area since the early history of Nevada. Indications of past mining activity include numerous prospects, shafts, and pits which can be found near the project site and are shown on topographic and geologic maps. Present mining activity in the project vicinity includes the C.E. Basic magnesite mine near Gabbs and numerous small gold and silver mines (typically two to three workers) concentrated in the Ione and Luning-Mina areas (Nevada Department of Industrial Relations 1983).

The Paradise Peak Project ore deposit is hosted in hydrothermally altered volcanic flows and tuffs of mid-to late-Tertiary age. Precious metal mineralization is found to some degree in each of the stratified volcanic rock units identified at the site except the basal latite unit. The majority of the economic mineralization occurs in silicified breccia and composite tuff units. In general, the higher grade precious metal mineralization corresponds to zones of higher silicification, but this is not always the case. High grade ore contains approximately 0.2 ounce of gold per ton, 12 ounces of silver per ton, and 290 parts per million of mercury. In addition to precious metal mineralization, pyrite, jarosite, iron oxides, barite, orpiment, realgar and native sulfur have also been identified.

FMC has explored the site for additional mineralization to avoid making future economic mineral reserves inaccessible. Future reserves could include extensions of the identified ore body, low-grade ore deposited as waste rock, and undiscovered ore bodies in the area. An existing inactive underground mercury mine is located within the southern portion of the proposed pit and consists of an approximately 70-foot deep shaft with several lateral drifts. There are numerous inactive mining claims in the area. These claims may be crossed by proposed alternative water pipelines and power transmission line corridors.

2.3 Paleontology

Paleontologic resources are an important geologic resource of the area. Unique occurrences of sedimentary sequences and fossils provide important keys to understanding ancient flora, fauna, geologic, and climatic conditions. A regional description of Mio-Pliocene floras of west-central Nevada is presented in Axelrod (1956).

The Union District, located approximately 16 miles northeast of the project site, contains exceptionally complete and unaltered stratigraphic sections of Paleozoic to middle Jurassic sediments. Within this District is Berlin-Ichthyosaur State Park which preserves for public display the mining ghost town of Berlin and the fossils of large marine reptiles (Silberling 1959).

FROM: J. H. COURTRIGHT 1-24-85

TO: RLB

CPI

1930 – 50
1967 – 100
Oct. 1984 – 312

Gold

$\frac{50}{312} \times \$300$ (present price) = \$48/oz

inflation adjusted real price

cc WLK, TDS, SAA

Newsbeat

Pretty fancy paper for austerity!!

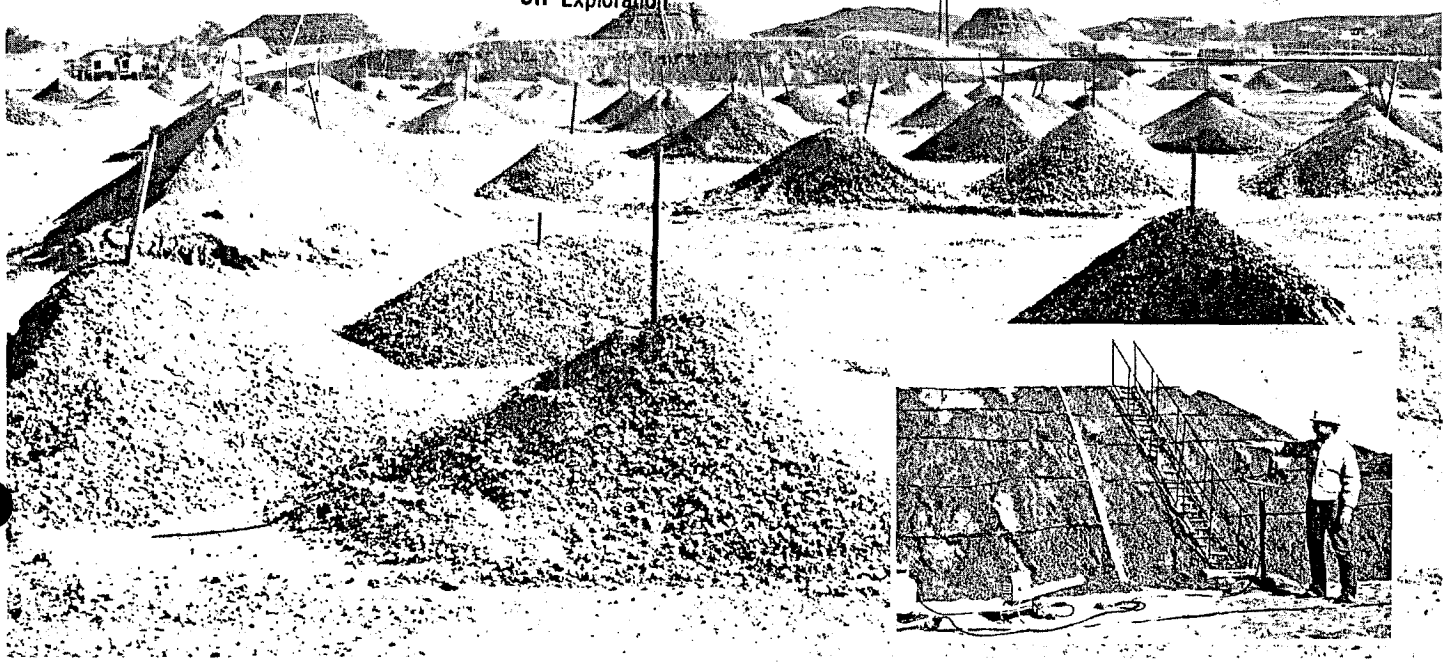
January 1985

ASARCO Incorporated

FEB 8 1985

SW Exploration

Sean Hagerty



Gold ore piles (left) before being carefully placed into mounds (background & inset) for "heap leaching" by a weak cyanide solution.

"Heap Leach" Gold Mining. . . the Mesquite Project

Gold in California! The thought conjurs up images of rugged prospectors, boom towns, and fabulous wealth for those lucky enough to find it, and avaricious and tough enough to keep it.

As with most images of early California, the gold mining picture has changed dramatically over the years. The prospector with his burro, blanket, gold pan, and six-shooter has given way to engineers in hard hats who use modern technology instead of sluice boxes.

Over 200 years ago, Indian stories of gold lured Spanish explorers into what is now eastern Imperial County, California.

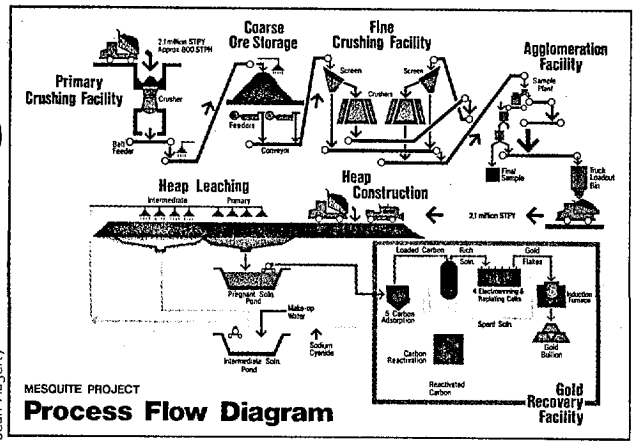
Ancient mine workings near the Colorado River just north of Yuma are thought to date from this period.

Prospecting intensified with the onset of the California gold rush in 1849.

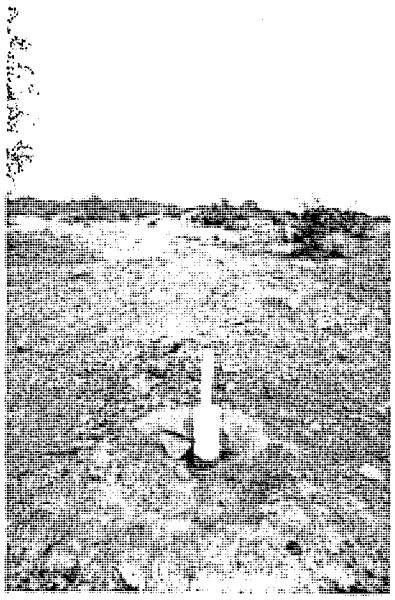
During the next ninety years, dozens of small gold deposits were discovered and worked in the mountain areas east of the Imperial Valley.

Prospectors have recently returned to many of these old mining areas. In 1981, Gold Fields discovered a significant low-grade gold deposit in the Mesquite Mining District and has continued exploration for additional reserves. continued page 4

Bureau of Land Management



Photos by Sean Hagerty



Test holes drilled to remove ore samples in hopes of finding gold.

Ore piles at crushing facility.

Historically, gold in California was found in easily-extracted placer deposits in stream beds or running in veins through rock.

Since then, a third type of gold deposit has been discovered which is of a very fine texture and rarely visible to the naked eye. Early California miners had no idea these deposits existed.

This kind of gold is present at the Mesquite site and its mining is made possible by an increase in the gold price, and by recent advances in metallurgical technology that facilitates the treatment of low-grade gold ores.

For three and a half years, the Gold Fields Mining Corporation has been exploring and testing the bedrock in the area, located a few miles northeast of the popular off-road vehicle recreation site at Glamis.

The company has drilled over 300 sample holes, tunneling into the mineralized bedrock to determine the extent of the deposit.

They have also been conducting small-scale tests to determine the best and most economical way of extracting the gold from the rock.

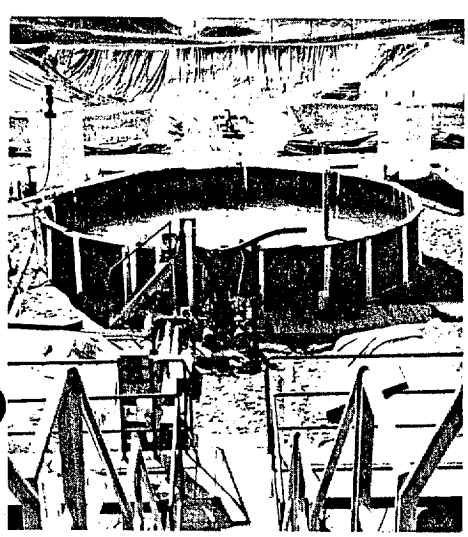
Up to now, Gold Fields spent over $10 million on the project, while actually producing only a few ounces of gold. The evaluation, however, has shown that the Mesquite deposit can be economically developed by Gold Fields.

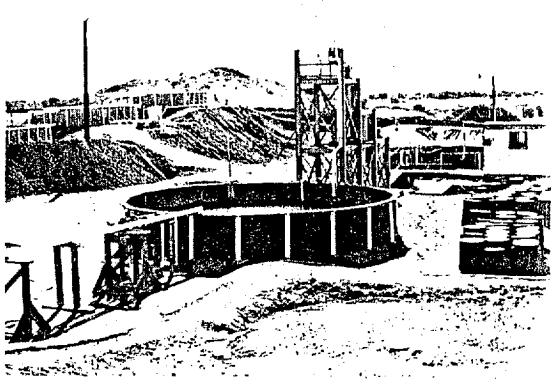
The company has submitted a plan to the Bureau of Land Management for mining construction and development which will reach major proportions.

Approximately 152 million tons of rock will be moved and treated to extract about 56 million tons of mineralized rock (gold ore).

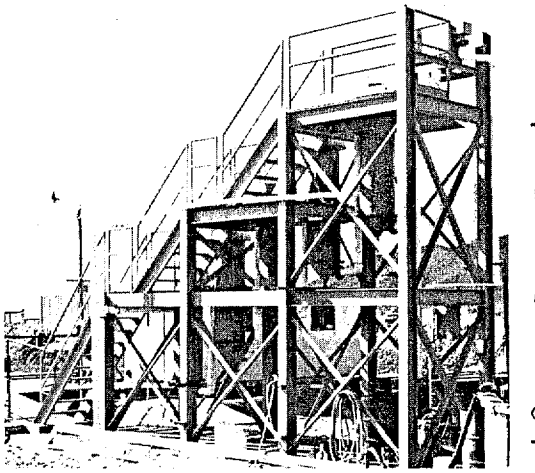
The gold ore will be transported from the mine by trucks and fed into crushers where rocks up to three feet in diameter will be reduced to fragments less than 5/8" in size.

The crushed ore will be heaped carefully onto impervious pads and then sprayed with a weak cyanide solution (0.1 percent) to dissolve the gold. This solution will dissolve only the gold, much like water dissolves sugar, in a process known as "heap leaching."





Solution ponds and adsorption towers at the Gold Fields site.



Gold Mining

continued from front

Gold-bearing solution flows by gravity from the ore heap to a solution pond. It is then pumped through a series of adsorption towers packed with carbon granules.

The gold-bearing solution will flow by gravity from the ore heaps and be pumped through a series of adsorption towers packed with activated carbon granules.

The carbon will remove the gold from the solution which will be brought back to its original chemical composition and returned to the leach pads to pick up more gold as part of a continual closed circuit system.

The gold deposited on the carbon will be stripped from the carbon granules by a hot caustic solution.

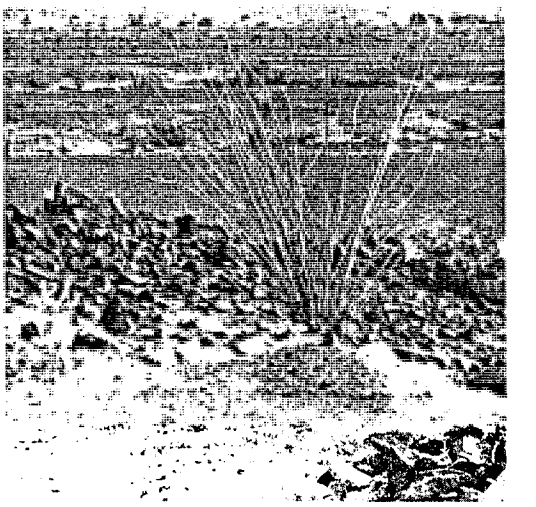
The rich, gold-bearing caustic solution will be piped through electrolytic cells where the gold will ultimately plate out as thin flakes of gold foil.

The flakes will be melted in a furnace to form "dore" bars which will be transported by armored car to an outside refinery for purification and marketing.

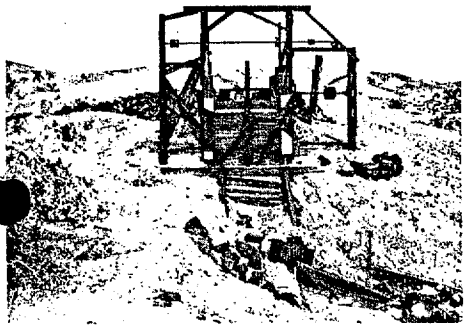
Gold Fields has estimated that the total value of the gold in the deposit could be as high as $940 million if the price of gold remains at about $300 an ounce.

The Mesquite Project was the subject of a joint environmental impact report/environmental assessment to comply with federal, state, and local environmental regulations.

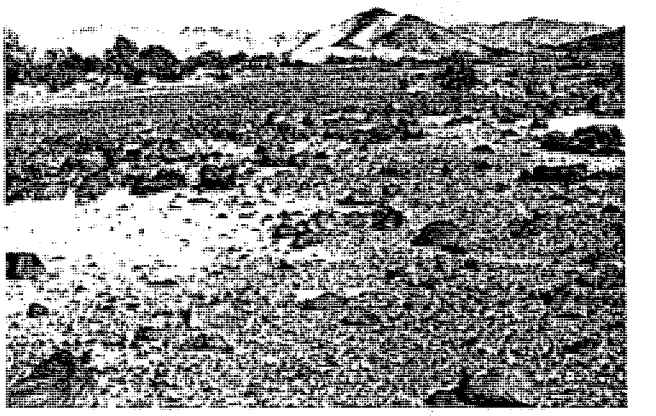
By Sean Hagerty, BLM Geologist, and Gold Fields Mining Corporation



Imperial County desert scenes in an area rich in gold.



Historic mine near the Gold Fields "heap leach" gold mining area.



Feb. 2, 1985

Vol. 74 No. 5

Skillings' Mining Review

ASARCO Incorporated

FEB 11 1985

SW Exploration

PUBLISHED EVERY WEEK

ESTABLISHED 1912

mine
cubic

GOLD

FROM: F. T. GRAYBEAL

TO: WL Kurtz
JD Sell
JC Balla
PG Vikre
DM Smith
DD Harper

Last P of attached is of interest

ASARCO Incorporated

FEB 15 1985

SW Exploration

2/7/85
File: Nevada Prospecting Syndicate

B.C. budget to address mining tax concerns

JDS → well full
Gold Curtis Trip

ASARCO

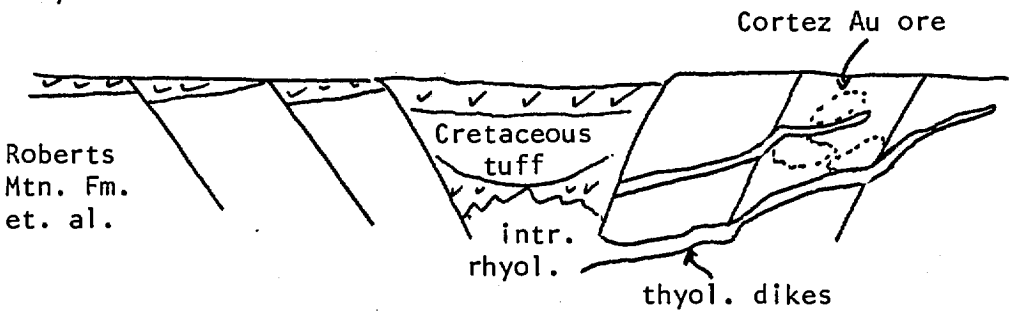
Great Basin Exploration Division

February 8, 1985

MEMORANDUM FOR FILES

Cortez caldera and gold mineralization
Lander-Eureka Counties, Nevada

At the 1st McKelvey Forum (USGS Research on Mineral Resources), Denver Colorado, Feb. 5-7, 1985, J. Rytuba related the sedimentary rock-hosted gold deposits at Cortez and Gold Acres to rhyolite dikes from the 32-33 m.y. old Cortez caldera. This caldera produced the Caetano tuff which covers parts of the Toiyabe Range, Cortez Mountains and Shoshone Range. According to Rytuba, the rhyolite dikes are intimately associated with 0.1 to 0.4 opt Au in Roberts Mountains Formation carbonate strata. The dikes themselves are unmineralized but highly altered. K-Ar ages on dikes average about 35 m.y. and may belong to early Caetano volcanism or be reset. Rytuba calls on the Vinini Formation as the source of Cortez et al., Au, Ag, Sb, As, Hg. A cross-section through the caldera and Cortez deposit simplified after a Rytuba slide:



Providing Rytuba is correct , gold targets remain in the caldera vicinity adjacent to rhyolite dikes. Because of intense exploration in the area most gold anomalies probably have been drilled. A program directing attention to associated element anomalies near Cortez rhyolite dikes might be worthwhile.

Note: W. Rehrig also likes Rhyolite Dikes, esp in swarms.
S. Keith likes 2-mica granites & dike swarms.
These for Au in CDCA & W-AZ regions.
JDS.

Peter G. Vikre

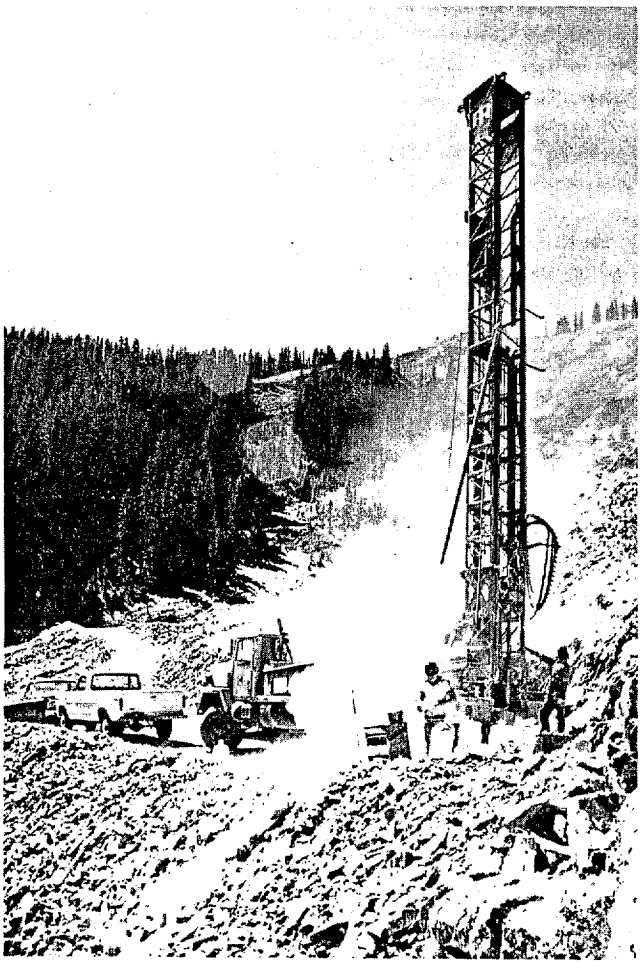
Peter G. Vikre

PGV/jn

cc: Mr. B. J. Maher
Mr. F. T. Graybeal
Mr. W. L. Kurtz ✓

RECEIVED
FEB 14 1985
EXPLORATION DEPARTMENT

Freeport-McMoRan Inc.
4th Quarter Report.



Exploration activities in search for precious metals

ning phase. The 90% net profits interest attributable to the Royalty Trust covers 16 of the 33 producing platforms and one of the platforms in the completion phase.

HARD MINERALS EXPLORATION

FMI's exploration program for hard minerals during the 1984 season was successful in identifying several prospects which hold promise as future projects in gold, diamonds and copper. Expenditures were increased 41% over 1983, an all-time high.

In the United States, the most notable successes were at Jerritt Canyon and Big Springs, Nevada, where we continued investigating gold mineralizations near FMI's joint-venture gold mine. Recently, snow cover ended the drilling in the Jerritt Canyon district, but not before the program's 379 holes indicated a geologic reserve of approximately 4,500,000 tons of gold-bearing ore with a grade of 0.14 ounce per ton. We are close to completion on engineering work to determine how much of this reserve is economically recoverable and can be added to Jerritt Canyon's minable reserves.

Last quarter, we reported the discovery of what appears to be a small, commercial-grade gold deposit at Sammy Creek in the Big Springs project area in Nevada. A total of 206 holes were drilled during 1984, the third year of this extensive exploratory drilling program. The results indicate two mineralized areas that are structurally complex and will require additional testing to determine their economic significance. Engineering and metallurgical studies to better define the recoverable reserves will be completed in 1985.

The Bow River diamond project in the Kimberley area of Western Australia continued to offer encouragement as the 1984 season drew to a close with the arrival of the annual wet season in northwest Australia. The work to date has indicated the presence of approximately five million metric tons of gravels with an average grade of better than 0.6 carats per metric ton. Additionally, mapping of surface and subsurface characteristics in conjunction with sampling data indicates that the presence of diamond-bearing gravels could be extensive within our work area. The potential of the project has been upgraded following the 1984 program. The 1,300 carats of diamonds recovered from approximately 2,300 metric tons of gravels mentioned in our last quarterly report have now been initially appraised at sufficient value to warrant an accelerated and larger sampling program in 1985. To this end, a pilot plant capable of processing 300 metric tons per day of gravels is currently being fabricated for use in 1985. The diamond parcels collected to date, which are thought to be too small to be representative, together with the larger parcels to be accumulated in 1985 must be further evaluated before the economic viability of this project can be determined.

We previously reported that FMI has been exploring for precious metals at Bald Mountain in northern Maine. During the fourth quarter, drill samples and data from this prospect were subjected to careful metallurgical testing and evaluation analysis. We have concluded from these studies that the development of the Bald Mountain property would not lead to financial results that meet FMI's standards, and the project has been discontinued.

Freeport Indonesia continues to work on an exploration adit toward what we believe to be a large potential reserve of copper ore of excellent grade at the Ertsberg East orebody in Irian Jaya. This discovery, based on drill-hole data, is beneath the existing underground mine and comprises two deposits at different elevations. At its completion, scheduled for early 1987, the adit will be approximately 6,000 feet long, and will allow more efficient production of ore from the existing higher-elevation mine as well as from the other orebodies which we hope to prove through this effort.

Other Matters

As mentioned previously, FMI intends to form an oil and

ASARCO

Southwestern Exploration Division

November 5, 1984

Memorandum to Staff

Some Gold Notes

Attached for your review on discovery and operating costs and cash flow potential are four articles:

1) North American Hardrock Gold Deposits - E&MJ October 1984.

2) Dean Witter Reynolds (Canada) article which was distributed at the Denver SME meeting.

3) SME/AIME Preprint #84-312 - The North American Gold Mines: Operating Costs, by A. W. Garson (gives added information on the Dean Witter Reynolds' diagrams).

4) SME/AIME Preprint #84-355 - Optimum Production Rate for High-Grade/Low Tonnage Mines, by R. Glanville.

James D. Sell

James D. Sell

JDS/cg

Attachments

cc: FTGraybeal (w/attach)

11/5/84

CIRCULATE

~~TCBenavidez~~
~~JHCourtright~~
~~WDGay~~
~~AGiesecke~~
~~EMKurtz~~
~~HCKreis~~

~~Dave Skidmore~~
~~WLKurtz~~
~~JDSell~~ - 4757
~~GJStathis~~
~~HMStone~~
~~JRStringham~~

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MONTREAL • 635 Dorchester Boulevard West, H3B 1S1 Phone (514) 871-0004

DIAGRAM 1

Mining Cost versus Ore Grade

($/ton) (oz./ton)

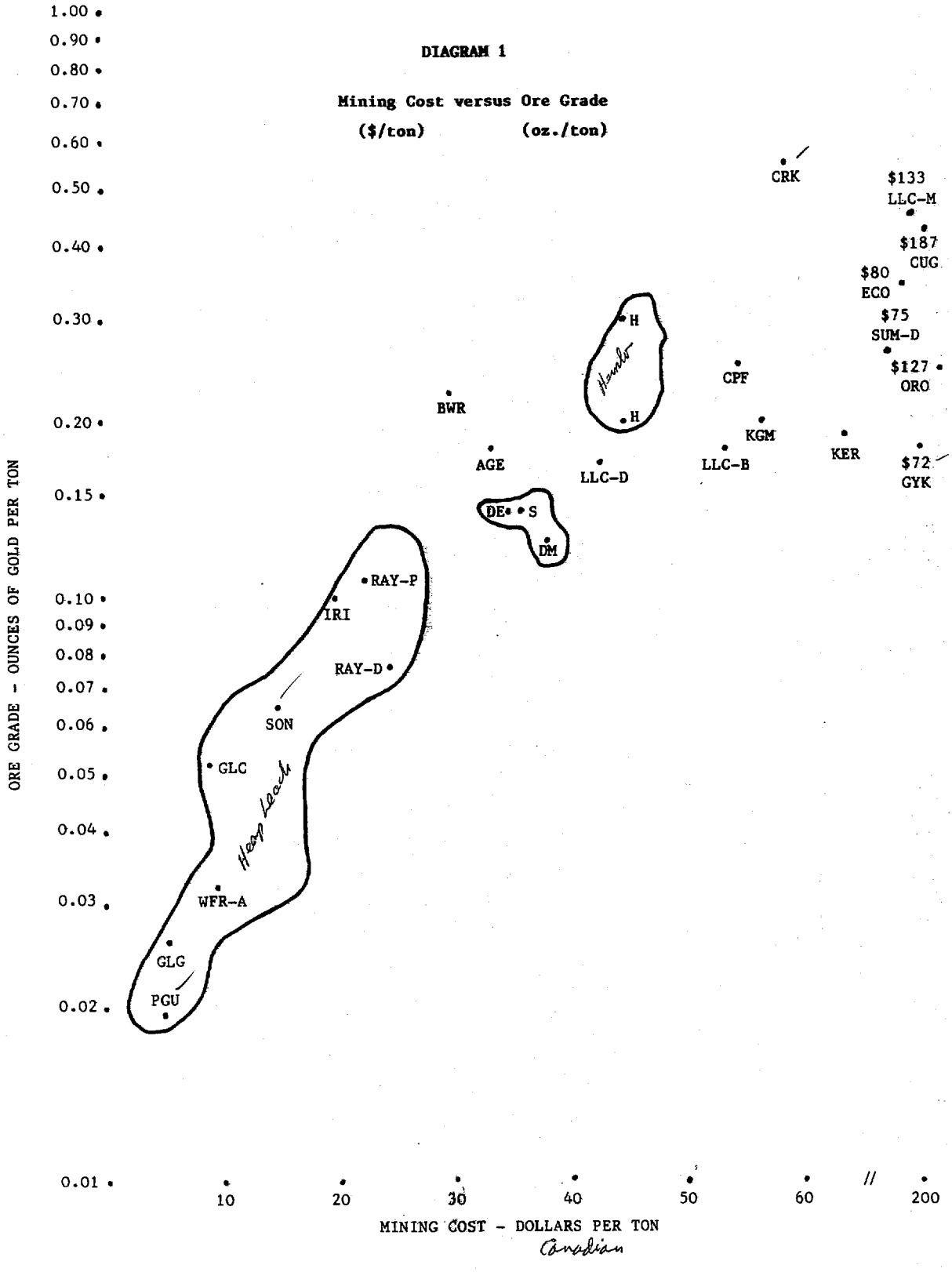

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DIAGRAM 2

Mining Cost versus Ore Grade

($/oz.) **(oz./ton)**

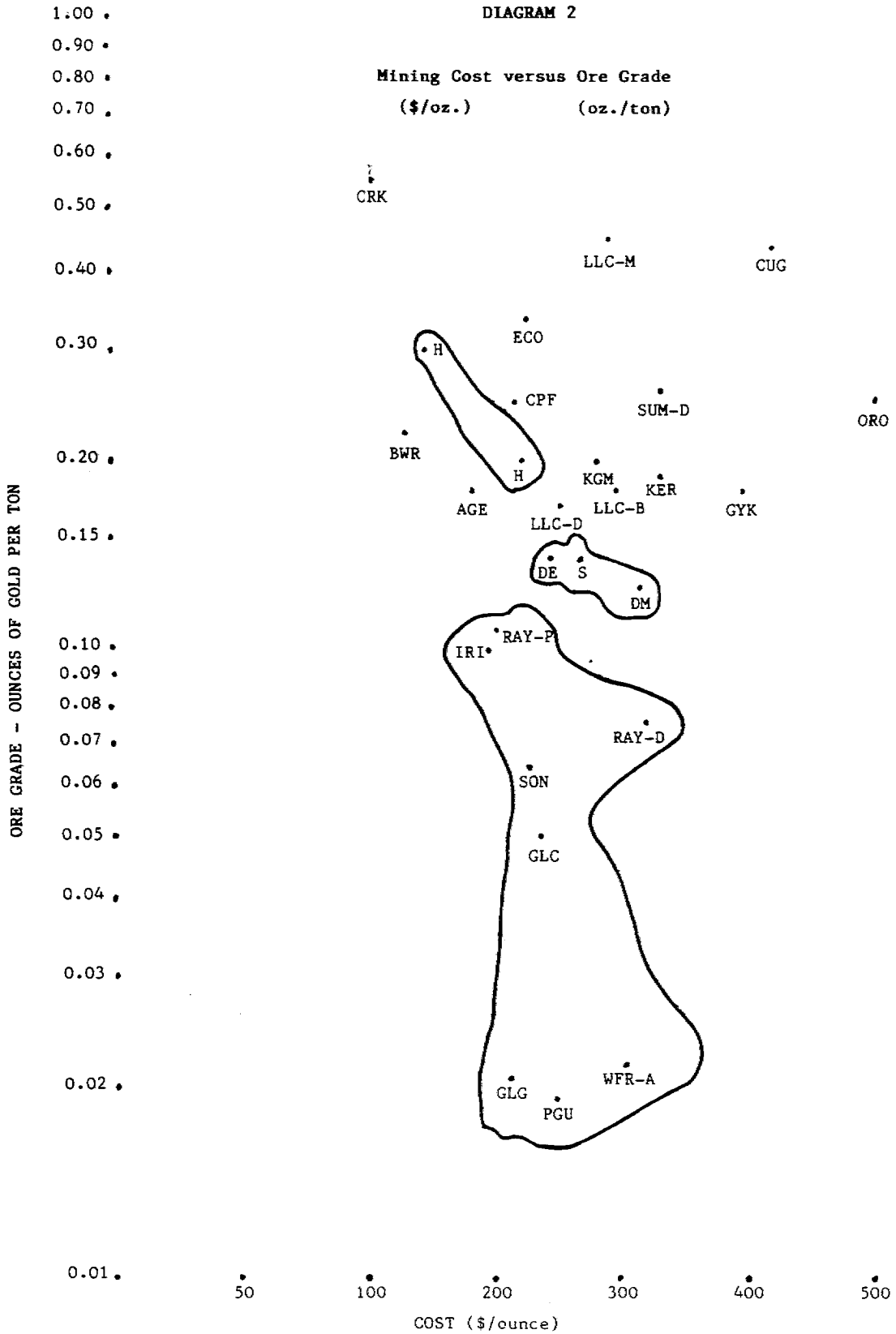
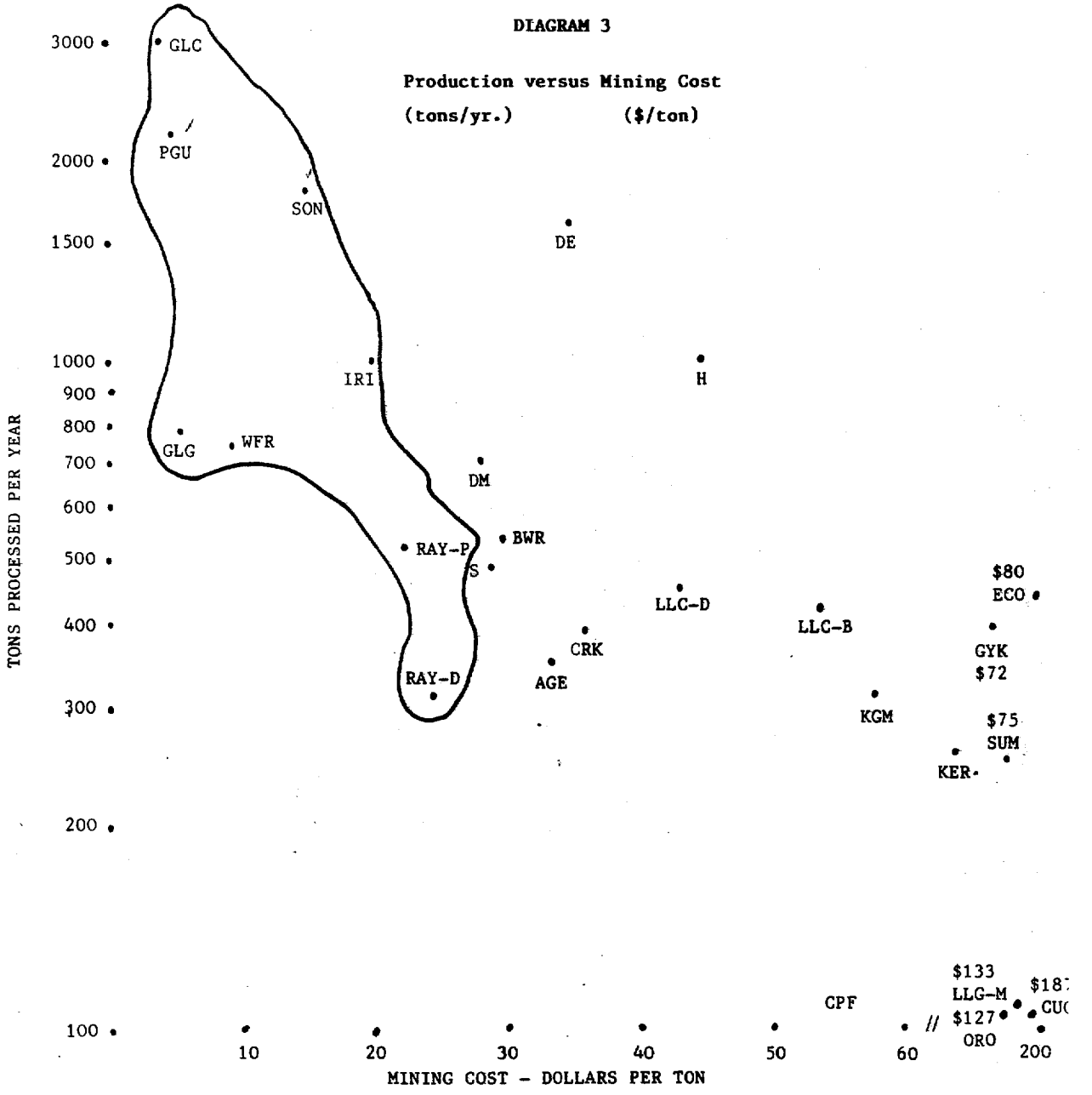


DIAGRAM 3

Production versus Mining Cost

(tons/yr.) ($/ton)



LEGEND

Low Grade Open-pit Producers	Stock Symbol
Pegasus Gold Limited	PGU - T
Glamis Gold Limited	GLG - V
Wharf Resources Limited	WFR-A - T, V (Annie Creek)
Sonora Gold Corporation*	SON - T
Rayrock Resources Limited	RAY-D - T (Dee Deposit)
Inca Resources Limited*	IRI - T, V

* Impending production

Senior Producers	
Dome Mines Limited	DM - T
Campbell Red Lake Limited	CRK - T
Sigma Mines Limited	S - T
Detour Lake	- (part of Campbell Red Lake)
Lac Minerals Limited	LLC-D - T (Doyon)
	LLC-M - T (Macassa)
	LLC-B - T (Bousquet)
Echo Bay Mines Limited	ECO - T

Other Producers	
Sullivan Mines Incorporated	SUM - T
Kerr Addison Mines Limited	KER - T
Giant Yellowknife Mines Limited	GYK - T
Cullaton Lake Gold Mines Limited	CUG - T
Agnico-Eagle Mines Limited	AGE - T
Kiena Gold Mines Limited	KGM - T

Impending Producers	
Hemlo - Participants	H
Orofino Mines Limited	ORO - T
Breakwater Resources Limited	BWR - T
Consolidated Professor Limited	CPF - T
Galactic Resources Limited	GLC - T

Anthony W. Garson, B.Sc., M.B.A.
Mines & Metals Analyst
(416) 868-0303

SOCIETY OF MINING ENGINEERS OF AIME

CALLER NO. D, LITTLETON, COLORADO 80127

PREPRINT NUMBER

84-355

OPTIMUM PRODUCTION RATE FOR HIGH-GRADE/LOW-TONNAGE MINES

Ross Glanville

Wright Engineers Limited
Vancouver, B.C., Canada

For presentation at the SME-AIME Fall Meeting
Denver, Colorado - October 24-26, 1984

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Optimum Production Rate For High-Grade/Low-Tonnage Mines

Introduction

The Optimum Production Rate (OPR) is one of the most important parameters in the evaluation of a mineral deposit. The OPR can also be expressed as the Optimum Mine Life (OML) in years since the expected mine life is determined by dividing the OPR per year into the estimated ore reserves.

Unfortunately, very little time and effort has been directed towards the determination of the OPR. Instead, "rules of thumb" are often applied to select a mine life without due consideration of the economic implications of such a selection. A "justification" for a particular production rate is often based on a pre-conceived arbitrary requirement for a mine life of 5, 10, or 15 years, for example. In this paper it is demonstrated that such arbitrary selections for high-grade/low-tonnage mines (such as many underground gold/silver deposits) often lead to sub-optimal results. In fact, an apparently uneconomic deposit at an arbitrary production rate may be economic at the OPR. Consequently, investment opportunities may be overlooked if one analyzes mine properties based on arbitrary production rates.

This paper analyzes the interrelationships of variables such as production rates, capital and operating costs, cut-off grades, discount rates, metal prices, etc. The analysis shows that the OML for high-grade/low-tonnage deposits is often in the range of two to four years. Although such mine lives intuitively appear too short to many individuals, it should be noted that, in the past, much of the mining industry became familiar with the economics of large scale mining operations. The rules of thumb developed for such large scale operations should not be applied to high-grade/low-tonnage mines such as many of the underground gold deposits.

Definition of Optimum Production Rate

The OPR is selected on the basis of maximizing the net present value of the after-tax cash flows. However, as discussed later in this paper, the production rate selected for a particular mine may be somewhat higher or lower than the theoretical OPR as a result of other variables such as:

1) the probability or expectation of additional reserves being discovered
2) the likelihood of being able to custom-mill nearby deposits owned by others
3) the configuration or attitude of the orebody, which may place upper limits on the production rate
4) the residual, or salvage values, of the mine/mill facilities

Comparison Between Different Production Rates

For an orebody with definite physical cut-offs*, or boundaries (such as faults or unconformities), the total ore reserves at different production rates may be identical. Consequently, for such an orebody, varying production rates result in differing mine lives. Although the metallurgical recoveries, the mine dilution, and other similar physical parameters may vary slightly at different production rates, the major differences are the capital costs and operating costs per ton. The ramifications of these differences are especially important at low production rates (up to approximately 1000 tons per day), but can also be significant at higher production rates. In addition to the cost differences between various production rates is the fact that the cash inflows (return to invested capital) are realized much sooner at higher production rates.

The combinations of the above factors cause significantly different cash flow profiles, which, in turn, result in substantial differences in net present values between different production rates. Before presenting and discussing the graphs of net present values versus production rates, the capital and operating costs at several different production rates for a specific deposit will be analyzed.

Characteristics of the Ore Deposit

For purposes of illustration, we have assumed an underground gold mine in an accessible area of central British Columbia with diluted mining reserves of 500,000 tons grading 0.45 ounces per ton. We also assumed good mining conditions, a 500-ft shaft, long hole open stopes, primary crusher underground, average ore hardness, rod and ball grinding, and 90% gold recovery. The remaining assumptions are provided in the detailed cash flow outputs. However, a summary of the 400 ton per day case is presented in Table 13.

Although this particular example is used for purposes of illustration, we have analyzed many other high-grade/low-tonnage deposits in different parts of the world, and found that the principles presented in this paper also apply to these other properties.

Capital Costs

Based on Wright Engineers' extensive experience and computerized data bases, capital costs were developed for operations with production rates of 100, 200, 400, 600, 800 and 1000 tons per day. With mining reserves of 500,000 tons, these daily tonnages translate into mine lives of approximately 15, 7, 3½, 2½, 2 and 1½ years, respectively. The actual "breakdown" of the various capital cost

*Later in this paper, the aspect of an economic cut-off, as opposed to a physical cut-off, is introduced.

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PREPRINT NUMBER
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THE NORTH AMERICAN GOLD MINES: OPERATING COSTS

Anthony W. Garson

Dean Witter Reynolds (Canada) Inc.
Toronto, Ontario, Canada

For presentation at the SME-AIME Fall Meeting
Denver, Colorado - October 24-26, 1984

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84-312

Good morning, ladies and gentlemen:

It is a pleasure to be here this morning as a representative of Dean Witter Reynolds, one of the foremost international investment firms in North America. My colleague, Hugh Wilmer (who will be speaking to you a little later), and I are involved as research analysts in the research and writing of published reports containing recommendations on the buying or selling of North American and South African gold and silver mining shares.

Our talk this morning will focus on the gold mining operating costs of North American and South African gold producers. However, I am not going to bore you with reams of statistics on the operating costs of large numbers of gold mining operations; how meaningful would this data be to you? Instead, I will discuss what the investor should look for, and how he should use this knowledge in making his investment decision.

I am not going to talk to you as a mining engineer, a cost accountant, or even as a government statistician. I come to you today as a financial analyst, hoping to shed some light on the following question: How can one use operating costs as a guide to buying or selling gold shares? Let's review the topic of costs for about ten minutes. Before we begin, however, may I suggest you keep this in mind: costs are only one criterion, albeit an important one, which will help one make a decision about the current or future earnings prospects of a gold mining company. This said, we may proceed to our discussion of costs.

Costs must be directly comparable between companies. The investor should have a clear definition of what constitutes cost. This is particularly important in a world where G.A.A.P. (Generally Accepted Accounting Principles) have many acceptable definitions of accounting concepts. One should be careful in isolating each company's definition of cost.

When we speak of cost, we refer to DIRECT OPERATING COST. This includes the direct cost of operating machinery, equipment, labour, and development of the working faces. Mining and milling of the ore generally accounts for some 70% of the total operating cost; of this amount, labour constitutes almost 50% of the cost. The remaining 30% comprises power and fuel and administration.

When referring to cost, do not include non-cash items, such as depreciation and amortization. There are a number of accounting principles that can be applied in presenting depreciation, among them, straight-line, units of production, and sum of digits. This variety of approaches makes it difficult to make direct comparisons between gold mining companies when non-cash items are included as direct operating costs.

Secondly, do not include interest charges on long-term debt used to finance the operation. Once again, this is a variable cost which does not allow for comparison between companies.

The main physical factor affecting cost is size of operation. Logically, it would seem that, the more tons hoisted per unit of time, the lower must be the cost per ton of ore extracted. All other things being equal, then, economy of scale lowers the cost. Thus, Dome Mines extracts 700,000 tons per year @ $40/ton, and a grade of 0.13 oz. Giant Yellowknife extracts 400,000 tons per year @ a cost greater than $70/ton, and a grade of 0.18 oz. In the final extreme, the large, open-pit producers, like Pegasus Gold, produce 2.2 m.t.p.y. (millions of tons per year) at a cost of $4.50/ton U.S. Our accompanying diagrams illustrate mining cost per ton versus production for a number of producers and potential producers.

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LEGEND

[redacted]

Internal Correspondence James D. [illegible]

Date: March 20, 1984

Subject: MESQUITE GOLD DEPOSIT, IMPERIAL COUNTY, CALIFORNIA

From/Location:

To/Location:

SUMMARY

Gold Fields Mining has a published reserve of 41.5 million tons of 0.056 opt gold at their Mesquite or Big Chief deposit on the flanks of the Chocolate Mountains in southeast California. Micron-size native gold is present in a largely oxidized, pear-shaped zone from the gravel-covered surface downwards for up to 450'. Mineralization is hosted by intensely and multiply deformed Precambrian gneiss. The major host is a leucocratic quartz-feldspar-biotite gneiss. Although much of the surrounding gneiss is strongly foliated, lineated, and partially mylonitized, it was implied that the ore is associated with a complex, thick zone of fracturing and brecciation within the gneiss package. This saucer-shaped, fractured, mineralized zone may be one of the low-angle faults in the region, although this was not stated. This fracturing event must be superimposed on the earlier, ductile shearing or stretching deformation. High-angle faults, possibly related to the San Andreas system, are also present in the deposit.

Gold is associated with limonite or pyrite. There are no other economically significant metals. There is no silicification and only minor quartz veining. Wall rock alteration is absent, though some brown carbonate veins and chlorite-epidote veinlets are present. Mineralization is strongly controlled by the structural preparation, principally fracturing and brecciation, of the gneisses. Neither the source of the gold, nor the age of mineralization are known.

Mineralized (0.16 to 3.8 ppm Au) surface samples of the footwall gneiss contain traces of arsenic, platinum and silver, but very little else. The usual gold epithermal suite is not anomalous. Ore fluids at Mesquite are interpreted to be of metamorphic derivation and possibly CO_2-rich. I have also suggested several structural, mineralogical, and geochemical exploration guides for use within the regional setting of metamorphic core complexes in the southwest. The combination of anomalous gold with open-space structural deformation seems to be the most essential feature of this type of deposit.

INTRODUCTION

On February 25, 1984, I visited the recently-discovered Mesquite gold deposit of Gold Fields Mining Corporation. The tour, part of an SEG field trip to the Mojave Desert, was the first public trip allowed at the property. This memo is to document and distribute the geologic information presented. C. Grabau, G. Smith, B. Stone and J. Dupree of

Anaconda were also on the trip. Sample descriptions and geochemical results are attached.

Gold Fields' project geologist, Stan Wadovich, lectured and patiently answered questions on the general geology. Geochemistry, geophysics, regional exploration and clear descriptions of the structural geometry were specifically excluded from the discussions. We examined one large outcrop of a footwall gneiss unit, selected core samples, and generalized geologic and grade maps. The total visit lasted only 2 1/2 hours. I have slides of the rocks and maps available for viewing. Due to time limitations, no attempt has been made in this memo to obtain additional information or to integrate the deposit geology into the regional setting and tectonics.

LOCATION AND HISTORY

The Mesquite deposit is located in the Mojave Desert at the southeast end of the Chocolate Mountains in Imperial County, California. It lies immediately north of State Highway 78 and 30 miles northeast of Brawley, California. Minor placer workings, prospect pits, and the Big Chief shaft (50 and 100 levels only) date from the early 1900's. They mined coarse gold in the old Mesquite mining district. The Gold Field deposit is located beneath a major wash. Bedrock outcrops are scarce.

Gold Fields first drilled the property in September, 1981, and in November, 1982, announced a geologic reserve of 41 million tons grading 0.057 opt Au in an open pit, heap leach situation. They drill RDW to the water table and then HQ core below that. They have already completed feasibility studies and a decline for bulk sampling. Permitting should be done shortly. The company has cooperated closely with the BLM, particularly since the deposit is in the California Desert Conservation Area.

HOST ROCKS

The deposit is within a Precambrian gneiss package called the Chuckwalla Formation. North of the orebody by a few miles, gneiss is in fault contact with intermediate (?) Mesozoic metavolcanics. The volcanics dip south but are tightly folded, overturned, and silicified (?). Tertiary quartz-porphyry plugs are present even further north in the range. Most of the deposit is covered by gravels. Post-mineral olivine basalts cover some of the Tertiary fanglomerates.

Gold Fields distinguishes four stratigraphic units in the Chuckwalla. From top to bottom, they are:

1) Muscovite schist : with pyrophyllite(?)

2) Leucocratic feldspar-quartz-biotite gneiss : major ore host. The core sample of this rock contained minor pyrite disseminated on the foliation planes.

3) Hornblende-biotite gneiss : footwall unit and partial ore host. Local feldspar augens and sphene. Abundant quartz-feldspar pegmatites.

4) Mafic gneiss : only known from core (to 1500'). Intruded by leucocratic, anatectic-type gneiss.

Very generally, Units 1 and 2 could be considered hanging wall, and Unit 3, footwall, to the ore and presumably to the major structural zone. The hornblende-biotite gneiss, as seen in outcrop on the southeast corner of the deposit, showed strong foliation and banding. Lineations plunged down dip. Six-inch to 3 foot-wide quartz-feldspar pegmatites are interlayered with more mafic schistose portions.

STRUCTURE

The Gold Fields' geologist repeatedly emphasized the profound importance of the strong structural preparation, principally fracturing and brecciation, in the host gneisses. He suggested that there were multiple structural systems, including late-stage, high-angle veins that may remobilize gold. These approximately parallel and could be related to the San Andreas system. He noted the physical difference between the more brittle intensely fractured, felsic Chuckwalla gneisses, and the tightly folded Mesozoic volcanics.

Though admitting that "numerous low-angle faults may be present in the region," Gold Fields never referred to any specific detachment fault or other of the popular models. The gneiss formations are much flatter within the Big Chief area than outside where dips range up to 70°. On the exposure visited, the foliations dipped 25° to 40°. Low-angle faulting is common and generally concordant to shearing and foliation within the biotite gneiss ore host. Mineralization and presumably the breccia-fracture zone that contains it, is dish-like or synformal in attitude. From their description, the synform axis should trend approximately east-west. The variety of structural styles present was demonstrated by the core samples of barren, dense mylonite, fault breccia with a gouge matrix, and oxidized, open space breccia with limonite and visible gold.

MINERALIZATION AND ALTERATION

"The gold mineralization occurs in structurally deformed metamorphic rocks and forms an irregular deposit that has the general cross-sectional shape of an upturned saucer." In plan view, the deposit outline resembles a flattened pear approximately 3200' long (in a northwest-southeast direction) and 1500' across. The thickness of the mineralized zone on the displayed section ranged from 50' to 400' with interlayered ore and sub-ore intervals. According to the Gold Fields geologist, the ore footwall is not a lithologic contact, but may be a breccia one. The leucrocratic biotite gneiss is the major ore host, possibly because it is very brittle and fractures well. However, mineralization transgresses lithology. It is unclear whether mineralization or ore transgress the major

fragmentation zones, or if they are confined to the fractured zones. The latter case is considered more likely.

Native gold is associated with limonite in the oxidized zone which makes up the bulk of the deposit. The gold is very pure and leaches very nicely (±80 percent recoveries). In the unoxidized part of the deposit, gold is associated with pyrite. The gold particles are typically about 10 microns in diameter, but range from submicron to millimeter size. Mineralization and oxidation extend from near-surface to as much as 450' depth. There is no obvious difference in grade between the sulfide and oxide ore.

In the unoxidized zone, pyrite is about 90 percent of the total sulfide content. Other sulfides are chalcopyrite, sphalerite, galena, and gersdorffite (NiAsS). According to the Gold Fields geologist, there is no arsenopyrite, fluorite, tourmaline, specularite, magnetite, graphite, kyanite, or visible barite in the deposit.

Furthermore, "there is no wallrock alteration." There is no silicification and no major amount of quartz veining. Brown carbonate veins are present, and apparently they are more abundant than the quartz veins. Relative timing of the veining and the relation of the veining to the structural deformation was not explained. However, the geologist did say that the carbonate veins are locally associated with higher grade zones. Minor chlorite is present on fractures and replacing biotite in the gneiss. Minor sericite is also present. Alteration at Newmont's Cargo Muchacha property is reportedly even more subtle.

The outcrop and core samples generally confirmed the lack of alteration. The gneiss in outcrop (tens to hundreds of feet from ore) was pristine, except for a few pieces with epidote-chlorite fracture coatings. A few prospect pits and trenches did have dumps with iron-stained pegmatitic, milky-white quartz and sheared, sericitic pegmatite with sparse quartz veinlets. The one core sample of ore was a vuggy breccia with visible gold. It had subrounded clasts of bleached and weakly-argillized gneiss with limonite and gold partially filling the open space between clasts. The bleaching may well have been due to supergene alteration. Gold Fields did not say how representative this breccia sample was of the mineralized zone.

GEOCHEMISTRY

Analytical results and sample descriptions for the 6 samples taken are attached. All of the samples were from outcrop or shallow pits in the footwall gneiss unit peripheral to the southeast edge of the deposit. The geochemical data is most noticeable for the nearly total absence of trace element anomalies. The unaltered gneiss contains detectable gold (17-30 ppb) and palladium (up to 269 ppb) but these could be regional background values. The two iron-stained dump samples have anomalous gold, including 0.11 opt Au in 299866, along with trace silver, platinum and arsenic, but very little else. Manganese appears to be depleted, but this may be a

result of oxidation. Neither base metals nor the ultramafic suite (Ni, Co, U) are enriched. The very low levels of the epithermal gold suite (Hg, As, Sb, F, Ba) combined with the lack of silicification or clay alteration is particularly significant. It suggests that the gold-depositing fluids at Mesquite were unrelated to an epithermal or hot spring source. The geochemical and alteration anomalies, as well as gold mineralization, appear to me to be limited to the zone of fracturing and brecciation. The carbonate veinlets, chloritic alteration and mineralogical and geochemical signatures suggest a greenschist facies metamorphic environment, where CO_2-rich (?), ore-bearing fluids were nearly in equilibrium with their host rocks.

EXPLORATION GUIDELINES

The following features are inferred by me to be useful exploration guides within the gneiss, metamorphic core complex, and detachment fault setting.

1) Gold above 100 ppb, including placers and lode districts.

2) Brecciation and fracturing with open space.

3) Iron-staining or pyrite.

4) Associated low-level silver, platinum, palladium, and/or arsenic anomalies.

5) Quartz veins.

6) Brown carbonate veins.

7) Chlorite-epidote veins.

8) Synformal portions of the detachment (?) fault zones.

9) Multiple deformational events.

10) Brittle horizons within the gneiss package or other formations.

11) Late Tertiary, high-angle faults?

12) "Metamorphic" zoning which may suggest local gradients in CO_2/H_2O fluid compositions. For example:

$$\underset{}{CO_2} + \underset{\text{Sphene}}{CaTiSiO_5} = \underset{\text{Calcite}}{CaCO_3} + \underset{\text{Rutile}}{TiO_2} + \underset{\text{Quartz}}{SiO_2}$$

Subject: Mesquite Gold Deposit
(Continued - Page 6)

The assemblage on the right-hand side may be associated with mineralization at Mesquite.

MESQUITE SAMPLE DESCRIPTIONS

Geochem No.	Field No.	Description
299861	M-1	Drill cuttings: SM-610
299862	M-2	Quartz-feldspar-biotite gneiss; fine-grained with quartz and feldspar augen or segregations. Granulated, sheared look. Mafics 10-15%. Mapped as biotite-hornblende gneiss.
299863	-	STANDARD Rew-M (0.055 ppm)
299864	M-3	Hornblende-biotite gneiss; medium-grained poorly-banded gneiss with 10% hornblende and 20% biotite. Trace chlorite after biotite and oxidation of mafics.
299865	M-4	Biotite gneiss; medium-grained, schistose with 35% biotite, quartz-feldspar augens. Epidote and chlorite coat sparse fractures.
299866	M-5	Iron-stained pegmatite, from dump of trench. Coarse-grained quartz and white K-feldspar with 5-10% fine-grained, grey green sericite. About 1% disseminated, fine-grained limonite, or unknown mineral described below. Sparse goethite-coated quartz veinlets without alteration envelopes. Some pieces show hints of breccia texture with more limonite in crackled open space.
299867	M-6	Miscellaneous grab of pieces from dump and float. Predominantly moderately iron-stained, quartz-rich, medium- to coarse-grained, sheared and fractured pegmatite with clots of biotite and sericite. Feldspars fresh. Some oxidation of mafics. 1-2% disseminated unknown metallic mineral: silver-grey color, slight glassy luster, non-magnetic, red-brown scratch, moderate hardness. Possibly ilmenite, rutile, gersdorffite, or goethite? Sample includes one piece of fault breccia (float) with brecciated vein quartz cemented by white bull quartz.

GEOCHEMICAL ANALYSIS SAMPLE SUBMITTAL FORM

DATE SAMPLES RECEIVED 9/7/84

Lab Report No 76184-04500-1

Assay Values (ppm Unless Noted)

Sample Field No	Sample Type	Sample I.D. No	Au	Ag	Hg	S(%)	F(%)	Sb	B	As	Ba	Co	Cr	Cu	Au
299	861	00001	.027	<0.5	0.03	0.01	0.05	21	<10	39	845	<10	19	<10	
	862	00002	.024	<0.5	0.02	0.01	0.02	<10	<10	18	1290	<10	17	<10	
STD	863	00003	.091	2.7	0.98	0.07	0.13	101	<10	1528	145	34	42	92	STD
	864	00004	.029	<0.5	0.02	0.01	0.08	10	<10	11	690	<10	15	19	
	865	00005	.017	<0.5	0.02	0.01	0.12	26	<10	18	1649	<10	12	<10	
	866	00006	3.85	<10	0.03	0.01	<0.05	<10	<10	45	700	<10	14	<10	3.60
299	867	00007	.158	<0.5	0.02	0.01	<0.05	<10	<10	20	590	<10	17	<10	

			Pt	Pd	V	Fe(%)	Mn	Mo	Ni	Pb	Zn
299	861	1	<.050	.269	58	1.9	337	<10	<10	<10	51
	862	2	<.050	.201	14	0.8	189	<10	<10	<10	13
STD	863	3	<.050	.046	211	18.7	1956	<10	41	33	313
	864	4	<.050	.098	73	2.3	444	<10	<10	<10	29
	865	5	<.050	.048	74	2.4	440	<10	<10	<10	32
	866	6	.091	.046	25	0.9	90	<10	<10	<10	13
	867	7	<.050	.029	17	0.6	77	<10	<10	<10	<10

; line for Tl, Te

Mercuro

GEOCHEMICAL ANALYSIS SAMPLE SUBMITTAL FORM

DATE SAMPLES RECEIVED 8/2

Lab Report No 76184-04500-1 Assay Values (ppm Unless Noted)

Sample Field No	Sample Type	Sample Lab No	Au	Ag	Hg	S(%)	F(%)	Sb	B	As	Ba	Co	Cr	Cu	Au(%)
299	861	00001	.027	<0.5	0.03	0.01	0.05	21	<10	39	845	<10	13	<10	
	862	00002	.024	<0.5	0.02	0.01	0.02	<10	<10	18	1290	<10	17	<10	
STD	863	00003	.091	2.7	0.98	0.07	0.13	101	<10	1528	145	34	42	92	STD
	864	00004	.029	<0.5	0.02	0.01	0.08	18	<10	11	690	<10	15	19	
	865	00005	.017	<0.5	0.02	0.01	0.12	26	<10	18	1649	<10	12	<10	
	866	00006	3.85	<10	0.03	0.01	<0.05	<10	<10	45	700	<10	14	<10	3.60
299	867	00007	.158	<0.5	0.02	0.01	<0.05	<10	<10	20	590	<10	17	<10	

			Pt	Pd	V	Fe(%)	Mn	Mo	Ni	Pb	Zn
299	861	1	<.050	.269	58	1.9	337	<10	<10	<10	51
	862	2	<.050	.201	14	0.8	189	<10	<10	<10	13
STD	863	3	<.050	.046	211	18.7	1986	<10	41	33	313
	864	4	<.050	.098	73	2.3	444	<10	<10	<10	29
	865	5	<.050	.048	74	2.4	440	<10	<10	<10	32
	866	6	.081	.046	25	0.9	90	<10	<10	<10	13
	867	7	<.050	.029	17	0.6	77	<10	<10	<10	<10

; like for Tl, Te

Mesquite

SKYLINE LABS, INC.
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REPORT OF ANALYSIS

JOB NO. DBZ 908
April 9, 1984
PROJECT CODE: X545-045000079

Additional Mesquite Analyses

Analysis of 7 Pulps

	ITEM	SAMPLE NO.	Te (ppm)	Tl (ppm)	Ti (ppm)	
	1	299861	.085	<1.	2700.	
	2	299862	.016	<1.	920.	
STD	3	299863	.760	3.	1050.	
	4	299864	.050	<1.	3350.	
	5	299865	.007	<1.	3350.	
	6	299866	.035	<1.	600.	sphene destruction?
	7	299867	<.005	<1.	220.	

Gordon H. VanSickle
Manager

cc:

Confidential

Benjamin C. Koskiniemi
1020 W. Comobabi
Tucson, Arizona 85704

May 8, 1984

Mr. Charles P. Miller
AMAX Exploration, Inc.
10 East Broadway, Suite 400
Tucson, Arizona 85701

Dear Charlie:

Enclosed is my report describing the results of the preliminary ore reserve estimates for the Margarita property.

This report includes preliminary estimates for the geologic and mineable reserves. The reserve estimates should be of sufficient accuracy for preliminary "order-of-magnitude" feasibility analysis. If preliminary feasibility is favorably established, it is recommended that refinements be made in the reserve estimates and pit designs.

I have enjoyed working with you on this project and if you have any questions, please let me know.

Sincerely,

Ben

B. C. Koskiniemi, P.E.

BCK:sar
Enclosures

ORE RESERVE ESTIMATES

MARGARITA PROPERTY
SANTA CRUZ COUNTY, ARIZONA

Prepared for
AMAX EXPLORATION, INC.

BY

B. C. KOSKINIEMI, P.E.

May 7, 1984

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INTRODUCTION

The scope of this study was to expeditiously prepare preliminary estimates of the geological and mineable ore reserves for the Margarita property.

This was to be a limited study to verify previous reserve estimates and provide adequate reserve information for preliminary feasibility studies.

SUMMARY

The reserves developed in this study are considered to be of a very preliminary nature and refinements must be made if preliminary feasibility is established favorably. The results are summarized in Table 1.

Geologic reserves are estimated at 516,000 tons at 0.054 ounces Au per ton using a 0.020 ounce grade cutoff.

Mineable ore reserves were estimated from open pit designs taking into account economic stripping limits. Two pits were designed on the basis of different metallurgical gold recoveries, (1) 60% recovery, and (2) 100% recovery. Mineable reserves within the pit outlines at a 0.020 ounce cutoff are as follows:

Pit Design	Ore		Stripping Ratio
	Tons	Oz Au/Ton	W:O
60%	361,000	0.064	1.21:1.00
100%	442,000	0.058	1.35:1.00

The entire study was done in AMAX's Tucson Office during the period of April 26 1984 to May 4, 1984, where all cross-sections, maps, pit designs, reports, and back-up data used and/or developed in this study are on file.

TABLE 1

MARGARITA PROPERTY
SUMMARY OF ESTIMATED ORE RESERVES

	Sec.	Geological Tons	Geological Oz. Au/Ton	Mineable (Cutoff w/60% Recovery) Tons	Mineable (Cutoff w/60% Recovery) Oz. Au/Ton	Mineable (Cutoff w/100% Recovery) Tons	Mineable (Cutoff w/100% Recovery) Oz. Au/Ton
Dew Cascabel	2	7,173	.084	4,782	.102	7,173	.084
	3	16,825	.064	16,825	.064	16,825	.064
Sectotal		23,998	.070	21,607	.072	23,998	.070
North Hill	6	3,754	.032	-	-	-	-
	7	12,404	.032	3,998	.041	10,486	.033
	8	12,749	.072	4,817	.149	7,020	.110
	9	3,072	.089	1,536	.152	3,072	.089
	10	24,898	.053	22,630	.056	22,630	.056
	11	4,586	.060	3,182	.074	3,182	.074
	12	13,888	.051	3,456	.118	9,280	.066
	13	13,490	.050	10,902	.056	10,902	.056
	14	26,037	.043	21,216	.046	26,037	.043
	15	18,480	.060	18,480	.060	18,480	.060
	16	17,875	.055	8,781	.084	8,781	.084
	17	4,266	.060	1,466	.141	3,868	.073
	18	8,015	.058	6,054	.069	8,015	.058
	19	10,160	.172	10,160	.172	10,160	.172
	20	1,696	.032	-	-	1,696	.032
Sectotal		175,370	.059	116,778	.076	143,609	.067
north Margarita	23	10,150	.042	2,342	.099	5,660	.058
	24	41,459	.033	12,135	.045	26,797	.038
Sectotal		51,609	.035	14,477	.054	32,457	.041
Margarita	27	17,617	.038	9,240	.045	9,240	.045
	28	10,521	.066	10,521	.066	10,521	.066
	29	36,600	.042	36,600	.042	36,600	.042
	30	29,744	.046	19,844	.052	27,581	.047
	31	51,304	.052	33,496	.061	48,887	.053
	32	41,472	.051	35,208	.055	41,472	.051
	33	37,296	.063	30,528	.071	33,408	.066
	34	36,036	.057	30,852	.063	32,472	.061
	35	1,504	.025	-	-	-	-
	36	1,440	.085	1,440	.085	1,440	.085
	37	1,408	.038	-	-	-	-
Sectotal		264,942	.052	207,729	.057	241,621	.054
TOTAL		515,919	.054	360,591	.064	441,685	.058
Ratio (Stripping)				1.21		1.35	

GEOLOGICAL RESERVES

The geological reserves were estimated from east-west ross-sections. The sections were AMAX generated computer rintouts at a scale of 1" = 50'. Information on the sections ncluded surface topography and drill holes. Generally, assays ere shown for five foot assay intervals where the grade xceeded 0.010 ounces of gold per ton. Cross-sections were enerally about 50 feet apart which follows the drill spacing f ± 50 feet between holes.

Cross-section locations are shown on area topography maps t a scale of 1" = 100'. A total of 37 cross-sections were enerated. Sections 1 to 26 are located on Topography Map-1, nd sections 27 to 37 are on Map-2, both of which are on file n the AMAX office. Cross-section coordinates and distance of nfluence are shown in Appendix I.

The sections did not include any geology and no geological nterpretations were added to the sections. AMAX geologists re in the process of analyzing and/or developing geological ata and maps.

The ore block model was developed on the cross-sections sing a block height of ten feet with an average assay of 0.020 z. Au, or greater. Generally, the horizontal block dimensions xtended midway between drill holes on sections and midway to he adjoining sections. Blocks were not located by levels but nly by assay intervals. However, a ten foot bench height was lso assumed for pit design in determining mineable reserves. As per Brost, September 1981).

Tonnages were based on a factor of 12.5 cubic feet per ton Brost and others).

For end holes it was assumed that the mineralization xtends 20 feet beyond the hole. Also, in areas where a ection does not have any drilling, the mineralization was ssumed to be continuous if both adjoining sections contained re blocks in the area concerned.

Total geological ore reserves are estimated at 516,000 ons, averaging 0.054 ounces of Au per ton at a 0.020 ounce utoff (See Table I).

Appendix II includes a listing of ore blocks by ross-section and drill hole. Also included are block imensions, tonnages and grades.

PIT DESIGN - GEOMETRY

The pit designs were prepared as overlays to the 1" = 100' cale topography maps. The following criteria were used to rrive at the final pit designs:

1) Bench height - Bench height is ten feet. The bench elevation is the elevation, in feet above sea level, of the bench floor, i.e., the 4,265 bench has a floor elevation of 4,265. All bench outlines are plotted at the bench median elevation; therefore the 4,265 bench is plotted to coincide with the 4,270 contours.

2) Bench elevations - Area topo maps have a contour interval of five feet with contour elevations ending in 0 or 5. In analyzing the locations of the pit bottoms on the sections, it was determined that a majority of the bottom bench ore blocks were nearer the 5 elevation, rather than the 0 elevation and for this reason, the mine design was done using the five elevation, i.e., 4,265.

3) Slope angle - The pit slopes were designed at 60°. This angle was selected because of the shallow depth from pit perimeter to pit bottom (0 to 80 feet). In most large open pits where bench heights are +50 feet the bench face angles range between 55 to 65°. These angles can generally be increased substantially (almost vertical in some cases) with controlled drilling and blasting.

4) Haul roads - Haul road locations are not shown in the pit designs but because of the shallow pits and favorable topography haul roads can be incorporated into the pit designs with minimal additional stripping and/or loss of ore.

The pit design overlays are on file in the AMAX office.

PIT DESIGN - ECONOMICS

The scope of this study did not include detailed estimating f capital and operating costs, however cost estimates are equired to prepare even a preliminary economic pit design. A ursory review of cost estimates by Brost (September, 1981), nd Bahamian Refining Corporation (BRC)(March, 1984) provided ost data judged to be adequate for this study.

Mr. C. P. Miller indicates that the Margarita property ould be developed and operated on a "no frills" basis. Each f the above mentioned estimates is based on this type of an peration.

I believe Brost to be optimistic in his capital and perating cost estimates, and he also expresses some eservations about his estimates. However, the BRC estimates f March, 1984, although incomplete, indicates Brost may be onservative in some areas of capital costs and less costly lternatives may be available. Purchase prices for crushing nd processing equipment are estimated at $357,100 by Brost 1981) and at $284,650 by BRC (1984). See comparison details n Appendix III.

Alternative operating practices might also result in educed operating costs as quoted in BRC reports of March 2 and , 1984. On the basis of this cost analysis for a "no frills" peration, it was decided to use Brosts' 1981 costs, without scalation, for the economic pit designs.

The following costs were used in the economic pit design:

1) Capital - Brost estimated total capital costs at $1,143,000. This was increased by 20% to $1,372,000 to take into account the cost of capitalization, as per discussions with C. P. Miller.

2) Operating Costs - Brost estimated operating costs of $16,000 per week at a production rate of 2,500 tons per week (mine 500 TPD, five days per week, and leach-precip at seven days per week).

3) Other capital expenses - Other expenses are estimated at $500,000 to take into account capitalization of investment by present limited partners and very minimal expenses for further drilling, metallurgical testing, and feasibility studies. This estimate provided by C. P. Miller.

Brosts' operating cost estimate did not have any drill, last, load or haul costs for waste mining. The only stripping osts were for a dozer stripping at eight hours per week versus ormal ore mining at 12 hours per week. For purposes of this nalysis it was assumed that Brost's $16,000 per week operating ost did not include any waste removal.

For economic design purposes the mineable ore reserves were stimated at 500,000 tons.

The total cost per ton of ore, without stripping, was stimated at $10.14 per ton as summarized in Table 2.

Two pits were designed on the basis of different etallurgical gold recoveries, (1) 60% recovery and, (2) 100% ecovery. A gold price of $400 per ounce is used in this nalysis. On this basis the design cutoffs, without stripping, re estimated at 0.042 and 0.025 oz Au per ton of ore for the 60% and 100% recoveries, respectively (See Table 2).

In order to determine grade cutoffs at various stripping atios a stripping cost was developed on the basis of Brost's ine operating cost. The stripping cost is estimated at $1.10 er ton of waste mined. See cost summary in Table 3 which is eveloped on the basis of a 1:1 (W:O) strip ratio.

Using the $1.10 cost for waste mining a stripping ratio ersus cutoff grade chart was developed (See Table 4).

TABLE 2

Ore Cutoff Analysis
(w/o Stripping)

	$/Ton Ore
Operating Cost $16,000 - 2,500 tons	$ 6.40
Capital $1,372,000 - 500,000 tons	2.74
Other Expenses $500,000 - 500,000 tons	1.00
TOTAL COST	$10.14

Grade Cutoffs

@ 60% Recovery
$10.14/ton ÷ ($400/oz Au x 0.6) = 0.042 oz Au/ton

@ 100% Recovery
$10.14/ton ÷ ($400/oz Au x 1.0) = 0.025 oz Au/ton

TABLE 3

Waste Mining Cost Summary
(@ 1:1 Strip Ratio)

			Cost/Week
manpower - Operations			
Drill	24 Hours		
Blast	8 Hours		
Load	12 Hours		
Haul	24 Hours		
Subtotal	68 Hours		
Strip -	8 Hours		
TOTAL	60 Hours @ $10/Hour =		$ 600
manpower - Maintenance 20 Hours @ $10/Hour =			200
Fuel - 180 gal. @ $1.16			209
Lube			68
Tires			121
Bucket			120
Repair Reserve			172
Bits			225
Explosives			1,036
Sundries			33
TOTAL			$2,784
Tons @ 1:1 Strip Ratio			2,500 Tons
Cost/Ton Waste Mined			$1.11/Ton
		Use	$1.10/Ton

TABLE 4

Strip Ratio versus Cutoff Grade

Strip Ratio Waste:Ore	Cost/Ton of Ore	Cutoff Grades 60% Recovery	Cutoff Grades 100% Recovery
0:1	$10.14	0.042	0.025
1:1	11.24	.047	.028
2:1	12.34	.051	.031
3:1	13.44	.056	.034
4:1	14.54	.061	.036
5:1	15.64	.065	.039
6:1	16.74	.070	.042
7:1	17.84	.074	.045
8:1	18.94	.079	.047
9:1	20.04	.084	.050
10:1	21.14	.088	.053

MINEABLE RESERVES

Mineable reserves are defined as the total ore reserve thin the pit limits.

The mineable reserves were developed by:

1) Developing the mineral block model on the cross-sections (geologic reserve),

2) Locating approximate economic pit limits on the cross-sections,

3) Transfer the pit limits from cross-sections to plan,

4) Pit design in plan (topography overlay),

5) Transfer designed pit limits from plan to cross-sections,

6) Determine mineable ore reserves from cross-sections,

7) Measure areas of levels from pit plan to determine tons of total material within pit limits.

8) Subtract total mineable ore from total material within pit limits and calculate strip ratio.

Mineable ore reserves were estimated for two pits on the asis of different metallurgical gold recoveries, (1) 60% ecovery, and (2) 100% recovery. Mineable reserves within the it outlines at a 0.020 ounce cutoff are as follows:

Pit Design	Ore		Stripping Ratio
	Tons	Oz Au/Ton	W:O
60%	361,000	0.064	1.21:1.00
100%	442,000	0.058	1.35:1.00

The mineable reserves are summarized in Table 1. Appendix I includes a summary of ore blocks by cross-section and drill ole. Also included, are block dimensions, tonnages and grades.

It should be emphasized again, that the ore tonnages are erived from the ore blocks as shown on the cross-sections. he ore block elevations are controlled by the grade assay tervals and not by the level designations shown on the pit lans. Therefore, the ore tonnages have not been broken down y level.

The total tonnages mined were estimated from the pit plans nd these tonnages are summarized by level and areas in ppendix IV for both pit designs.

No attempt was made to determine to what extent the ore onnage and grades would be affected by dilution. It is ecommended that geostatistical techniques (variograms, etc) be sed to analyze drilling densities and assay reliabilities.

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?Dec '84

WLK - JDS - GJS - FRK - AG
Please read this draft and criticize

-- DRAFT -- -- DRAFT --

File Memorandum

Supergene Gold--A Possible Exploration Lead

A presentation entitled "Metallogeny and Tectonic Evolution of the Precambrian of Brazil" was given by Nigel Grant, Billiton Exploration, at a recent AGS meeting in Tucson.

The talk included a brief description of (1) the fabulous Carajas iron occurrence containing 60 billion tons of 60% Fe, located 550 km SSW of Belem at the mouth of the Amazon, (2) the nearby Salabro copper deposit with over one billion tons of 1.0% Cu (in metasediments enclosed in gneiss), and (3) the Cerro Pelado gold mine, an occurrence of supergene gold as nuggets in a 100m thick zone of saprolitic weathering. The deposit, located , was worked for many years as a placer by several hundred individual small claim miners. The grade is, of course, unknown; however, "waste" dumps of some 7 million tons average around 7 gms per ton. The deposit is underlain by a quartz vein stockwork containing relatively fine gold ranging from 1 to 2 gms per ton.

According to Grant, the nuggets (ranging up to several kilos in weight) were formed in the zone of weathering by the "nucleation" process --progressive accumulation on particles of iron oxide, the gold having been originally dissolved from the "protore" by cyanide released from decaying vegetation.

This "growing of nuggets" has been reported elsewhere, such as at Stirrup Lake, B.C.,* where placers contain gold crystal growths believed

*Supergene gold crystals at Stirrup Lake, B.C., Western Miner, June 1979. (From George Stathis file.)

to have formed from solution and redeposition of micron gold in bedrock; and, at the Osceola placers, Nevada,** where the source of plus one ounce nuggets has not been identified, suggesting they may have "grown" from small particles of gold found in bedrock. The old timers mined placer gold at Carlin, but failed to recognize the values in bedrock simply because they were too fine to be visible in the pan (intercepts up to 5.0 oz/ton were cut in the diamond drilling). Since no bedrock source for the placers at Carlin has (according to my information) been identified, it is possible, if not probable that it (the coarse gold) was formed by supergene gold bearing solutions derived from the micron gold in bedrock.

It is concluded that some placer occurrences (excluding, say, river gravels with very large drainage basins) may offer clues to "no-see-um" gold deposits in bedrock--either exposed or concealed by post-mineral cover. Accordingly an aggressive search should be undertaken, which would logically start on file and literature reviews with the objective of first selecting those placers located in areas where the occurrence of disseminated gold, particularly micron gold deposits, is geologically permissable.

J. H. Courtright

**J. D. Sell file memo, 10/5/84.

JHC/cg

cc: WLK
JDS
GJS
FRK
AEG

J108

UNITED STATES DEPARTMENT OF INTERIOR GEOLOGICAL SURVEY

Controls on Base and Precious Metal Mineralization in Deeper Epithermal Environments.[1]

by

Philip M. Bethke

U. S. Geological Survey, Reston, VA 22092

[1] Transcript of a talk given before the Society of Economic Geologists Symposium on Geothermal Systems and Ore Deposits in honor of Donald E. White: AIME 113th Annual Meeting, February 26-March 1, 1984, Los Angeles, California.

Open-File Report 84-890

This report is preliminary and has not been reviewed for conformity with U. S. Geological Survey editorial standards and stratigraphic nomenclature.

ck OF-83-710 : Heald-Wetherfer, P., et al.

LIST OF SLIDES

1. Model for epithermal vein systems. After Buchanan, 1981
2. Listing; Goldfield-type vs. Creede-type epithermal deposits.
3. Distinctive characteristics; Goldfield-type & Creede-type deposits from Heald-Wetlaufer and others, 1983.
4. Generalized geologic map of the Eastern San Juan Caldera Complex.
5. SW-NE cross section, Summitville mining district, Colorado. After Steven and Ratte, 1960.
6. Schematic cross section, alteration zoning, Summitville. After Steven and Ratte, 1960.
7. Repeat of Slide 3
8. Photograph of R. W. Henley (slide omitted from this report)
9. Comparative schematic cross sections of Goldfield-type and Creede-type geothermal systems. After Henley and Ellis, 1982.
10. Sulfuric acid-producing reactions important in acid-sulfate alteration.
11. Repeat of Slide 9.
12. Repeat of Slide 10.
13. Sulfur isotopic ratios of Goldfield sulfides and sulfates. After Jensen and others, 1971.
14. Oxygen and hydrogen isotopic composition of hydrothermal and supergene kaolinites from Japan. After Marumo and others, 1982.
15. Isotopic characteristics of acid-sulfate alteration of different origins.
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17. Enlargement of upper portion of Slide 16.
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21. Fluid inclusion homogenization temperature-salinity data, locality NJP-X, OH vein, Creede, Colorado.
22. Longitudinal section, OH vein, Creede, Colorado showing distribution of illitic alteration cap, vein adularia and localities showing fluid inclusion evidence for boiling.
23. Primary and pseudosecondary fluid inclusions in quartz crystal, OH vein, Creede, Colorado. Courtesy, N. Foley.
24. Fluid inclusion homogenization temperature-salinity data, OH vein quartz and NJP-X sphalerite. Courtesy, N. Foley.
25. Repeat of Slide 23.
26. Repeat of Slide 24.
27. Typical photograph, R. W. Henley. (slide omitted from this report)

ABSTRACT[1]

Two distinct chemical environments are recognized for epithermal mineralization in volcanic settings. In the first, typified by Goldfield, Nevada, mineralization is closely tied to magmatism; the ores are deposited in a chemical environment dominated by disproportionation of magmatic SO_2 and interaction of the resulting acid solutions with wallrock in steep thermal gradients near shallow plutons. In the second, typified by Creede, Colorado, mineralization is controlled by deeply circulating cells analogous to many modern geothermal systems; ores are deposited beneath an interface between deep, near neutral brines and overlying surface waters as a result of cooling and loss of acid volatiles.

TRANSCRIPT OF TALK

First of all let me voice my pleasure at having been asked to participate in this symposium honoring Don White; the man who has led all of us - geothermalists and economic geologists alike - to the ground which this symposium attempts to cover - the relationships between active geothermal systems and hydrothermal ore deposits. Don, your focus, thru the years, on the nature and origins of natural thermal waters has not been universally accepted by your colleagues - or supervisors - at the Survey as being pertinent to the study of ore deposits, particularly in the early days of your studies; but look where it has led us! No other single approach has contributed so much to our understanding of hydrothermal ore deposits, particularly the epithermal ores, as has the study of active geothermal systems, as this symposium so well demonstrates. You must be very proud, and deservedly so; but Don, I think perhaps the shoe is on the wrong foot today; it is we, the speakers who are honored, by being asked to participate in YOUR symposium!

Let me begin by defining what I am, and am not, going to attempt to cover in this talk. I will restrict my remarks to volcanic-hosted epithermal ores. There are two reasons for this: first of all, Bill Bagby and Art Radke are going to discuss the sediment-hosted deposits right after lunch, and secondly, I've never worked in any of those deposits, and you deserve better than to hear me talk about them.

In the first part of this talk I am going to draw some distinctions between what I consider to be two very different - although related - epithermal environments. I'll then discuss the first of those environments, emphasizing some of the conflicting viewpoints concerning the nature of the ore-forming processes operative within it, and presenting an approach which may allow us to choose between some of the alternative mechanisms. Finally, I will turn our attention - at least mine - to the second environment and discuss it in terms of the chemical and hydrologic structure and processes known to occur in some active geothermal systems, with particular emphasis on how we might recognize some of the characteristics of active systems in the ore deposits which we are studying. I will, of course, use Creede as an example of this second type.

SLIDE 1

Several years ago Larry Buchanan (1981) published a comprehensive, and

[1] Abstract published in Program and Abstracts, AIME 113th Annual Meeting, February 26 - March 1, 1984, Los Angeles, California

very important synthesis wherein he related the distribution of mineral assemblages, zoning of metals, mineral textures and structure of volcanic-hosted epithermal ore deposits to one another as summarized on this slide on which I've added a little color to Larry's original illustration. Larry interpreted the model in terms of processes going on in the ore fluid, with particular emphasis on boiling. He also had a few words of advice for those of you who actually work for a living on how you might use the model find orebodies. I think this synthesis of Larry's was a singular achievement, and helped, at least in my mind, bring a considerable amount of order to a variety of observations that have been made on epithermal orebodies. Some aspects of Larry's model can stand refinement, however, and I'm particularly interested in whether this kaolinite + alunite acid-sulfate alteration zone, shown here in red, is universally present - or even occasionally present in most epithermal vein systems. In fact, I am going to contend that the epithermal ores containing intense primary alunite + kaolinite acid-sulfate alteration form in a very different environment from the more common vein-type deposits characterized by illitic to sericitic alteration. Three of our colleaques in Reston, Penny Heald-Wetlaufer, Dan Hayba and Nora Foley, have undertaken, with the help of Jim Goss, a comprehensive analysis of the characteristics of 16 volcanic-hosted epithermal ore deposits, selected primarily on the basis of the amount of data available in the literature on which their comparative study was based. A number of you may have heard Penny present the results of their analysis at the GAC-MAC meetings in Vancouver last May. A copy of Penny's remarks and slides has been issued as a USGS open-file report (Heald-Wetlaufer and others, 1983). In general their study corroborates and adds documentation to Larry Buchanan's model, but they have drawn what I consider to be an important distinction between those ore deposits which are characterized by intense acid-sulfate alteration to kaolinite-alunite-quartz assemblages and those which are characterized by alteration to illite or sericite and usually contain adularia as a vein constituent.

SLIDE 2

The deposits on which they compiled information are grouped as shown on this slide, which contrasts those deposits characterized by acid-sulfate alteration (which I have chosen to call the Goldfield-type) with those characterized by illite-sericite alteration (which scientific chauvinism compels me to call the Creede-type). Although I am emphasizing the differences in wallrock alteration, there are other systematic differences between the two types of deposits which help validate the distinction between them.

SLIDE 3

The Goldfield-type ores contain a sulfide assemblage signifying a high chemical potential of sulfur and including enargite, famatinite, luzonite, and, usually, covellite in contrast to the chalcopyrite and tetrahedrite-tennantite characteristic of the Creede-type ores. In all of the Goldfield-type districts studied the K/Ar age of the ore overlaps, within analytical precision, that of the rhyodacitic volcanic dome or neck to which it is related, and which usually hosts the ore. In contrast, the ores of the Creede-type are usually substantially younger than their volcanic host rocks, and can seldom be tied to any particular intrusive or volcanic unit. The Goldfield-type ores also tend to be more Copper-rich than do those of

the Creede-type. These relations are remarkably consistent within each of the deposit classes although within each class there are marked differences in other characteristics, particularly in base:precious metal ratios and Ag:Au ratios.

Although I have chosen to call the acid-sulfate-bearing deposits the Goldfield-type, I would like to example them by briefly discussing the Summitville district in Colorado.

SLIDE 4

The Summitville district lies in the eastern San Juan Mountains, about 30 miles Southeast of Creede. The ores are contained within a rhyodacite (= quartz latite) volcanic dome which appears to have formed along the structural margin of the Platoro caldera (Lipman, 1975).

SLIDE 5

The important relations are illustrated on this cross section, rather poorly prepared after a plate from the Professional Paper by Tom Steven and Jim Ratte (1960). The quartz latitic volcanic dome was emplaced about 23 million years ago, cutting the 29 million year old andesite which flooded the Platoro caldera. The orebodies, indicated in red, are entirely contained within the Quartz latite, but not necessarily within the coarsely porphyritic phase as implied by this cross section. The wallrock alteration related to the ores affects both the the quartz latite and the andesite. The alunite in the ores has been dated at 22 million years, a figure consistent with the geologic relations which show that the 20 million year old Rhyolite of Cropsy Peak lies on an eroded surface of the altered Quartz latite dome, but has not been affected by the alteration. The dates shown are from Mehnert, Lipman and Steven (1973), and have been supplemented by additional age determinations by the Anaconda crew which has been working at Summitville for the past several years. The Anaconda dates extend the range of values for each of the units - including the alunite - but still drive home the point that the mineralization and accompanying alteration were integral parts of the volcanic evolution of the dome itself.

SLIDE 6

The distribution of the alteration zones is shown in this slide, again from the 1960 Steven and Ratte Professional Paper. In the upper part of the ore zone the alteration tends to be poddy, becoming more tabular to veinlike with increasing depth, but the assemblges and zonal pattern found in the deepest workings are the same as those exposed at the surface. The sulfide mineral assemblage is dominated by pyrite which occurs as vug and vein filling, but mainly as widespread disseminations. Enargite and covellite, with smaller amounts of luzonite, and famatinite make up the bulk of the ore which contains only minor amounts of sphalerite, galena and native gold. The copper minerals characterize the assemblage as being of the high sulfur variety. Thus, Summitville displays very well the characteristic features of the acid-sulfate (or Goldfield-type) of epithermal ore deposits, of which I remind you by repeating the earlier slide.

SLIDE 7 (Repeat of 3)

Now, in order to give some authority to my statements concerning the fundamental differences between the Goldfield and Creede-type environments, I have borrowed, combined, and colored up a couple of illustrations constructed

SLIDE 8 (Slide omitted from this report)

by this happy fellow - Dick Henley, The Great Guru of Wairakei. Those of you sitting up front can see, first of all, that Henley is wearing a proper T-shirt, and, secondly, that he is being supported by a stout sign bearing the Maori inscription: "TE ARA MOKOROA," translated for us below as "the long abiding path of knowledge" To what greater authority could I appeal?

SLIDE 9

In this slide, the Goldfield environment, as I envision it, is on the left; the Creede environment on the right. The fundamental difference between the two is immediately obvious; the Goldfield-type ores were formed at shallow depths in the upper part of a volcanic edifice. This one I borrowed from Henley is perhaps too much of a proper volcano, and too large in size to accurately represent most of the Goldfield-type deposits, but I chose it purposely. Dick drew it to represent a typical geothermal system in calc-alkaline stratavolcano and then used it to represent a possible environment for porphyry copper mineralization, in my mind an environment not far removed from that of the Goldfield-type, although the Goldfield-type systems are substantially shallower. The Goldfield-type ores are, in general, hosted by volcanics derived from the heat source which drove the system and are not far removed from the heat source. In the Creede-type system, the ores were again formed at shallow depths, but they were plastered along the top of a deeply circulating hydrothermal system, and were deposited a long way from the heat source. We could continue the comparison by discussing the relative importance of magmatic components and the implications regarding the chemical evolution of the two systems, but we would be getting further and further adrift on a sea of speculation with few areas of documentation on which to set an anchor. The point I hope I have made is that I consider the Goldfield-type and the Creede-type systems to be fundamentally different, and that that difference is a consequence of the fact that the Goldfield-type system is driven by a near surface - and near ore - heat source, whereas the Creede-type systems are driven by a deep heat source, far removed from the ore zone.

The next obvious question to ask is why the acid-sulfate alteration is characteristic of the Goldfield-type environment but is seldom, if ever, seen as a primary assemblage in the Creede-type systems. This brings us into an area of controversy concerning the origin of the acid-sulfate alteration, which in turn revolves around the mechanism by which the sulfuric acid, required to form the alunite + kaolinite assemblage, is generated. I think I know, but so do a number of other people, and they don't all agree with me.

SLIDE 10

This slide shows three reactions; all leading to the formation of sulfuric acid. The top reaction is the simple oxidation of hydrogen sulfide and is the reaction that produces the spectacular solfataric alteration that we see at the surface in active geothermal systems.

SLIDE 11 (repeat of 9)

It arises from the escape of H_2S from the deeply circulating waters, shown in this slide in yellow, as they begin to boil on nearing the surface. The H_2S rises along with the steam and other volatiles, such as CO_2 , and is oxidized upon encountering the overlying ground waters which are saturated in atmospheric oxygen. These overlying groundwaters become heated by the recondensation of steam and are referred to as steam-heated waters. They are shown in green on this slide, for both the Goldfield-type and Creede-type environments, and they are essential elements of almost all high temperature geothermal systems in silicic or andesitic volcanic terranes. The alteration produced by this mechanism is very intense, but is a surficial phenomenon, rarely extending down to depths of even fifty meters below the surface. I think that we see very few of these alteration zones in ore deposits because they are very soft, shallow and easily eroded; but many people have appealed to this mechanism to explain the acid sulfate alteration in a number of districts.

SLIDE 12 (repeat of 10)

I prefer the second reaction as the mechanism by which sulfuric acid is formed in Goldfield-type environments. SO_2 gas is unstable below temperatures of about 400 degrees centigrade in the presence of water, and it disproportionates, as shown in the reaction on the slide, into H_2S and sulfuric acid. There are a lot of attractive reasons for preferring this mechanism, but there isn't time to defend my choice right now. Rather I'm going to propose a test whereby we can distinguish between these mechanisms. First, however, we must note that the kaolinite + alunite assemblage can also be produced by sulfuric acid attack during supergene alteration of sulfide ore bodies as suggested by the third reaction on the slide; written for pyrite being oxidized to something approximating limonite. This has happened at Creede, at Goldfield, I think at Round Mountain, and I'm sure in many other districts. Getting back to the proposed test, I am going to try to convince you that we can use a little isotope geochemistry to help us distinguish among these three mechanisms of acid-sulfate alteration.

SLIDE 13

Back in 1966, Cy Field proposed (Field, 1966), as did Roy Jensen in 1967 (Jensen, 1967), that the measurement of sulfur isotopes in alunites could distinguish between those formed by hydrothermal precipitation and those formed by supergene alteration. In 1971 Jensen, Roger Ashley and John Albers (Jensen et al., 1971) applied this technique to Goldfield as shown on this slide. As you can see, the alunites judged to be primary on field and petrographic grounds were much heavier than those formed by secondary processes which showed the same range of sulfur isotopic composition as did the primary sulfides. The reason for this is that there is no sulfur isotopic fractionation during low temperature oxidation of sulfides due to kinetic effects, as has been demonstrated many times. Sulfuric acid generated by the oxidation of sulfides has the same sulfur isotopic composition as the sulfides! On the other hand, at temperatures above about 200 degrees celsius considerable isotopic exchange between aqueous sulfide and sulfate species can take place, particularly in acid environments, as has been discussed by Ohmoto and Lasaga (1982). Primary alunite formed at these temperatures will be much heavier than coexisting sulfides because of the

strong fractionation between aqueous sulfide and sulfate species. So now we know how to tell primary from secondary alunite. But we really don't, because it turns out that when H_2S is oxidized in the steam-heated waters it too doesn't undergo much sulfur isotopic exchange with the sulfate produced and, as has been demonstrated by Shoen and Rye (1970) for Yellowstone. The sulfate in the steam-heated waters in active geothermal systems has the same sulfur isotopic composition as the H_2S from which it was derived by oxidation, so the alunites formed in supergene and steam-heated water environments will both have sulfur isotopic compositions approximately the same as their coexisting sulfides. So now we can tell alunites that formed by disproportionation of SO_2 from that formed from steam-heated or supergene waters, but, based on the sulfur isotope data alone, we can't tell apart the alunites formed from steam-heated versus supergene waters. Fortunately, if we look at the associated kaolinites we should be able to distinguish between them.

SLIDE 14

This slide shows us how it works. In the top figure a group of kaolinites and halloysites formed by supergene alteration of some Japanese orebodies are plotted (Marumo et al., 1982). You can see that their compositions fall very close to the KAOLINITE LINE of Savin and Epstein (1970) and Lawrence and Taylor (1971) which is the line representing the locus of hydrogen and oxygen isotopic compositions for kaolinites formed by weathering and assumed to be in equilibrium with meteoric waters along the meteoric water line. Note that the compositions of some of the surface waters plot, as expected, near the meteroric water line and are connected to their corresponding kaolinite values. In the bottom figure are plotted the hydrogen and oxygen isotopic compositions of a group of kaolinites from acid-sulfate alteration zones from several geothermal areas in Japan along with some hydrothermal kaolinites from several ore deposits - indicated by open circles. These kaolinites, formed from steam-heated waters, are considerably removed from the meteoric water line and very close to the dashed line which marks the compositions of kaolinites that would be in equilibrium with meteroric waters at temperatures of 100 degrees celsius. So the hydrogen and oxygen isotopic composition of the kaolinites in acid sulfate alteration zones may allow us to distinguish between primary and supergene origins for acid sulfate alteration zones. This is not an original concept, for Taylor (1974) pointed this out years ago.

SLIDE 15

If we put this all together, as on this slide, we see that we can use the isotopic signatures of the alunite and kaolinite in acid sulfate alteration zones to tell us something about the origin of the acid-sulfate alteration. As you can see, I've added a row to show that the K/Ar age determinations on alunite, if we trust them, can also be used to distinguish between supergene and primary origins, as we have done at Creede and as has been done for a number of other districts. Alunite itself contains both hydrogen and oxygen as well as sulfur, so if we can calibrate, and learn to trust, the isotope systematics we can do the whole thing on alunite. Right now Dick Henley, Bob Rye and I are in the beginning stages of trying to apply these isotopic discriminators to a number of epithermal and geothermal environments to see if they really work, and to find out what the mechanism of acid sulfate alteration in Goldfield-type environments really is.

Now let's look at the Creede-type environment for the time remaining.

SLIDE 16

I've borrowed another slide from Henley - Good old Dick; he's easy. This one is a cross section of the Wairakei geothermal field as it looked prior to its development for geothermal power; which changed a lot of things. I begged this slide to illustrate the overall chemical structure of a deeply circulating geothermal system in a silicic volcanic terrane. The yellow represents the plume of heated, near neutral chloride waters which displaces the cooler, fresh groundwater shown surrounding it in blue. Mixing occurs between the two resulting in dilute chloride waters. Note that the isotherms are displaced to the right by the gravity-driven lateral flow along the top of the system, and that boiling occurs in the upper 400 meters of the chloride-water plume. Let's move a little closer to examine the detail along the top of the system.

SLIDE 17

Here we see our old friend the steam-heated waters forming a shallow cap to the hottest part of the system. These steam-heated waters mix with the chloride waters on the downflow side, and a small area dominated by steam is shown in red. I'm told that this is the general structure of geothermal systems of this type the world over. If this is so, we should be able to find evidences that such a structure existed in some epithermal systems of the Creede-type; if we are correct in our assumption of the doctrine of uniformitarianism in this case. Before we go looking, though, lets glance briefly at only a few of the results of a very nice and very useful study of the Waiotapu geothermal system in New Zealand done by Jeff Hedenquist for his PhD dissertation at the University of Aukland (Hedenquist, 1983). I'm sorry that Jeff is not here in person telling us about Waiotapu as part of this symposium, because Jeff approached the study thru the eyes of an ore deposit geologist, and the slant he gave it makes it somewhat unique, and of particular interest to us.

SLIDE 18

The advanced argillic or acid sulfate alteration shown in yellow across the top marks the position of the steam-heated waters responsible for the alteration. Note the patches of disseminated lead-zinc mineralization shown in red, some distance below the acid sulfate alteration, and the suggestion that gold is being transported thru the zone where lead and zinc are being deposited. The circulation patterns are shown by the arrows. Note that, in addition to the deep convective flow, Jeff has identified a lateral flow component at about the 150 meter depth. Several different waters are denoted by alpha-numeric symbols. A-1 is the deep chloride water which rises and is shown to be boiling at about the 300 degree isotherm. As it rises it continues to boil, loses heat, and becomes more concentrated as it moves toward A-2. The chemical consequences of the boiling are that the fluid becomes more oxidized, principally thru the loss of hydrogen, and more alkaline thru the loss of the acid volatiles such as CO_2 and H_2S which, like hydrogen, are very strongly partitioned toward the vapor phase. Due to its much lower viscosity, the vapor phase rises until it mainly condenses - but partially escapes thru fumeroles - heating and acidifying the waters in which it condenses, as we have discussed before. The steam-heated waters are denoted here by the

letter B. The waters denoted by the A-3 symbol flow laterally across the top of the system, mixing, as we shall see, with the A-2 waters. So Jeff has identified two processes, boiling and mixing, both going on at the top of the deeply circulating chloride cell, in the area of base metal precipitation. This is good because we economic geologists have been appealing to mixing and boiling for some time, particularly boiling in the past several years; and Ed Drummond in an important thesis, that we all hope will be published soon, has demonstrated the effectiveness of both mechanisms and provided an excellent theoretical base on which quantitative models can be constructed (Drummond, 1980). But these aren't the only possible mechanisms of precipitation, and as economic geologists we need to seek evidences of the occurrence of boiling and mixing in ore-forming systems and use them both to help locate ourselves in the system and to develop meaningful quantitative models of ore deposition.

Jeff's study was based on drill core, downhole and well-head temperatures, and well-head fluid samples - all gathered some 17 years prior to his study; and on spring samples gathered thru the years. One of the many techniques Jeff used in reconstructing the Waiotapu system was:

SLIDE 19

this plot of the heat content vs. the chloride content of the various waters. (Note that I've plotted temperature along the top of the heat content axis) This kind of plot is particularly useful in deciphering the origins of - and the relationships between - fluids from different springs or wells in geothermal systems. I hope I can show you that it can be equally useful in the study of ore deposits. Let's see how its used. The A-1 water at about 300 degrees and 1300 ppm Chloride is the deep, unmodified chloride brine. As this brine rises to the upper part of the system it begins to boil. When that happens it cools because it loses heat to the vapor, and becomes more concentrated because the chloride is retained in the liquid phase. On this heat-content vs. chloride diagram, its position moves along the line as indicated by the arrow, radial to the steam point (the heat content - about 2800 kilojoules/mole - of pure water at its boiling point). Jeff's A-2 water has been derived simply by boiling of A-1. The A-3 water is that water that was seen to be entering the top of the system by lateral flow. It's nearly fresh, but has been heated to about 170°C. The yellow triangles fall very tightly about a line between the boiled, deep, A-2 waters and the fresh, heated A-3 waters; and are interpreted to show a mixing trend between the two. Some of them might better have been interpreted as mixing between the A-2 and cool meteoric waters; or even between the A-1 and A-3 waters. The steam-heated waters lie at B, about 100°C and don't show any well defined mixing trend with any of the other end-member waters.

Now lets look at ore deposits. One of the easiest sets of observations we can make on fluid inclusions is the temperature of homogenization on heating - and the freezing point of the liquid which is a measure of its salinity, or (for most epithermal systems) its chlorinity.

SLIDE 20

Ed Roedder (1977) has done a detailed growth-zone by growth-zone study of the homogenization and freezing temperatures of individual primary fluid

inclusions from 20 growth zones in this large sphalerite crystal from the OH vein at Creede. He found that each growth zone had a narrow range of both temperature and salinity values, and that these ranges were quite different for different growth zones, as shown on the following slide.

SLIDE 21

In this slide the numbers beside each temperature-salinity field indicate the paragenetic position of the growth-zone, 1 being the oldest and 20 the youngest. Note that this temperature-salinity diagram is almost equivalent to the heat content chloride diagram which Jeff Hedenquist used to evaluate his Waiotapu data, and that Ed's temperature-salinity fields for each growth zone can be considered equivalent to Jeff's well or spring samples. We can imagine that the highest salinity water - 9 - could have been derived by boiling of a higher temperature water like 8 - and that the group of waters indicated by yellow triangles were derived by mixing of waters like 9 with cooler, fresher waters. If we draw a mixing line thru the data (ignoring the values from zones 15 thru 20 for the moment) we can propose that these waters in here were derived by mixing a boiled water like 9 with a fresh water heated to about 200 degrees C. Not unlike the situation Waiotapu, huh? It's particularly interesting that the density of a water like 9 and fresh water at 200 degrees are nearly identical and that the mixing line is nearly an isodensity line - a condition that greatly favors mixing. This same relationship between temperature and salinity of fluid inclusions has been documented now for several other localities on the OH vein at Creede, and Dan Hayba on our project is making good progress in the extremely painstaking job of defining the temperature-salinity relationships in space along the OH vein as well as in time - and of interpreting the data in terms of the boiling and mixing of different end-member waters. Well now, what evidence is there that boiling occurred in the OH vein or that heated, fresh waters were available for mixing with the hot brines? It turns out that there are several lines of evidence, some direct and some inferential. The first was Ed Roedder's discovery, more than 20 years ago that some quartz crystals from the OH vein contained co-eval fluid inclusions with widely varying liquid vapor ratios. He interpreted them to have resulted from the trapping of a mixed liquid-vapor fluid. This was the first such demonstration of boiling on any ore deposit; an observation and interpretation since duplicated for hundreds of other deposits.

SLIDE 22

Here I have plotted the 5 localities where we have found, from such observations, definite evidence of boiling along the OH vein. I have also shown two other features interpreted to be the results of boiling; the distribution of vein adularia mainly below the zone of documented boiling, and the zone of intense illitic alteration which caps the OH vein above the zone of documented boiling, and is spectacular in its contrast and sharpness of contact with the unaltered wallrock below which hosts the productive part of the vein. I should note that the distribution of the alteration cap is much more regular along the simple open fracture of the OH vein than it is in other parts of the district where the hydrology was much more complex. The vein adularia and illitic alteration are two sides of the same coin. The adularia precipitated in response to the rise of the pH of the ore fluid on boiling, due to the strong fractionation of acid-forming volatiles such as CO_2 and H_2S into the vapor phase. The illitic alteration resul-

ted from the recondensation and subsequent hydrolysis reactions of those volatiles in the overlying, probably fresh waters. OK, so boiling did occur along the OH vein at Creede. How about those heated, fresh waters? Well, Bob Rye and I had a problem with the interpretation of the isotope geochemistry of the fluid inclusions in quartz from Creede which suggested that the quartz, which was intergrown in vugs with the sulfides, formed from waters that were isotopically much different from those which deposited the sphalerite and altered the wall rocks. When Nora Foley came to work for us we set her about trying to find out what was wrong. Well, she did; and lots more.

SLIDE 23

By very painstaking work Nora found (Foley and others, 1982) that the quartz crystals that Bob and I had sampled for isotopic analysis of the fluid inclusions contained two very different populations of inclusions; and therein hung the tale. The relatively large, juicy primary inclusions (the kind that all of us measure because they're so easy to see) gave a range of temperature-salinity values

SLIDE 24

pretty much the same as those measured from sphalerite (Ed Roedder's data, previously discussed, are plotted here for reference)

SLIDE 25 (repeat of 23)

But, along with the relatively few primary inclusions Nora found thousands of tiny pseudosecondary inclusions along healed fractures - the kind that none of us measure (particularly in freezing studies) because its so hard to see what's going on because they're so small. These little buggers showed a nearly identical temperature range to that of the primaries,

SLIDE 26 (repeat of 24)

but their salinities were much lower - zero in many cases. Nora also found that about 15% of the primaries had been intersected by the fractures and gave intermediate salinity values. So Nora has shown that heated, fresh waters did exist in the Creede system - presumably along the top of the system - and were available for mixing with the upwelling brines as suggested by Roedder's data and by that being gathered by Dan Hayba. Nora solved our isotope problem, too, by tediously cutting out, with a wire saw, areas in a quartz crystal containing either only primary or only pseudosecondary inclusions in large enough quantity to allow hydrogen isotopic analysis. Bob Rye found that the primary inclusions had hydrogen isotopic signatures just like those of the sphalerites, whereas the pseudosecondaries were about 35 permil lighter. This result allowed the much more comfortable interpretation that the quartz and sulfides grew together from the same fluid, but that the quartz was fractured episodically - probably due to thermal shock resulting from the descent of cooler, overlying groundwaters into the ore zone as a result of some hydrologic flucuation. As these fractures healed, they trapped these fresh waters as pseudeosecondary fluid inclusions. The original analyses of the hydrogen isotopic composition of the quartz fluid inclusions by Bethke and Rye (1979), assumed to represent only the primary inclusion waters, were, in fact, strongly biased through the contamination

by these much lighter (in delta D) waters contained in the pseudosecondary inclusions.

Before closing I should note that Rick Robinson and John Rice have both, in their Master's theses, presented fluid inclusion evidences for mixing of fluids in the southern part of the Creede district - evidence which I just couldn't work into the timing of this talk.

SLIDE 27 (Slide omitted from this report)

Well, I see I've put Henley to sleep - again. If someone will wake him up we can all go to lunch. Thank you.

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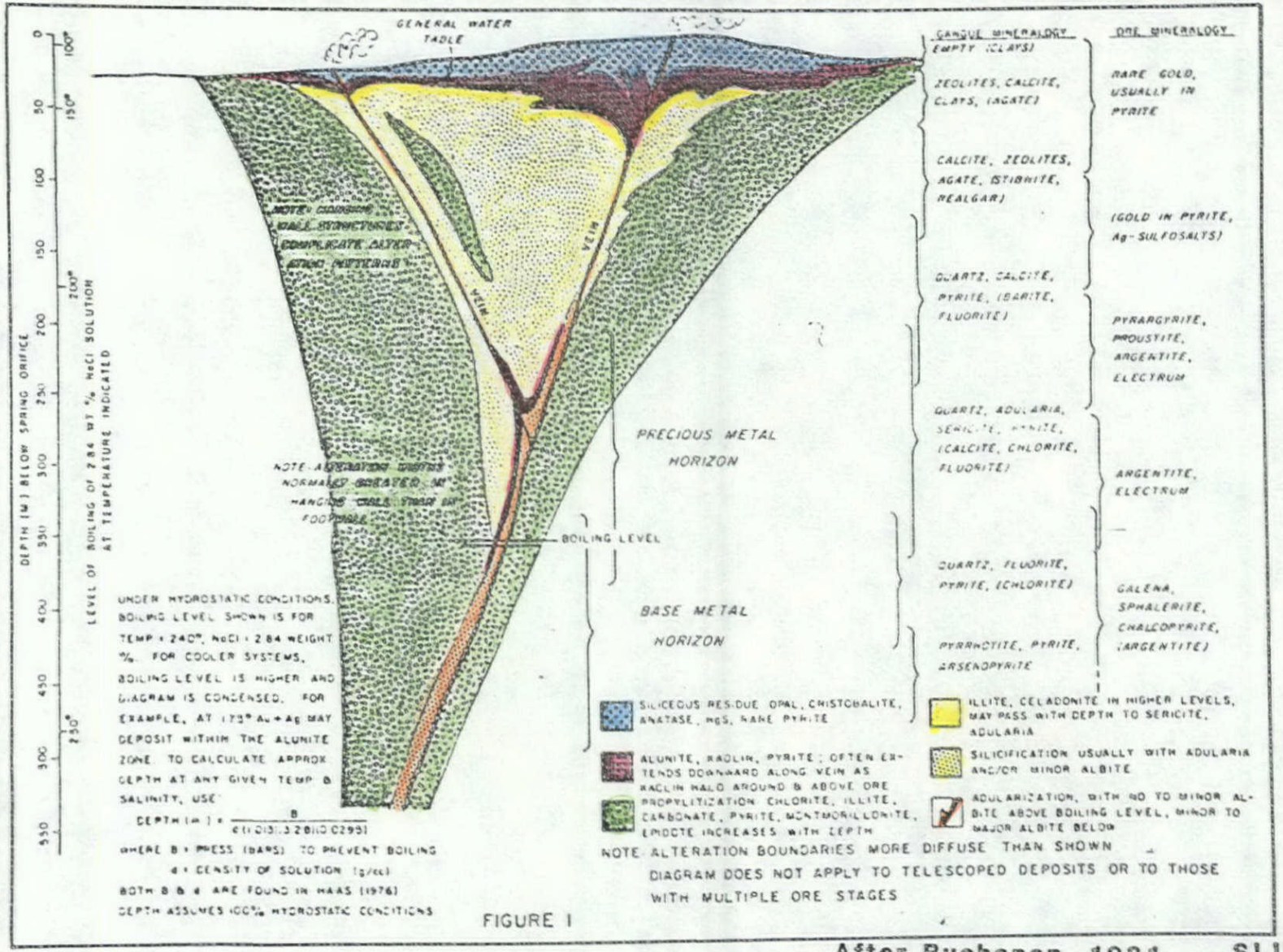


FIGURE 1

After Buchanan, 1981. SLIDE 1

ACID - SULFATE ALTERATION	ADULARIA - SERICITE ALTERATION	
Goldfield, NV	Creede, CO	Tonopah, NV
Summitville, CO	Comstock, NV	Silver City, ID
Red Mountain, CO	Round Mountain, NV	Oatman, AZ
Julcani, Peru	Colqui, Peru	Guanojuato. Mexico
	Eureka, CO	Pachuca, Mexico

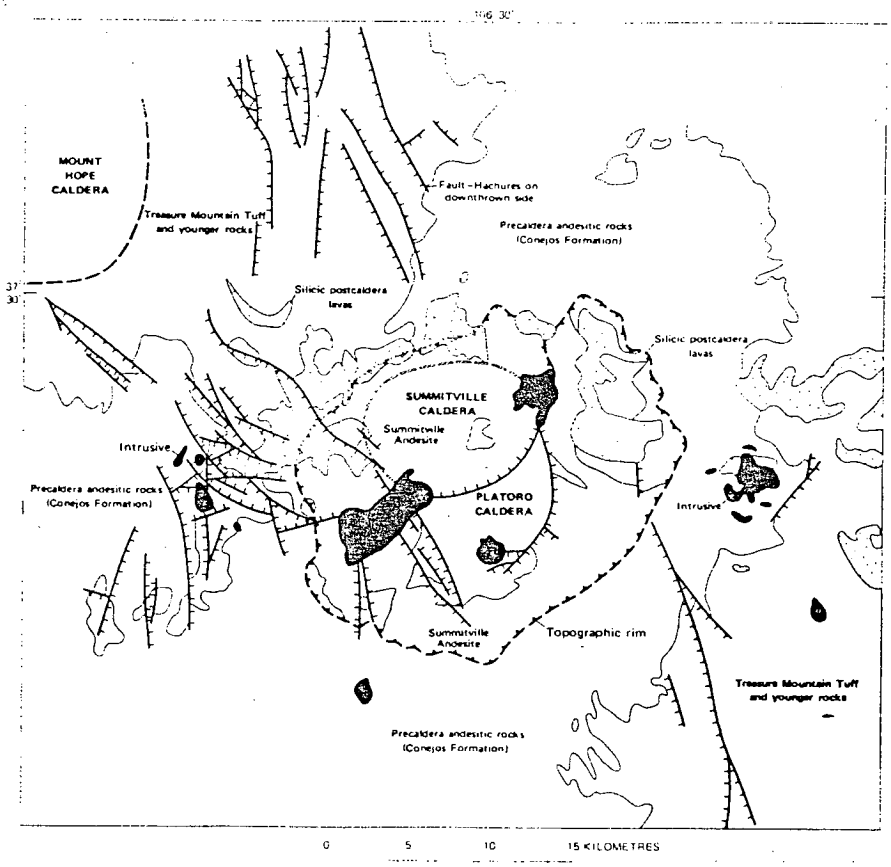
SLIDE 2

After Heald-Wetlaufer, Hayba, Foley & Goss, 1983

DISTINCTIVE CHARACTERISTICS

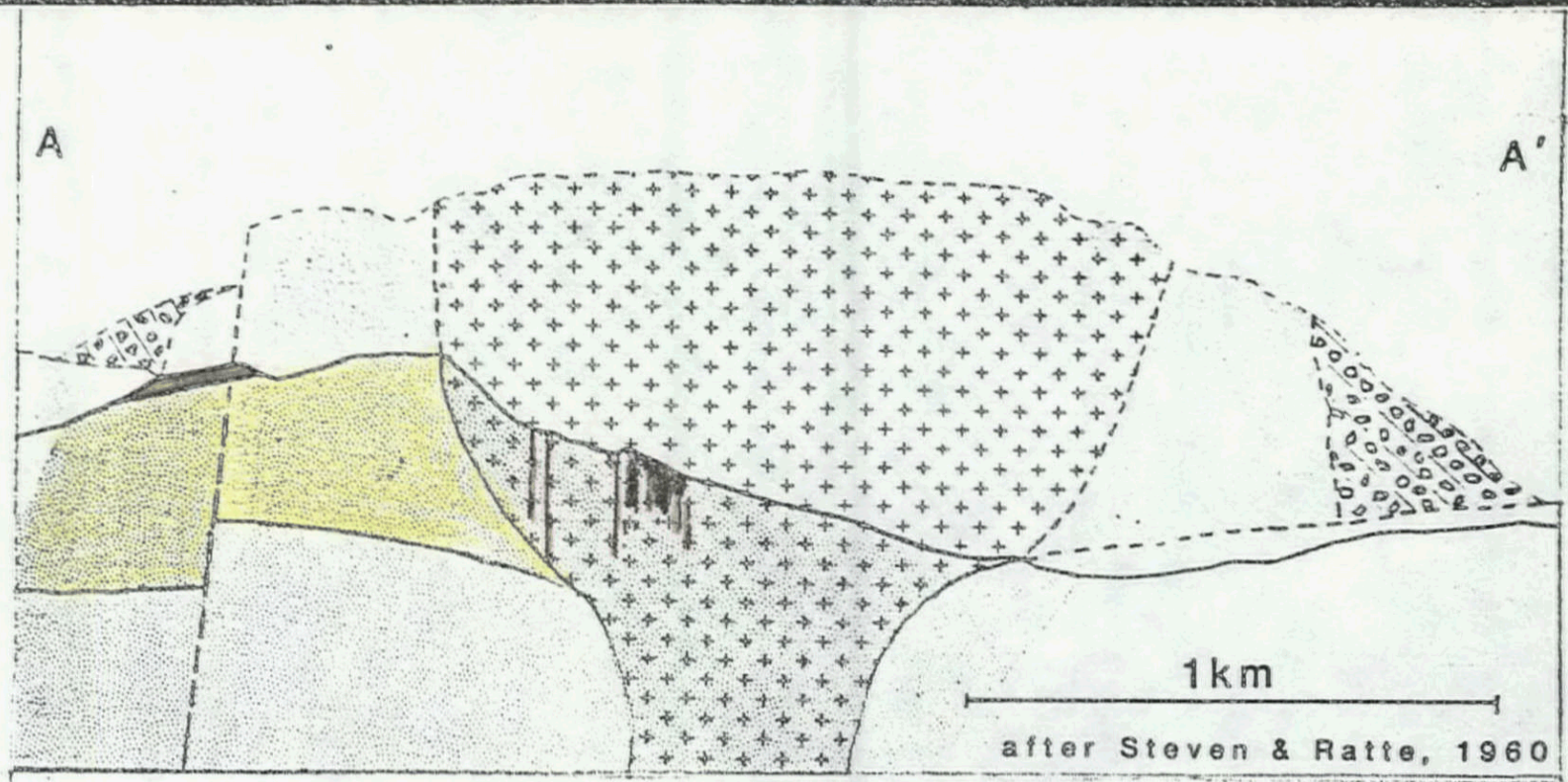
ACID - SULFATE ALTERATION	ADULARIA - SERICITE ALTERATION
Higher Sulfur Mineral Assemblege	Lower Sulfur Mineral Assemblege
Advanced Argillic Alteration	Sericite - Adularia Alteration
Ages of Host and Ore Equivalent	Ore Distinctly Later Than Host
Noteworthy Copper Production	Low Copper Production

SLIDE 3 After Heald-Wetlaufer, Hayba, Foley and Goss, 1983



From Steven & Lipman, 1976

SLIDE 4



Rhyolite 20.2ma

Quartz Latite 22.9ma

Andesite ~29ma

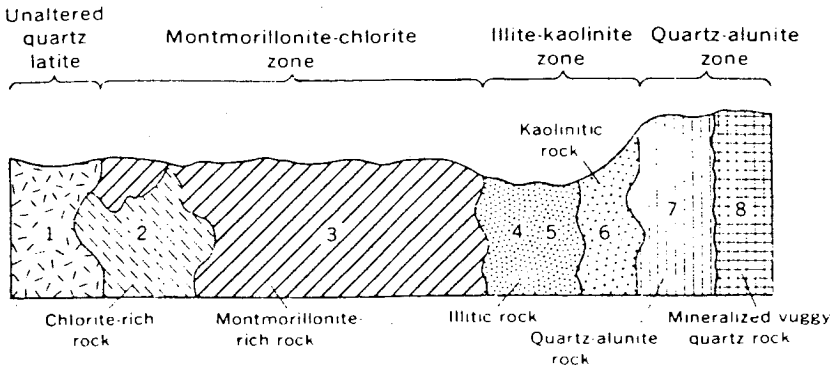
Coarse Porphyry

Quartz-Alunite 22.3ma

Altered Area

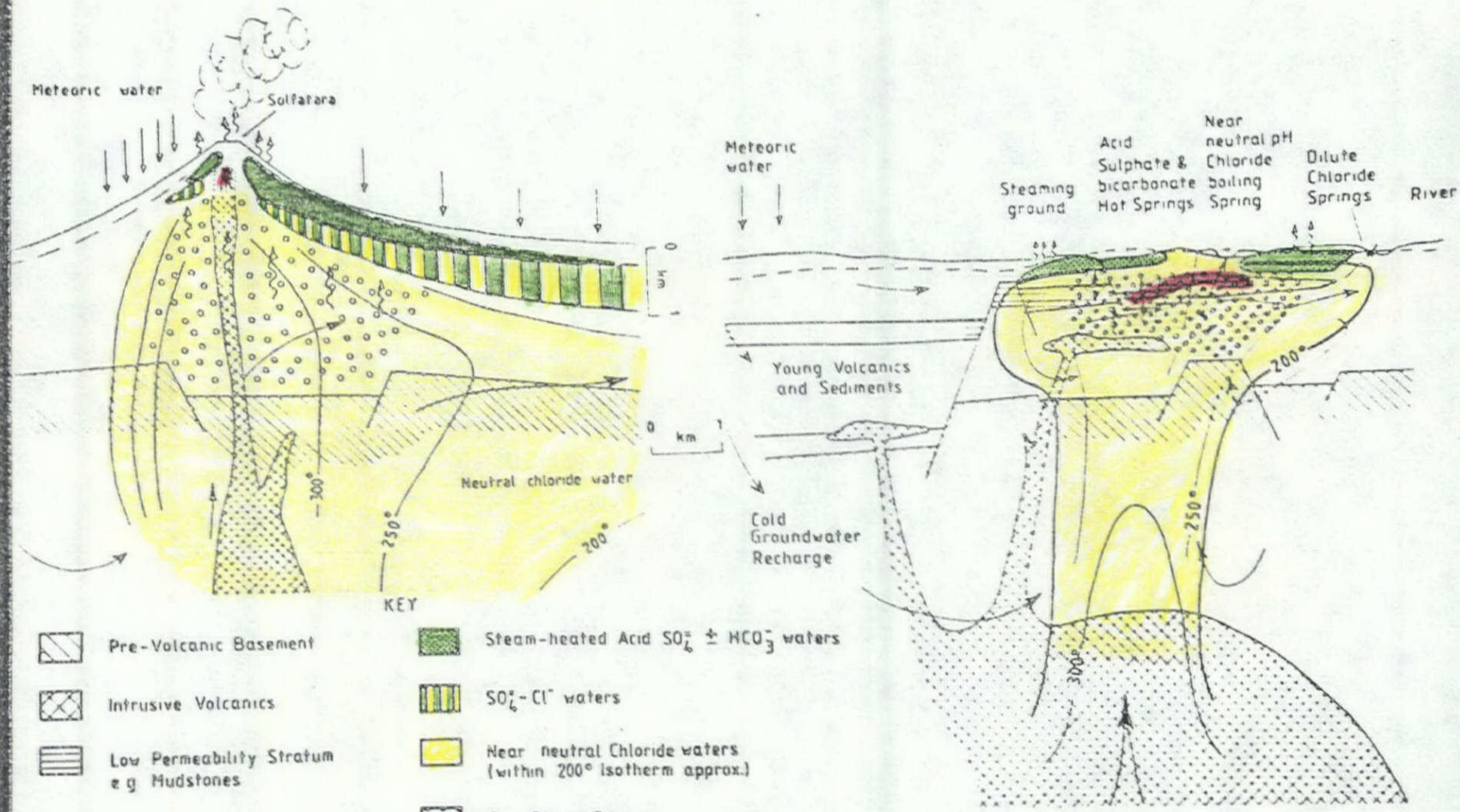
After Steven & Ratte, 1960

SLIDE 5



From Steven & Ratte, 1960

SLIDE 6



KEY

- Pre-Volcanic Basement
- Intrusive Volcanics
- Low Permeability Stratum e.g. Mudstones
- Steam-heated Acid $SO_4^{=} \pm HCO_3^-$ waters
- $SO_4^{=}$-Cl^- waters
- Near neutral Chloride waters (within 200° Isotherm approx.)
- Two Phase Region Water Liquid + Steam (+ Gas)

HEAT AND MASS (NaCl, CO_2, SO_2, H_2O) TRANSFER FROM MAGMA SYSTEMS.

After Henley & Ellis, 1983

SLIDE 9

$$4H_2S + 8O_2 \rightarrow 4H_2SO_4$$

$$4SO_2 + 4H_2O \rightarrow 3H_2SO_4 + H_2S$$

$$2FeS_2 + 7H_2O + 9O_2 \rightarrow 2Fe_2O_3 \cdot 3H_2O + 4H_2SO_4$$

SLIDE 10

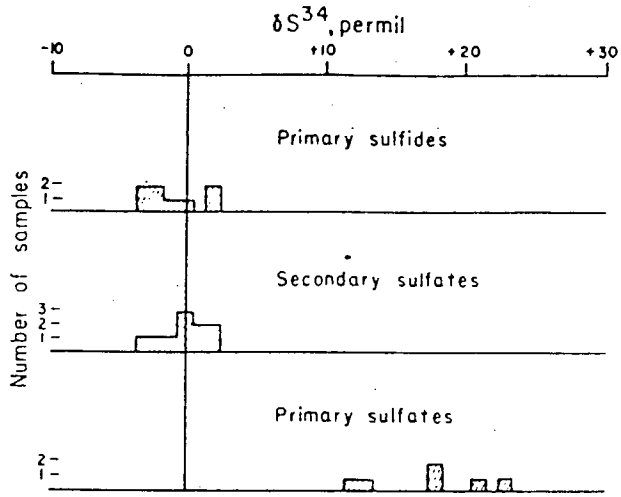
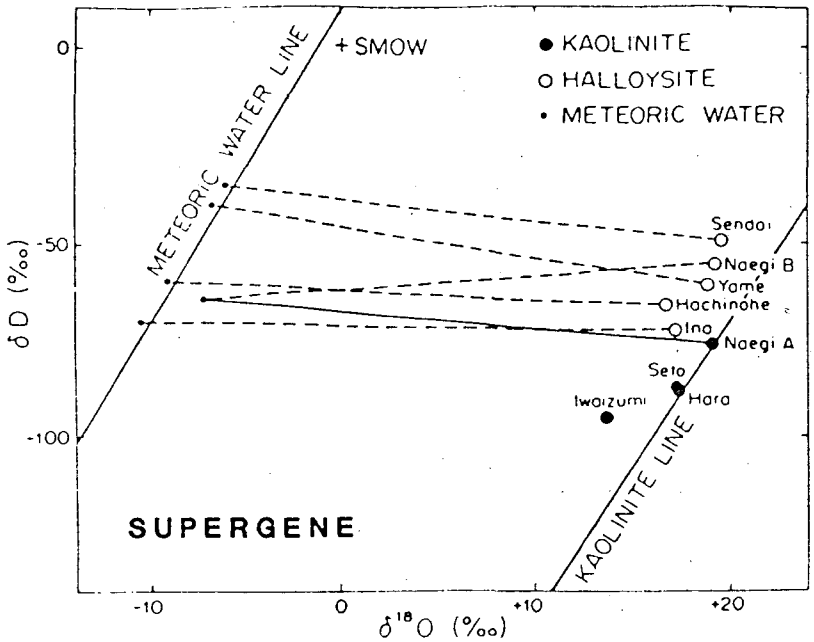
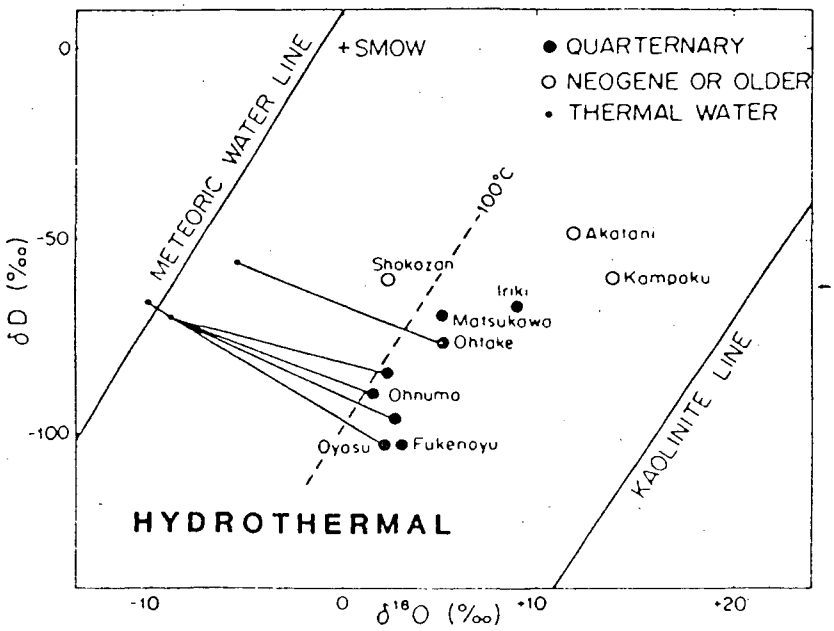


FIG. 3. Sulfur isotopic ratios of Goldfield sulfides and sulfates.

after Jensen, Ashley & Albers, 1971

SLIDE 13





after Marumo, Matsuhisa & Nagasawa, 1982

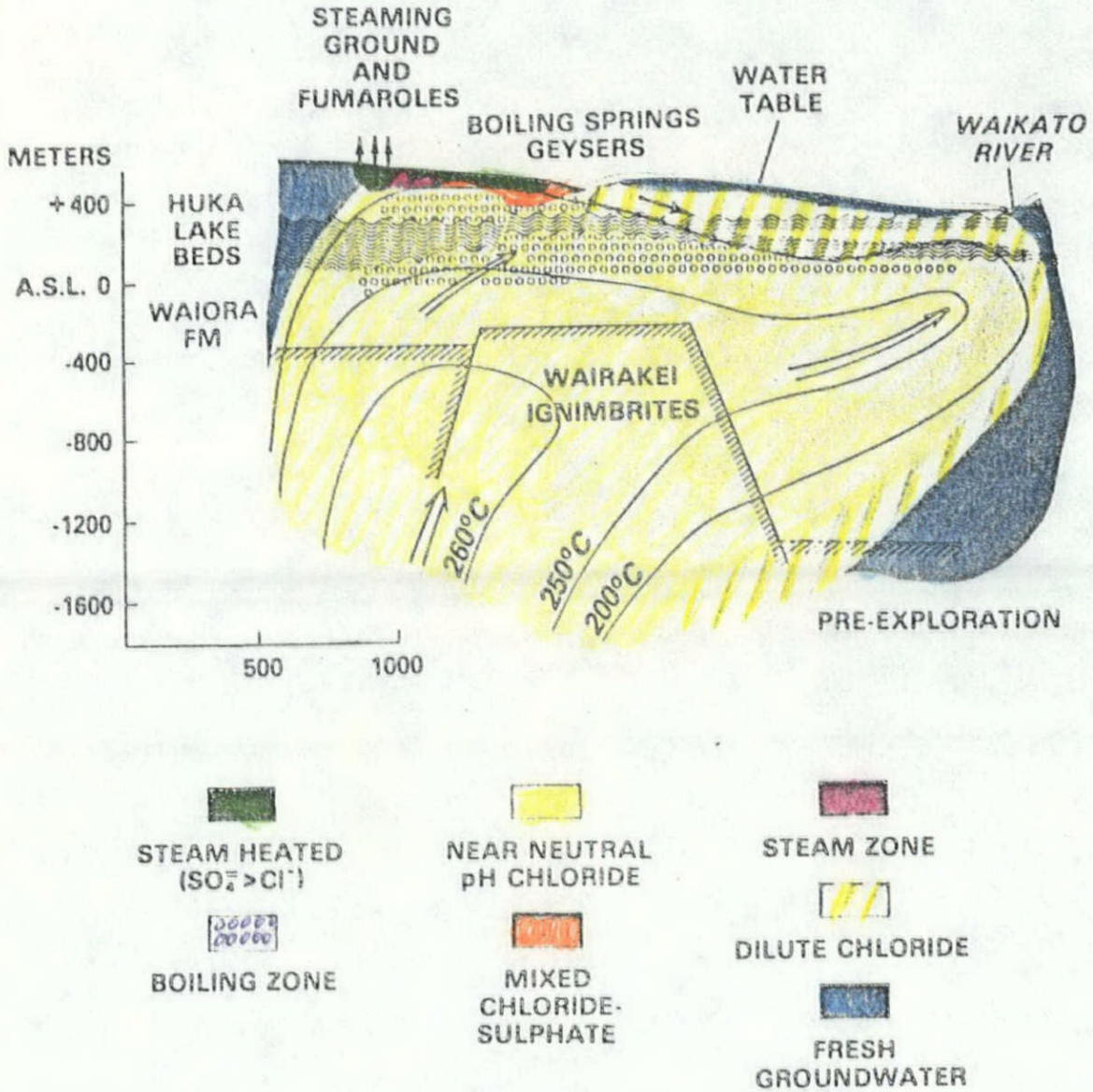
SLIDE 14

	Oxidation of H_2S	Disproportionation of SO_2	Supergene Oxidation
$\delta^{34}S_{alunite}$	= Sulfides	$\gg$ Sulfides	= Sulfides
$\delta^{18}O_{kaolinite}$	Removed from Kaolinite Line	Removed from Kaolinite Line	Near Kaolinite Line
$K/Ar_{alunite}$	Concordant	Concordant	Younger

ISOTOPIC CHARACTERISTICS OF ACID-SULFATE ALTERATION OF DIFFERING ORIGINS

SLIDE 15

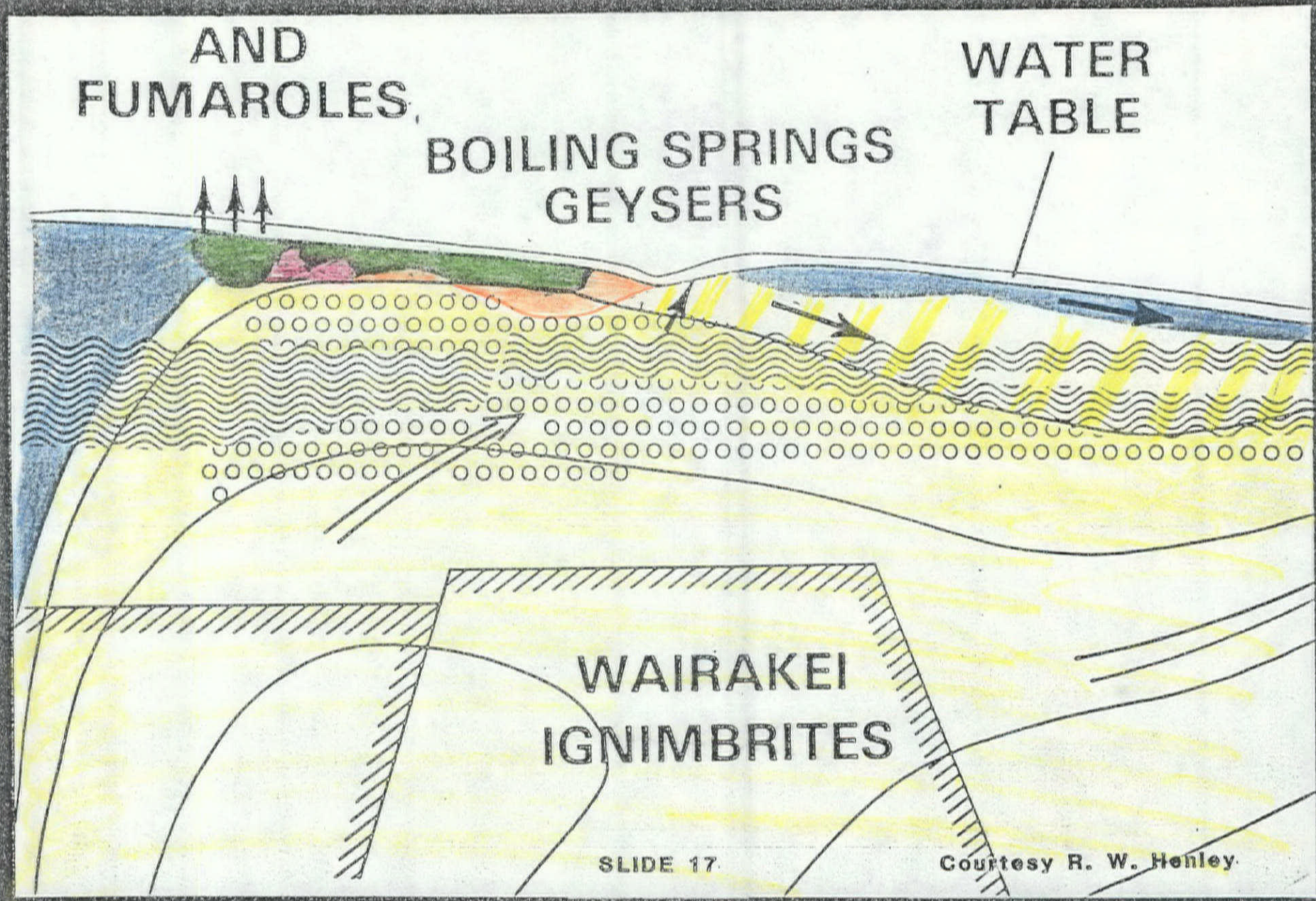
WAIRAKEI GEOTHERMAL FIELD, NEW ZEALAND
LONGITUDINAL SECTION, PRE-EXPLOITATION



ALL FLUID BOUNDARIES GRADATIONAL

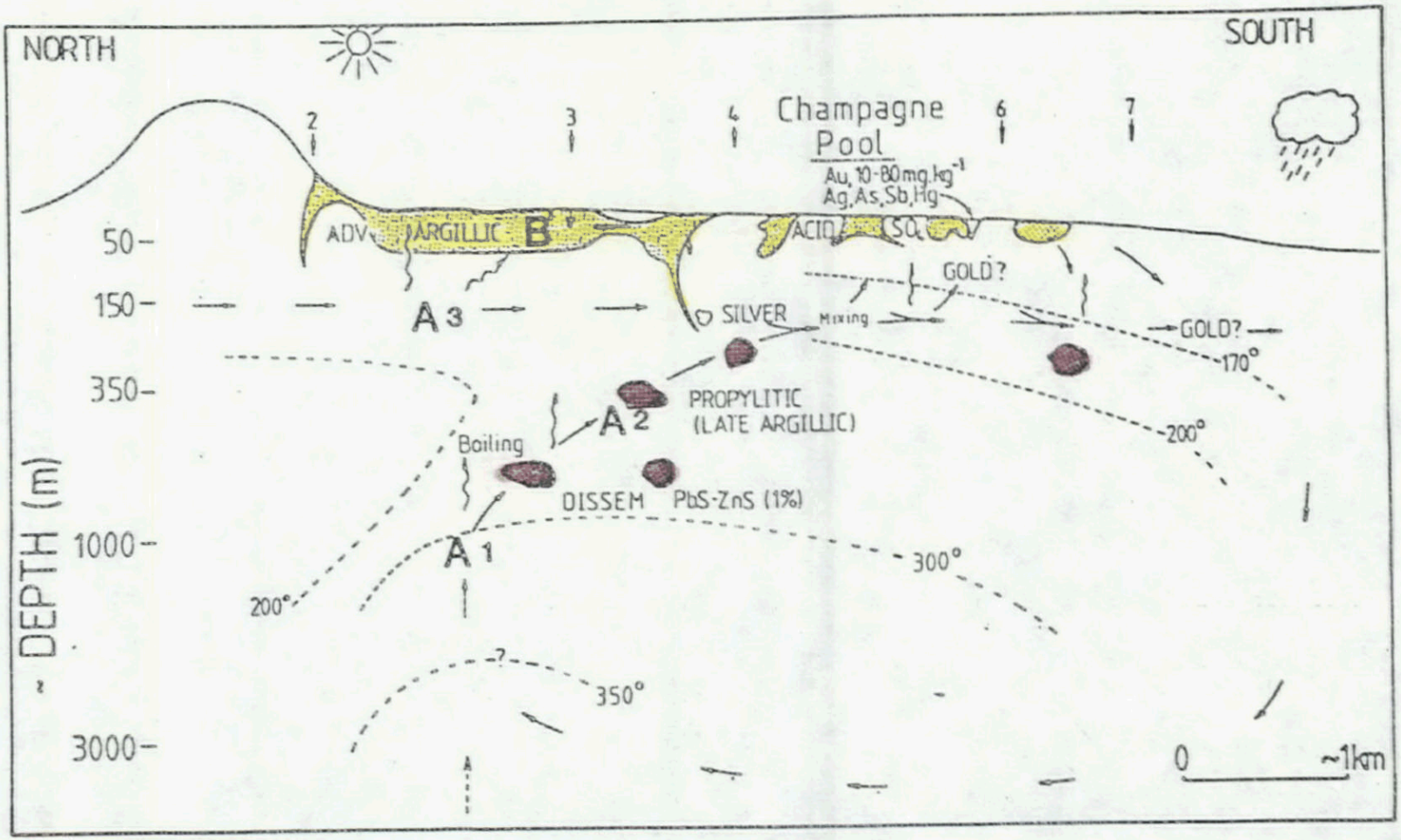
SLIDE 16

Courtesy R. W. Henley



SLIDE 17

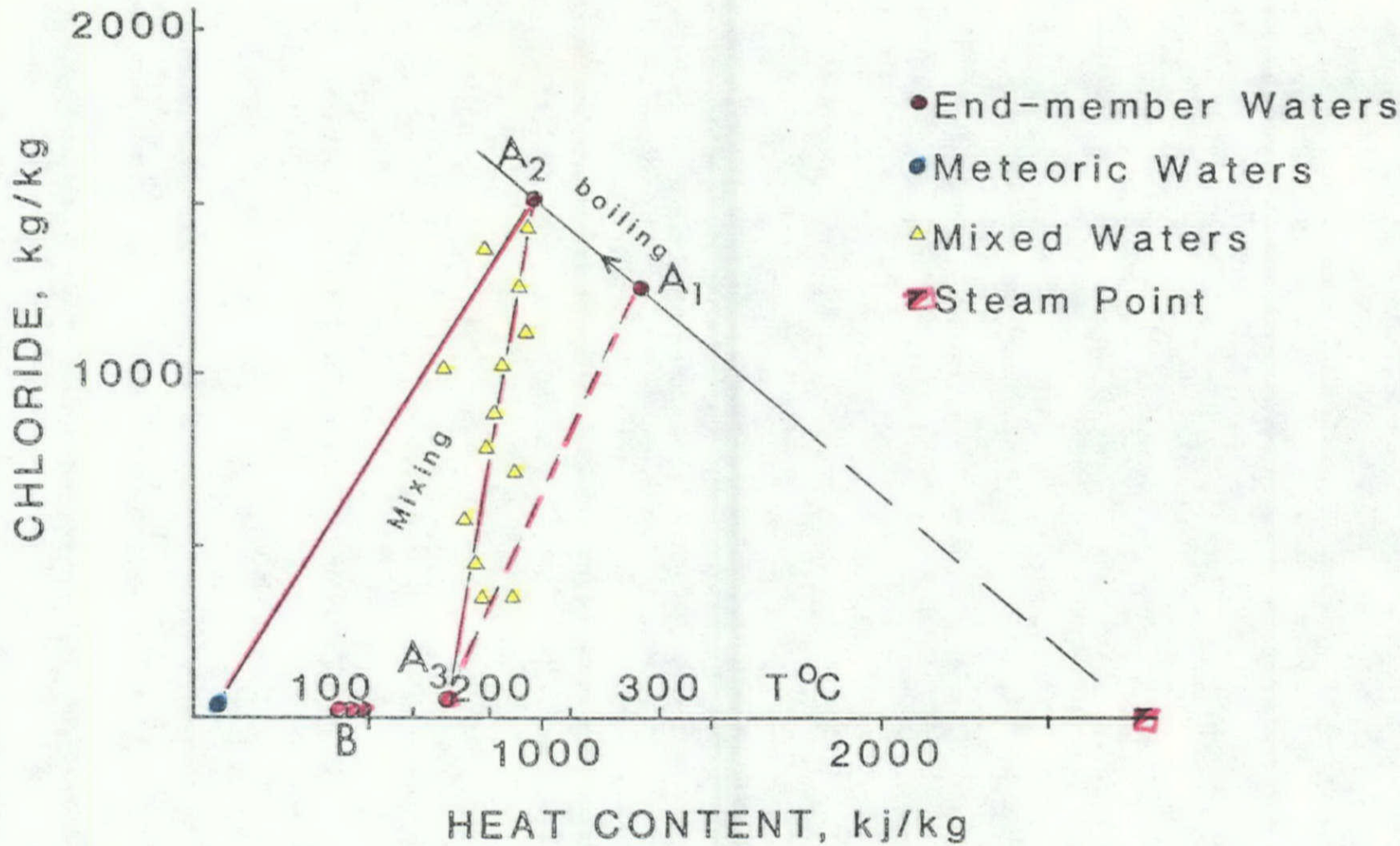
Courtesy R. W. Henley



Longitudinal Section Waiotapu

SLIDE 18

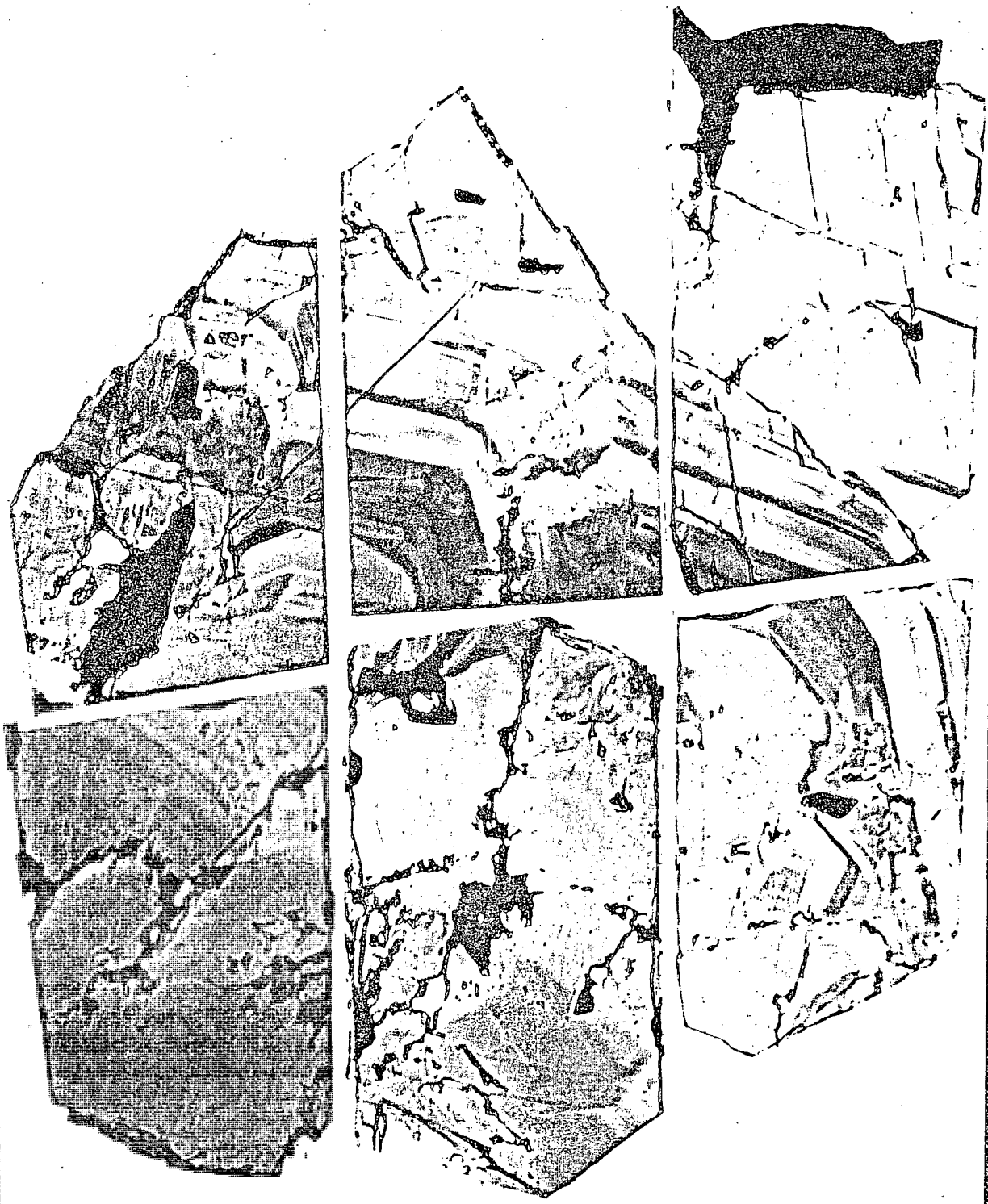
after Hedenquist, 1983



MIXING DIAGRAM WAIOTAPU

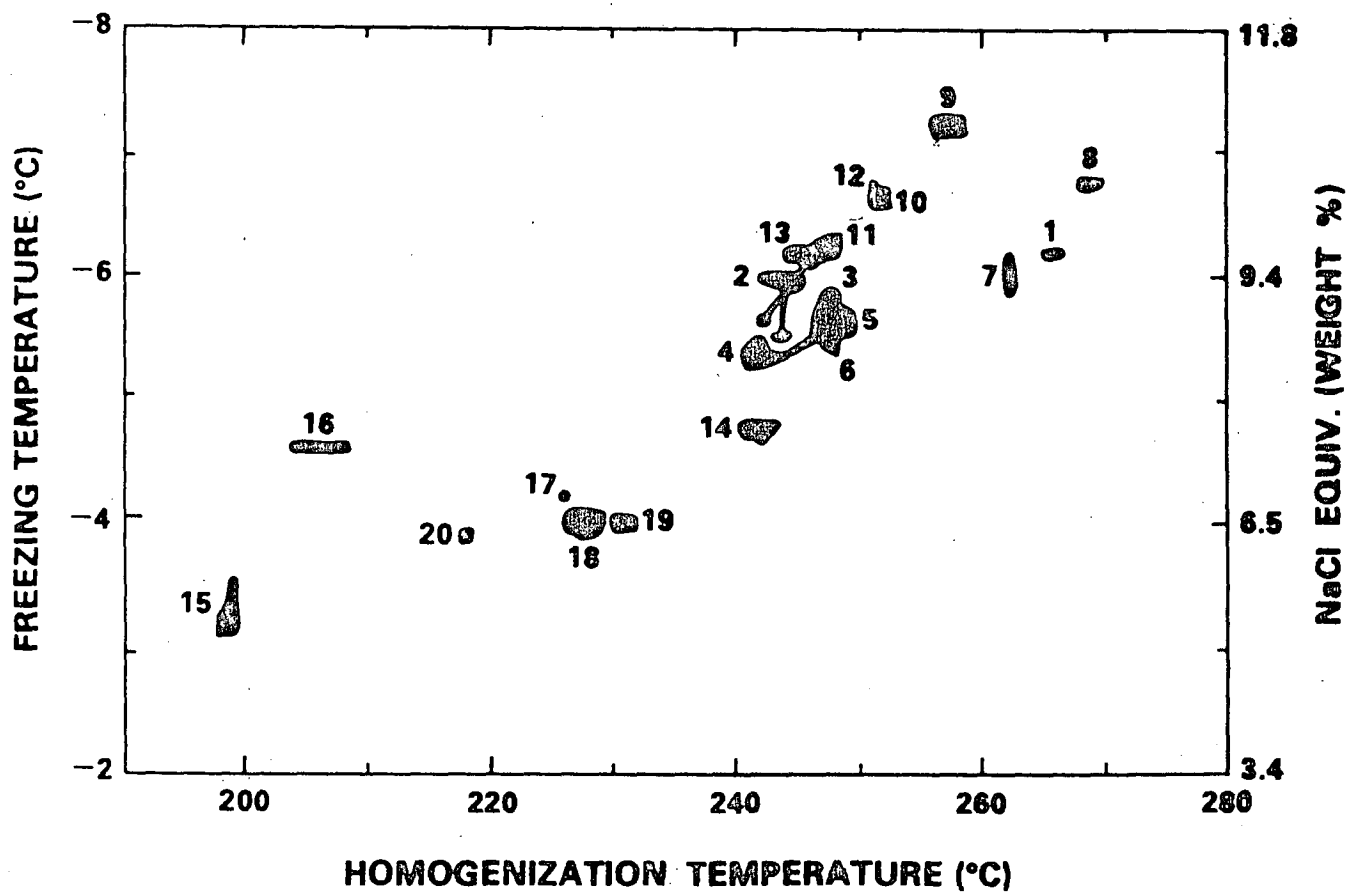
SLIDE 19

modified after Hedenquist, 1983



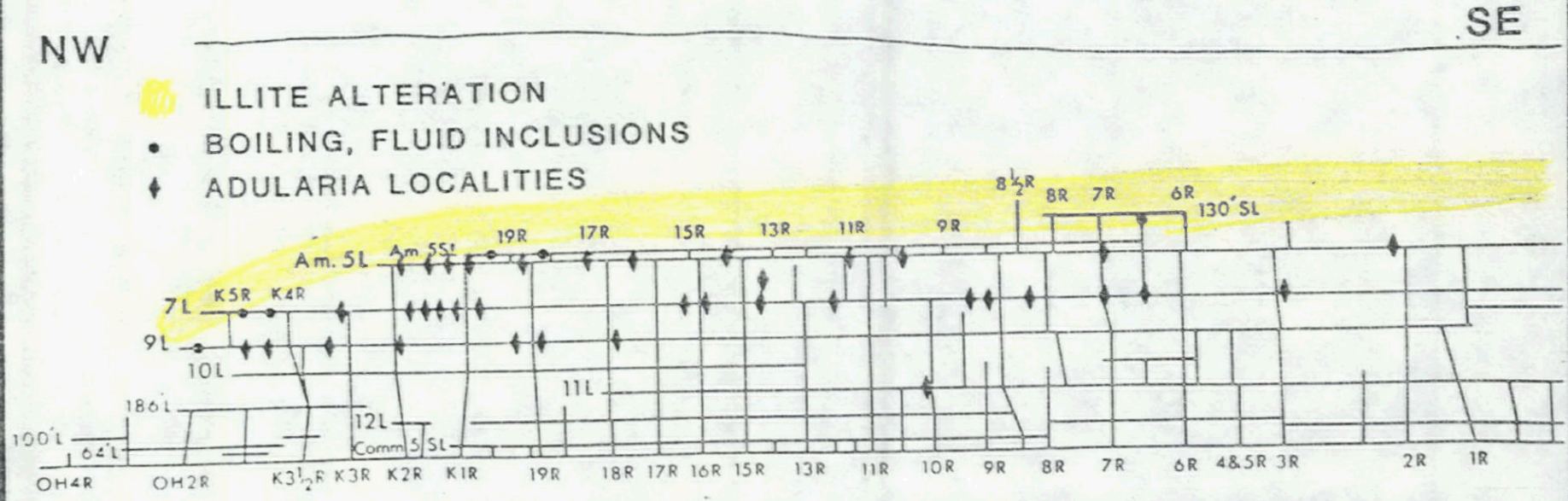
1cm

SLIDE 20



From Roedder, 1977

SLIDE 21

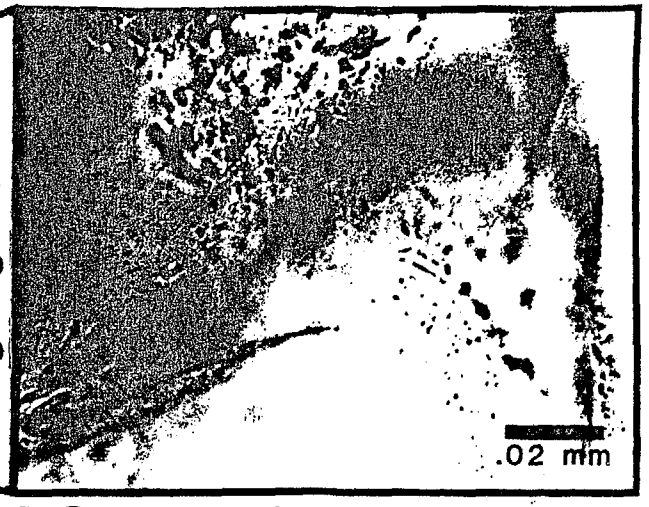


LONGITUDINAL SECTION – OH VEIN, CREEDE, COLORADO

SLIDE 22

PMB-BY-244-65

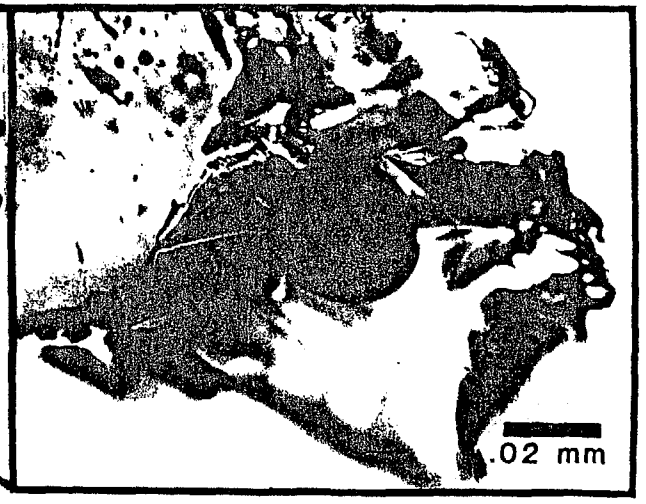
177 - 270°C
0 - 0.5 WT%
δD = -100‰



PSEUDOSECONDARY

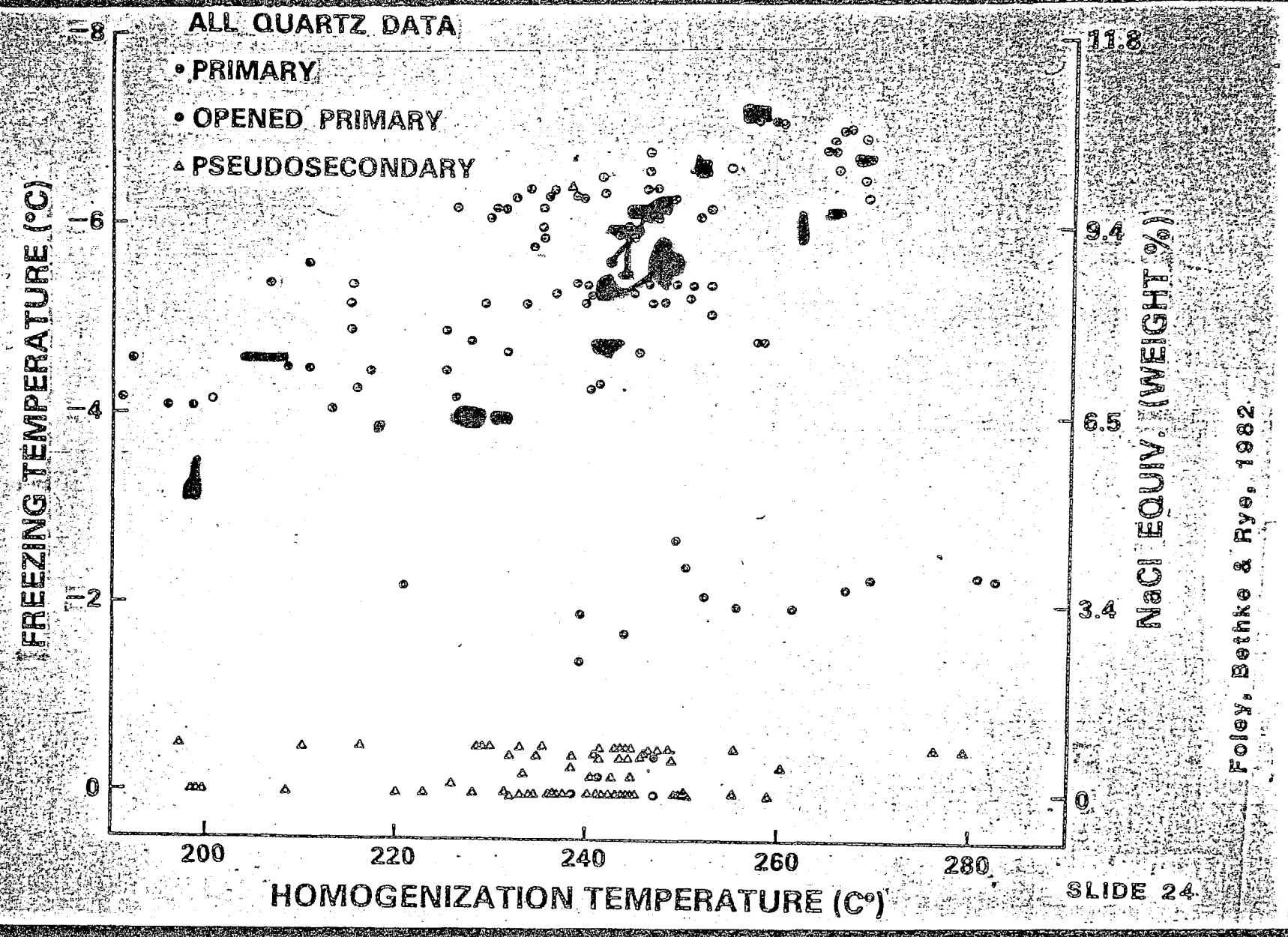
PRIMARY

190 - 276°C
4 -12 WT%
δD = -70‰



SLIDE 23

Foley, Bethke & Rye, 1982



Foley, Bethke & Rye, 1982

SLIDE 24