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NEVADA BUREAU OF MINES & GEOLOGY
REPORT 36

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

NEVADA BUREAU OF MINES & GEOLOGY

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NORTHERN NEVADA SECTION OF AIME

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1983

# CONTENTS

*in order of presentation at Symposium*


## GEOLOGY

**Gold deposits at Copper Canyon, Lander County, Nevada,** *by David W. Blake, Duval Corp.; and Edward L. Kretschmer, Pinson Mining Co.* . . . . . 3

**Sedimentology as applied to the exploration of fossil placer deposits,** *by James M. Prudden, Prudden Geoscience Services* . . . . . 11

**Geology and ore deposits of the Taylor mining district, White Pine County, Nevada,** *by Stuart R. Havenstrite, Silver King Mines* . . . . . 14

**Geology of the Cinola deposit, Queen Charlotte Islands, British Columbia,** *by Michael G. Cruson and Fred W. Limbach, Cruson & Pansze; Robert A. Brooks, Energy Reserves Group, Inc.; and Kenneth G. Sanders, Duncan Bain, and Steve Lacy, Consolidated Cinola Mines, Ltd.* . . . . . 27

**Bulk-tonnage, low-grade silver deposits—update 1980,** *by Barry N. Watson, U.S. Borax* . . . . . 36

**The Ortiz gold deposit (Cunningham Hill)—geology and exploration,** *by Alan Wright, Consolidated Gold Fields* . . . . . 42

**Styles of low-grade gold mineralization in volcano-plutonic arcs,** *by Richard H. Sillitoe, Consultant* . . . . . 52

## HEAP LEACHING

**Some factors in the design of heap leaching operations,** *by George M. Potter, Consultant* . . . . . 69

**Heap leaching and pilot testing of gold and silver ores,** *by Paul D. Chamberlin, Occidental Minerals Corp.* . . . . . 77

**Leaching practices at Smoky Valley Mine,** *by C. Arthur Lefler, Copper Range Co.* . . . . . 84

**Percolation leaching of clayey gold-silver ores,** *by Harold J. Heinen, Heinen-Lindstrom Associates, Inc.* . . . . . 87

## PLACER DEPOSITS

**Legal problems of placer locations,** *by Richard W. Harris, Attorney at Law* . . . . . 92

**Some notes on placer investigation,** *by John H. Wells, Consultant* . . . . . 98

**Evaluation of Tertiary gravels in the Sierra Nevada,** *by James J. Hodos, Placer Service Corp.* . . . . . 107

## MILLING

**Operation of an optical ore sorting system,** *by Daniel D. McLaughlin, Black Pine Mining Co.* . . . . . 111

**Application of the Lamella™ Gravity Settler/Thickener in precious metal recovery plants,** *by Russell L. Cook, Parkson Corp.* . . . . . 118

**Metallurgical practices at Homestake, Lead operation,** *by Harold L. Hinds and Larry Trautman, Homestake Mining Co.* . . . . . 124

## SAMPLING AND GEOSTATISTICS

**Calculation of sample requirements and the development of preparation procedures for reliable analyses of particulate materials,** *by Derek J. Ottley, Mineral Systems, Inc.* . . . . . 132

**Practical points of bulk material sampling,** *by Harrison R. Cooper, Harrison R. Cooper Systems, Inc.* . . . . . 145

**Sampling and ore reserve estimation for the Ortiz gold deposit, New Mexico,** *by Mark Springett, Gold Fields Mining Corp.* . . . . . 152

**Geostatistics at the feasibility stage, Golden Sunlight deposit, Montana,** *by A. J. Sinclair, University of British Columbia; E. T. Lonergan and W. E. McConechy, Placer Development, Ltd.* . . . . . 165

# PREFACE

The 1980 Precious-Metals Symposium was held in Sparks, Nevada, November 17–19, at John Ascuaga's Nugget Convention Center. The meeting was the result of endeavors of the Northern Nevada Section of the AIME; the Nevada Division of Mineral Resources, Department of Conservation and Natural Resources; and the Nevada Bureau of Mines and Geology. Aside from these organizations, many individuals involved in Nevada's mineral industry contributed their time and efforts. Major credit goes to Joyce Hall, then chairman of the AIME Section and Administrator of the Nevada Division of Mineral Resources, Department of Conservation and Natural Resources. Without the conscientious work of Paula Fieberling, secretary at the Division of Mineral Resources at the time, registration problems would have been insurmountable.

A few of the presentations are not included in this report for various reasons. The papers may not have been written; they may be in the process of publication by a governmental unit, or publication may not have been desirable for the author or his superiors.

John Schilling, Director of the Nevada Bureau of Mines and Geology, offered to print the papers and therefore made this publication possible. His help, and the help of the staff of NBMG, is greatly appreciated by the organizations directly involved as well as by those in the mineral industry who have patiently awaited the release of this report. The first step in the publication process was editing the papers. Those asked to assist were more than willing to help, and I take personal responsibility for the longer than normal delay in transmitting the edited copy to the Nevada Bureau of Mines and Geology. Special thanks are due Susan Tingley, NBMG Publication Manager, who advised in the editing process and arranged the typesetting.

Again, I apologize for the delay in the publication of these papers and sincerely appreciate the patience of the many people who have waited for this report.

Victor E. Kral, *Editor*
March, 1983

# GOLD DEPOSITS AT COPPER CANYON, LANDER COUNTY, NEVADA

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## 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.

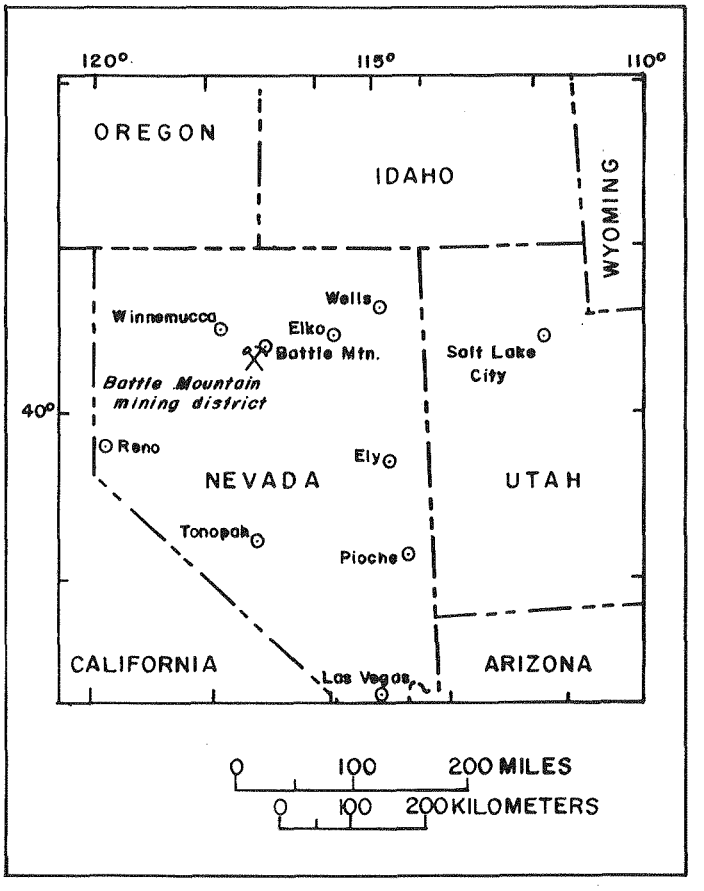



FIGURE 1.

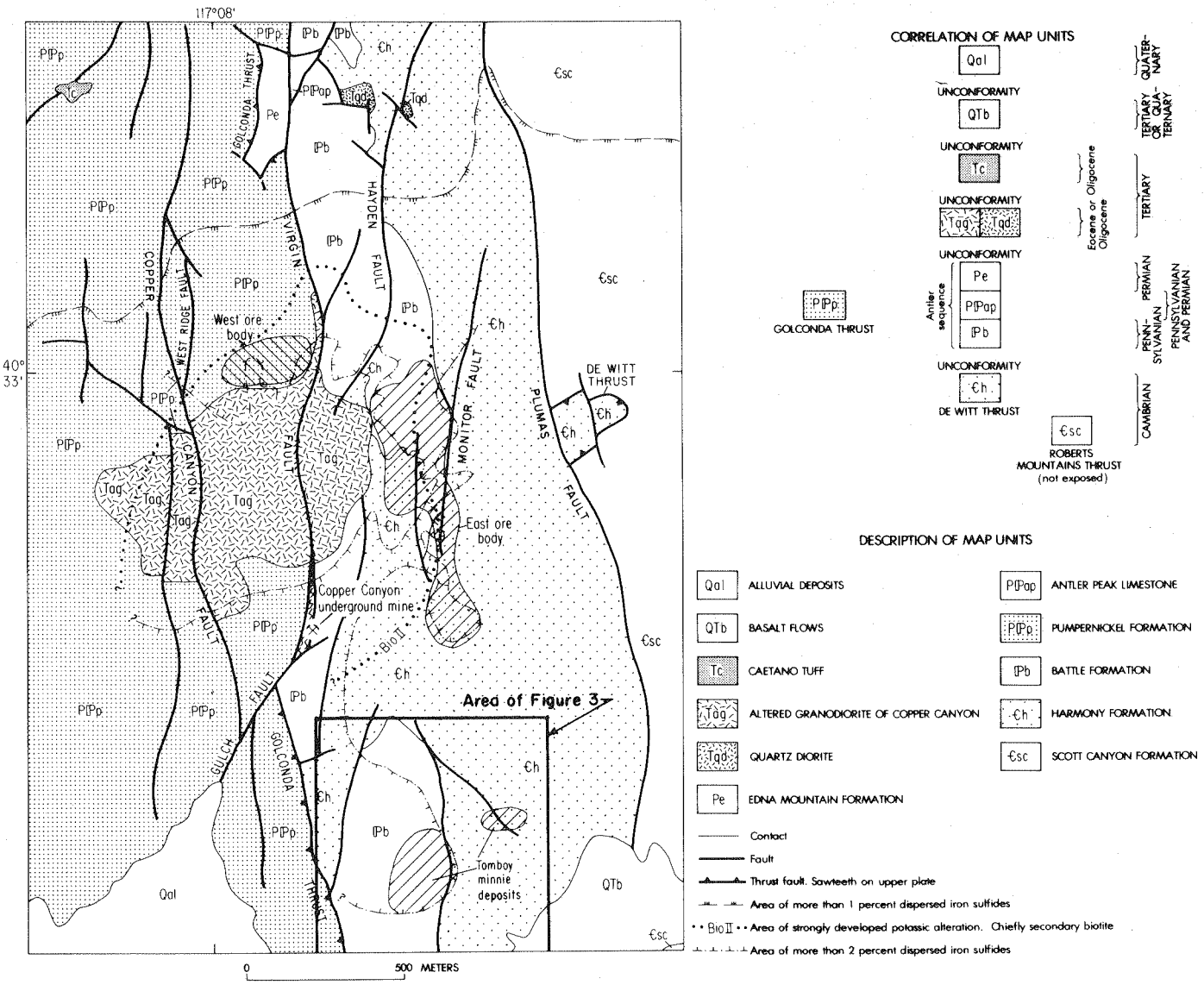



**FIGURE 2. Geology of the Copper Canyon area.**

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

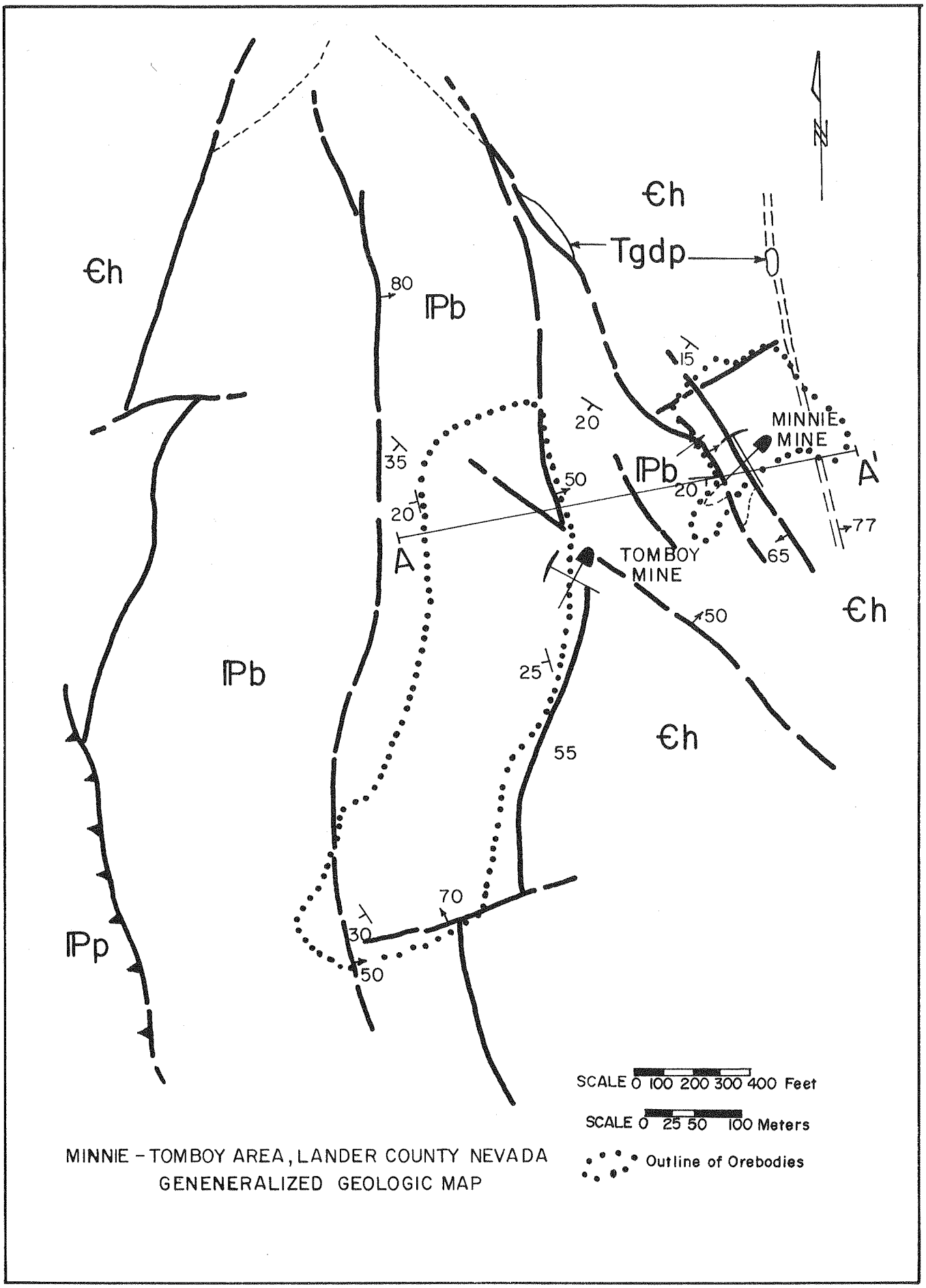
Duval Corporation started producing gold ore from the Tomboy and Minnie open pits in late 1978. Since that time the geologic setting for these two deposits has been studied using standard pit-mapping techniques. Detailed studies pertaining to the genesis of these gold deposits have only recently been 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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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.**

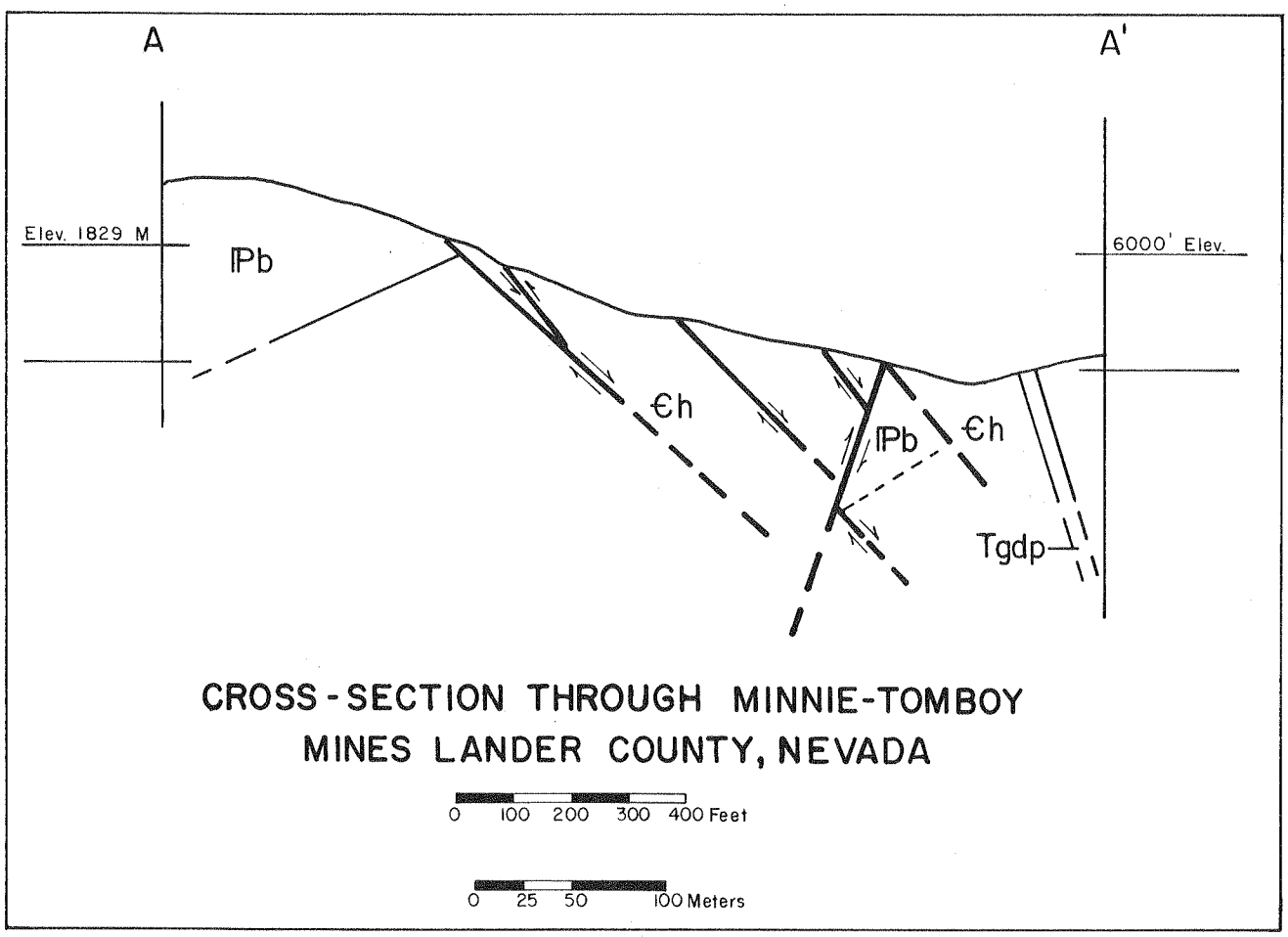



FIGURE 4.

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.

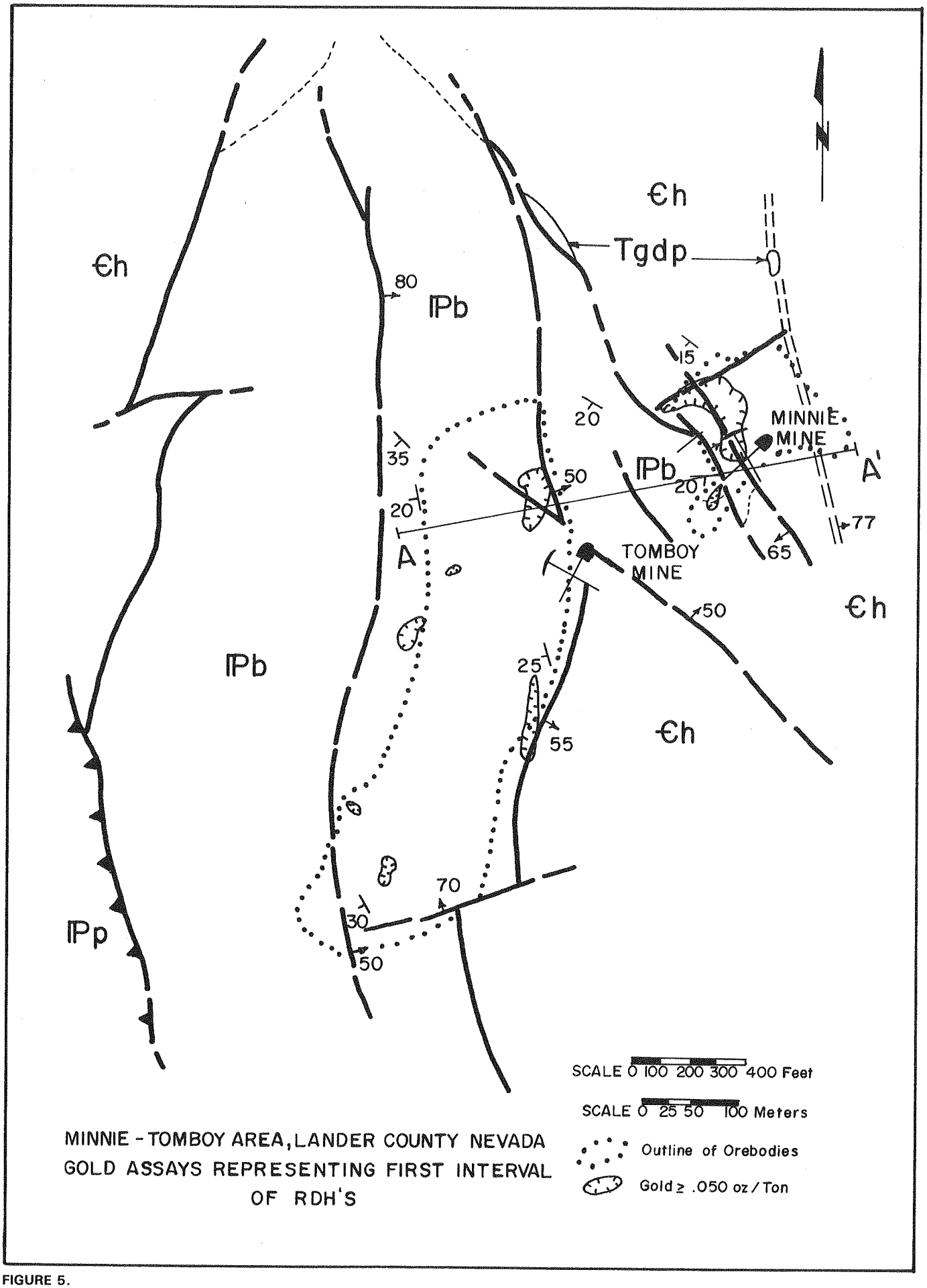
Thermal metamorphism of the conglomerate has produced a tremolite-bearing siliceous conglomerate. Hydrothermal alteration associated with gold deposition is reflected in the replacement of tremolite and other metamorphic products by epidote, chlorite and clay minerals. Epidote also occurs in fractures. 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.

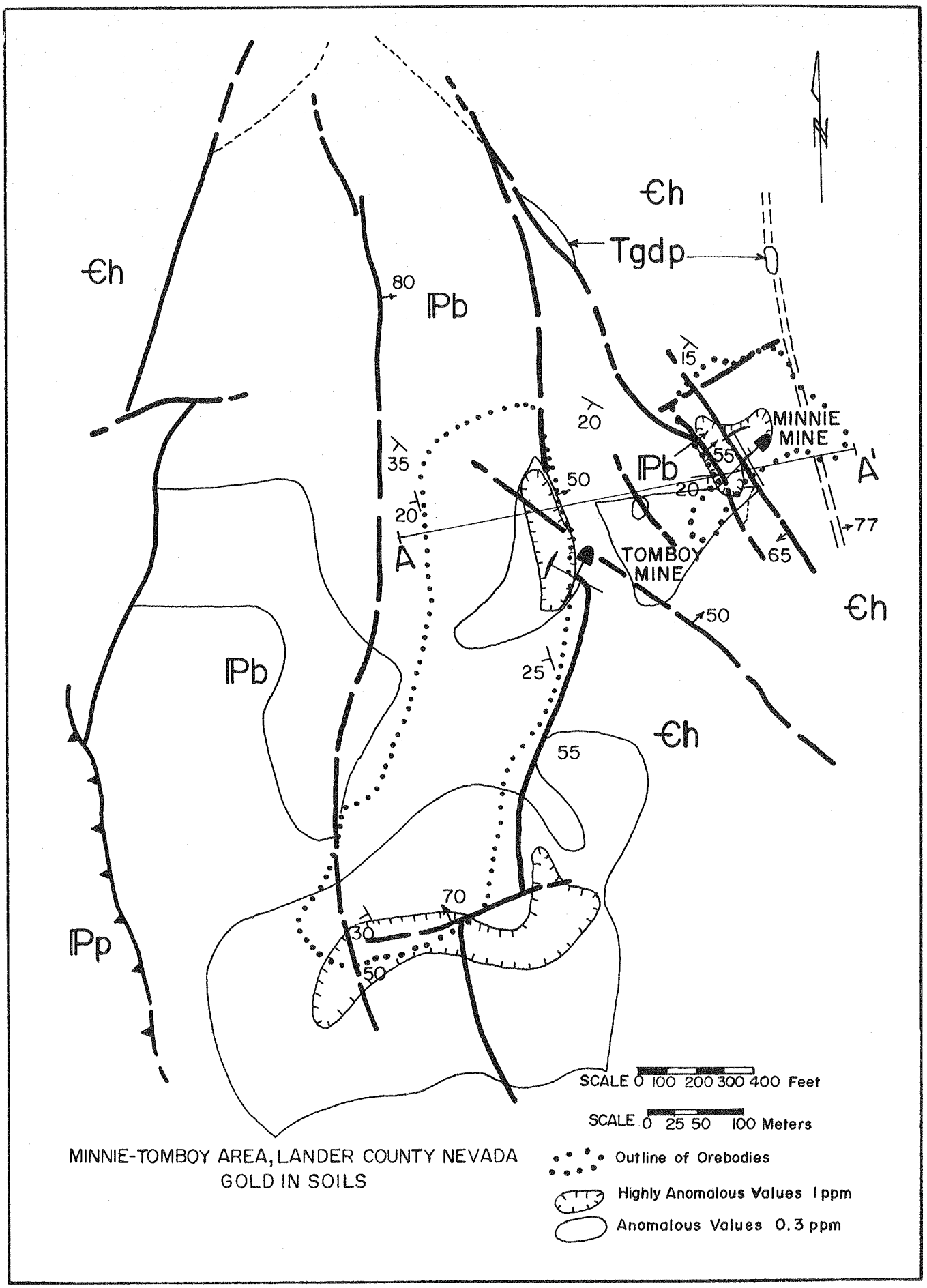
Gold concentrations greater than .050 oz/ton (1.7 ppm) clearly reflect the two deposits based on rock material assayed from the first interval of drilling (Figure 5). The broken north-south elongated pattern of the gold distribution in the Tomboy orebody reflects primarily the exposed basal Au-bearing 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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# SEDIMENTOLOGY AS APPLIED TO THE EXPLORATION OF FOSSIL PLACER DEPOSITS

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## ABSTRACT

The Witswatersrand Basin in South Africa is the most famous sequence of fossil placer deposits, having produced 55% of all mined gold. This geological model possesses the required sedimentological signatures fundamentally necessary to evaluate economic ore deposits in other Precambrian and younger geological settings. Provenance area, basin structure, external and internal geometry of the host sediments and sedimentary history of the enclosing basin are the major components of the formation of Precambrian fossil placer deposits. Younger fossil placer ore deposits must also contain the above ingredients to form the necessary concentration of detrital minerals. The proposed Precambrian climatic change from a reducing to an oxidizing atmosphere had a dramatic effect on the genesis of sedimentary uranium deposits. However, the nature of detrital gold deposits remains primarily unchanged throughout geological time. Consequently, the Witswatersrand is considered the geological model by which contemporary and younger detrital gold deposits may be evaluated.

Sedimentology must play an integral role in the evaluation of any placer deposit. The sampling problems associated with particulate gold are only too well known and cause—at times—insurmountable evaluation problems. Consequently, properly directed and executed sedimentological evaluation of carefully selected clastic sediments is a reliable method for the exploration and development of potential auriferous horizons.

Geological exploration for any ore type is best oriented by using a well-defined geological model. It is my proposal that the Witwatersrand Basin in South Africa be such a model. This series of clastic horizons have produced 2.8 billion tons of ore averaging 10 grams/ton or an estimated *55 per cent of ALL* mined gold (Papenfus, J.A. 1968). Contemporary and younger clastic horizons generally conform to the Witwatersrand sedimentation pattern.

The schematic diagram in Figure 1 represents the geological model for this famous deposit picturing a portion of the closed yoked basin, consisting of an active fault bounded margin(s) and a geochemically favorable provenance or source area being actively eroded by braided streams during a long, humid climatic cycle. These streams formed active coalescing alluvial fans vigorously reworked in a progressively shrinking basin. Offshore winnowing by lacustrine currents, as illustrated by the skewed shape of the alluvial fan base, constitutes a subordinate, but important variation to this fluviatile environment. The degree of sediment reworking and consequent heavy mineral concentration would be dependent on the interaction of local geologic and geomorphologic factors. The duration of these events would partially govern the payability of these fluviatile clastic sediments as formed and reworked in this braided-stream system.

This model would be very similar to the East Rand gold field where the mid-fan dimensions would be 40 kilometers, fan base 90 kilometers and the longitudinal or down-fan axis would be 40 kilometers. The East Rand has produced almost one billion tons of ore averaging 8.5 grams per ton gold from 27 major mines over a period of 75 years (Whiteside, H.C.M. 1968).

There are several other well-known Proterozoic clastic sequences which have payable detrital gold and/or uranium mineralization deposited in similar braided-stream environments. The variabilities within and between each deposit are more dependent on local source area configurations, geological structures and geomorphological environment than their gross dissimilarities with the Witswatersrand model. To take this important observation one step further, current sedimentation features are widely used in the evaluation of these ancient gravels, implying that the study and usage of clastic sedimentation transects geological time (Sestini G. 1973; Goss, W.H. 1968; and Bateman, J. D. 1958).

There are four basic parameters involved in effective basin analysis fully utilizing the effectiveness of our geological model. The first category is provenance area which would be the prime prerequisite for the formation of an auriferous placer deposit. Greenstone belts commonly portray the highest geochemical levels of gold and commonly host narrow hydrothermal quartz veins frequently mined for gold. The type area would be the Archean Swazi System (Barberton Mountain area) in South Africa (Viljoen, R. P. et.al. 1969). However, other metamorphic terrains are also sources for auriferous sediments. Examples would include the gneissic terrain in the Jacobina area of Brazil (Bateman, J. D. 1958), the quartz-veined Paleozoic schists at Manhattan, Nevada (Ferguson, H. G. 1917) and also the California Mother Lode (Clark, W. B. 1970 and Moore, L. 1968). Granitic and granite gneiss terrains with pegmatite swarms would be the logical source for particulate uranium minerals in the case of Lower Proterzoic uraniferous placer deposits.

Gross basin structure and sedimentation characteristics are interrelated. Our model is typically a large basin with tectonically active margins supplying detritus in concentrated troughs known as yoked basins. In this model, terrestrial sedimentation in a humid climate supplies vast quantities of sediment from the elevated source bordering the basin to be concentrated within a braided-stream environment

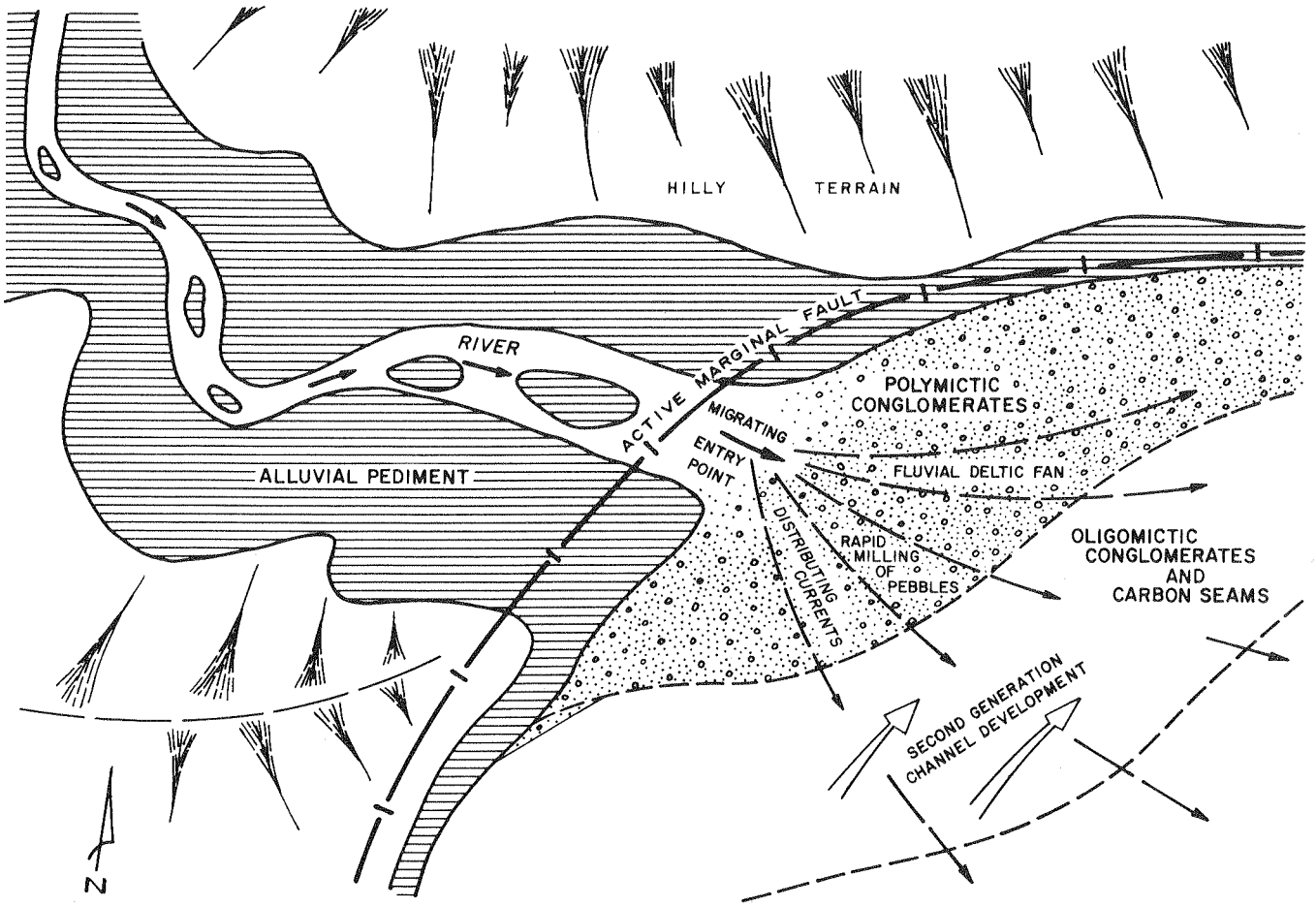


**FIGURE 1. Schematic diagram illustrating braided stream sedimentation patterns.**

in the form of coalescing fluvial fans. A further example would be the Basin and Range fluviatile sedimentation formed during the Pleistocene in the western United States.

A regressive sedimentation pattern, resulting from rejuvenated basin margin tectonics, provides the energy necessary to prolong the source erosion and also the reworking of earlier sediments. It has been estimated that a conglomerate grading 0.12 parts per million averaging 290 meters thick and outcropping for 1700 meters at a 10° dip *downstream* will liberate 810 kilograms of gold per one meter of vertical erosion (Papenfus, J. A., 1968). Hence, continual basin margin uplift of an auriferous provenance area in alternating transgressive-regressive cycles during a humid climate will eventually produce economic concentrations of heavy minerals. Offshore reworking of the alluvial fan base by offshore lacustrine-marine currents provides an important heavy mineral concentration phase to this ore deposit model.

After the explorationist has confirmed a favorable provenance and lengthy fluviatile sedimentary history necessary to concentrate economic heavy minerals, the external geometry of the sediments must then be evaluated. Essentially, this involves the classification of the basin sedimentation history into various cycles to ascertain the most favorable zone(s) for detailed examination. The most useful parameters are:

1. Percentage conglomerate within defined strategraphic zones
2. Delineation of braided-stream channels and associated fluvial fans
3. Transgressive—regressive cycles in the stratigraphic column
4. Gross paleocurrent trends.

Some idea can thus be gleaned as to the tectonic activity of the basin margin during sedimentation and the size and gross economic potential of the clastic sedimentation sequence.

The next phase in the geological evaluation of a potential economic clastic sequence is selecting which specific horizon(s) have the most potential for developing viable fossil placer deposits. Each clastic sedimentary unit must be evaluated by combining numerous internal geometry aspects. The more useful of these detailed measurements are:

1. Thickness of the individual sedimentary cycle
2. Conglomerate thickness, which can also be presented as a percentage of the total sedimentary cycle presented as isopach maps

3. Systematic pebble counts, obtained from residual weathering products, outcrops, underground exposures and drill core. These critical factors will measure roundness, pebble lithology, sorting and packing. This raw data can then be plotted to produce percentage of clast lithology, mean pebble size, standard deviation, distinguish variances in provenance areas, predict paleo channel confluences and in general delineate paleoslope gradients within the braided-stream system
4. Paleocurrent measurements as cross-bedding and pebble imbrication provide the necessary directional vectors to clasify and evaluate the broader sedimentological projections mentioned above.

Processing these measurable parameters will determine the relative tonnage potential of each horizon and can be used as an empirical technique to establish exploration priorities within the various clastic units.

The above systematic approach to the evaluation of conglomerate horizons might appear long winded and academic. However, a concerted continent-wide gold exploration program requires a specific geological model and the methodology in selecting and evaluating exploration targets. Sampling for particulate gold, with the best of intentions, can be so subjective that a placer exploration program could be doomed to failure before it commences. However, the selection of exploration targets on well-formulated geological parameters assures a cost-and time-effective exploration project enjoying the highest success potential.

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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-

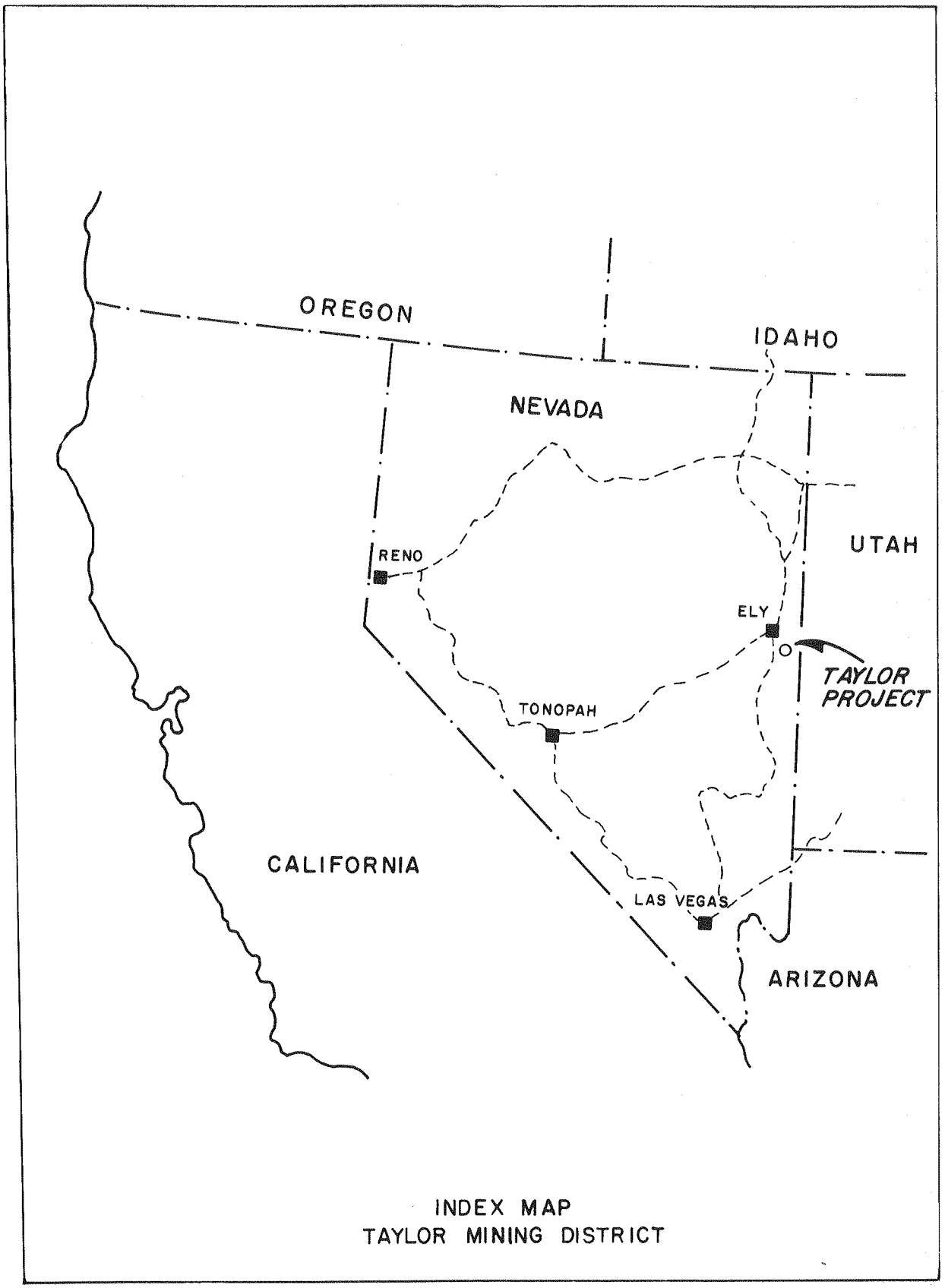
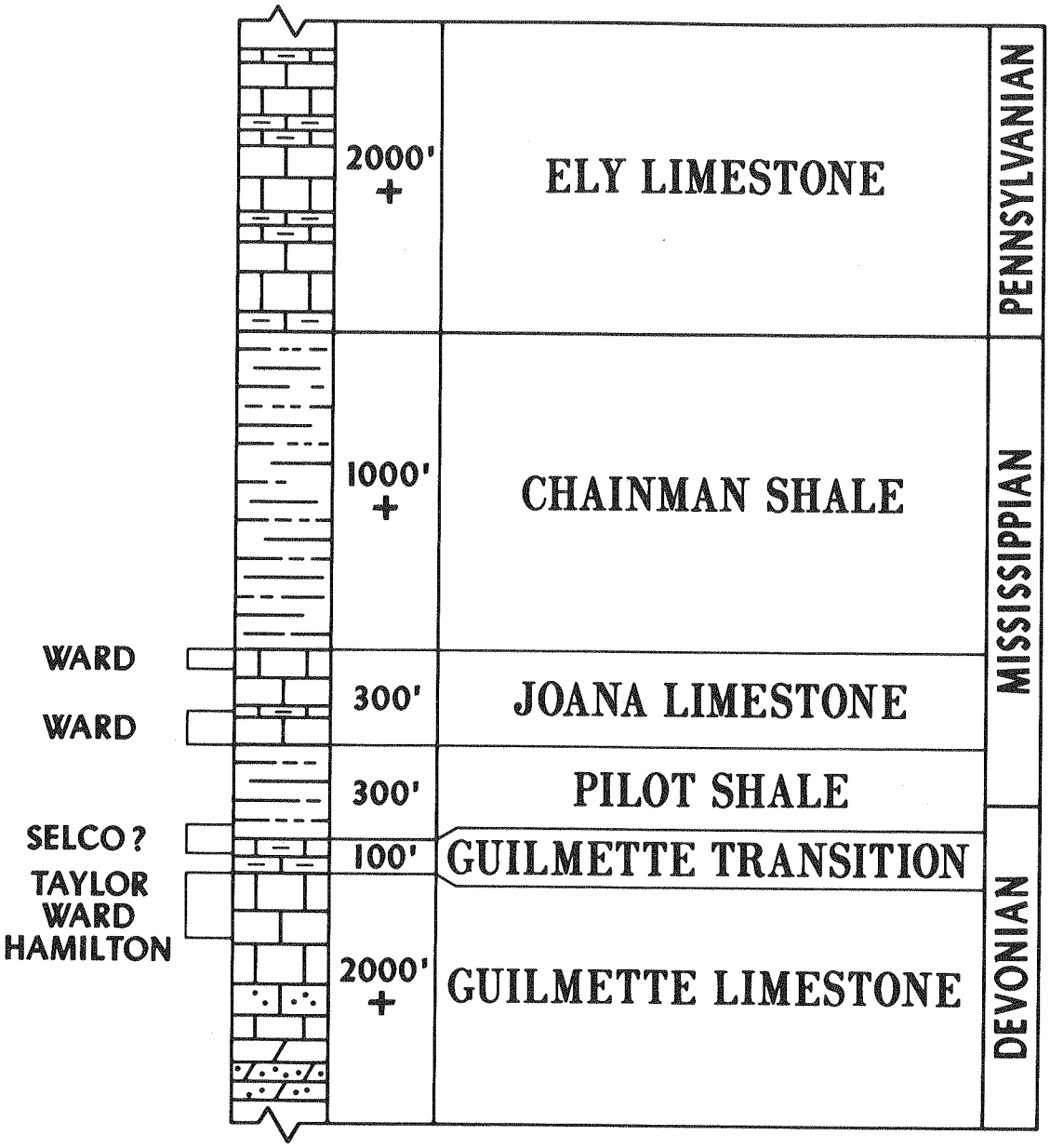

OREGON
IDAHO
NEVADA
UTAH
RENO
ELY
TAYLOR PROJECT
TONOPAH
CALIFORNIA
LAS VEGAS
ARIZONA
INDEX MAP
TAYLOR MINING DISTRICT

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.

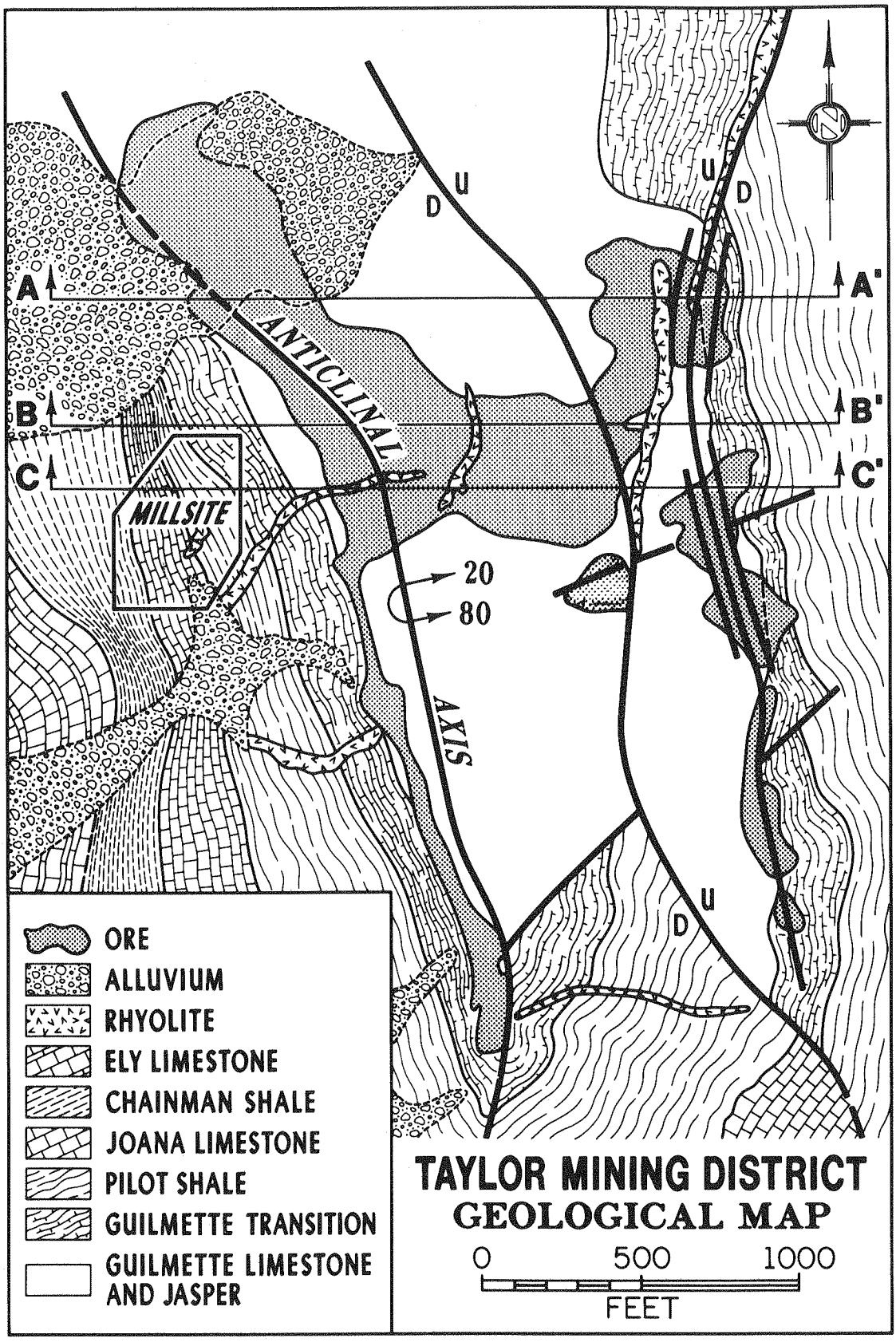
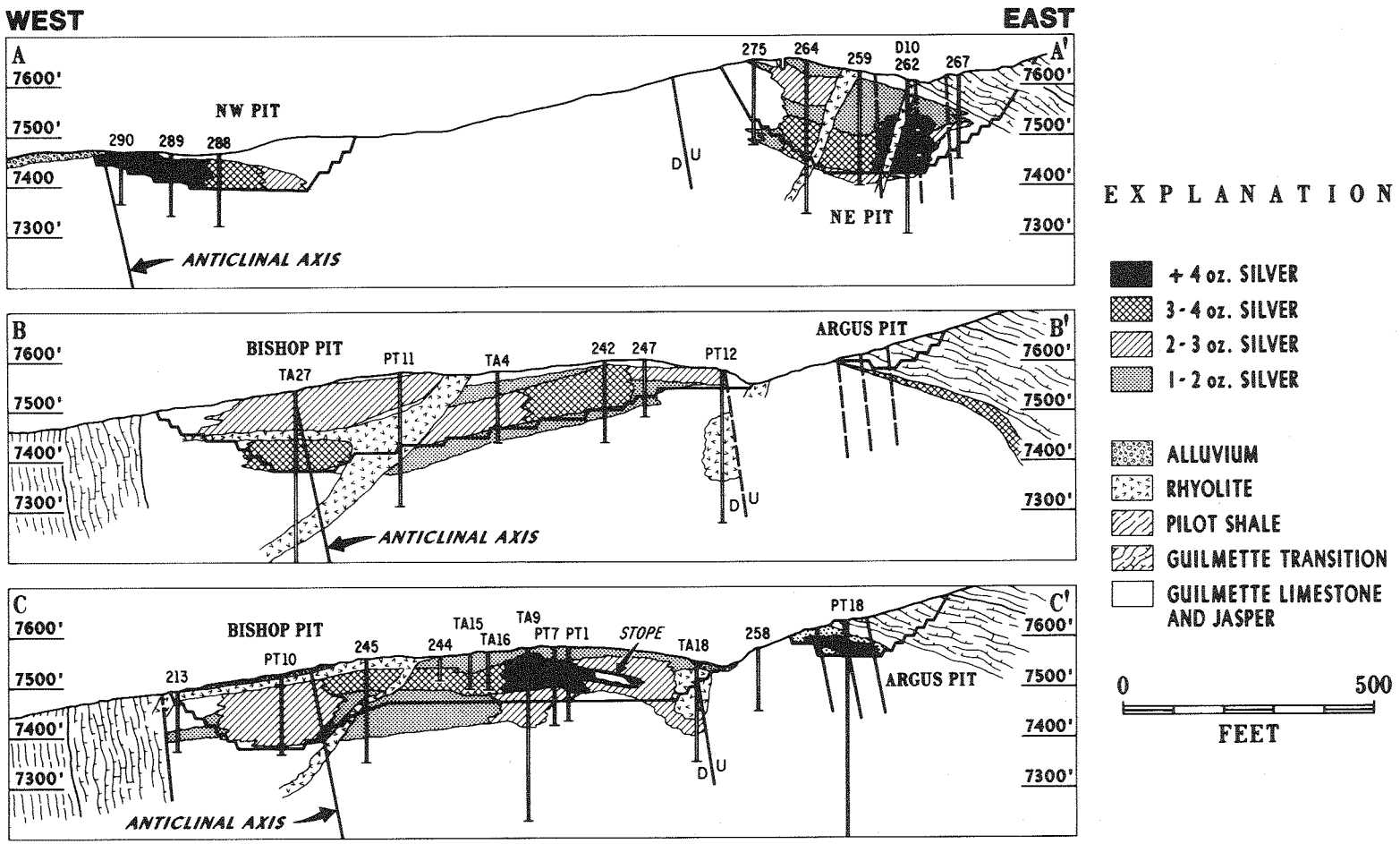

ANTICLINAL
AXIS
MILLSITE
A
A'
B
B'
C
C'
U
D
20
80
ORE
ALLUVIUM
RHYOLITE
ELY LIMESTONE
CHAINMAN SHALE
JOANA LIMESTONE
PILOT SHALE
GUILMETTE TRANSITION
GUILMETTE LIMESTONE AND JASPER
TAYLOR MINING DISTRICT
GEOLOGICAL MAP
0
500
1000
FEET

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

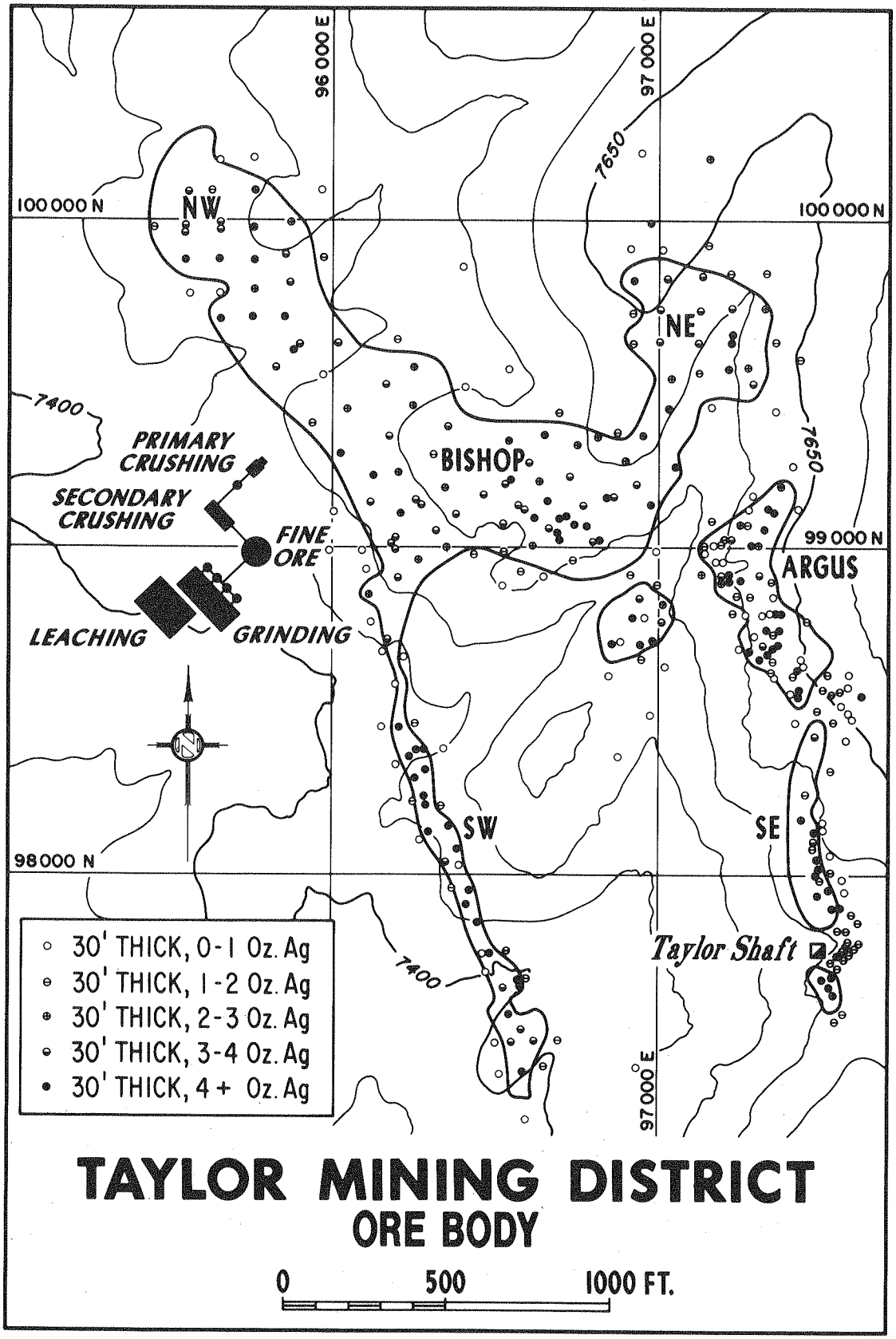
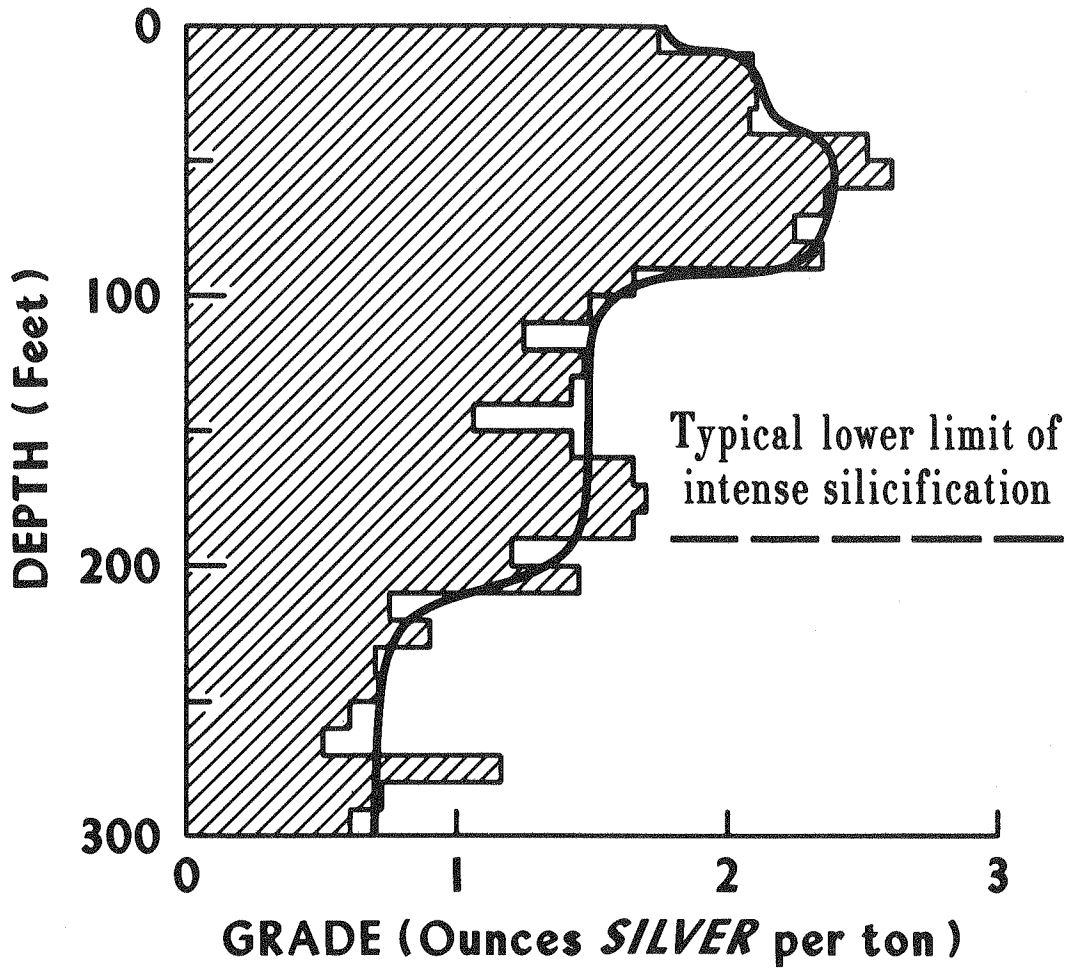

96 000 E
97 000 E
100 000 N
99 000 N
98 000 N
97 000 E
NW
NE
SW
SE
BISHOP
ARGUS
7650
7400
PRIMARY CRUSHING
SECONDARY CRUSHING
FINE ORE
LEACHING
GRINDING
Taylor Shaft
30' THICK, 0-1 Oz. Ag
30' THICK, 1-2 Oz. Ag
30' THICK, 2-3 Oz. Ag
30' THICK, 3-4 Oz. Ag
30' THICK, 4 + Oz. Ag
TAYLOR MINING DISTRICT
ORE 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

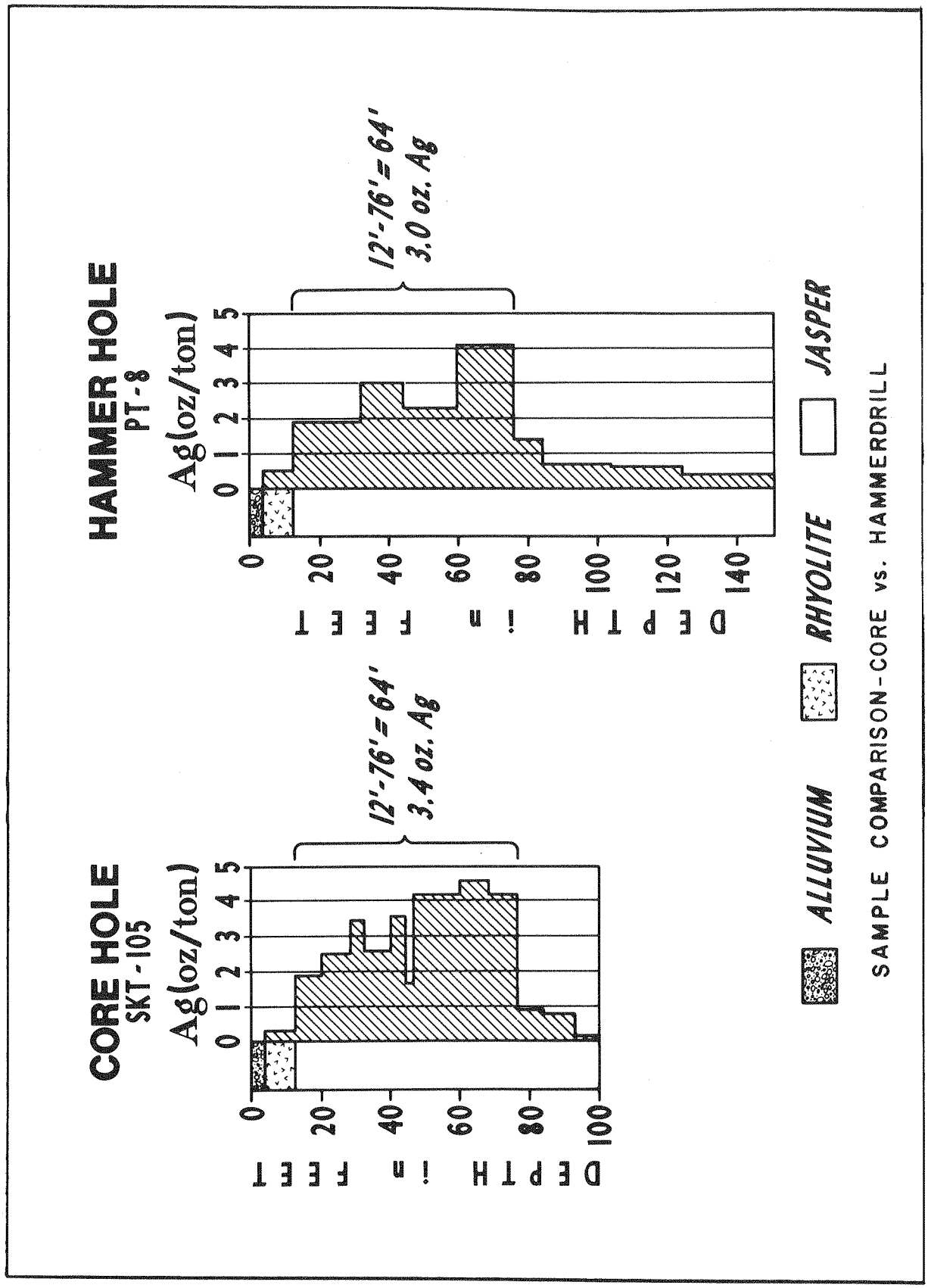

HAMMER HOLE
PT-8
Ag(oz/ton)
0 1 2 3 4 5
12'-76' = 64'
3.0 oz. Ag
DEPTH in FEET
0 20 40 60 80 100 120 140
CORE HOLE
SKT-105
Ag(oz/ton)
0 1 2 3 4 5
12'-76' = 64'
3.4 oz. Ag
DEPTH in FEET
0 20 40 60 80 100
ALLUVIUM
RHYOLITE
JASPER
SAMPLE COMPARISON-CORE vs. HAMMERDRILL

FIGURE 7.

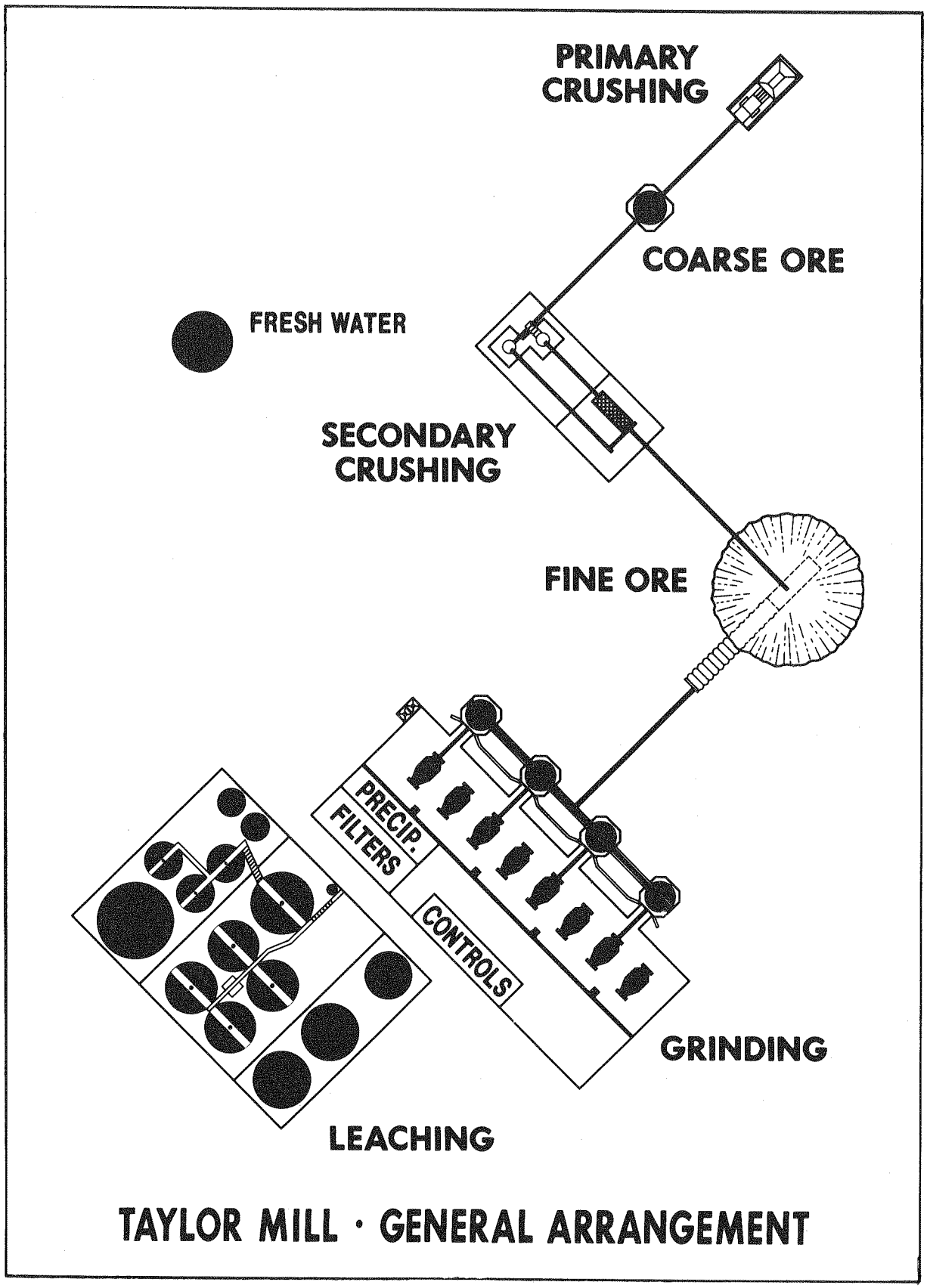

PRIMARY CRUSHING
COARSE ORE
FRESH WATER
SECONDARY CRUSHING
FINE ORE
PRECIP.
FILTERS
CONTROLS
GRINDING
LEACHING
TAYLOR MILL · GENERAL ARRANGEMENT

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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# GEOLOGY OF THE CINOLA DEPOSIT, QUEEN CHARLOTTE ISLANDS, BRITISH COLUMBIA

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**FRED W. LIMBACH** Cruson & Pansze, Geologists
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**DUNCAN BAIN** Consolidated Cinola Mines Ltd.
**STEVE LACY** Consolidated Cinola Mines Ltd.

## ABSTRACT

The Cinola deposit is a gold orebody located on Graham Island of the Queen Charlotte Islands, British Columbia. Exploration by several different companies has been carried out since the discovery in 1970. Recent drilling has delineated a large tonnage, low grade gold deposit in a unique geologic setting. The deposit is found in Tertiary intrusive subvolcanic felsic rocks and volcaniclastic sediments that have been intensely brecciated and silicified. The mineralization is bounded at its base and on the west side by a younger normal fault that probably had substantial lateral movement.

Gold mineralization is closely associated with silicification and spatially associated with the felsic intrusive rocks. The mineralization occurs in a quartz stockwork zone above a near surface rhyolite intrusive. Two modes of gold mineralization are present: thick, moderate grade disseminations and thin, high grade veins. Most of the gold is less than 0.5 microns in size but free gold may be visually observed in quartz veins. The sedimentary stratigraphy has no apparent control on the gold mineralization. Similarly, the gold content is not directly related to the pyrite or carbon content of the host rocks. The only sulfides of significance in the deposit are pyrite and marcasite. Other sulfides have been reported but occur in only trace amounts. Alteration consists of silicification and argillization. Reserves at the Cinola gold property are in excess of 45 million tons containing 0.054 oz gold per ton; significant gold has been encountered in drilling outside the area containing the reported reserves and additional surface exploration targets are present.

## INTRODUCTION

### PURPOSE AND LOCATION

The purposes of this report are to describe the geology of the Cinola gold deposit and present a genetic model. The Cinola gold deposit, formerly known as the Babe or Specogna deposit, is located at lat 53° 32′ N., long 132° 13′ W. The deposit is in the south central part of Graham Island in the Queen Charlotte Islands, British Columbia (Fig. 1). The deposit occurs near the border between the Skidegate Plateau and the Queen Charlotte Lowlands on a small hill with an elevation of 90-220 m. The area is generally heavily timbered and the climate is mild but wet.

### EXPLORATION HISTORY

Champigny, Sinclair and Sanders (1980) have prepared an excellent report on the discovery and history of exploration of the deposit. In summary, the property was located by two local prospectors in 1970 and was optioned to a succession of companies through 1977. These early companies did extensive geochemical soil sampling, trench sampling and moderate amounts of drilling. Consolidated Cinola Mines Ltd. optioned the property in 1977 and excercised their option in 1978. In 1979 Energy Reserves Canada, Ltd. (a subsidiary of Energy Reserves Group, Inc.) formed a joint venture with Consolidated Cinola to explore and develop the prop-

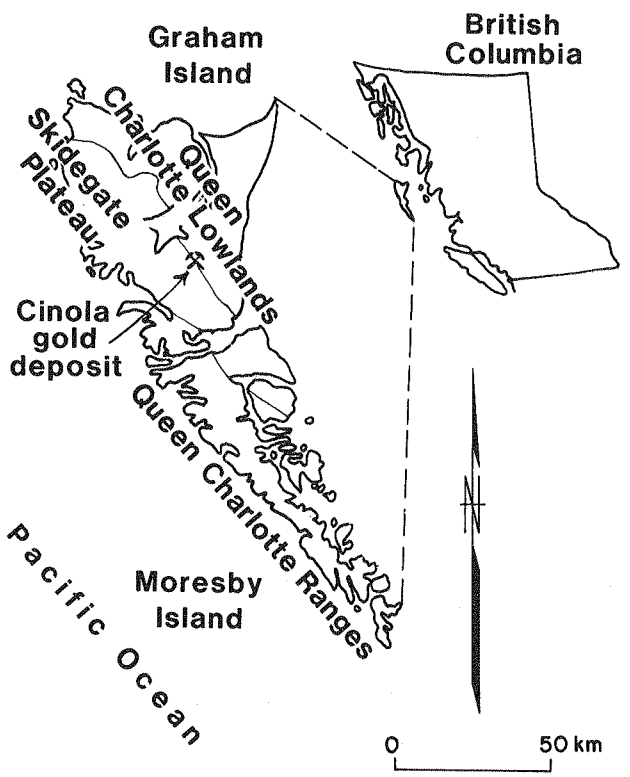


**FIGURE 1. Location map of Queen Charlotte Islands.**

erty. The joint venture, which became active in August of 1979, enlarged the higher grade ore reserves that had been discovered by Consolidated Cinola in late 1978 and early 1979. The dramatic increase in the price of gold during late 1979 and early 1980 strongly influenced the development decisions. Consolidated Cinola Mines, Ltd. is the operator for the joint venture.

The geology of the Queen Charlotte Islands is described by Sutherland-Brown (1968). Early reports on the Cinola deposit include Sutherland-Brown and Schroeter (1975 and 1977) and Richards, Christie, and Wolfhard (1976). Champigny and Sinclair (1979) have summarized the geology and exploration results prior to the formation of the joint venture.

The authors represent the principal geologic and administrative staff for the joint venture from August 1979 to the present. This included supervision of diamond core drilling and sample preparation. Over 13,700 meters of core were logged, split, and analyzed for gold. Supplemental work included relogging of old core and surface geologic reconnaissance. Contract and in-house laboratory work were carried out in support of the field operations. Principal laboratory methods included thin section analysis, microprobe scanning, and quantitative analysis.

## REGIONAL GEOLOGY

The Queen Charlotte Islands lie in the western system of the Canadian Cordillera within the Insular Fold Belt which contains late Paleozoic, Mesozoic, and Tertiary rocks. The Pacific continental shelf is narrow in this area and terminates a few miles west of the Islands. The Cinola property is located immediately west of the boundary between the Skidgate Plateau and the Queen Charlotte Lowlands. This physiographic break coincides with the Sandspit fault system (Fig. 2). The Sandspit fault can be traced across Graham Island for more than 60 km and has a displacement of thousands of meters with the down dropped block to the east (Sutherland-Brown, 1968). A fault, believed to be part of the Sandspit fault system, displaces the Cinola orebody (Fig. 3).

West of the Sandspit fault in the Skidgate Plateau, west-dipping rhyolite tuffs of the Masset Formation of early Tertiary age unconformably overlie folded sediments of the Haida Formation of Cretaceous age (Sutherland-Brown, 1968). East of the fault is the Queen Charlotte Lowland with limited exposures of unconsolidated clastics of the Mio-Pliocene Skonun Formation, the host for most of the Cinola gold orebody. Pleistocene glaciation has caused extensive modification of the plateau and thin till deposits mantle much of the lowlands and plateau.

There are numerous other gold occurrences on the Queen Charlotte Islands. A belt of gold mineralization at least 30 miles wide and 150 miles long contains over 70 gold occurrences. Some of these active gold prospects are shown in Figure 2.

## STRATIGRAPHY

The rocks in the immediate vicinity of the deposit have been divided into two broad groups: the footwall section and the hanging wall section. This is illustrated by Figure 4 and cross-sections A-A' and B-B' (Figs. 5 and 6.).

### FOOTWALL SECTION

The oldest rock unit found in the immediate vicinity of the deposit is a dark mudstone which is correlated with the upper member of the Cretaceous Haida Formation of the Queen Charlotte Group. The mudstone, which is not mineralized, forms the footwall fault block with the mineralized section. Only the upper 36 m of the mudstone has been penetrated by drilling and much of this section is sheared and fractured by the faulting.

The mudstone is massive with no visible indication of bedding. It is a soft, dark grey to dark brown mudstone that is carbonaceous and commonly calcareous. Irregular veinlets of white calcite less than 1.0 mm wide are common. Pyrite, that is likely syngenetic, occurs as disseminations and nodules in amounts up to 5 percent. For several meters within

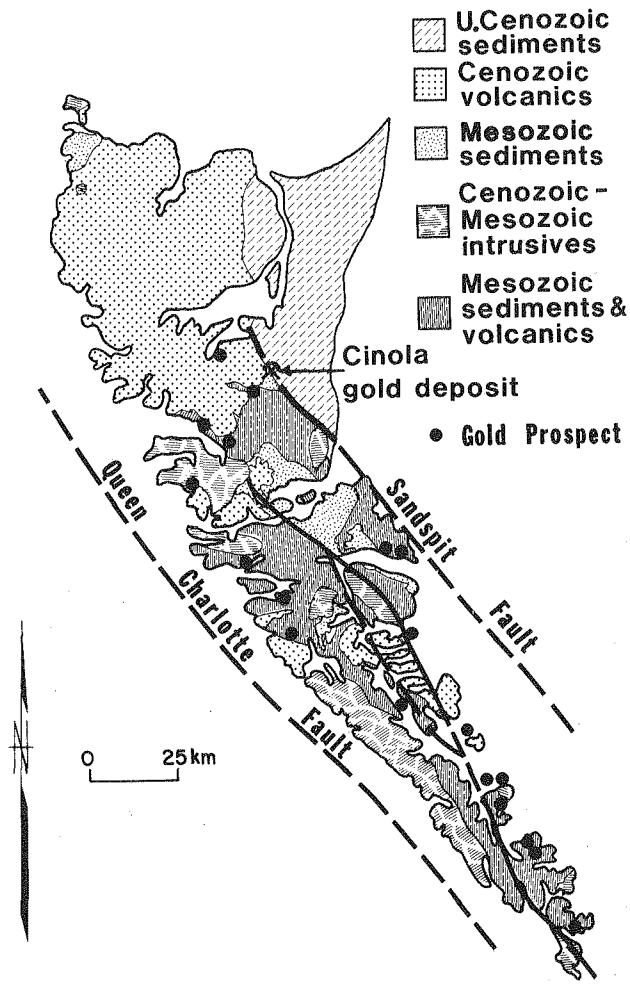


FIGURE 2. Regional geologic map.

the fault zone, the mudstone is silicified and brecciated with clear quartz veins. Within this zone, the rock has the appearance of an argillite.

According to Sutherland-Brown (1968) the Haida Formation was deposited in a Lower to Upper Cretaceous marine basin. The mudstone member of the Haida formation is at least 300 m thick.

Rhyolite porphyry is found in the footwall block intruding the Haida shale at the Marino showing and in Barbie Creek northwest of the deposit (Fig. 3). Identical rhyolite porphyry is found in the hanging wall block and is described in the section dealing with the hanging wall section.

## HANGING WALL SEDIMENTARY SECTION

The mineralized section on the hanging wall of the fault is a complex interfingering of coarse conglomerates and minor fine-grained clastics which are intruded by rhyolitic subvolcanic units. An extensive zone of brecciation, quartz veining, silica flooding, and argillic alteration obscures precise stratigraphic correlation of some of the units on the hanging wall.

The sedimentary section above the fault has been correlated with the Mio-Pliocene Skonun Formation based on lithology. Sutherland-Brown (1968) describes the Skonun as marine to nonmarine, poorly consolidated siltstones, sandstones and conglomerates that total over 1800 m in thickness. The

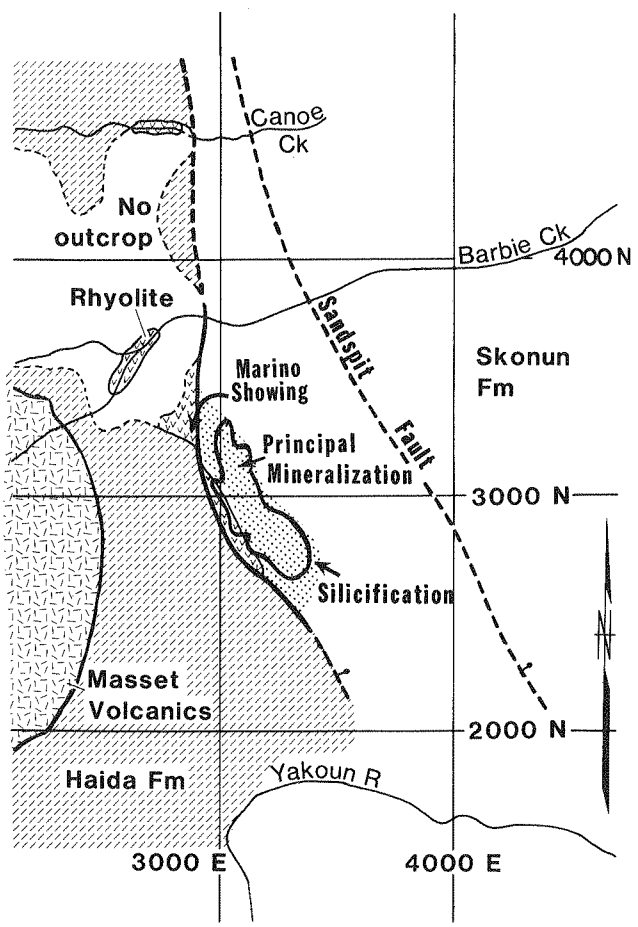


FIGURE 3. Geologic map of the cinola deposit (grid in meters).

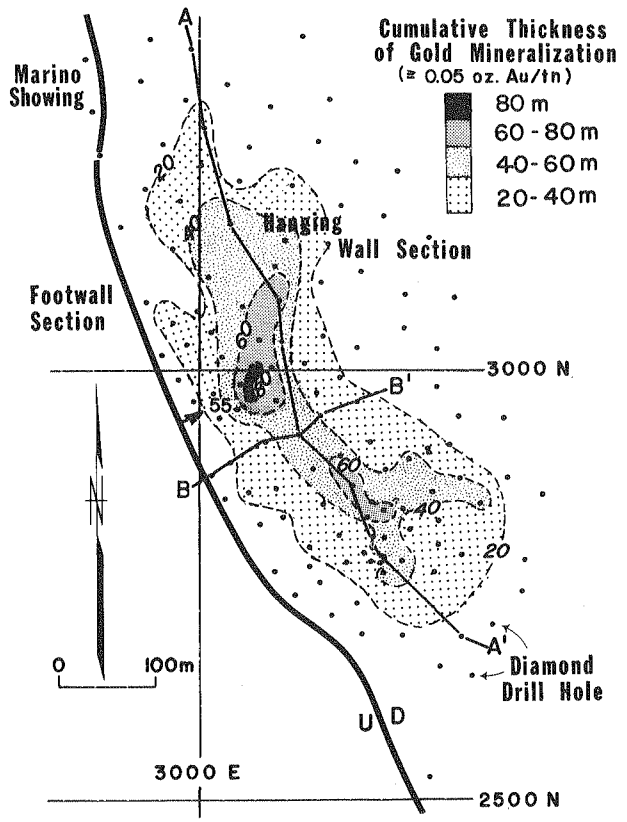


FIGURE 4. Cinola deposit, showing drill hole locations, cross section lines, and cummulative thickness of gold mineralization.

section penetrated by drilling is over 400 m vertical thickness and consists of approximately 20 percent mature fine clastics and 80 percent immature coarse clastics that generally dip gently to the northeast. The hanging wall sedimentary section can be divided into two broad units: a massive lower conglomerate and an upper conglomerate with interbedded sands and silts.

Coarse clastic sediments predominate in the lower parts of the deeper drill holes on the east side of the property. These sediments consist dominantly of tan, pebble to cobble size, rounded to subrounded fragments of aphanitic felsic volcanic rocks; but, boulder size clasts are not uncommon. A small percentage of the clasts are basic volcanic rocks, phaneritic plutonic rocks, and sedimentary rocks. The fragments are generally clast supported. Carbonized wood fragments and pyrite each make up less than 3 percent of the rock. No well developed bedding is present, but crude graded bedding is discernible in massive cobble-boulder conglomerate units greater than 60 m thick.

The upper portion of the sedimentary section consists of coarse clastic with interbeds of fine clastics. The dominant rock type is a tan to grey volcanic conglomerate composed of a sandy matrix with mostly felsic aphanitic volcanic fragments that are either clast or matrix supported. The clasts, which vary in size from 1-10 cm, make up 30-70 percent of the rock. The clasts are generally subrounded and often show a weathered or altered rind. Carbon fragments

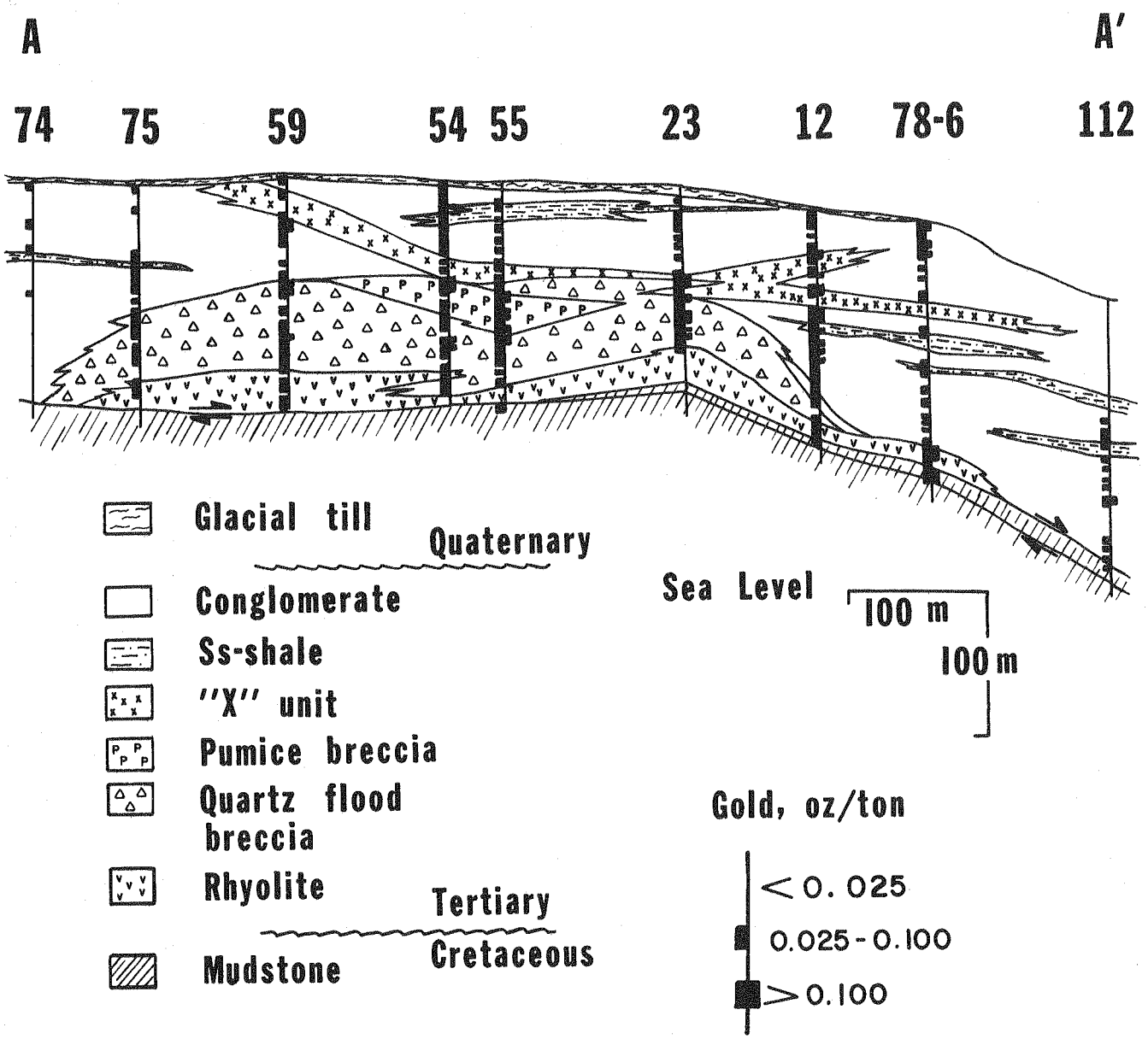


**FIGURE 5. Geologic section A-A′ with gold assay histograms through diamond drill holes.**

and pyrite are common constituents of this conglomerate. The conglomerates within the upper sedimentary section are up to 30 m thick but generally consist of fining upward sequences from conglomerates through sandstone to siltstone in thickness of 1-3 m.

Units that are interfingered with the upper conglomerate sequence and that are used for stratigraphic correlation between holes include a green pebble or mafic rich conglomerate, siltstones and sandstones, a crowded-pebble conglomerate and distinctive lithology termed the X-unit. The origin of the X-unit is uncertain; it may be a mud or debris flow or an intrusive pebble dike complex.

The green mafic conglomerate consists of a predominance of mafic volcanic pebbles. Most of the clasts have a distinct chlorite-epidote alteration. The subrounded clasts are 2-5 cm in size and generally show clast support. Carbon fragments and disseminated pyrite occur as common accessories. This mafic rich conglomerate is generally less than 5 meters thick and has a lenticular shape suggestive of a channel deposit.

Mature siltstones and sandstones that are used for correlations are grey to dark grey. (Tan siltstones and sandstones commonly cap the upward fining conglomerate sequences but these tan, fine clastics are generally not laterally continuous.) The grey siltstone and sandstone units are generally less than 10 m thick. These units are thin-bedded, well-sorted and organic-rich sediments. Ripple laminations are common in the sandstone. The sandstones are very fine to fine-grained with well-rounded grains. Pyrite is present in amounts up to 3 percent. Pelecypod

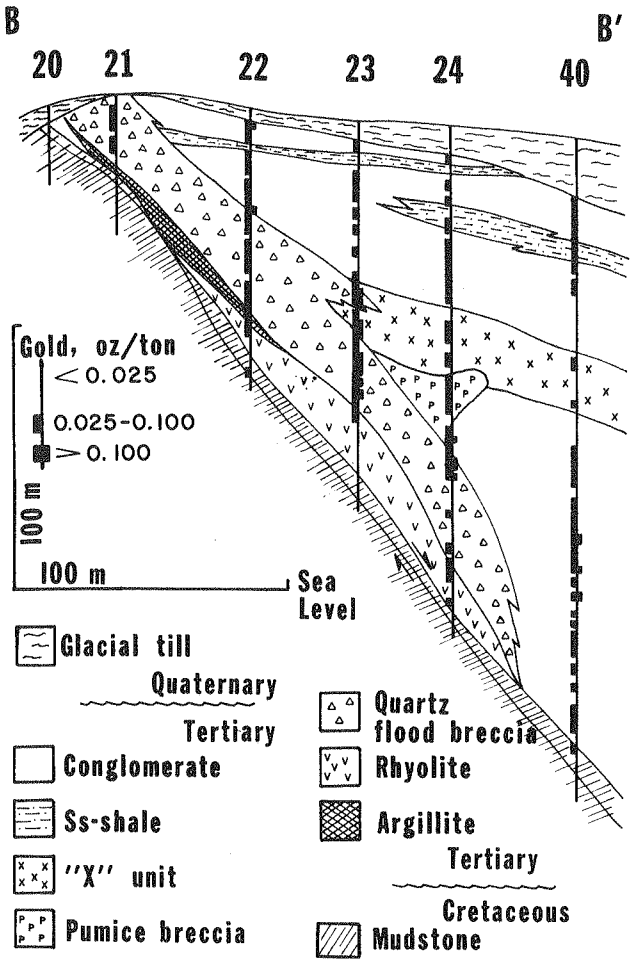


FIGURE 6. Geologic section B-B' with gold assay histograms through diamond drill holes.

shell fragments are found in a grey sandstone interbedded with a pebble conglomerate at the surface on the north end of the deposit. Minor glauconite is present in sandstone in a drill hole 200 m east of the mineralized zone. Both of these constituents are suggestive of a marine to nearshore environment for most of the sedimentary section. Deposition must have taken place during rapid sedimentation with periods of relative quiescence.

The crowded-pebble conglomerate and the X-unit are two closely related lithologies and have been combined on the cross-sections. Both are local units found above the rhyolite intrusive. The crowded-pebble conglomerate consists of 80-90 percent pea-sized clasts with very little matrix. The clasts are subrounded and are mostly aphanitic volcanic fragments. Carbon fragments and pyrite are again common constituents. The crowded-pebble unit commonly overlies the X-unit.

The X-unit has the lithology of a conglomeratic siltstone with a distinctive bimodal assemblage of constituents. It is a poorly sorted grey to greyish brown siltstone with pebbles of subrounded lithic clasts and angular carbon fragments evenly distributed in the siltstone matrix. The clasts, which are mostly volcanic, make up 10-20 percent of the rock and are completely matrix supported. Clay galls with a diameter of less than 1.0 cm make up a minor percentage of the clasts. Pyrite content is variable and can be up to 5 percent.

## HANGING WALL INTRUSIVE SECTION

A rhyolitic subvolcanic unit is found in both the foot wall and the hanging wall sections. The rhyolite phases are best developed on the hanging wall but rhyolite porphyry occurs on both sides of the fault. The rhyolite porphyry contains 5-15 percent phenocrysts set in a light grey aphanitic groundmass. The phenocysts are typically 1-2 mm in diameter and consist of equal amounts of anhedral quartz and altered, subhedral potassium feldspar. Disseminated pyrite makes up less than one percent of the rock.

In addition to the rhyolite porphyry, several other distinctive rhyolitic phases are present in the mineralized zone above the fault. One of these phases is a rhyolite porphyry that has been brecciated and veined with medium grey quartz veinlets. Several stages of quartz veining are evident. The quartz veinlets are commonly 2 cm in width and often have vugs which are partially filled with clear terminated quartz. Also present is a rhyolite phase that has a crackle breccia appearance. This unit, called a rhyolite stockwork, is a rhyolite porphyry that has been shattered and cross-cut by numerous veinlets of quartz with pyrite and hematite. The aphanitic groundmass is generally tan to light grey but also has a greenish cast from the presence of finely distributed chlorite. An isopach map of the combined thickness of the rhyolite units above the fault is shown in Figure 7.

Closely related to the rhyolite phases and occurring immediately above them are two other breccia units. The pumice breccia is a cream to pinkish tan rock which consists of finely, vesicular siliceous fragments, that resemble pumice in texture. Quartz veinlets are common and up to 5 percent disseminated pyrite is present. The pumice breccia may represent a frothy, volatile rich rhyolite phase or a siliceous sinter deposit above the rhyolite body (Fig. 7). Occurring with the pumice breccia and generally forming the contact zone between the rhyolite phases and the sediments is a unit termed the quartz flood breccia. The quartz flood breccia consists of brecciated and silicified sediments, with minor pyrite, that have had much of their original texture destroyed. The quartz flood breccia is believed to represent the contact zone between the rhyolite intrusion and the sediments. Thin pebble or breccia dikes (less than 1.0 m thick) have been observed in this unit and also may be present in the normal sedimentary section where they are more difficult to recognize. A complex swarm of these dikes may overlie the pumice breccia and the thickest portion of the rhyolite.

## STRUCTURE

### BEDDING

The lenticular shape of many of the stratigraphic units make structural interpretation difficult. Since no key marker beds have been recognized, the fine-grained clastic units are used to determine bedding attitude. The beds making up the mineralized section strike N5-10W and dip 20° to the northeast. No folding within the mineralized section has yet been recognized. The attitude of the mudstone in the footwall has not been determined.

### FAULTING

The major structural feature of the property is the fault that separates the silicified, gold bearing rocks from the mudstone. The fault zone consists of a 1-20 m zone of fault gouge separated by sheared and broken rock. The footwall section contains numerous slickenside surfaces on brecciated fragments of mudstone. There are also thin (0.2-3.0 m) fault slivers of non-silicified rhyolite porphyry faulted in with the mudstone.

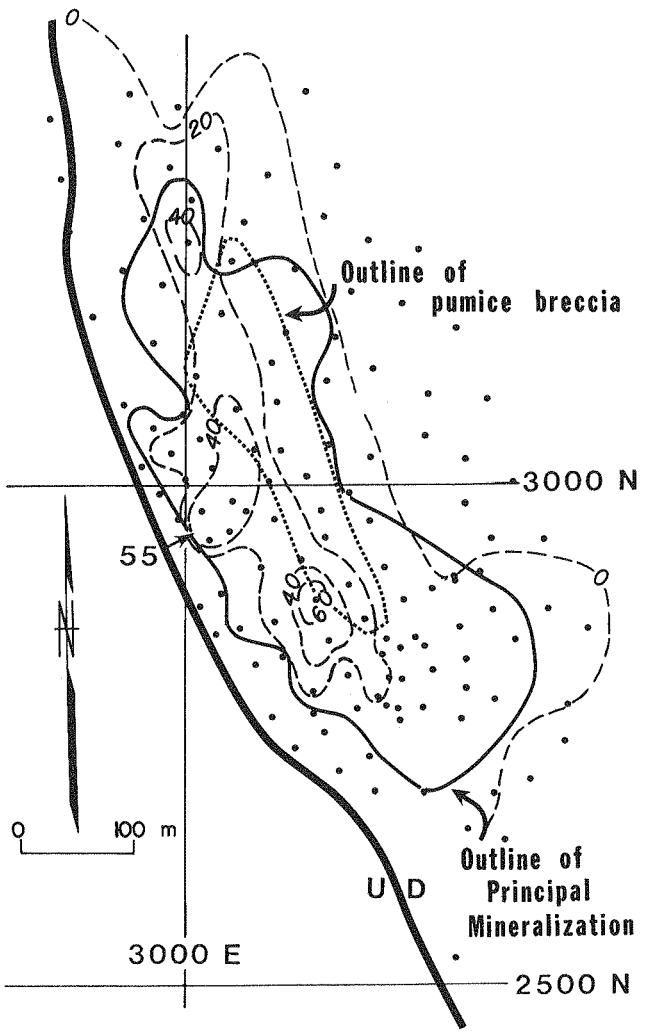


**FIGURE 7. Isopach map of rhyolite above the fault.**

The fault zone strikes N15-20W and dips 55° to the northeast (Fig. 8). The amount of displacement has not been determined from core logging. Slickenslide striations in several holes indicate a major strike slip component. This fault is part of the major Sandspit fault that is traced across much of southeastern Graham Island. Regional studies suggest the system is a normal fault with some horizontal displacement (Sutherland-Brown, 1968).

The faulting appears to be, in part, post-mineralization. The fault system may have controlled the location of the rhyolite intrusion initially, but more recent faulting has displaced the orebody. Limited amounts of deep drilling to the east side of the property indicate that the rhyolite is thin or not present to the east. Mineralized rhyolite porphyry is found below the fault to the northwest of the principal mineralization at the Marino showing, suggesting that this area may be the root zone of the rhyolite intrusion and the mineralization (Fig. 3).

## ALTERATION

### SILICIFICATION

The rocks in the central portion of the Cinola deposit are moderately to strongly silicified. All rock types above the fault, including the rhyolite porphyry, have been subjected to silicification and veining. The strongly silicified sections often have a boxwork type of texture where much of the original rock has been leached away. The silica veining occurs in widths of 1 mm to greater than 2 m. Silica veining is of several generations with the color ranging from dark grey, light grey, white, brown and clear. The paragenetic sequence is complex and has not been clearly established. The veins are generally massive and only the clear or latest stage of quartz has open space filling.

The silicification is strongest above the rhyolite units and pumice breccia. The quartz flood breccia unit is an extreme alteration phase in which the original rock texture has been obliterated by brecciation, silicification and silica veining. The silicification grows weaker away from the rhyolite units and grades into a halo of advanced argillic alteration as shown in Figure 8.

### ARGILLIZATION

The advanced argillic alteration is typically developed in the sediments and equally affects both the fine and the coarse clastics. Within the conglomerates both the matrix and the clast are equally altered. The advanced argillic alteration causes an almost complete change to a light tan clay.

## MINERALIZATION

### SOIL GEOCHEMISTRY

Figure 9 is a regional geochemical contour map of gold in soils from the Cinola deposit and outlying

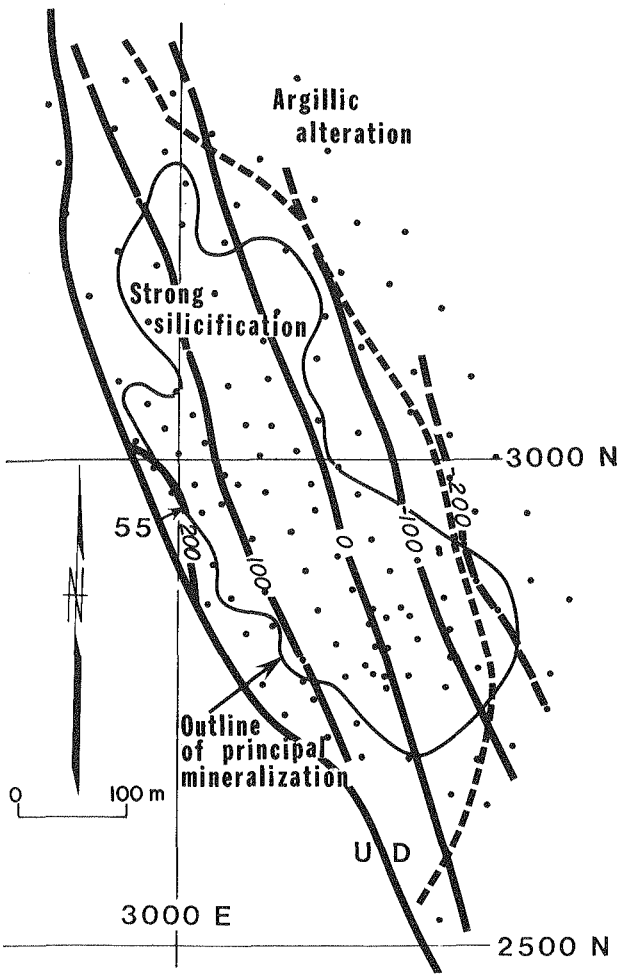


**FIGURE 8. Structure contour of the fault plane and surface alternation (values in meters above or below sea level).**

areas. It is a compilation of three separate surveys with all the samples from the 'B' soil horizon. The outline of the principal gold mineralization indicates a good positive correlation between soil geochemistry and gold mineralization. Other elements analyzed were Hg, As, Ni and Cu. None of these elements exhibits a good positive correlation with the gold mineralization.

## MINERALOGY

*Gold.* The mineralogy of the deposit is quite simple with only iron sulfides occurring in any appreciable amount with the gold. The gold assays from the core range from trace amounts to over 4.00 oz per ton averaged over a 2.0 m interval. Much of the gold is evenly distributed throughout the silicified host. Because of this, the gold has no direct relationship to the pyrite-marcasite content or the carbon content. Within the conglomerates, approximately equal amounts of gold are found in both the clasts and the matrix indicating the pervasive nature of the mineralizing process.

Mineralogical studies (Gasparrini, 1979) have shown that much of the gold is too fine to be resolved by the highest microscope magnification of 1250 times (0.5 microns). Most of the free gold observed is found in transecting veinlets of quartz with the free grains generally being less than 20 microns. This free gold is present in the native form, dominantly in discreet grains not associated with pyrite or other sulfides. The chemical content of the gold particles includes approximately 10 percent silver but no appreciable amounts of other elements (Gasparrini, 1979). The ratios of gold to silver within the host rocks vary widely, but overall is 0.5-0.3. No silver minerals have been recognized.

*Sulfides.* Iron sulfides are the dominant sulfide in the deposit and occur ubiquitously. Undoubtedly more than one generation of pyrite and marcasite occurs within the mineralized zone. Some of the pyrite in the sedimentary section is syngenetic. However, the marcasite and vein pyrite is related to the mineralizing process. Pyrite-marcasite forms 1-5 percent of the rock volume but locally ranges over 15 percent. The pyrite occurs as crystalline disseminations, replacement of rims and centers of pebble and breccia fragments, and replacement of carbon fragments. Pyrite is also present in thin quartz veins in the rhyolite stockwork. The pyrite form is commonly granular to massive but crystal forms such as cubes and octahedrons are also pres-

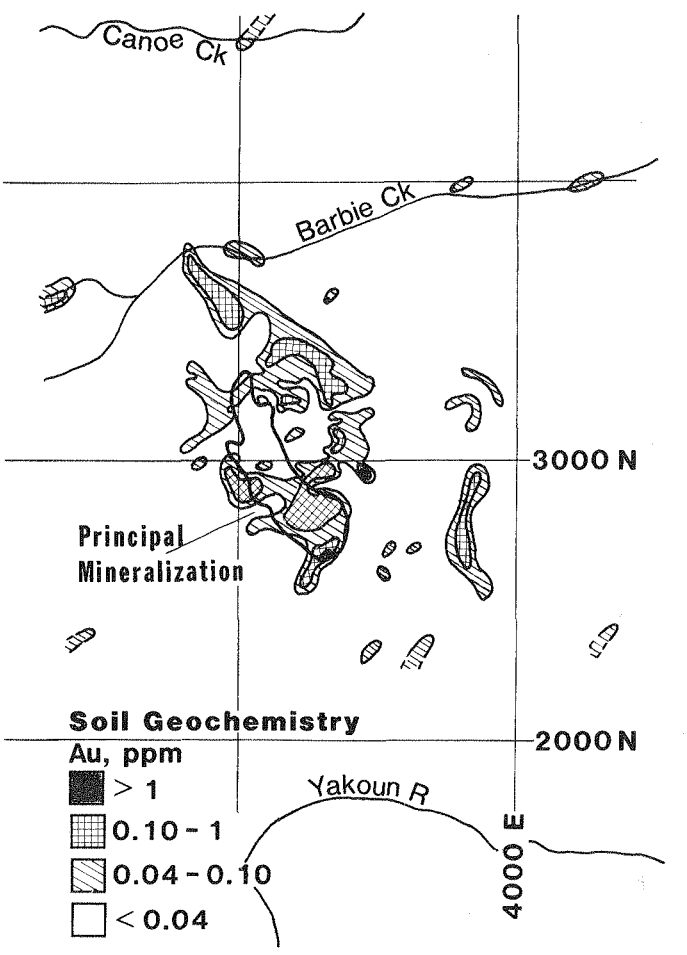


**FIGURE 9. Soil geochemical map of the Cinola deposit (grid in meters).**

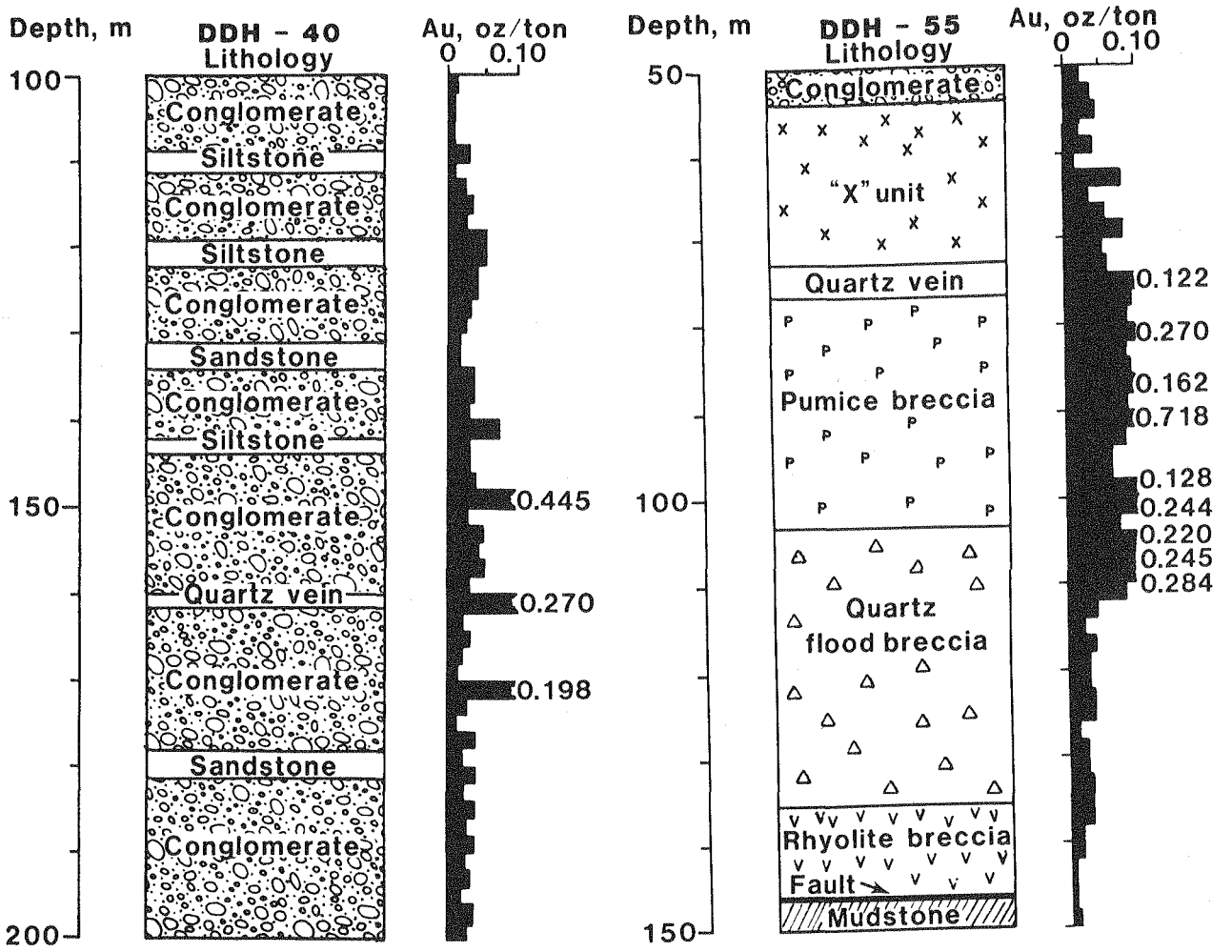


**FIGURE 10. Lithologic logs and gold assay histograms of diamond drill holes 40 and 55.**

ent. Marcasite, which is mixed in with the pyrite and is hard to distinguish, generally occurs as radiating blades.

Other sulfides that have been reported but occur in only trace amounts are sphalerite, chalcopyrite and pyrrhotite (Gasparrini, 1979).

*Other Minerals.* Native copper, cinnabar and rutile also occur in minute amounts in the deposit. Mercury concentrations as high as 19,200 ppb have been noted. Other anomalous element concentrations include arsenic and antimony. White calcite veins less than 10 cm wide are present, but rare, within the silicified section. The calcite is apparently late in the mineralization sequence. Hematite is found in the quartz-pyrite veins in the rhyolite stockwork and also disseminated in the groundmass of this particular unit.

## OREBODY SHAPE

In plan view, the main mineralized zone covers an oval area of 700 by 200 meters (Fig. 4). The contour map of gold mineralization shown in Figure 4 does not take into account the depth of the hole. Because holes to the east are considerably deeper than those to the west, our contour map has slightly shifted the mineralized zone to the east. Gold is present from the surface to depths of 300 meters. The fault zone forms a lower boundary to the main zone of gold mineralization. The gold mineralization has a close association with the degree of silicification and proximity to the rhyolite units (Figs. 5 and 6). Where the rock is moderately silicified, regardless of lithology, gold values generally exceed 0.025 oz/ton. The argillically altered rocks generally have very low gold values. Individual quartz veins carry gold values in excess of 0.100 oz/ton. The gold mineralization is generally thicker and of higher grade above areas where the rhyolite units are greater than 20 m thick (Fig. 7).

## GOLD DISTRIBUTION

In general there are two types of mineralization. A low grade, disseminated type is characterized by sub-microscopic gold and grades of 0.020-0.100 oz/ton. The high grade type occurs in silica veins, has visible gold and has grades in excess of 0.100 oz per ton. In Figure 10, the close association between quartz veining and high grade gold values is apparent in hole 40. Also illustrated is the strong gold mineralization above the rhyolite in the intense-

ly silicified quartz flood and pumice breccia units in hole 55.

Table 1 presents the results of four duplicate assays of core from hole 16. The higher grade intervals are more erratic because of the presence of coarse, visible gold. The low grade assay intervals indicate that the gold is very evenly distributed throughout the rock. Separate assays of both the clasts and the matrix in the mineralized conglomerate give near identical results, indicating that the mineral-bearing fluids were quite penetrative.

## GENESIS

The gold mineralization is directly related to the degree of silicification. The silicification appears to be controlled by proximity to the rhyolite. As suggested by Champigny and Sinclair (1979), the intrusion of the porphyry in a near surface environment created a hydrothermal system in which fluids rich in gold migrated through the clastic sequence. Cooling of these hydrothermal fluids resulted in supersaturation and precipitation of quartz and gold with traces of silver, mercury, arsenic and antimony. In summary, the Cinola deposit occurs in a quartz stockwork zone above a rhyolite intrusion.

Subsequent to the intrusion of the rhyolite, faulting displaced the orebody. Outcrops of rhyolite with minor amounts of gold to the northwest suggest that this area may have been the root zone of the rhyolite porphyry of the hanging wall.

Richards, Christie and Wolfhard (1976) suggested that the Cinola deposit be classified as a Carlin type based on metallic mineral assemblage, alteration mineralogy, permeability control of mineralization, occurrence of gold and proximity to a major structure. Differences between the Cinola deposit and the Carlin type are regional geologic setting, lithology of the host rocks, the extensive nature of the alteration and the close association of the rhyolite intrusion. Better analogues may be the Pueblo Viejo deposit in the Dominican Republic (Argall, 1975), deposits in the Hauraki goldfields, New Zealand (Williams, 1965), or the Porgera deposit in Papua New Guinea (Cotton, 1975). These deposits may represent a new class of large-tonnage, low-grade gold deposits in which gold is associated with quartz stockworks above near surface felsic intrusives emplaced in a complex tectonic setting.

**TABLE 1. Comparative assays of diamond drill hole 16 from four duplicate analyses**

| Depth, m | BW | S1 | S2 | UN |
|---|---|---|---|---|
| 14–16 | 0.065 | 0.065 | 0.070 | 0.076 |
| 16–18 | 0.035 | 0.025 | 0.030 | 0.030 |
| 18–20 | 0.088 | 0.110 | 0.110 | 0.056 |
| 20–22 | 0.030 | 0.020 | 0.020 | 0.028 |
| 22–24 | 0.025 | 0.020 | 0.020 | 0.018 |
| 24–26 | 0.025 | 0.025 | 0.020 | 0.036 |
| 26–28 | 0.360 | 0.320 | 0.310 | 0.300 |
| 28–30 | 0.240 | 0.150 | 0.120 | 0.070 |

Values in oz. Au/tn.

## SUMMARY

The Cinola deposit occurs in a silicified breccia zone immediately above a subvolcanic rhyolite intrusive complex. Argillic alteration forms an outer concentric zone around the silicified core. There are no apparent stratigraphic influences on mineralization but the gold is disseminated in an unusual suite of coarse volcaniclastic sediments. There are two types of gold mineralization. One is low grade, submicroscopic and disseminated and the other type is structurally controlled and consists of visible, high grade gold mineralization in quartz veins. Much of this high grade mineralization has not been fully evaluated. Current reserves for the Cinola deposit are 45 million tons of 0.054 oz of gold per ton.

## ACKNOWLEDGEMENTS

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

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## 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,

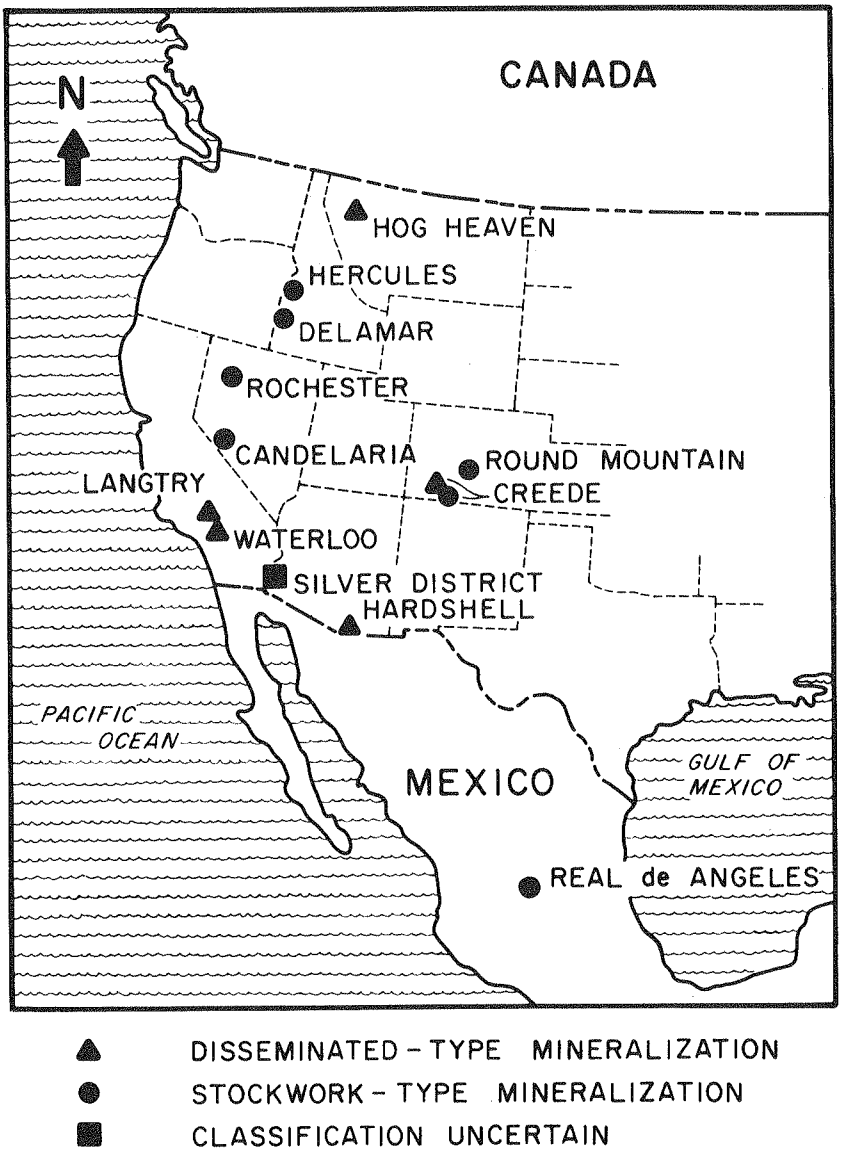


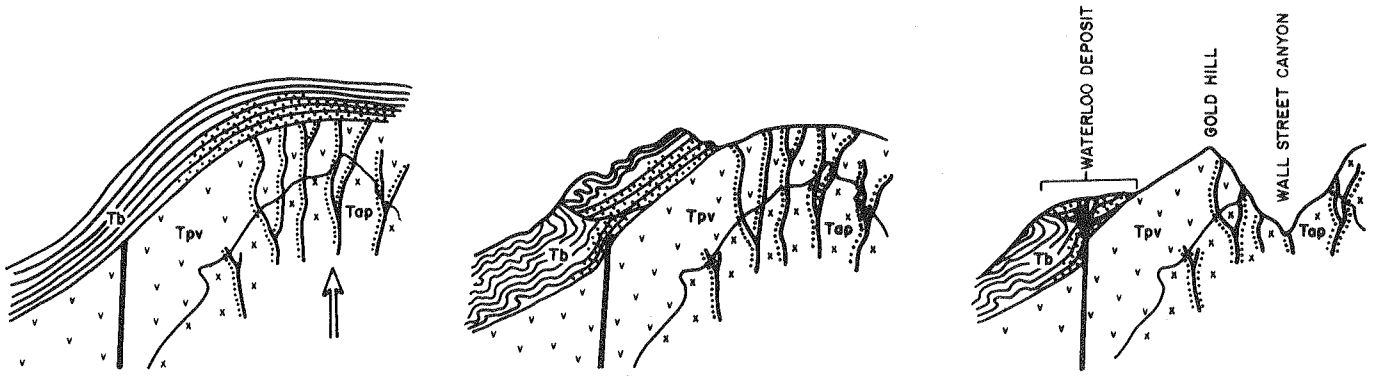

**FIGURE 1. Location of some bulk-tonnage silver deposits.**

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 by-product 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.

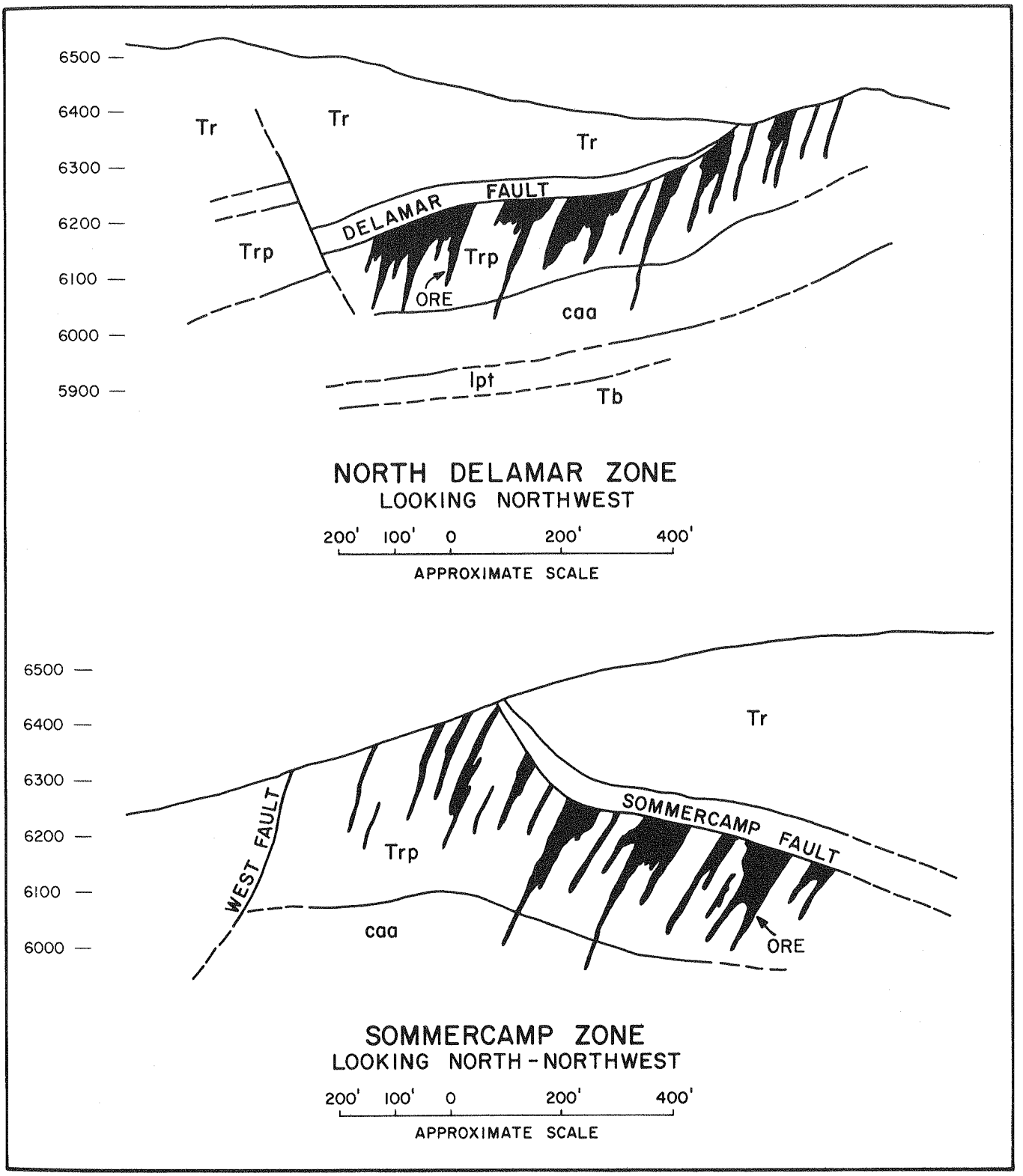



**FIGURE 3. Diagrammatical cross-sections showing relationships between structure and ore in rhyolitic volcanic rocks at Delamar, Idaho. (Adapted from Delamar Mine handout sheets.)**

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.).

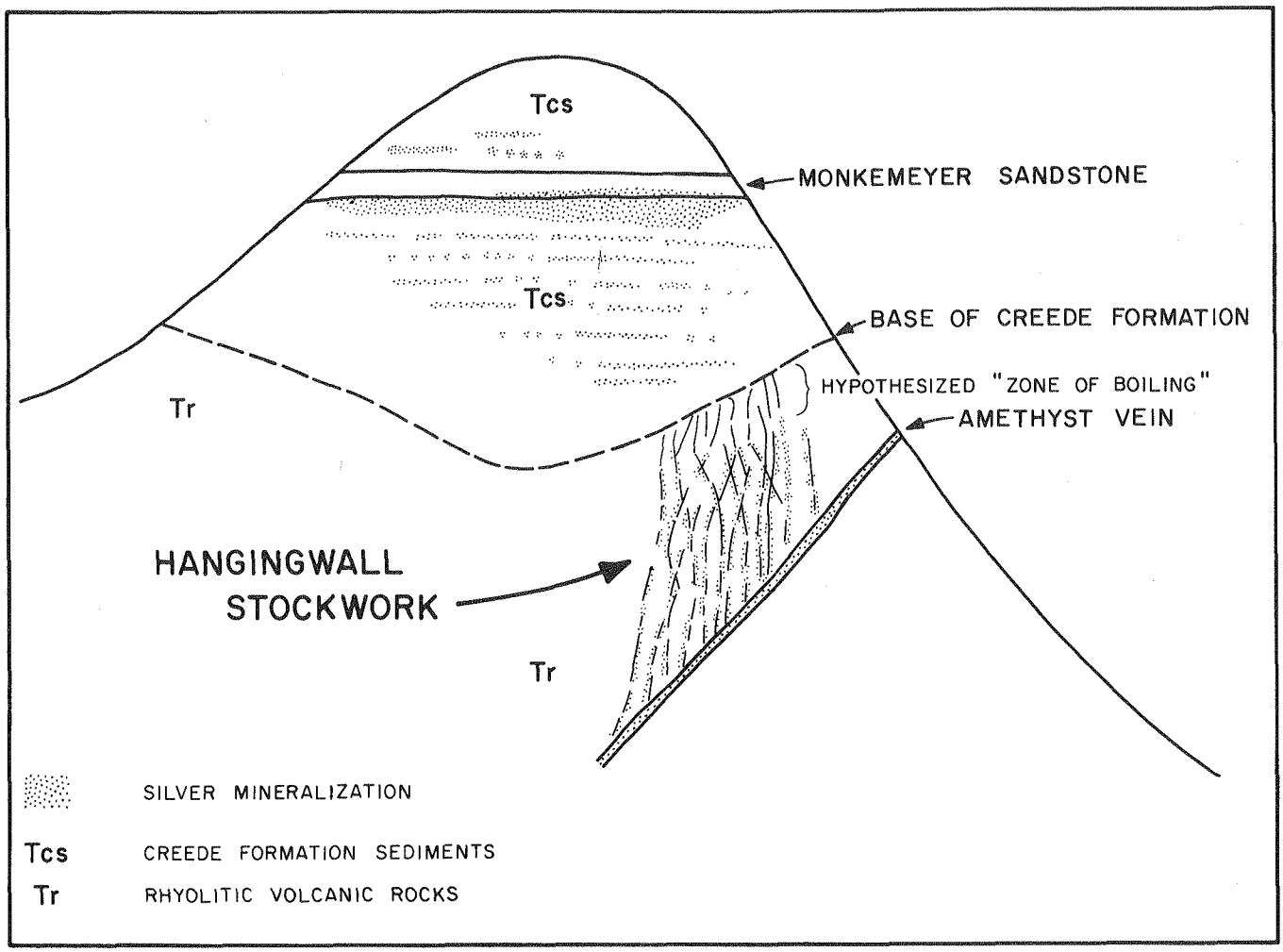



**FIGURE 4. Diagrammatical cross-section through Batchelor Mountain, Creede, Colorado, showing spatial and probable genetic relationship between disseminated-type silver in the Creede Formation sediments underlain by stockwork-type silver in rhyolitic volcanic rocks. (Not to scale.)**

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.

# THE ORTIZ GOLD DEPOSIT (CUNNINGHAM HILL)—GEOLOGY AND EXPLORATION

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## ABSTRACT

The Ortiz (Cunningham Hill) gold deposit is located 40 miles northeast of Albuquerque on the eastern side of the Ortiz Mountains in an area known as the "Old Placers" District. Gold was discovered here in 1828 and for the next 70 years many attempts were made to work both the gold-bearing pediment gravels and numerous vein deposits in the region. The Ortiz (Cunningham Hill) deposit is roughly 700 feet long and 400 feet wide at surface and is hosted in an outcropping pipe-like body of brecciated Mesaverde Quartzite. The pipe was probably formed by explosive activity in Oligocene times. Porous areas of the quartzite breccia acted as pathways for late volcanic hydrothermal fluids which caused severe alteration of feldspars in the arkosic quartzite to clay minerals, sericite and carbonates and precipitated calcite, pyrite, iron oxides, and minor gold, scheelite and copper sulfides in the voids. Although the deposit had been known since the late 1800's and had been partially drilled out in the 1940's, the gold and tungsten values were too low for exploitation at the prevailing metal prices. Gold Fields acquired the property in mid-1973 and after limited exploration of the Ortiz Basin volcanics it concentrated on a program of angled diamond drilling, underground exploration, bulk sampling and bench scale metallurgical testwork on the Cunningham Hill deposit. On-site heap leach tests were carried out on drift round material in 1976 and in March 1978 a decision was made to develop an open-pit, heap-leach gold operation. Startup was in early 1980. Recently, further exploration has continued outside of the open-pit site.

## INTRODUCTION

A paper treating the same subject as this was presented by Bill Lindqvist to the Colorado Mining Association in Denver, February 6-8, 1980(4). Much of the information is, therefore, similar although some new ideas are introduced as a result of more recent detailed mapping in the area and some of the sections have been reorganized and expanded.

The title has been changed slightly to avoid confusion in the perception of the name "Ortiz Mine". The current operation by Gold Fields is known as the Ortiz Mine, but will be referred to in this paper as the "Cunningham Hill" deposit to distinguish it from the old lode mine, discovered in 1828, which has traditionally been known as the Ortiz Mine and which lies approximately 7/8 mile west of the current pit.

Thanks are accorded to Gold Fields Mining Corporation for permission to publish this paper and to all the staff of Gold Fields both in Exploration and at the Cunningham Hill Mine for invaluable assistance and discussion.

## HISTORICAL REVIEW

It is not unlikely that native Pueblo Indians knew of the existence of gold in the Ortiz Mountains. They were aware of, and mined, turquoise in the neighboring Cerrillos Hills.

Published information credits the discovery of gold in the area in 1828 to a Mexican shepherd attached (?) to a freighting operation carrying merchandise from the Mississippi to the West Coast. He apparently noted the similarity of rocks in the Dolores Ranch area to auriferous rocks in his native Sonora.

This was the first significant gold occurrence reported west of the Mississippi and a small placer mining camp quickly became established. Water was always a problem and primitive dry washing techniques were universally employed to concentrate the gold. Numerous small surface and underground placer operations were carried out and within a period of four years, as many as 4,000 people occupied the town of Dolores. Lode mining dates from 1832 when the Ortiz vein was discovered. In 1833, a land grant of approximately 60,000 acres was awarded by the Republic of Mexico to Don Jose Francisco Ortiz, a merchant and the commander of the local garrison. A Spanish engineer, Don Cano, was later engaged to sink a shaft on the vein after the surface-accessible vein ore had been mined out.

Activity was intense in the period up to 1840 with continuing placer mining and additional other lode production from the Benton, Live Oak, Cunningham, English and Florencio Mines. The low-grade ore in the vicinity of Cunningham Hill was probably discovered at this time and may have been recognized as a principal contributor to the local placer gold accumulations.

In 1840 gold was discovered in the San Pedro Mountains and the "New Placers" district was established with a resulting depopulation of the Dolores area which then became known as the "Old Placers". During the period from 1828 to 1840, the lode and placer deposits of the southern and western Ortiz Mountains were probably also discovered. They include the Golden, Bonanza, and Carache Placers and the Lucas, Old Reliable, Pat Collins, Alpine Gulch, Gold Leaf Gulch, Gypsy Queen, and Candelaria Mines.

In 1864, the New Mexican Mining Co. acquired

the grant and erected a 20 stamp mill and amalgamation plant to treat crushed ore from the Ortiz Mine. The operation failed, apparently due to diminishing ore grades at depth, although this may have been more a factor of poor recovery from refractory ores.

From 1870 through 1887 activity was insignificant but was followed by renewed work evidenced by the need for, and establishment of, a post office at Dolores (1890-1894 and 1899-1903). Thomas Edison erected a plant in 1900 to try to separate gold from the placer gravels by static electricity. The presence of too much moisture resulted in failure of the process and shutdown of the operation.

In 1920, the Ortiz Mine was refurbished by a group of outside investors. Some production was maintained, but the effort eventually proved unprofitable and was closed about 1930. Dolores, during this time, degenerated to a ghost town and today only one habitable building remains.

Universal Placer Mining Corporation erected a large mechanized dry washing plant in 1939-1940 and treated 200,000 tons of placer gravels but recovered only 33% of the gold content, again being frustrated in their attempt to establish a viable placer operation by high moisture content of the gravels. From 1940-1946 the property was largely idle with respect to exploration and mining activities.

More recent work on the property up to present day operations is reported in the next section.

## GEOLOGIC SETTING

The Cunningham Hill gold deposit is located in the Ortiz Mountains approximately midway between Albuquerque and Santa Fe, New Mexico. Figure 1 shows the general setting of area within the Cerrillos Uplift fault block (La Bajada Constriction) which modifies the eastern margin of the Rio Grande Graben. The uplift block separates the Hagan Basin from the Santa Fe Embayment.

Recent tectonic activity in north central New Mexico has resulted in the exposure of Precambrian basement rocks on the east flank of the Rio Grande Depression and in the Tijeras/Gutierrez fault block graben and horst structure (3). The sedimentary cover strikes approximately N-S, dips east and begins with Mississippian rocks, only scattered remnants of which remain. Pennsylvanian units commonly rest directly on the Precambrian and are followed by Permian, Triassic, Jurassic, Cretaceous, Tertiary and Quaternary deposits.

The Tijeras fault zone cuts the SW corner of the Ortiz Mountains and in places has a reported post-Cretaceous, left-lateral strike-slip movement up to 1½ miles. The continuation of this fault zone to the NE is disjointed but the San Lazarus, Los Angeles, Lamy & Canoncito systems are probably genetically related to the Tijeras fault (5).

The San Pedro-Ortiz Porphyry Belt occupies an axial position in the uplift block and consists of four Oligocene intrusive centers. The Ortiz Mountains occupy the central part of this 30-mile zone.

On the Ortiz Mine Grant, the regional easterly dip of sediments is apparent in the disposition of Jurassic Morrison formation and Cretaceous Mancos Shale and Mesaverde quartzite (Figure 2), although locally, structural modifications are extreme, especially in the southwest corner where the Tijeras fault system meets the mountains. Strike-slip movement appears to offset the sediments but exposure is poor and cross faulting confuses the picture. Erosion of the uppermost Cretaceous units and non-deposition of Palaeocene rocks followed by continental Eocene Galisteo sedimentation with only slight angular unconformity is reported in adjacent areas and presupposed on the mine grant. In the surrounding areas Galisteo sediments are followed by the Oligocene Espinaso volcanics consisting of volcaniclastic breccias, conglomerates, sandstones, mudstones, volcanic flows and waterlain and airfall tuffs (7,8).

The core of the Ortiz Mountains consists of a subcircular intrusive stock of relatively fresh nepheline-bearing augite monzonite. The intrusive is in contact with and surrounded by a large number of laccoliths, sills, dikes, and small plugs of latite and latite/andesite porphyry. Most of these intrusives are omitted in Figure 2 for the sake of simplicity.

To the east of the stock is an area of ash-flow tuffs occupying the Ortiz Basin, which were preserved by down-faulting or deposition in a topographic depression. The ash-flow tuffs in the Ortiz Basin may well be time equivalent to the Espinaso volcanics.

The ash-flow tuff unit in the Ortiz Basin is fairly consistent over much of its exposed extent and, in places, displays an apparent crude stratification. Lithic fragments of sediments, porphyry and monzonite are commonly present. Local areas display extreme heterogeneity. Near Cunningham Hill, a volcanic breccia unit has been distinguished which consists of a chaotic assemblage of larger mixed lithic blocks with a subordinate igneous matrix. In the Florencio mine area and north of Cunningham Hill, a similar volcanic breccia has been mapped but is distinguished by the presence of dominant quartzite clasts. An area also exists where sedimentary structures are seen and these are interpreted as reworked ash-flow deposits formed during quiescent periods between volcanic events.

A number of latite and latite/andesite dikes cut the ash-flow tuff unit and may be related to the latites, latite porphyries and intrusive breccias of the Magnetic Hill wedge.

On the south side of the ash-flow tuff unit is an irregular breccia pipe which is the host for the Cunningham Hill gold deposit. The breccia pipe is composed of approximately 90% Mesaverde quartzite and argillite clasts and 10% latite porphyry fragments. Clasts in the pipe ranges from rock flour to boulders several feet across and vary from angular to rounded. The resulting body is vuggy and acted as an excellent conduit for penecontemporaneous or later hydrothermal fluids from which iron oxides,

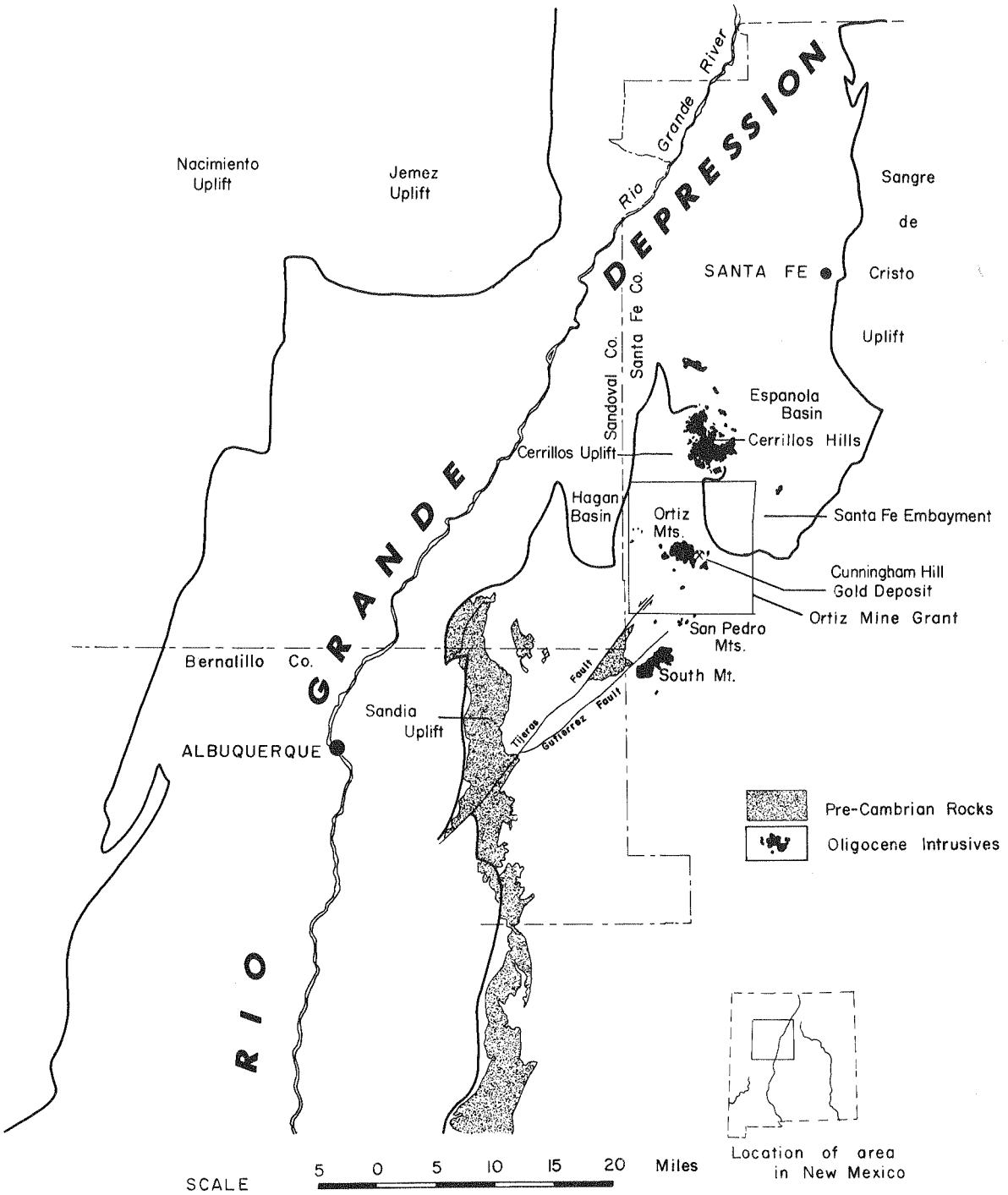



**FIGURE 1.**

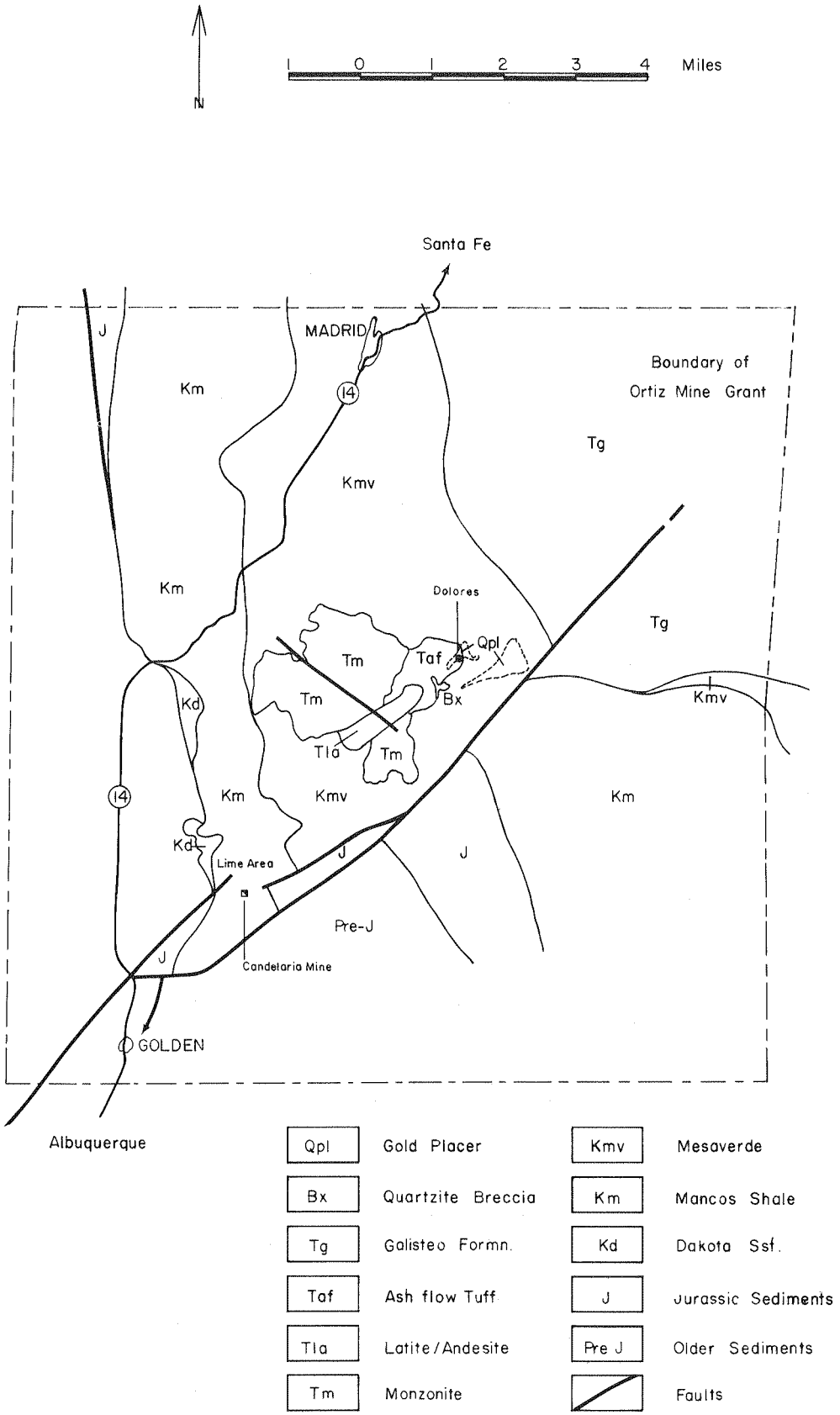
pyrite, gold and other minerals were deposited.

Approximately 60% of the grant is covered with pediment gravels and alluvial deposits which contain placer deposits of gold in the proximity of mineralized areas.

## MINERALIZATION

Records of gold production from the individual mines in the Old and New Placer districts are unknown and only qualitative estimates can be made based on the size of the associated waste dumps. Scores of gold-bearing shears and fractures have been located in the Ortiz Mountains, but only the Ortiz, Benton and Candelaria veins have been mined to any extent.

Judging by the intensity of surface activity in the "Lime Area" on the southern flank of the Ortiz Mountains, a significant production of gold can be assumed in this area despite the absence of large waste dumps. The mineralization here is localized in a garnetized limestone unit of the Mancos Shale and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Kmv
Km
Kd
Lime Area
J
J
Pre-J
J
Candelaria Mine
GOLDEN
Albuquerque
Qpl Gold Placer
Bx Quartzite Breccia
Tg Galisteo Formn.
Taf Ash flow Tuff
Tla Latite/Andesite
Tm Monzonite
Kmv Mesaverde
Km Mancos Shale
Kd Dakota Ssf.
J Jurassic Sediments
Pre J Older Sediments
Faults

**FIGURE 2.**

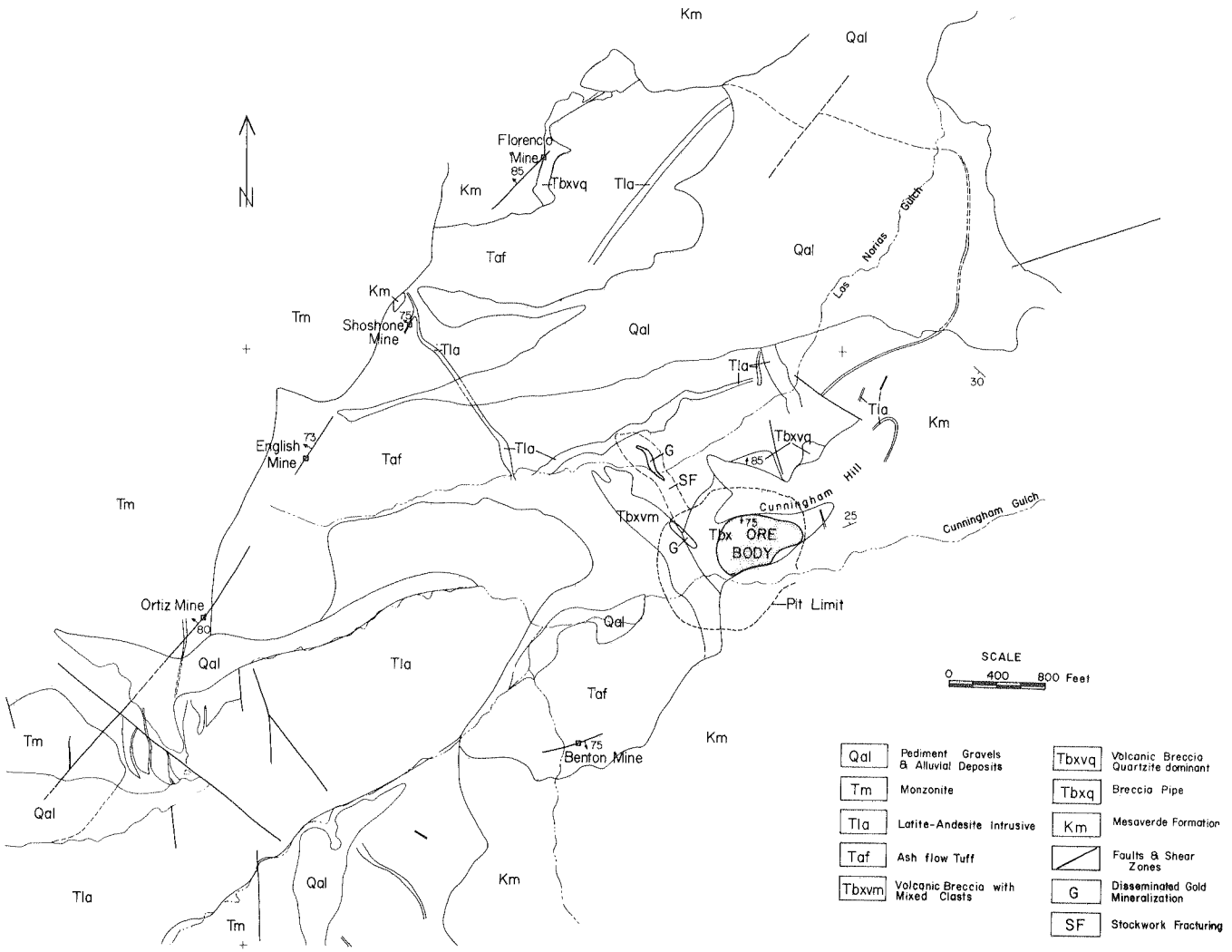


**FIGURE 3.**

is associated with copper mineralization.

Little is known of the Candelaria mine mineralization except that gold and copper mineralization are present in a fault zone which apparently parallels the Ortiz shear zone.

The Ortiz vein is a steeply-dipping, magnetite/hematite and pyrite-filled fracture with very minor quartz veining. The mineralization averages 4 feet in thickness and has been mined for about 600 feet along strike and to a depth of 350 feet below surface. The Ortiz vein is part of a fracture system that extends for at least 6,000 feet through the English, Shoshone and Florencio mines (Figure 3).

In the Florencio Mine area, the fracture system encounters a volcanic breccia with quartzite clasts containing low-grade gold-mineralized intervals. Recent work in this area has shown that the volcanic breccia/country rock contact is steep but not obviously faulted and that heavy pyrite mineralization in the vuggy breccia close to the contact is not as auriferous as the more typical ash-flow tuff which contains disseminated pyrite in this area.

The Benton vein is a narrow quartz, pyrite-filled shear near the southern contact of the ash-flow tuff and it produced a small tonnage of gold-tungsten ore.

The Cunningham Hill ore deposit is exposed on the southern margin of the ash-flow tuff unit which fills the Ortiz Basin and forms a distinctive unit at surface in which quartzite blocks and other rock fragments are partially cemented by multicolored hydrous oxides of iron (limonite, hematite, jarosite) and, rarely, alunite (Figure 4). Contacts of the breccia with the Mesaverde sediments are commonly very sharp (Figure 5). Mineralogic investigation of drill core samples shows that the ore body is zoned vertically (Figure 6).

*Oxidized ore* forms a capping averaging 90 feet in thickness where the breccia is cemented mainly by indigenous and transported limonite. Some granular gold exists in this zone, but most occurs as thin sheets or films usually rolled or knurled. Current mining has uncovered some blocks of remnant, unoxided breccia containing pyrite in sharp contact with oxidized ore (Figure 7).

*Semi-oxidized ore,* approximately 60 feet thick, underlies the oxidized zone and the breccia is cemented by indigenous limonite with grains of residual pyrite. Gold is present as thin coatings on

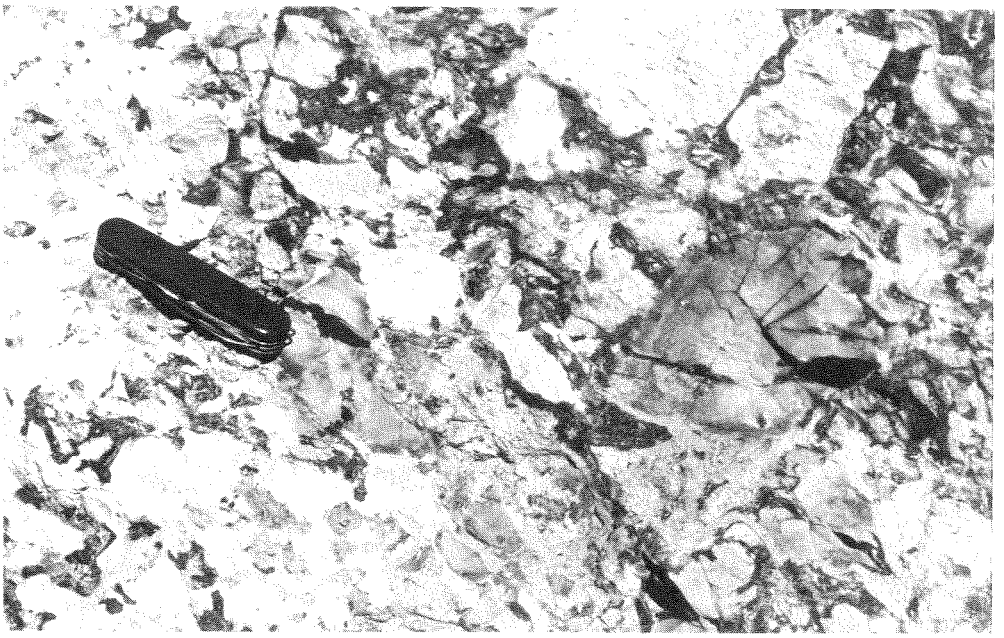

FIGURE 4.

FIGURE 5.

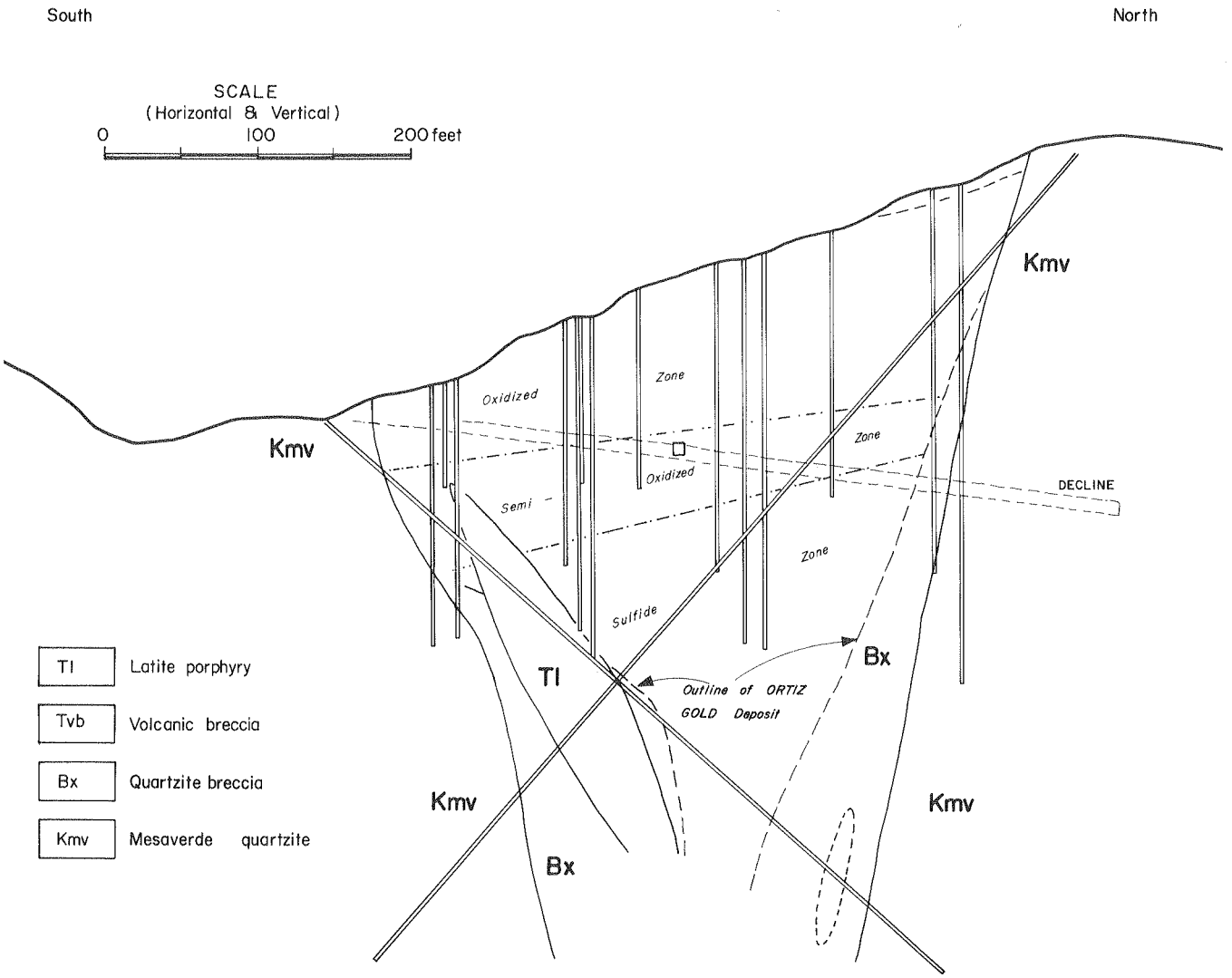


FIGURE 6.

tarnished pyrite and as granular inclusions in pyrite. Some scheelite appears in this zone.

*The Unoxidized zone* represents the primary ore and forms approximately 60% of the deposit. Magnetite and hematite are intimately associated and fresh pyrite is present. Gold is present as granular inclusions in fractured pyrite and in late-stage gangue minerals (chiefly carbonates). In addition to gangue, the fractures in pyrite contain rare pyrrhotite, chalcopyrite and galena. Abundant scheelite is present in this zone. The gold particle grain size varies from 5 to 400 microns and scheelite grains mostly occupy the 0.2 - 0.4 mm size range. Gold values are leaner in depth and primary iron oxides become less abundant.

Hydrothermal mineralization in the breccia pipe has also caused alteration in the country rock and the clasts. The original felspars in the quartzite clasts have suffered clay-sericite alteration and the metashales consist of microscopic sericite embedded in a finely divided matrix of quartz and felspar. Phenocrysts and ground mass of the common porphyry clasts are intensely altered to clay-sericite-carbonate minerals. In the surrounding area propylitic alteration is common, but not continuous and no regional zonation has been recognized.

Copper mineralization is very patchy and rarely exceeds 0.5% except in isolated samples. Elsewhere on the Ortiz grant there is a correlation between high copper and gold assays although copper-gold ratios are highly variable.

If gold deposits in the area are the result of a single mineralizing event, this would have occurred after deposition of the ash-flow tuffs since these rocks carry low-grade values in the Dolores-Good Friday stockwork-fractured area (Figure 8).

The age relationships of the ash-flow tuff to the breccia are presently uncertain. If the breccia pipe is pre-ash-flow tuff, the present ore body may have been recently uncovered and other pipes may be buried beneath the tuff blanket. The common feature of rebrecciation in breccia pipes and the possibility of separate mineralizing events may well be present and continued observation will be necessary to construct the complete history.

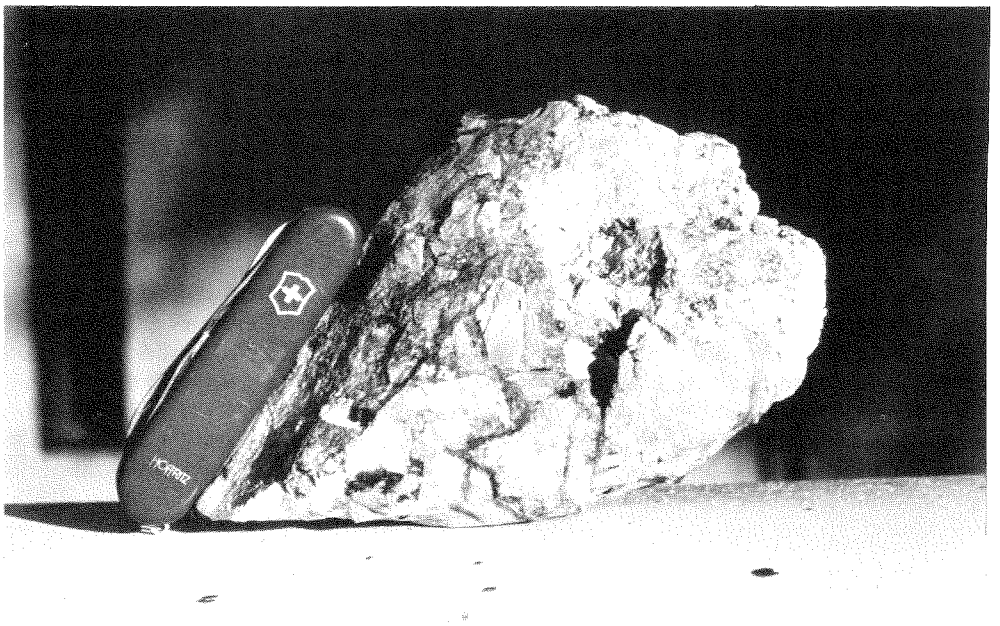

FIGURE 7.

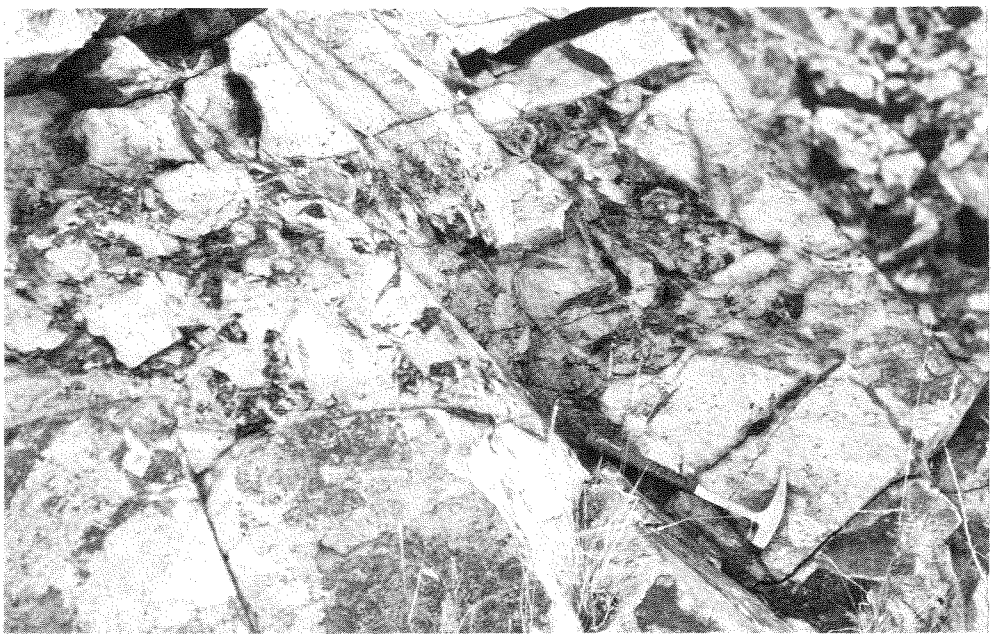

FIGURE 8.

## EXPLORATION

Exploration on the Ortiz Mine Grant has been essentially continuous since the discovery of gold in 1828 and the mountains are riddled with old prospect pits, trenches, shafts and adits. These activities have provided invaluable bedrock exposures and sampling points.

In this section only the exploration activity carried out since World War II is discussed.

In 1946, George W. Potter of Joplin, Missouri, purchased the grant subject to an existing option to lease by the Silas Mason Co. of New York. This company surrendered the lease in 1947, having failed to prove up a viable placer deposit on Cunningham Mesa. The surface and minerals were severed in 1947, and Potter commenced a major exploration effort consisting of geological mapping, geophysical surveys, prospect drilling and metallurgical studies. The drilling at Cunningham Hill consisted of more than 50 vertical percussion, churn and diamond core holes and indicated a large disseminated gold-tungsten deposit amenable to open-pit mining, but the grades were found to be too low to permit development.

The minerals were leased subsequently to Ortiz Mines, Inc. and then subleased to the Molybdenum Corporation of America, who conducted induced polarization studies and drilling to try to locate extensions of the known mineralized body and find other ore bodies of a similar or larger extent. They surrendered their rights in 1969.

In 1972, approximately 17,000 acres of the western portion of the grant was subleased by Ortiz Mines, Inc., to Conoco who was primarily interested in the copper porphyry potential of the ground. This sublease is current although exploration activity has ceased.

The property was subleased by Gold Fields (then operating as Azcon) in 1973 at the recommendation of Dan Stewart, who had been associated with the exploration work conducted by Potter and Sims. Because of the many minor gold-bearing fractures and shears within the ash flow tuff unit, which at that time was considered to be a diatreme, Gold Fields began with a program of trenching and shallow, vertical diamond drill holes on approximately 1000-foot centers to test the large low grade gold potential of this unit. The results were not altogether encouraging.

In July, 1973, a vertical diamond drill hole sited over the highest-known ore grade mineralization confirmed the reliability of the previous open hole drilling. The former drilling had not adequately defined the walls of the deposit, and no useful material for metallurgical testwork was available. It was, therefore, decided to drill out the deposit with angled NQ diamond drill holes on section lines 100 feet

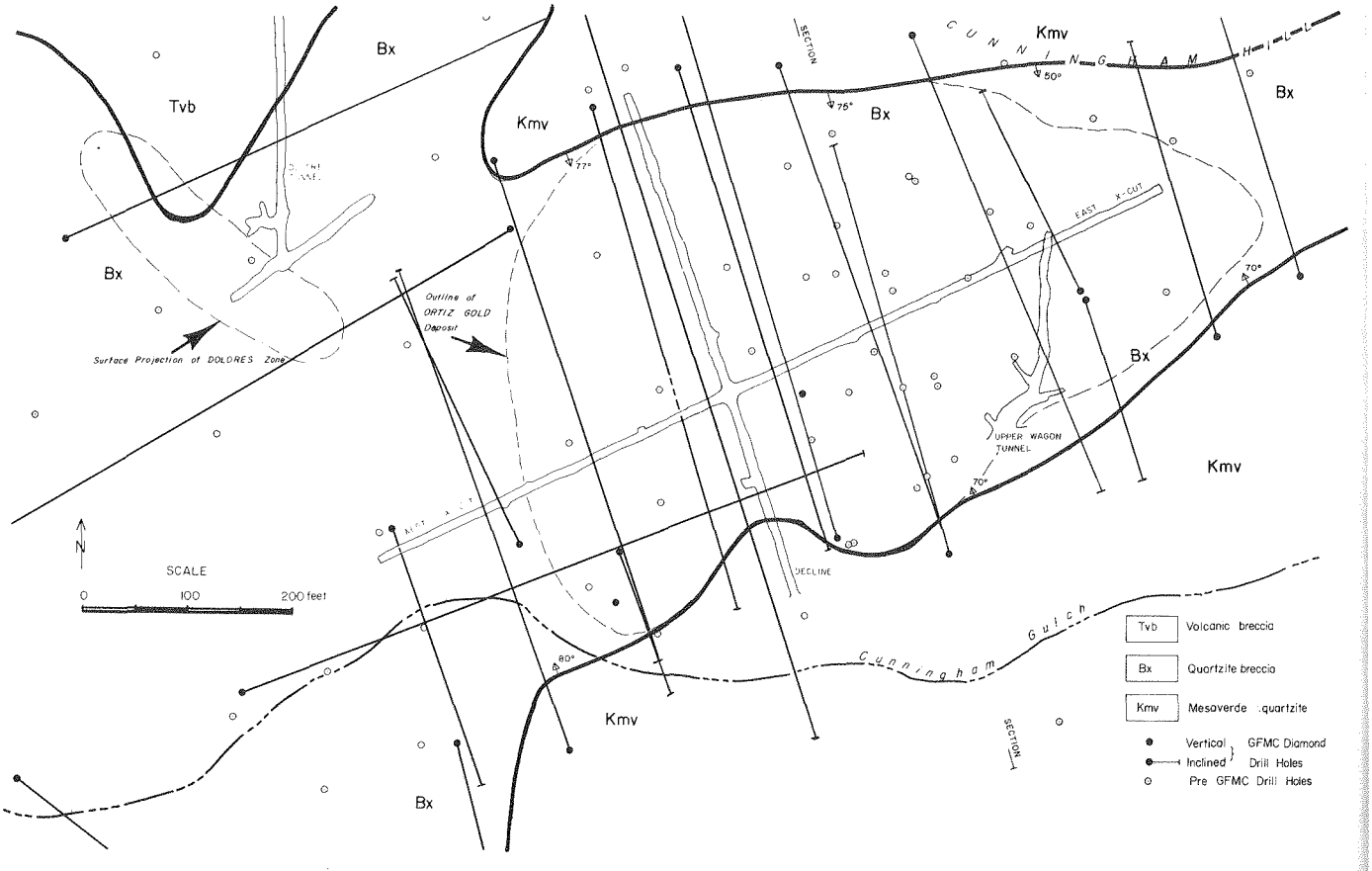


FIGURE 9.

apart. Eighteen holes were completed by late 1974 and these, together with previous drill data, outlined a deposit with surface dimensions of 700 feet by 400 feet and plunging about 50 degrees to the west (Figure 9). Ore grade values were found to extend out to the breccia walls in the uppermost, central part of the deposit but to retreat gradually from the walls to the east and west and at depth. Despite the existence of 70 drill holes in and around the deposit, there was still concern about the reliability of the drill-indicated grade estimates for this low-grade gold mineralization so it was decided to drive a decline northwards along the short axis of the deposit and cross-cuts along the long axis to check grade continuity. One side of the Upper Wagon tunnel was also slabbed for sampling purposes. Details on the handling and sub-sampling of the tunnel rounds as well as the drill core are published by M. Springett (6). Underground work was completed at the end of 1975 and the grade estimates made from the bulk sample assays compared closely to those made from drill holes.

Bench scale metallurgical testwork, undertaken during 1976, indicated that although satisfactory recoveries could be achieved for gold, it would be difficult to prepare a saleable tungsten concentrate at an acceptable recovery. At the prevailing gold price, conventional milling did not look economically attractive so tests were conducted on the amenability of the ore to heap leaching. The tunnel round reject material that had been crushed to minus 5/8 inch was available and in the summer of 1976, heap leach tests were carried out on two 500 ton ore heaps with satisfactory results. A decision was made in March 1978 to develop the Ortiz deposit as an open-pit, heap leach operation with construction to begin in 1979 and start-up early in 1980.

In the interim period after completing underground testing and until the end of 1979, no surface exploration was carried out. Beginning in 1980, exploration began again with slightly different priorities in view of the current high gold price and the presence of an established facility. First priority is still low-grade disseminated gold mineralization although the constraints of size are less onerous. Of second priority is the reevaluation of the old lode deposits particularly along the Ortiz Shear Zone and in the vicinity of the Old Benton Mine. For this program detailed geologic mapping and reconnaissance I.P., magnetic, VLF-EM gravity techniques are being employed. Exploration drilling is currently underway on the property.

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# STYLES OF LOW-GRADE GOLD MINERALIZATION IN VOLCANO-PLUTONIC ARCS

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## ABSTRACT

The current high gold prices prevailing on World markets require a fundamental redefinition of what constitutes an economic gold deposit; in many areas, deposits containing ≤2 g/t can be profitably worked. In Phanerozoic magmatic arcs, a wide spectrum of partly overlapping geological habitats possesses bulk-minable, low-grade gold deposits. A World-wide review of deposits and prospects leads to recognition of the following broad mineralization types with low-grade gold potential: 1. porphyry copper deposits; 2. intrusive-hosted deposits of epithermal character; 3. contact-metasomatic skarn deposits; 4. contact-metasomatic deposits of distal type generated beyond the influence of high-temperature skarns; 5. Carlin-type deposits; 6. hydrothermal breccia-hosted deposits; 7. enargite-luzonite-bearing massive sulfide deposits of replacement origin; 8. epithermal deposits dispersed in felsic or intermediate-composition volcanic rocks; 9. vein-type epithermal deposits; and 10. Kuroko-type massive sulfide deposits.

## INTRODUCTION

Volcano-plutonic arcs, constructed during post-Precambrian subduction (and subordinate rifting) events along both island arcs and continental (Andean-type) margins, have been and remain significant producers of hard-rock gold. Traditional and most widespread are high-grade (≥10 g/t), low-tonnage vein deposits, either of epithermal (bonanza) type in volcanic terranes or within or peripheral to calc-alkaline stocks or batholiths (e.g., Emmons, 1937). However, a number of other larger-tonnage but substantially lower-grade producers have also long been operative. From many of these gold has been a by- or coproduct of copper mining, but during the last 15 or 20 years bulk-minable gold deposits have started to come on stream. In response to the markedly higher gold price since 1972, and its dramatic upward swing in 1979, the search for and recognition of large-tonnage, low-grade gold deposits has been intensified in diverse parts of the World. Employing low-cost mining and recovery techniques, many deposits containing ≤10 million tonnes of ≤2 g/t Au are economically viable with a gold price exceeding US 400/ounce.

Here an attempt is made to categorize proven and potential low-grade gold deposits in Phanerozoic magmatic arcs, while recognizing that many of the new discoveries are geologically complex and currently either poorly understood and/or subject to confidentiality. Nevertheless, the diversity of geological habitats with potential for bulk-minable, low-grade gold is clearly apparent.

## PORPHYRY COPPER-GOLD DEPOSITS

At least thirteen porphryr copper deposits possess ≥0.5 g/t Au (Sillitoe, 1979), which is, or has the potential to be, extracted as an important byproduct. The largest gold-rich porphyry copper deposit is Conzinc Riotinto of Australia Ltd.'s Panguna deposit on Bougainville Island, Papua New Guinea, which contains some 900 million tonnes of 0.55 g/t Au. The richest in gold is Benguet Corporation's new mine at Dizon in the Philippines, where some 90 million tonnes average 0.93 g/t Au and a smaller tonnage exceeds 1.5 g/t Au. An unusual case of residual gold enrichment occurs at Ok Tedi in Papua New Guinea (Fig. 4), where 31 million tonnes of leached capping above a major Cu-Au-Mo-bearing supergene copper enrichment blanket (Bamford, 1972) contains 3.1 g/t Au, and will constitute the first part of the deposit to be exploited by a consortium led by the Broken Hill Proprietary Company Ltd. Copper-poor porphyry gold deposits have not been recognized but are considered by Sillitoe (1979) as an exploration target.

As summarized by Sillitoe (1979), high gold values are mainly present in rocks characterized by feldspar-stable alteration assemblages, commonly of biotite-rich potassium silicate type, although Lowder and Dow (1978) state that high gold values occur partly in advanced argillic alteration at Cabang Kiri, Sulawesi. Alteration may affect intrusive rocks of either calc-alkaline or shoshonitic affiliation and, at Mamut in Sabah, as felsic as adamellite. Gold is closely associated with chalcopyrite and/or bornite mineralization, occurs largely in the native state, and is present in quantities proportional to the copper grade. No obvious relationship exists between gold and pyrite contents. Gold-rich deposits tend to be poor in molybdenum but are far from being devoid of it as shown by deposits at Saindak in Pakistan, Bajo de La Alumbrera in Argentina (Fig. 1), and Ok Tedi in Papua New Guinea. Many gold-rich porphyry deposits contain abundant hydrothermal magnetite of veinlet, disseminated and irregular replacement origin, commonly accompanied by replacement quartz, in the feldspar-stable assemblages (Fig. 1) (Sillitoe, 1979).

## INTRUSIVE-HOSTED EPITHERMAL DEPOSITS

Besides gold-rich porphryr copper deposits, which of course are hosted by intrusives, an additional category of intrusive-hosted deposits of broadly epithermal category seems to be justified. A deposit of this type at Porgera, Papua New Guinea, is currently under investigation by a consortium comprising Placer Development, MIM Holdings and Consoli-

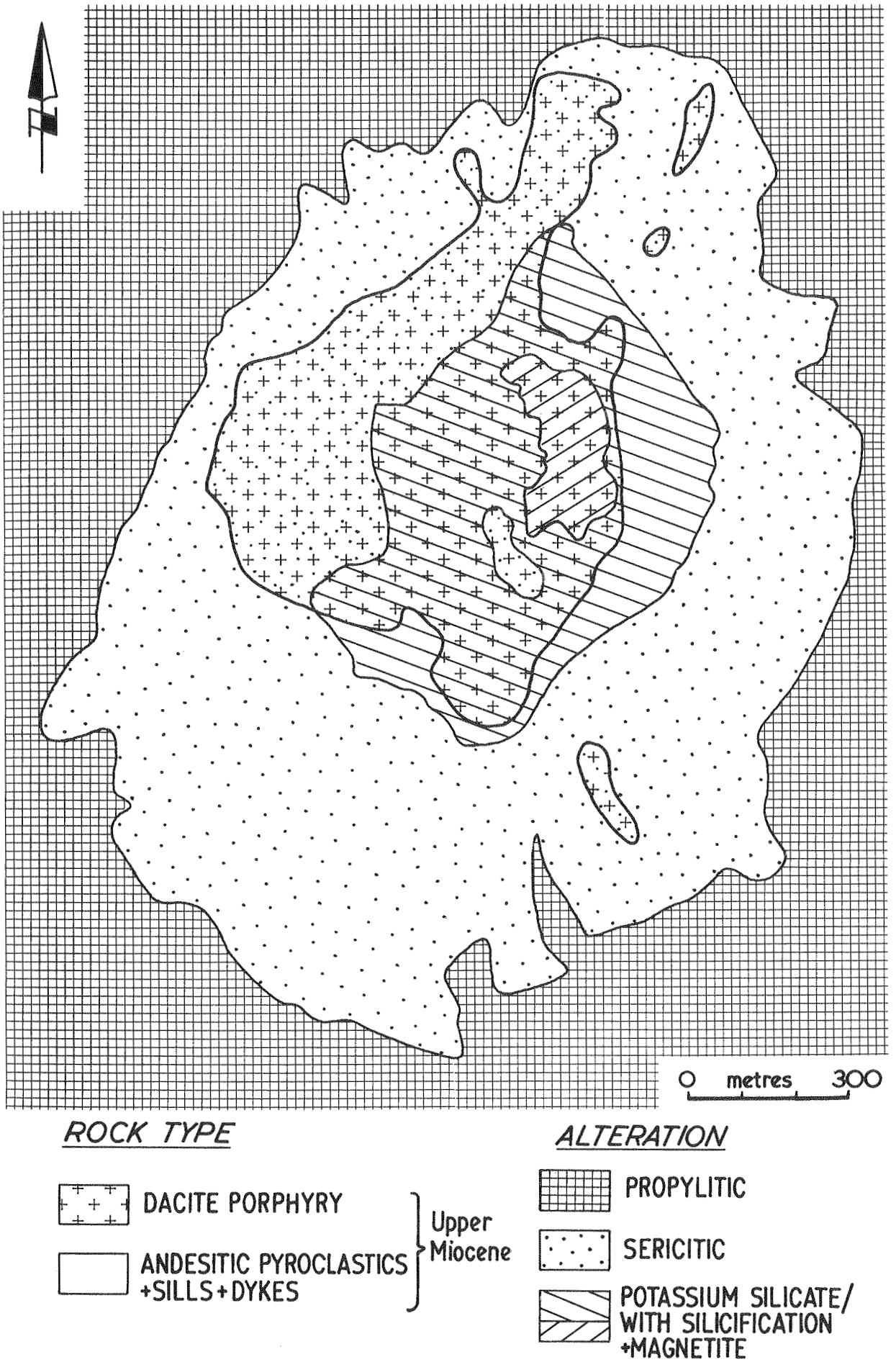


FIGURE 1. Geology and alteration of the Bajo de La Alumbrera gold-rich porphyry copper deposit, northwest Argentina (after Sillitoe, 1979). Highest gold grades occur in areas of heavy silicification and magnetite replacement.

dated Gold Fields of Australia, and may become a major ore body; some 125 million tonnes averaging 2.67 g/t Au and 12 g/t Ag have been announced to date. Gold occurs in several generations of multidirectional veinlets, which exhibit crustiform banding and open-space fillings and carry several of quartz, calcite, dolomite, chlorite, hematite, pyrite, sphalerite (for ~1 % Zn) and galena (for ~0.5 % Pb), and minor chalcopyrite, tetrahedite and silver sulfosalts (O'Dea, 1980). Hydrothermal breccias are also mineralized. The ore zone occurs largely in partly calcareous and carbonaceous shallow-marine shale and mudstone of Upper Cretaceous age in close proximity to irregular stocks and dikes of diorite and andesite porphyries (Fig. 2) (O'Dea, 1980) of probable shoshonitic affinity and late Cenozoic age. The stocks, some intra-mineral, are themselves locally mineralized. Alteration is dominantly propylitic, but with selvages of sericite-kaolinite on some veinlet generations.

A stockwork gold deposit at Salave, northwest Spain, is currently under development by Anglo-American Corporation. It occurs in a Hercynian granodiorite and its gabbroic and Canbro-Ordovician metasedimentary host rocks. Gold grades exceeding 1 g/t accompany pyrite, arsenopyrite, stibnite, molybdenite and minor sphalerite and chalcopyrite in propylitized rock, high in sericite, carbonate and albite (Harris, 1980).

Gold deposits such as the Yellow Pine in the Stibnite district of Idaho, USA, could also be included in this category. Gold accompanies disseminated pyrite and arsenopyrite in zones of sericite and fine-grained quartz introduction; this assemblage is post-dated by locally commercial concentrations of scheelite and stibnite (Fig. 3) (Cooper, 1951). The ore bodies are controlled by a roughly northeast-trending shear zone within a quartz monzonite phase of the Idaho batholith (Fig. 3). Cooper (1951) quoted inferred reserves in the Stibnite district of 14.5 million tonnes containing 1.7 to 6.8 g/t Au, quite similar to an up-to-date reserve of 16.3 million tonnes containing 2.0 to 3.4 g/t Au which is due to be mined.

## CONTACT-METASOMATIC: SKARN DEPOSITS

Several skarn-type deposits may be considered as large, low-grade gold resources, although most of them are primarily copper deposits. The skarn bodies occur either adjoining unmineralized or only weakly mineralized intrusives or border porphyry copper-bearing stocks.

The first category includes: the Copper Canyon deposit, 9 million tonnes of 0.57 % Cu and 0.93 g/t Au, in the Battle Mountain district of Nevada, USA, which is contained in biotite-, K-feldspar- and calc-silicate-bearing Paleozoic sediments abutting a 37-m.y.-old granodiorite stock (Theodore and Blake, 1975); Ertsberg (30 million tonnes of 2.5 % Cu and 0.86 g/t Au), Irian Jaya, Indonesia, in a pipe-like mass of magnetite-rich, calc-silicate skarn developed from a Mesozoic-Tertiary sedimentary sequence intruded by Pliocene diorite and monzonite (Flint et al., 1976); and La Luz (~20 million tonnes of 0.4 % Cu and 3.7 g/t Au), Nicaragua, a calc-silicate skarn developed in Lower Cretaceous carbonates overlain by andesitic volcanics and cut by Tertiary granodiorite (Plecash et al., 1963).

The second category, skarns marginal to porphyry copper deposits, is exemplified by magnetite-rich bodies in Oligo-Miocene limestones adjoining a 1.2-m.y.-old monzonite porphyry stock at Ok Tedi, Papua New Guinea (Fig. 4) (Bamford, 1972). Part of the skarn reserve is quoted as 22.7 million tonnes of 2.3 % Cu and 1.1 g/t Au.

## CONTACT-METASOMATIC: DISTAL DEPOSITS

This category incorporates deposits of a number of different types, all believed to be related to, but not necessarily immediately contiguous with, intrusive stocks. They are all situated beyond the influence of any high-temperature skarn-type alteration.

At Battle Mountain, Nevada, Duval Corporation's Minnie and Tomboy gold deposits occur beyond the Copper Canyon skarn, some 1200 m south of the Copper Canyon granodiorite (see above). Some 9 million tonnes of 2.5 g/t Au ore have been reported to date. They occur in the 30-m-thick basal portion of a calcareous conglomerate as fracture fillings and disseminated replacements, partly as a result of leaching of carbonate cement (Blake and Kretschmer, this Volume). Gold accompanies pyrite, pyrrhotite, clays, tremolite and epidote, and lesser galena, sphalerite, chalcopyrite, chlorite and muscovite. This gold mineralization lies beyond the main base-metal zone in rocks carrying more than 4 vol. % iron sulfides (Blake et al., 1978).

The Windfall mine in the Eureka district, Nevada, USA, is being worked for low-grade (1.2 g/t) gold ore present in sandy dolomite of Cambrian age. The dolomite is locally hydrothermally brecciated and silicified, and is cut by a rhyodacite porphyry dike. Mineralization was probably emplaced as a result of late Cretaceous (100 m.y.) magmatism (Nolan and Hunt, 1968), and is located distally with respect to base-metal skarn-type deposits (Roberts et al., 1971).

The adjoining Landusky and Zortman deposits in Montana, USA, are also judged to possess attributes of this category. Together they contain 32.5 million tonnes of 1.0 g/t Au in calcareous sediments adjoining a lacolith of syenite and related porphyries, which are locally gold-bearing (Corry, 1933).

At Andacollo, Chile, stratabound gold mineralization occurs in four rhyolite to dacite lava flows from 20 to 50 m thick, which persist for more than 5 km along strike, and are interbedded in a Lower Cretaceous volcanic sequence of dominantly andesitic composition (Fig. 5) (Llaumett, 1980). Native gold

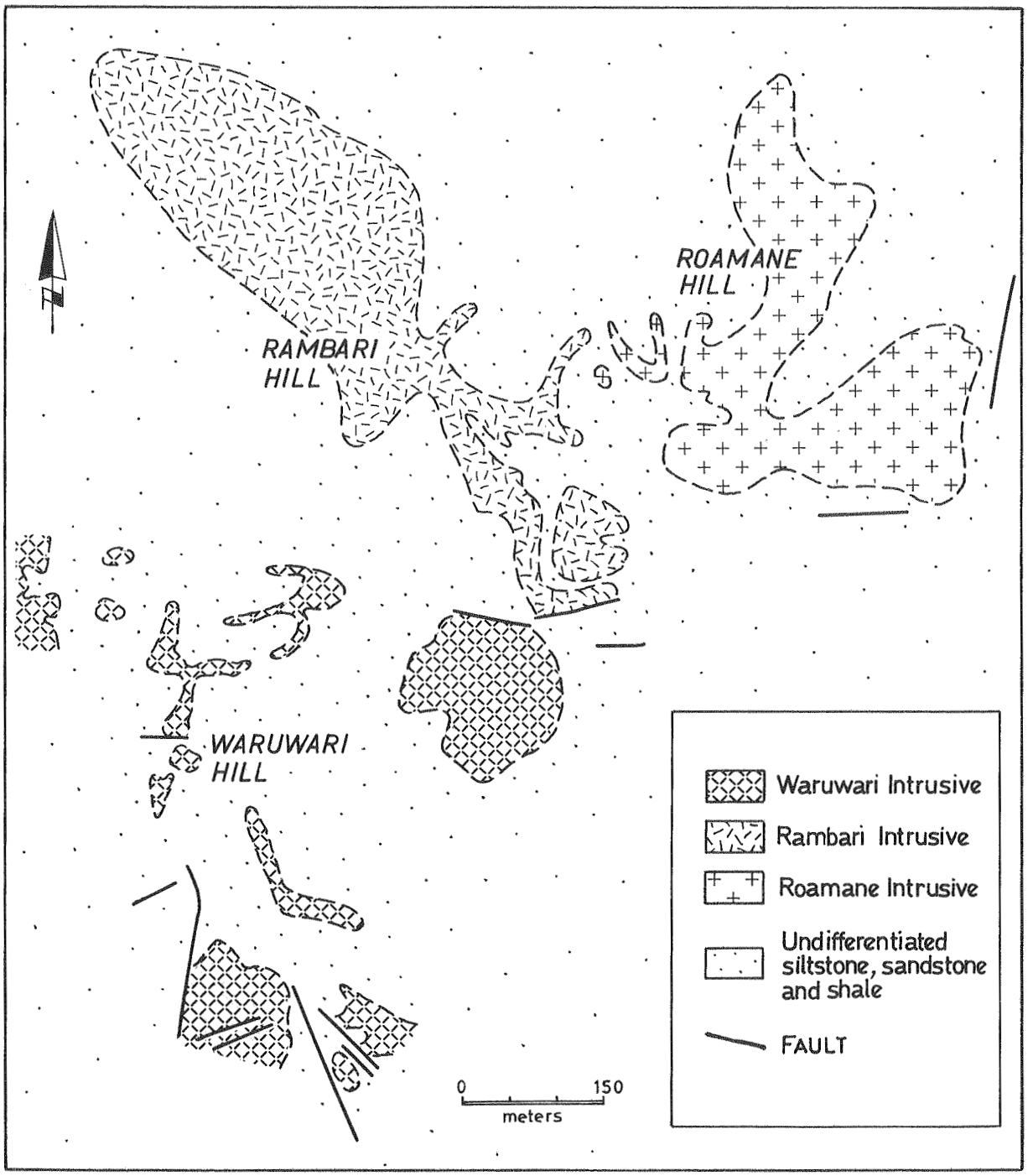


**FIGURE 2. Simplified geology of the Porgera gold deposit, Papua New Guinea (taken from O'Dea, 1980). Main proven mineralization in the Waruwari sector.**

accompanied pyrite (5-15 vol. %), quartz, chlorite, hematite and subsidiary chalcopyrite as vesicle fillings, disseminations and discontinuous veinlets in the weakly albitized flows. Interbedded andesites are devoid of sulfides and gold. To date, some 20 million tonnes of 2 g/t Au are claimed. The deposit is located from 1 to 3 km west of the Andacollo porphyry copper deposit (Fig. 5), which is the most gold-rich porphyry copper deposit in Chile (Sillitoe, 1981). Although the gold deposit occurs beyond the main pyrite halo of the Andacollo porphyry system, a metasomatic origin by solutions channelled along permeable felsic flows and numerous cross-cutting gold-quartz veins (Fig. 5) is favored. However, as suggested by Llaumett (1980), an origin of gold syngenetic with the enclosing flows cannot be discounted at this stage, a formational mechanism that would render the Andacollo gold deposit similar to Chilean manto-type copper deposits.

Sam Goosly in British Columbia, Canada, is a broadly stratabound silver-copper-antimony deposit carrying about 1 g/t Au. Herein it is assigned to a distal setting, alongside a copper-molybdenum-bearing quartz monzonite stock, although it is com-

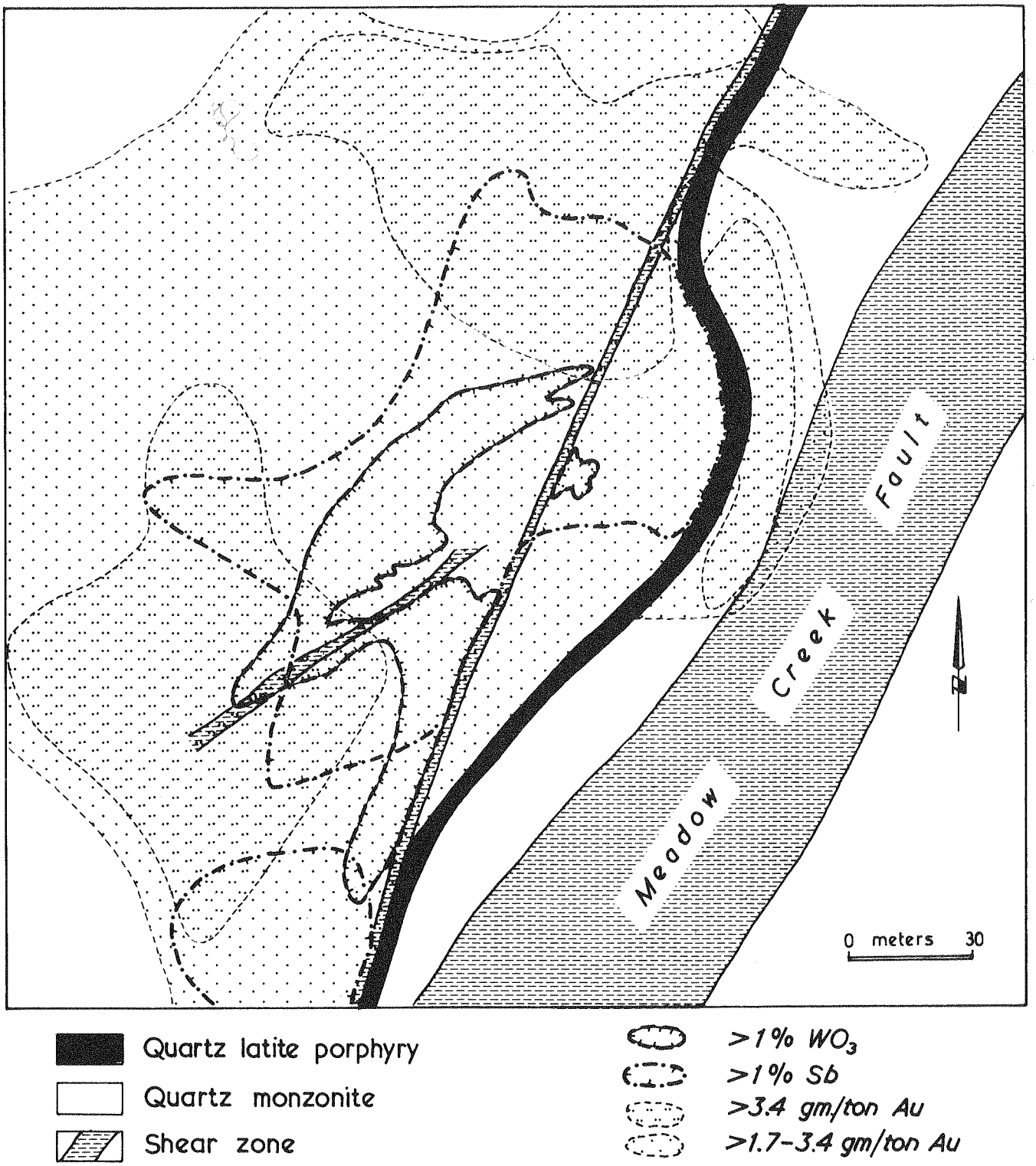


**FIGURE 3. Geology and distribution of Au, W and Sb on the no. 2 level, Yellow Pine mine, Stibnite district, Idaho (taken from Cooper, 1951).**

monly considered to be of volcanogenic origin and largely syngenetic with its host Lower Cretaceous felsic pyroclastic rocks (Ney et al., 1972).

## CARLIN-TYPE DEPOSITS

Carlin-type or disseminated (Roberts et al., 1971) gold deposits are considered here as a distinct category, although it is recognized that they could perhaps also be considered as distal contact-metasomatic deposits.

The best known deposits of this type occur in north-central Nevada and include Carlin, Cortez, Getchell, Gold Acres, Jerritt Canyon, Northumberland and Manhattan, but are also found farther east, as at Alligator Ridge, Nevada, and Mercur, Utah.

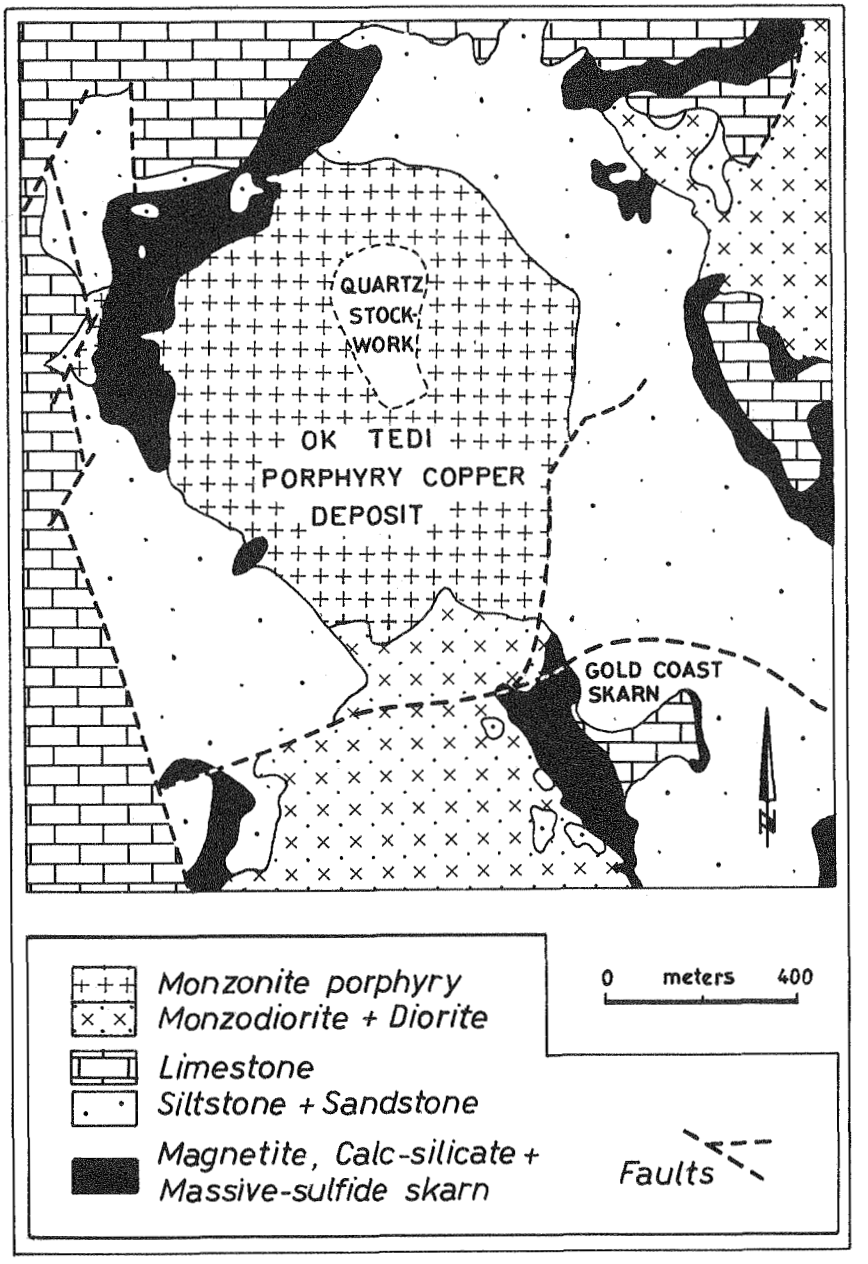


**FIGURE 4. Geology of the Ok Tedi porphyry and skarn-type copper-gold deposits, Papua New Guinea (after Howell et al., 1978).**

Although amenable to bulk mining, a number of these deposits are not strictly low-grade under present conditions: Newmont's Carlin deposit was originally reported to comprise 13.5 million tonnes of 10 g/t Au, and the proven part of Freeport Minerals' Jerritt Canyon deposit contains 9 million tonnes of 8.2 g/t Au. Others are, however, of lower grade, viz. up to 10 million tonnes of 6.5 g/t Getchell, 4.5 million tonnes of 3.4 g/t at Amselco's Alligator Ridge deposit, and some 10 million tonnes of 2.2 g/t at Cyprus Mines' Northumberland deposit.

As summarized by Radtke and Dickson (1974) and Radtke et al. (1980), most deposits of this type in north-central Nevada occur in thin-bedded, silty, carbonaceous dolomitic limestones of Cambrian to Mississippian age. Mineralization occurs along high-angle fault zones and as broadly stratabound bodies in permeable horizons extending away from the faults. Carlin-type ore is typically fine-grained and enriched in arsenic, antimony, mercury and thallium, which as realgar, orpiment, stibnite, cinnabar and other minerals accompanied by calcite and barite may fill late cross-cutting veinlets. Base-metals are uncommon, but are locally late additions. Mineralized limestones contain fine-grained introductions of pyrite (<2 vol. %), quartz and organic carbon. Massive silicification of limestones to form "jasperoid" is generally an accompaniment to Carlin-type deposits and in some, as at Northumberland, it hosts higher-grade ore. Hydrothermal breccias are also widespread and are well mineralized locally, as at Alligator Ridge. Unoxidized ores possess micron-size gold in carbonaceous material, with arsenic, antimony and mercury minerals present on the sur-

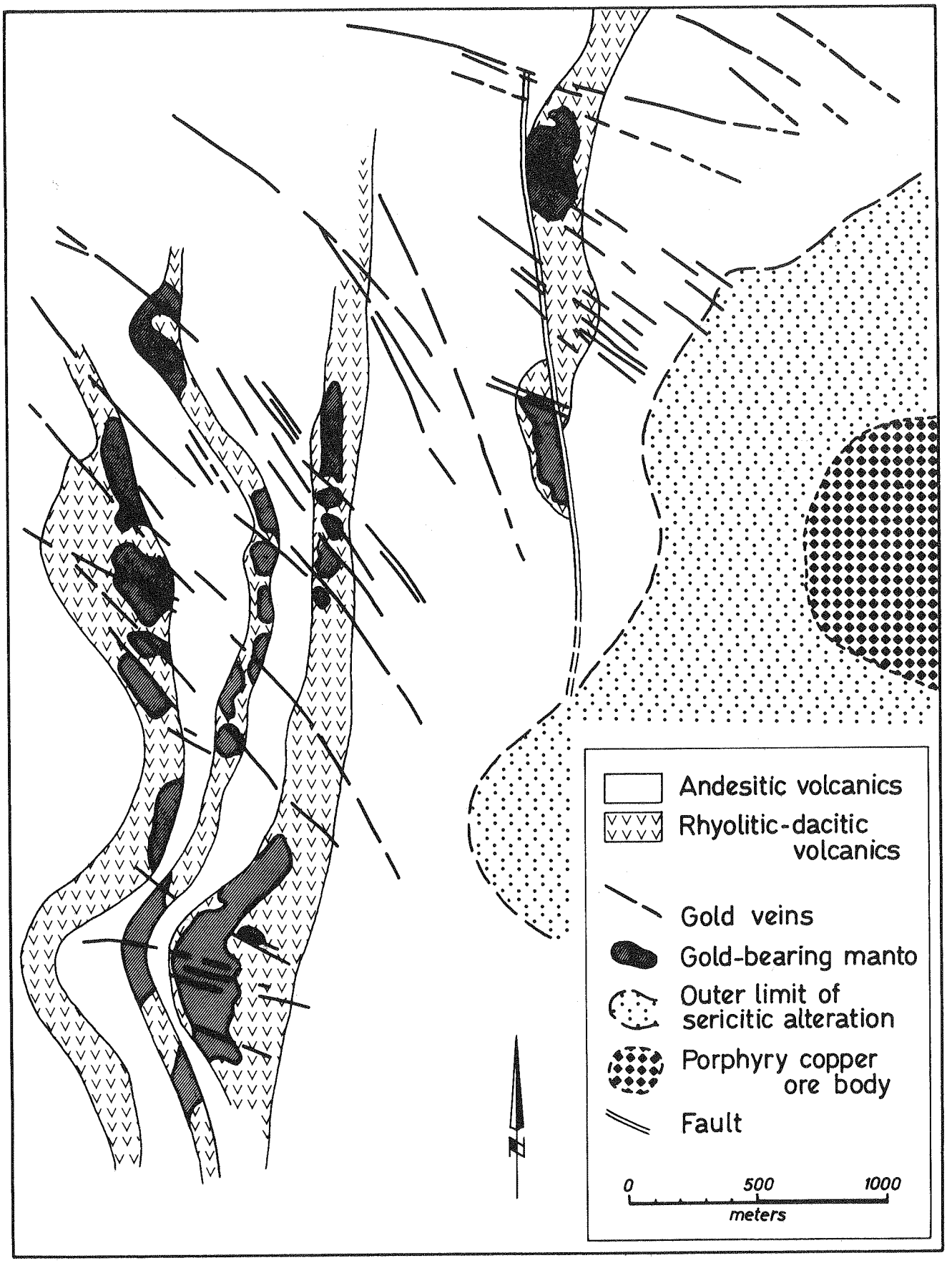


FIGURE 5. Stratabound and vein-type gold mineralization and porphyry copper deposit, Andacollo, Chile (after Llaumett, 1980).

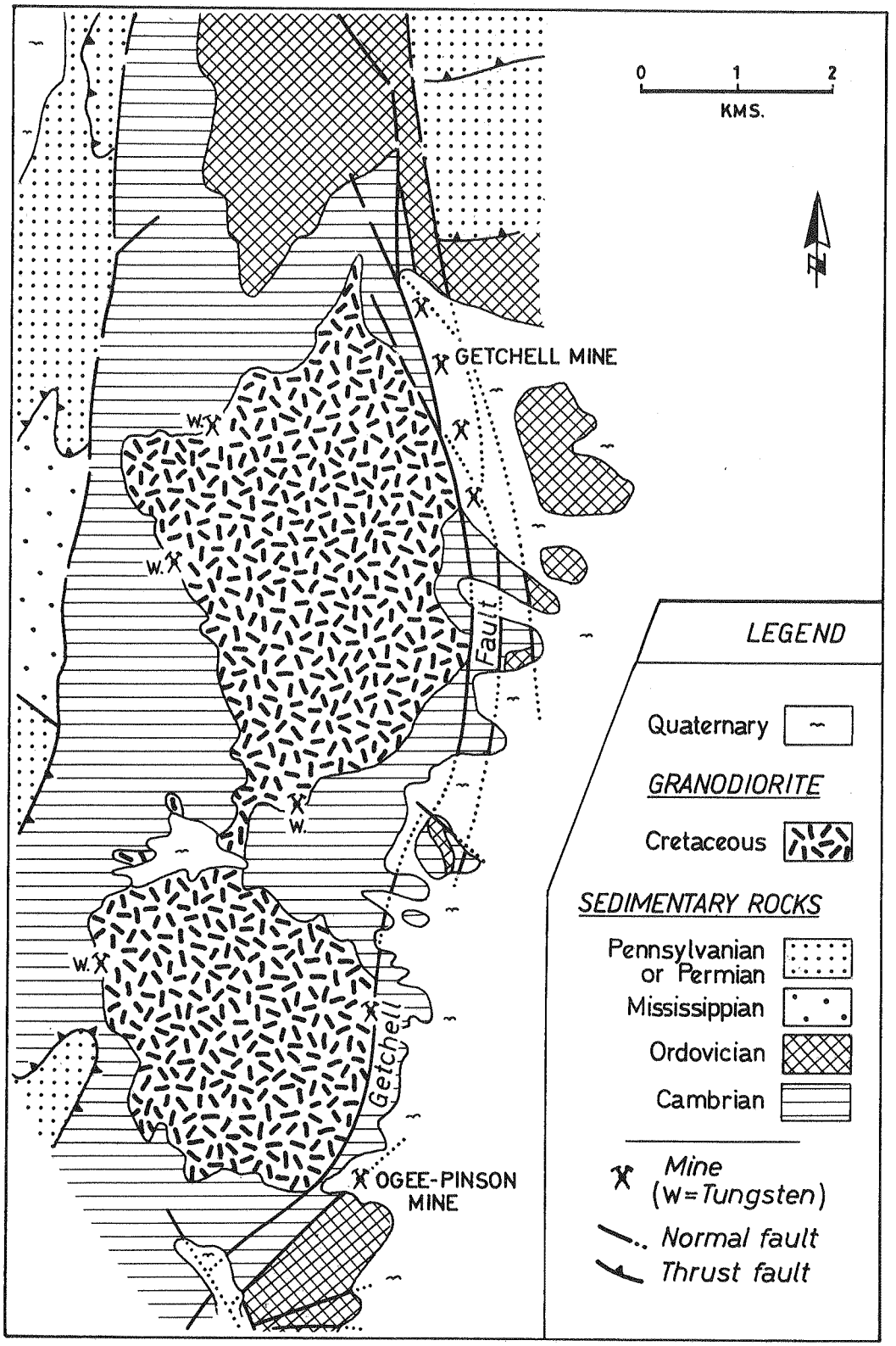


FIGURE 6. Geological setting of the Getchell gold deposit and nearby tungsten mineralization, Humboldt County, Nevada (taken from Silberman et al., 1974 and Berger, 1975).

faces of pyrite grains, and rarely in the native state in silica or clays (Radtke and Dickson, 1974).

The Nevada Carlin type deposits appear to have been emplaced at various times, with Getchell at 90 m.y., Gold Acres at 94-92 m.y., Cortez at 35 m.y. or later, and Manhattan at 16 m.y. (as summarized by Silberman et al., 1974, 1976), although some workers (e.g., Radtke and Dickson, 1974) prefer a late Cenozoic age, associated with Basin-Range faulting, for all deposits.

Some of the deposits possess no known relation to sizable intrusions, as at Carlin, but elsewhere intrusive rocks are spatially and perhaps genetically associated with the gold ore bodies: At Getchell, a granodiorite pluton associated with scheelite-bearing skarns abuts Carlin-type mineralization occurring along the high-angle Getchell fault (Fig. 6) (Berger, 1975); at Gold Acres, quartz monzonite was intersected at a depth of about 150 m during drilling beneath the heavily brecciated Roberts Mountain thrust zone, within which occur sericitized quartz porphyries, molybdenite- and sphalerite-bearing skarns and later Carlin-type mineralization (Wrucke and Armbrustmacher, 1975); at Cortez (Wells et al., 1969) and Northumberland, broadly sill-like intrusives of intermediate composition abut gold ore; and northwest of Carlin, at Blue Star and elsewhere, low-grade gold and copper mineralization occurs in intermediate-composition intrusive rocks beneath normal Carlin-type mineralization.

Similar gold deposits probably occur elsewhere, as at Moldova Noua in Romania, where gold mineralization associated with realgar is reported as the distal representative of a sequence comprising lead-zinc veinlets in limestone, copper-iron-bearing skarn and a porphyry copper-molybdenum deposit (Vlad, 1979).

## BRECCIA-HOSTED DEPOSITS

This category includes deposits in which essentially all gold is contained in hydrothermal breccias. It is distinguished from volcanic- and intrusive-hosted types in which part of the gold may occur in hydrothermally brecciated rock.

Hydrothermal breccias occur as pipe-like to irregular bodies and are characterized by angular to rounded fragments of one or more lithologies cemented by variable amounts of rock flour and hydrothermal minerals. Gold mineralization may occur in the rock-flour matrix, in cross-cutting veinlets or in inter-fragment voids. Two of the most important known examples are Placer Development Ltd. properties: Kidston in Queensland, Australia, reported to contain 39 million tonnes of 1.76 g/t Au, and Golden Sunlight in Montana, reported to contain 26 million tonnes of 1.7 g/t Au.

The Kidston pipe, 1200 x 600 m in surface dimensions, cuts Proterozoic granitic and metamorphic rocks and contains sericitized fragments of these rock types as well as fragments and bodies of felsite and rhyolite porphyry of probable Carboniferous age (Fig. 7) (Bain et al., 1978; Placer Exploration Limited, 1981). Although minor mineralization occurs with rock flour in the breccia matrix, most of the gold ore is present in a wide zone of quartz-filled sheeting which follows, and is partly superimposed on, the southwest marginal parts of the pipe (Fig. 7). Pyrite, sphalerite and galena, and minor chalcopyrite, pyrrhotite and arsenopyrite accompany the native gold.

At Golden Sunlight (Ageton et al., 1969), the shallow-dipping breccia pipe was emplaced during the early Tertiary into flat-lying Proterozoic clastic sediments cut by a latite porphyry sill and other minor intrusions. Breccia fragments are angular to rounded and are cemented by chalcedonic silica, which carries pyrite, gold and traces of sphalerite.

Gold Fields Mining Corporation's Ortiz deposit in New Mexico, USA, is also of this type. It is contained in a 215 x 120-m breccia pipe of Oligocene age in which fragments of Cretaceous quartzite predominate (Wright, this Volume). Minor latite porphyry fragments are argillized, sericitized and carbonated, and open spaces between fragments are partially filled by magnetite-hematite, pyrite, scheelite, copper sulfides, calcite and minor gold (Wright, this Volume).

The Cyprus Mines' Gilt Edge prospect in South Dakota is also dominated by hydrothermal breccias (Mukherjee, 1968). Several elongate breccia bodies accompany Tertiary latite and rhyolite porphyry stocks cutting a Proterozoic low-grade metamorphic basement. Breccias contain sericitized fragments in a silicified rock-flour matrix, and underwent introduction of pyrite, molybdenite and gold.

The so-called "Cresson Blowout" at Cripple Creek, Colorado, is a hydrothermal breccia pipe some 200 m in diameter, which transects an earlier diatreme filled with brecciated and pyritized rocks, dominantly latites and phonolites of Oligocene age (Lindgren and Ransome, 1906). Gold is present in tellurides with pyrite, sphalerite, tetrahedrite, stibnite and molybdenite in a quartz-fluorite-carbonate gangue, and was produced from the Cresson Blowout, epithermal vein systems and more irregular bodies. Recent surface sampling has outlined an area of 1150 x 150 m in the vicinity of the Cresson Blowout, where possible dispersed gold mineralization assays 2.5 g/t on average (Gott et al., 1969).

Similarly, at Wau-Edie Creek in Papua New Guinea, parts of large bodies of Pliocene hydrothermal breccia ("Namie Breccia"), composed of phyllite and dacite and andesite porphyry fragments and rock flour, have been open-pitted. Material from the leached capping carrying manganese-oxide (after rhodochrosite) veinlets is reported to assay 4.5 g/t Au whereas subjacent pyritic material is lower in grade (Rebek, 1975).

The so-called "Balatoc Plug" in the Baguio vein-type gold district of the Philippines is a major Quaternary breccia pipe, some 1000 x 600 m in surface dimensions and known for more than 1000 m vertically. Although it is only weakly propylitized and

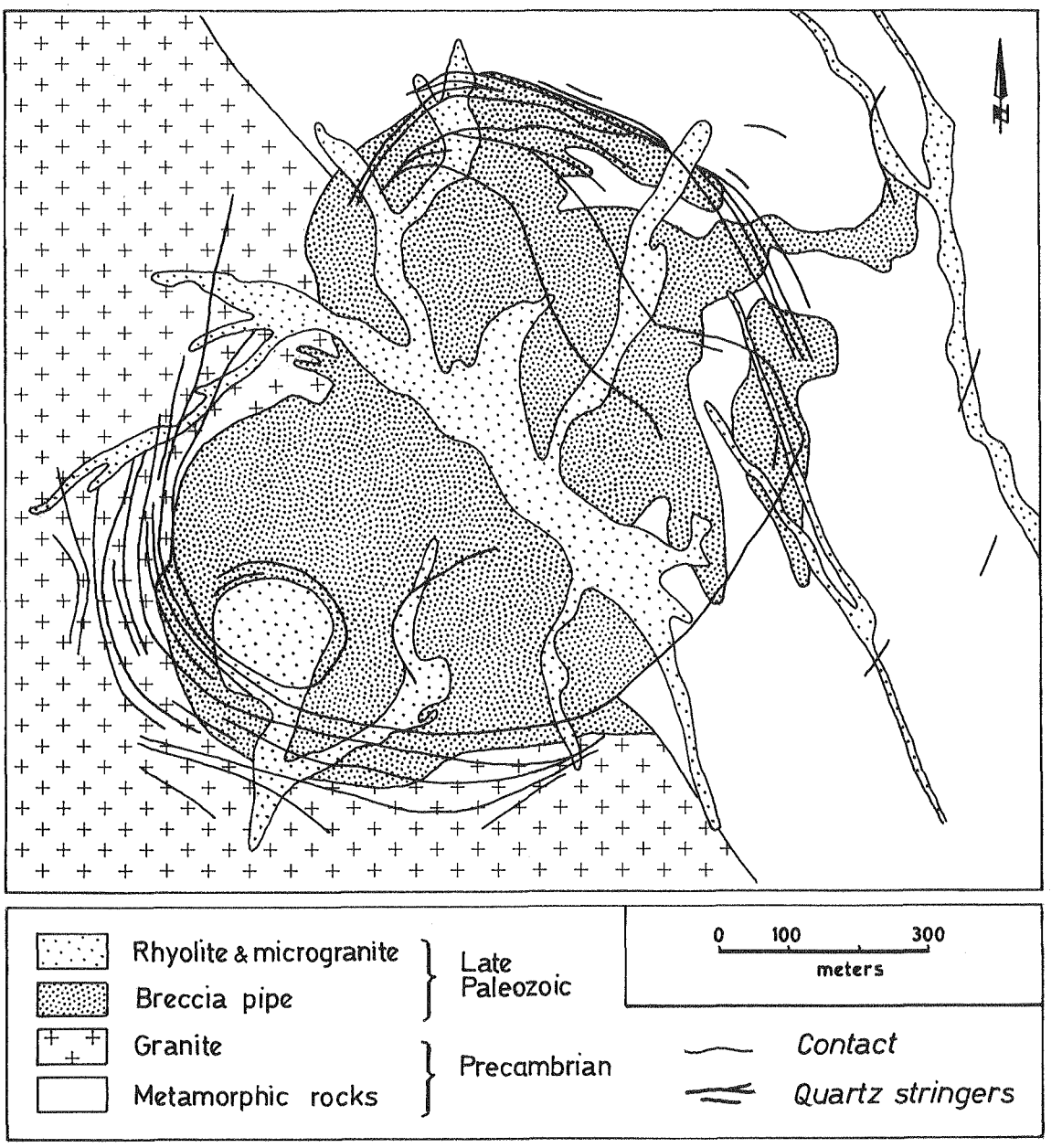


FIGURE 7. Geology of the Kidston gold deposit, Queensland, Australia (after Bain et al., 1978).

carries only low gold values, it is surrounded by a semi-continuous sleeve of more permeable breccia, up to 60 m wide, which hosts the nine G.W. orebodies (Worley, 1967; Damasco and de Guzman, 1977). The G.W. ore is present in pipe-like bodies, averaging 600 m in height, in which breccia fragments are cemented by chalcedonic silica carrying pyrite, marcasite, base-metal sulfides, stibnite and native gold. Damasco and de Guzman (1977) quoted ore reserves of 15 million tonnes of 3 g/t Au.

## ENARGITE-LUZONITE-BEARING MASSIVE SULFIDE DEPOSITS

In contrast to Kuroko-type massive sulfide deposits, treated below, the enargite-luzonite-bearing massive sulfide deposits are believed to form by replacement within a normally andesitic volcanic pile, commonly in the upper parts of porphyry copper systems (Sillitoe, 1980). Fine-grained pyrite is the principal component of the massive pods and lenses, which are flanked by pervasive silicification and advanced argillic alteration. Barite and native sulfur are common accompaniments to enargite and luzonite-famatinite. Hydrothermal breccias are widespread. Structural weaknesses, commonly faults, exerted a strong control on the location and form of the deposits.

The fourteen massive sulfide bodies at Bor, Yugoslavia, aggregate some 90 million tonnes of +1% Cu and significant but unreported gold. They occur in Senonian submarine andesites (Fig. 8) and contain enargite and hypogene covellite as the main copper-bearing sulfides (Jankovic et al., 1980). The Lepanto deposit in the Philippines contained some 27 million tonnes averaging 2 % Cu and 3.3 g/t Au,

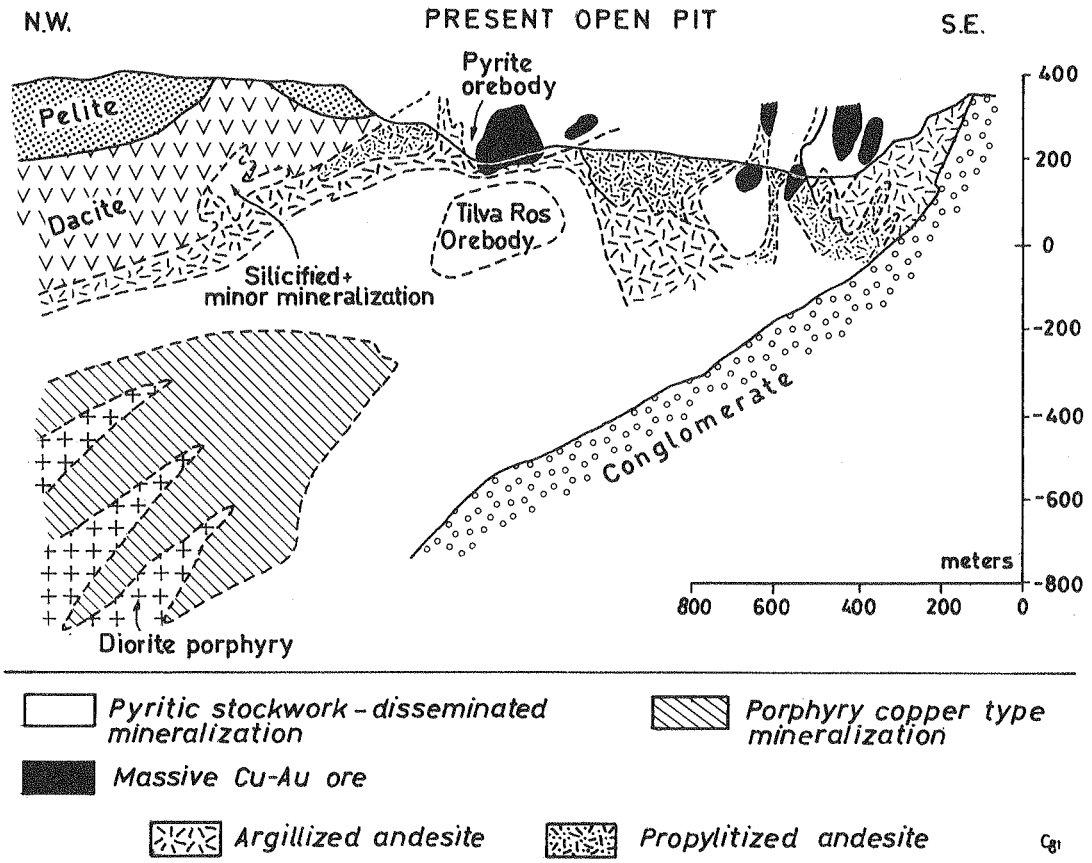


**FIGURE 8. Geology of the Bor massive pyritic copper-gold deposit, Yugoslavia, in relation to subjacent porphyry copper-type mineralization (simplified after Jankovic et al., 1980).**

and is localized by the intersection of a major fault and an unconformity between andesitic volcanics and Pliocene dacitic pyroclastics (Gonzales, 1956).

## VOLCANIC-HOSTED EPITHERMAL DEPOSITS

This broad category of low-grade epithermal gold deposits is distinguished from the two massive sulfide categories, which of course are also hosted by volcanic rocks. The deposits considered here are believed to be mainly of high-level epigenetic origin but to have been emplaced as integral components of volcanism—dominantly subaerial but locally subaqueous. Characterisitics and geometries of the deposits are diverse but here a subdivision is made on the basis of the associated hydrothermal alteration: acid leaching advanced argillic or more alkaline types.

Deposits associated with advanced argillic alteration are normally accompanied by pervasive fine-grained silicification. At Rosario Dominicana's Pueblo Viejo mine in the Dominican Republic, a major oxidized gold ore body underlain by sulfide mineralization occurs in a carbonaceous siltstone unit capping a bimodal volcanic sequence of Lower Cretaceous age (Russell et al., 1981). Submicroscopic gold mineralization is present in veinlets cutting silicified and pyrophyllitized siltstone and, in the sulfide zone, is accompanied by pyrite, sphalerite and enargite. There are a total of 37 million tonnes of oxide ore, 27 million of which average 4.2 g/t Au and 21.6 g/t Ag.

Dispersed gold mineralization at the old mining camps of Yalwal and Panbula in New South Wales, Australia, occurs in rhyolitic volcanic rocks of late Devonian age, which have undergone widespread pyritization and alteration to chalcedonic silica, pyrophyllite, sericite and kaolinite (McIlveen, 1974).

At Nalesbitan in the Philippines, a 125-m-thick, roughly stratbound body of low-grade oxidized gold mineralization occurs in late Cenozoic andesitic agglomerate and tuff. The volcanics are silicified, argillized and alunitized and underwent local hydrothermal brecciation. Gold originally accompanied some 10 vol. % pyrite, which was shown geochemically to have been associated with copper-, arsenic- and molybdenum-bearing minerals (J. P. McKibben, pers. comm., 1978). In Kyushu, Japan, low-grade gold (about 3 g/t) occurs with pyrite, enargite, luzonite and native sulfur in intensely silicified, alunitized and stockworked Neogene andesitic volcanics at Kasuga (Fig. 9) and Akeshi mines (Mukaiyama, 1970). Low gold values also accompany silicified, alunitized and pyrophyllitized trachyandesitic and rhyodacitic volcanics of early Miocene age over 40 km² at Goldfield, Nevada (Ashley and Keith, 1976), and silicified and sericitized Pliocene basaltic andesites over 4 km² at Gold Ridge, Guadalcanal, Solomon Islands (Hackman, 1980).

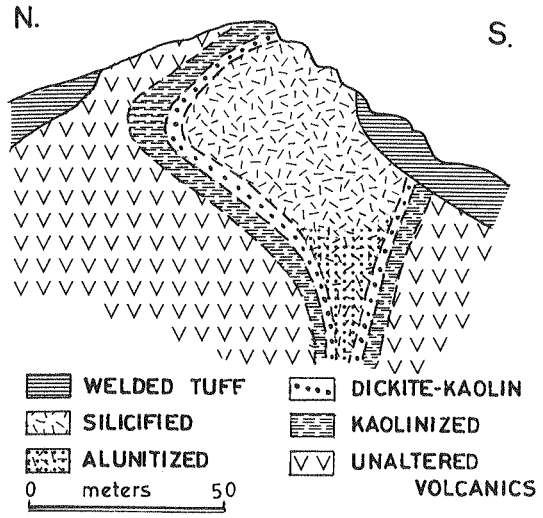


**FIGURE 9. Generalized geology and alteration of the Kasuga gold deposit, Kyushu, Japan (after Mukaiyama, 1970).**

Several examples of low-grade gold with less-acid alteration are known: at Round Mountain, Nevada, USA, the Smoky Valley Mining Co. exploits a 10-million-tonne oxidized gold ore body assaying 1.54 g/t Au, but has deeper reserves of 177 million tonnes containing 1.4 T g/t Au. As described by Berger and Tingley (oral commun., 1980), gold is present as disseminations and in now-oxidized quartz-pyrite and pyrite fracture fillings in a 300-m-thick rhyolitic ignimbrite flow of early Miocene age and intra-caldera origin. Alteration in the ore zone consists of relatively weak sericitization with some local adularia.

The Cinola deposit in the Queen Charlotte Islands of British Columbia, Canada, contains 41 million tonnes of 1.9 g/t Au ore in permeable carbonaceous conglomerates and sandstones of early Miocene age and in later (14 m.y.) silicified and brecciated rhyolite porphyry (Champigny and Sinclair, 1980; Kimbach et al., 1981; Cruson et al., this Volume). Auriferous pyrite-marcasite mineralization abuts a major high-angle fault zone, and is closely related to intense quartz veining and silicification of the brecciated contact of the rhyolite porphyry, and is also accompanied by minor sericite, illite, kaolinite and chlorite. Cinola is assigned to the Volcanic-hosted category because of the apparently near-surface environment of mineralization, and the presence of rhyolite tuffs only 500m to the west (Kimbach et al., 1981).

In the Santa Clara district of Costa Rica, United Hearne Resources and Canadian Barranca Corporation recently began exploitation of the Mondongo zone, where 5 million tonnes of 1.9 g/t Au is present in a partly oxidized pyritic stockwork cutting a propylitized andesitic sequence of probable Upper Miocene age.

These low-grade gold deposits in volcanic rocks may confidently be ascribed to shallow deposition over restricted vertical intervals in geothermal systems instigated by plutons at depth. As proposed by Barker et al. (1980), gold in the less-acid alteration types was probably precipitated at levels of boiling from low-salinity fluids of the type characteristic of modern geothermal systems, such as Broadlands, New Zealand. The highly silicified advanced argillic assemblages presumably reflect systems in which liberation of $H_2S$ during boiling gave rise to a highly acid zone of near-surface base leaching.

Some modern geothermal systems are actively precipitating gold. In New Zealand siliceous sinters contain gold grades of 2 g/t at Whakarewarewa, Rotorua and 5 g/t at Champagne Pool, Waiotapu. Antimony-rich siliceous precipitates forming as suspensions in hot-spring waters at Steamboat Springs, Nevada, contain 10 g/t gold (see Ellis and Mahon, 1977, p. 110-113).

## VEIN-TYPE EPITHERMAL DEPOSITS

There are two essentially epithermal vein-type situations in which bulk minable, low-grade gold potential may exist: a swarm of discrete but closely spaced veins amenable to extraction *en masse* or a single, exceedingly wide vein or shoot. An example of the first type at Marmato, Colombia, where a 6-m.y.-old composite stock of propylitized dacite and andesite porphyries is transected by gold-bearing vein zones characterized by quartz-pyrite-sphalerite and subordinate chalcopyrite and arsenopyrite, has been proposed as a bulk-minable proposition by Wokittel (1960) and others.

The second type is exemplified by two recently initiated open-pit gold operations: The Houston Oil and Minerals' open pit at the south end of the Comstock Lode, Nevada, USA, exploited the Con-Imperial ore shoot, bounded by hanging-wall splits of the shallow-dipping Comstock fault, where it cuts Miocene andesitic volcanics and interbedded sediments (Fig. 10) (McIntosh, 1980). The ore shoot, emplaced about 12 m.y. ago, comprises quartz veins, stringers and breccias carrying pyrite, base metals, native gold and silver sulfides and sulfosalts below the zone of oxidation and is associated with clay, sericite and adularia alteration. In some ways similar to the Comstock Lode is the Umuna Lode on Misima Island, Papua New Guinea, which is currently being investigated for its bulk potential. It is related to andesitic and dacitic porphyries intruded into greenschist-facies metamorphic rocks and carries quartz with lead, zinc and lesser copper sulfides as associates of fine native gold (de Keyser, 1961).

The second open-pit operation is that of Atlas Mining at Masbate, Philippines, which exploits submicroscopic native gold from a zone of oblique intersection, up to 120 m wide, of two quartz veins of probable Mio-Pliocene age. They cut Tertiary ande-

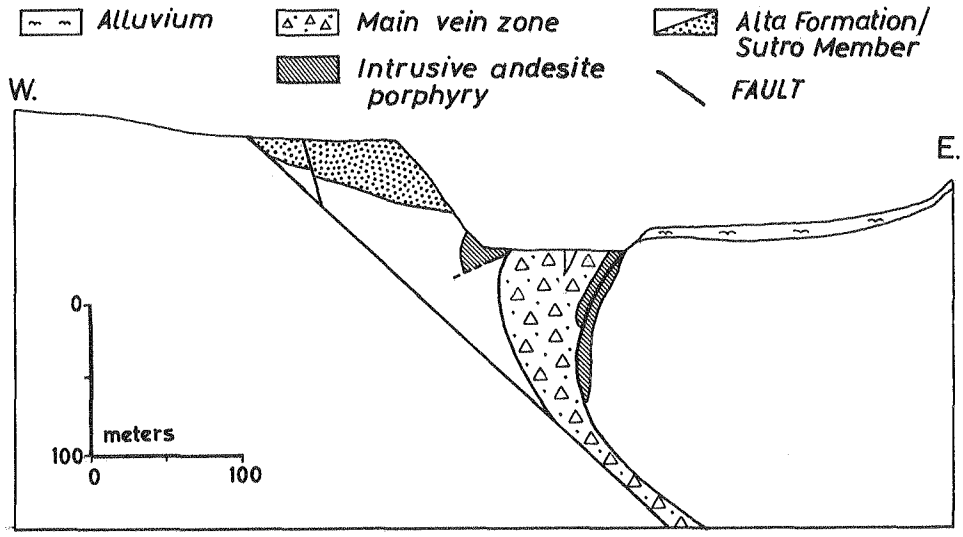


FIGURE 10. Geological section through the Con-Imperial open-pit gold-silver mine on the Comstock Lode, Nevada (after McIntosh, 1980).

sitic volcanic and volcaniclastic rocks and contain some 8 million tonnes of 2.7 g/t gold ore.

The Waihi deposit in North Island, New Zealand, could be considered as a combination of the two above-mentioned vein-type geometries and is currently under investigation by Amax Exploration. Well-defined quartz lodes cutting Mio-Pliocene dacitic flows attain 30 m in diameter and are associated with many subparallel subsidiary veins (Fig. 11). Gold is accompanied by pyrite, sphalerite, galena, chalcopyrite, calcite, adularia and chlorite beneath the oxidized zone (Williams, 1974, p. 110-114).

## KUROKO-TYPE MASSIVE SULFIDE DEPOSITS

A few large volcanogenic massive sulfide deposits, of broadly Kuroko type, contain significant quantities of gold. Of these, the most noteworthy is the Mt. Morgan deposit, Queensland, Australia, the volcanogenic origin of which has been confirmed by recent work (Taube, 1980). It is hosted by Middle Devonian felsic volcanic rocks and overlying cherts and tuffs. The gold-bearing ore body is pipe-like in form (Fig. 12) and originally graded downward from massive pyritic ore to massive silicification carrying veinlets and disseminations of pyrite. It has produced 51 million tonnes of about 2 % Cu and 4.6 g/t Au. At Mt. Lyell, Tasmania, Australia, disseminated and veinlet copper ore bodies were emplaced in a subsurface setting within Cambrian felsic volcanics and beneath only minor massive sulfide lenses. Although most ore bodies carry, only 0.25 to 0.45 g/t Au, the massive pyritic Blow deposit produced 5.6 million tonnes of 1.3 % Cu and 2 g/t Au (Reid, 1975).

In contrast to these copper-dominated volcanogenic ore bodies, 3.8 g/t Au occurs at Rosebery, Tasmania, in a +20-million-tonne, syngenetic sheet of massive pyrite-sphalerite-galena-chalcopyrite ore at a thin sedimentary horizon within Cambrian felsic volcanics (Burton, 1975). A smaller Lower Paleozoic example is open-pitted by Western Mines Ltd. on Vancouver Island, British Columbia, Canada, where three separate massive sulfide bodies possessed an original reserve of some 5 million tonnes of copper-lead-zinc-silver-gold ore with an average gold grade of about 2.2 g/t (Seraphim, 1980).

## THE LOW-GRADE GOLD SPECTRUM

The rather loosely defined categories of bulk low-grade gold deposits reviewed above may be considered as parts of a broad spectrum related to the various facets of arc magmatism (Fig. 13a, b). Deposit types span the interface between intrusive and volcanic processes and environments: Gold-rich porphyry copper and skarn deposits are the most closely related to intrusive activity and were generated by hydrothermal fluids of obvious magmatic parentage (e.g., Panguna, Eastoe, 1978; and Copper Canyon, Batchelder et al., 1976). Although the requisite data are not available, distal contact-metasomatic deposits and hydrothermal breccias associated with intrusive rocks (e.g., Golden Sunlight, Kidston and Ortiz) were probably also generated from magmatic-hydrothermal fluids, albeit with a greater probability of appreciable meteoric water involvement. Many of the other categories may be classed as epithermal and probably had any magmatic fluid contribution to their makeup masked by groundwater dilution or, alternatively, never had a significant magmatic input. Examples in which meteoric dominated fluids of typical low-temperature, low-salinity epithermal type deposited the precious metal ores include Goldfield, Comstock Lode and Round Mountain (O'Neil and Silberman, 1974), the Baguio district (Sawkins et al., 1979), Carlin (Radtke et al., 1980) and Cortez (Rye et al., 1974). At the Comstock Lode, however, a sample from the deeper levels provided light stable-isotope evidence for mixing of magmatic and meteoric fluids (O'Neil and Silberman, 1974). In contrast, the massive sulfide deposits of broadly

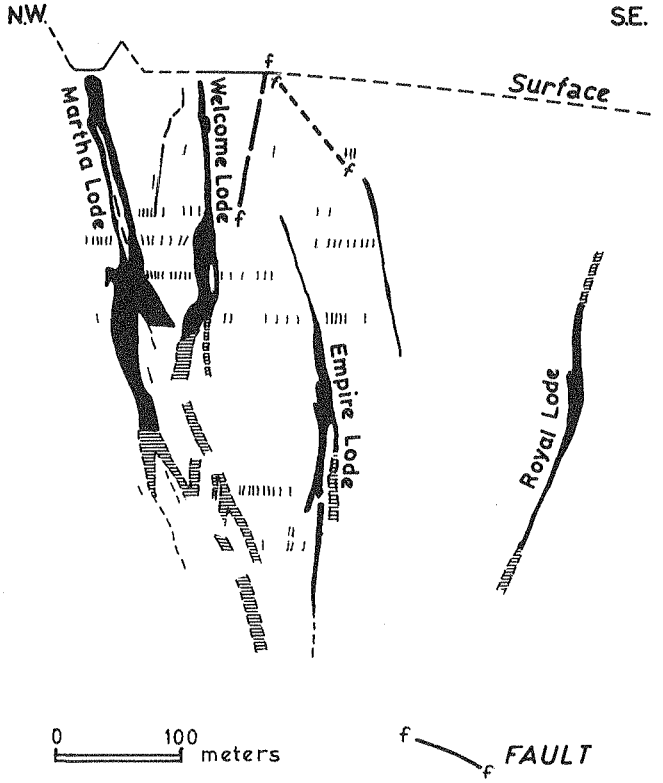


**FIGURE 11. Cross-section through the Waihi gold mine, North Island, New Zealand (after Williams, 1974, p. 113).**

Kuroko type were deposited from fluids containing appreciable sea-water, as recently demonstrated at Mt. Morgan (Golding and Wilson, 1981).

When considered in more detail, many of the categories are seen to possess transitional characteristics and even to overlap one another. This is very evident in the intrusive-related types, where porphyry coppers are abutted by skarns (Ok Tedi) or distal contact-metasomatic deposits (Andacollo), or skarns give way outwards to the distal contact-metasomatic type (Battle Mountain district) (Fig. 13a). Upwards in porphyry copper systems, gold-bearing potassium silicate alteration gives way to feldspar-destructive alteration types, in which high gold values are known at Cabang Kiri. Although nowhere proven, it is suggested that some of the gold-bearing hydrothermal intrusion breccias occur as upward extensions of porphyry-type mineralization. Some of the enargite-luzonite-bearing massive sulfide deposits are also believed to have been emplaced in feldspar-destructive habitats high in porphyry copper systems (Fig. 13a). Many of the volcanic-hosted deposits were probably emplaced in volcanic piles unrelated to porphyry systems (Fig. 13b), but examples hosted by andesitic volcanics and characterized by advanced argillic alteration (Nalesbitan, Goldfield) could be porphyry-related (Fig. 13a). Furthermore, some deposits associated with felsic volcanics (Pueblo Viejo, Sam Goosly) are not easily distinguishable from epigenetic feeder zones to Kuroko-type massive sulfide deposits that are gold-bearing at Mt. Morgan and Mt. Lyell (Fig. 13b, nos. 8 and 10).

Worthington et al. (1980) have suggested that some volcanic-hosted deposits, such as the late Precambrian Haile in South Carolina, USA, are partly syngenetic and were generated subaerially as part of hot-spring sinters. It is maintained that all the volcanic-hosted deposits listed above are epigenetic albeit near-surface phenomena. It is recognized, however, that gold-bearing sinters, comparable to those referred to above in active geothermal systems, may originally have capped some deposits. Moreover, at Pueblo Viejo, early pyrite in carbonaceous siltstone is certainly syngenetic and was probably deposited in a freshwater lake (Russell et al., 1981).

Other similarities between categories are also apparent. The stockwork style of mineralization in intrusive-hosted epithermal deposits at Porgera and Salave is reminiscent of porphyry-type deposits, although the characteristic mineralogy and alteration zoning are lacking—even though Salave contains molybdenum. The breccia-hosted deposits have clear transitions to both volcanic- and intrusive-hosted deposits, both of which may include important volumes of hydrothermal breccia (Nalesbitan, Porgera, Mondongo, Cinola), and locally appear to have an affinity to porphyry deposits, as in the case of the molybdenum-bearing Gilt Edge. Some Carlin-type deposits appear to occur distally with respect to skarn deposits (Getchell) (cf., Roberts et al., 1971) and may therefore possess spatial similarities with some members of the distal contact-metasomatic category (Minnie-Tomboy). Moreover, in the Getchell district (Fig. 6), low-grade molybdenum mineralization occurs in the adjoining granodiorite pluton (Berger, 1975), and in the Carlin district, intrusive rocks carry low-grade copper and gold mineralization. There seems to be no good reason why aspects of the "hot-spring" model for Carlin-type genesis (e.g., Radtke et al., 1980) should not be applied to the distal environment, either above or marginal to plutons, which may or may not possess metalliferous skarn developments, although this would tend to imply a magmatic origin for gold and its associated elements rather than one involving leaching of the sedimentary pile.

Some Carlin types may also be transitional to the volcanic-hosted category: at Round Mountain, for example, gold-bearing sedimentary rocks underlie the mineralized ignimbrite, and at Manhattan Nevada, small amounts of gold ore occur in Paleozoic schists and in silicified parts of a Tertiary volcanic succession beyond the main mineralized limestone horizon (Ferguson, 1924). Salave, the Stibnite district and the New Zealand hot-spring sinters could also be considered to possess similarities with Carlin-type deposits, in these cases because of the characteristic gold-arsenic-antimony association. Finally, it should be remarked that Cinola was assigned to the Carlin-type category by Champigny and Sinclair (1980), based on a number of characteristics including its anomalously high mercury content.

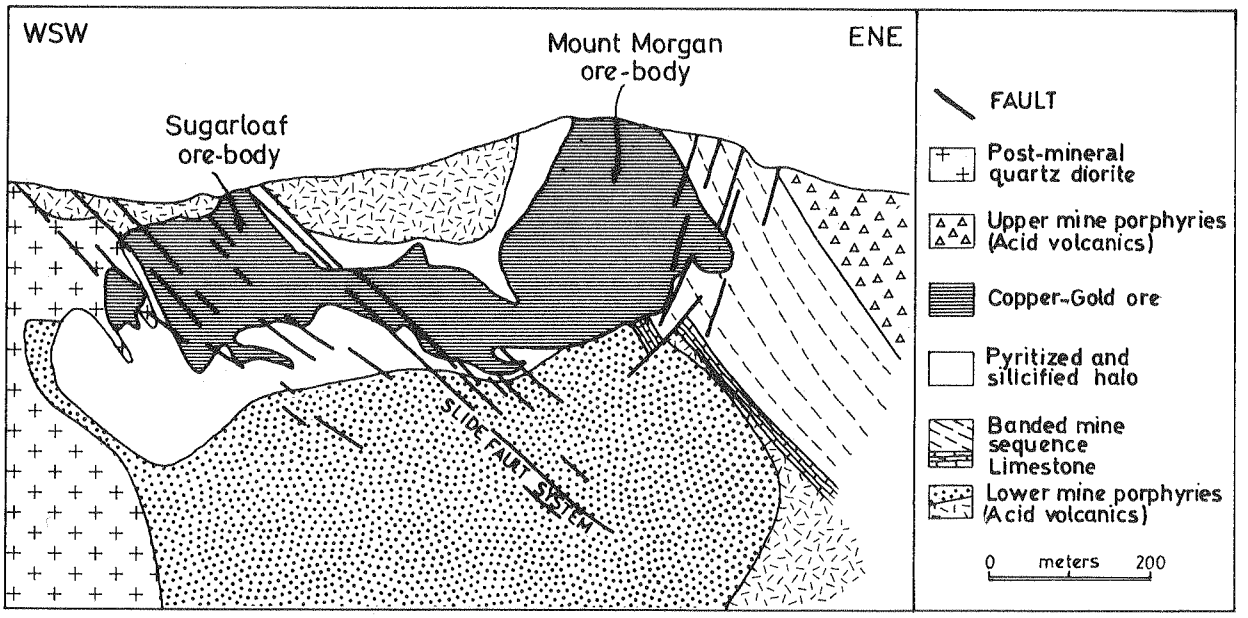


FIGURE 12. Generalized longitudinal section of the Mt. Morgan copper-gold mine, Queensland, Australia (after Taube, 1980). Typical massive sulfide geometry clarified by restoring Banded Mine Sequence (including exhalites) to horizontal and reversing movement on Slide Fault System, thereby placing Sugarloaf stockwork ore body immediately beneath Mouth Morgan massive ore body.

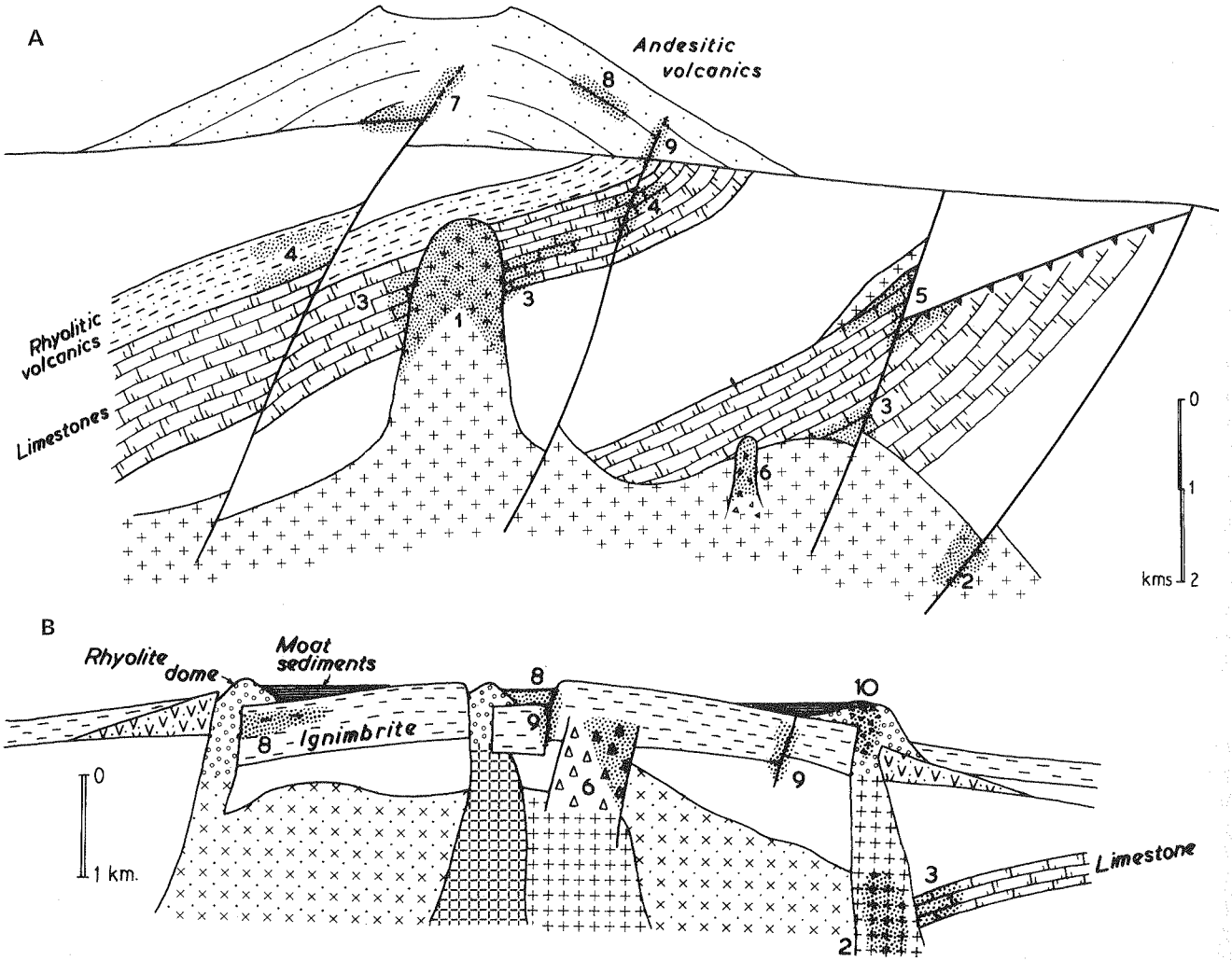


FIGURE 13. Schematic representations of possible positions for bulk low-grade gold deposits in two volcano-plutonic settings: A) a stratovolcano, and B) a submarine resurgent caldera. Note that all deposits in B, except for Kuroko-type massive sulfides, also may be emplaced in a subaerial setting. Key to numbers: 1. porphyry copper-gold; 2. intrusive-hosted epithermal; 3. contact-metasomatic: skarn; 4. contact-metasomatic: distal; 5. Carlin-type; 6. breccia-hosted; 7. massive enargite-luzonite; 8. volcanic-hosted epithermal; 9. vein-type epithermal; and 10. Kuroko-type massive sulfide.

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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# SOME FACTORS IN THE DESIGN OF HEAP LEACHING OPERATIONS

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## ABSTRACT

This report reviews and discusses some factors in the design of heap leaching operations using dilute alkaline cyanide solutions to dissolve gold and silver from relatively coarse ores. Some of the requirements for laboratory and pilot work to permit reliable scale-up for plant design are presented. The influences upon design of such factors as ore reserves, available terrain, crushing requirements, ore preparation, pad materials, leaching solution strengths, solution application rates, and precious metals recovery methods are briefly outlined. Alternative methods of recovering the precious metals from solution and their subsequent smelting to a doré bullion and some factors affecting choice of method are reviewed. The role of activated carbon in gold-silver recovery and various means of recovering the metals from subsequent stripping operations are discussed. Other viable precious metal recovery methods, including Merrill-Crowe zinc dust precipitation and soluble sulfide precipitation of silver, are described. A few notes on some recent published plant capital costs are mentioned.

## INTRODUCTION

During the late 1960's the mining industry became interested in the U.S. Bureau of Mines' efforts to develop the gold-silver heap leaching process. In the decade 1970-1980, successful commercial heap leaching applications increased. The process provides a relatively low-capital means of processing treatable ores with reasonable operating costs. Present gold and silver prices compared to capital and operating costs favor the continued growth of the heap leaching method.

The heap leaching process (Figure 1) is defined as the percolation leaching of relatively coarse gold-silver ore piled on an impervious surface. To date, dilute alkaline cyanide solution is the lixiviant of choice. However, other leaching reagents such as thiourea and hypo have been suggested, although not currently used in heap leaching practice to my knowledge. Ores may be mine run in size or crushed as fine as about 3/8-inch (1 cm.). Gold and silver-bearing solutions draining from the heaps may be processed in activated carbon columns to recover (adsorb) the metals followed by stripping and metal recovery, or treated by the Merrill-Crowe zinc dust precipitation method. All solutions are generally recycled to the leaching step, and the heap leaching system is relatively easy to control from the environmental viewpoint.

A number of helpful references from the Bureau of Mines, consultants, and industrial sources describing heap leaching are available in the literature. A selected list of references is attached, and these valuable references and supplementary personal communications have been instrumental in preparing this report.

The flexibility of the heap leaching process should be emphasized. Plants may be very simple, comprising leaching of uncrushed ore on a locally-obtained clay lined pad and shipment of gold and silver recovered in the form of loaded carbon or zinc precipitates; such a plant would have a relatively low capital investment. More elaborate plants for larger ore bodies and available financing may be planned to include crushing, cyclic pad use, and doré gold-silver recovery. The general planning is similar in each case.

The purpose of this report is to outline some key factors in the design of heap leaching operations and how they are applied to develop a design for a working operation. Individual design requirements and local construction costs vary so much that no attempt has been made to list detailed costs. A few general comments on costs are included.

## OREBODY, RESERVES, PROCESS EVALUATION AND PRELIMINARY FEASIBILITY STUDY

The evaluation of a given mine ore reserve in terms of tonnage, grade, and process amenability is a basic prerequisite for any process design including heap leaching. As mine sampling of a new or unknown orebody proceeds, preliminary metallurgical tests should be made concurrently so that continuously updated information on ore process response is available. By the time the ore body is reasonably well defined, preliminary cost feasibility studies should be made to determine profitability obtainable by various process alternatives using several assumed metals prices and process conditions. Thus, the first step in design is a feasibility study to enable:

1. Decision to continue with the project
2. Selection of an appropriate process (assumed to be heap leaching in this case)
3. Estimation of a tentative allowable budget for mine and process plants under assumed conditions
4. Determination of the estimated life of the operation, tonnage rate per day, yearly tonnage, ore grade, and metals recovery.

## PERMITS, SERVICES, AND ENVIRONMENTAL FACTORS

The availability and cost of power, water, and other services have a large influence on design and should be studied early. Similarly, operating permits and the effect of environmental regulations upon design should be outlined promptly because of their key role in design or proceeding with design. For ex-

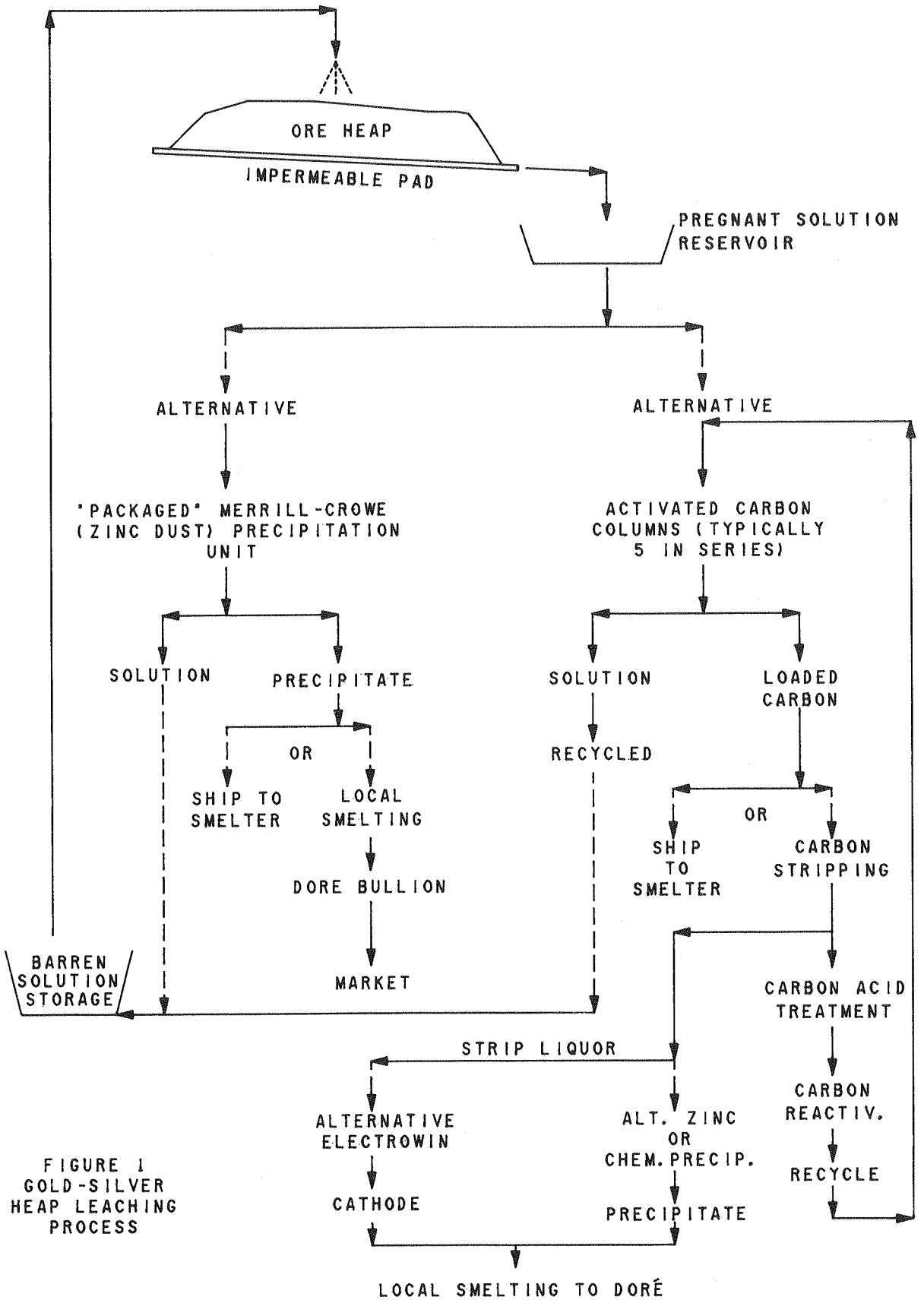


FIGURE 1
GOLD-SILVER
HEAP LEACHING
PROCESS

FIGURE 1.

ample, environmental requirements in tailings storage might influence the choice of pad design as to cyclic or one-time use, as well as the design of the storage area. The climate notably affects design through effect on the annual operating time under part-time freezing conditions, and the need for insulating and heating buildings and process units.

## TOPOGRAPHICAL MAPS AND PLOT PLAN

The best available topographical maps will already have been used to some extent in the preliminary feasibility study and are essential in preparing a plot plan. The final design of a heap leaching operation is greatly influenced by the topography and location of the leaching pads. Pad size, type, and pumping costs are linked to available areas and elevations. The location of solution reservoirs and the process plant is likewise a critical design decision.

## DETAILED METALLURGICAL TESTING FOR SCALE-UP

The ore mineralogy and rock permeability affect the amenability of ores to the heap leaching process. Treatable ores should be porous, competent, and satisfactorily low in cyanide and oxygen consuming substances. The gold and silver particles should be clean and fine in size. The clay content must be low enough so that permeability in a leaching heap can be maintained, or else the ore may be agglomerated as outlined in recent USBM work [10]. Each ore deposit presents different characteristics, so that detailed laboratory and pilot tests are required to determine the optimum flowsheet in each case. Tests may be made in bottle rolls, small columns, large columns, and actual heaps of various sizes. The use of all these methods as keys to scale-up and design is recommended, as outlined in the following discussion.

1. Preliminary tests in bottles and small columns are helpful in showing response to cyanidation at sizes up to 3/8-inch (1 cm) or so.
2. Larger column tests are a reliable guide to scaled up operations. Columns may be thought of as a core in the center of a large leaching heap, with the same access to solution and oxygen. The column diameter should be four times the largest rock size to avoid undue wall effects. Some essential design factors obtainable from columns include the following:
   - amount of solution needed to saturate the ore
   - amount of solution that drains from saturated ore
   - amount of moisture retained by drained ore
   - size of crushing versus leaching time
   - optimum cyanide strength and consumption
   - choice of alkali (frequently lime), strength, alkali consumption, pH
   - solution and wash application rate
   - height of the leaching ore column
   - amount of time of draining and washing
   - oxygen content of effluent
   - feasibility of countercurrent leaching and/or washing
3. Pilot heap tests are used to verify most of the factors listed under column tests and also to provide information on some physical factors of heap construction and leaching. However, heaps smaller than 500 tons or so, unless very shallow, have an excessive slope area compared to the level part of the heap and may mislead on oxygen availability. Heaps supplement columns by indicating:
   - mode of applying solution (sprinklers, wigglers, drip, etc.)
   - confirmation of solution application rate
   - physical factors of heap construction
   - general confirmation of column test factors
   - compaction under leaching conditions
4. Laboratory and pilot gold and silver recovery tests by the activated carbon, Merrill-Crowe zinc dust precipitation, soluble sulfide, and other methods are helpful supplements to column and heap leaching tests. However, the selection of the recovery method from leaching solutions is finally based upon economic analysis of the probable solution flow and analysis, together with experience.

## DEVELOPMENT OF THE METALLURGICAL FLOWSHEET AND DESIGN

Each ore body differs in important respects from others. Therefore, the flowsheet and plant design must be based upon individual tests as previously outlined. Nevertheless, there are a number of common factors derived from operational experience that are summarized in the following sections.

1. *THE MINE PLANT*

   The mining method for heap leaching has usually been open pit. Mines vary so much that the type and size of equipment cannot be specified here but adequate maintenance (usually at the property) is absolutely essential whether contract mining or company-operated. Ore sampling and assaying facilities near the mine are a requisite for grade control. The

degree of mechanization such as conveyor belt ore and waste handling is tied to ore reserves and the available investment budget.

2. *THE ORE PREPARATION*

The optimum degree of ore crushing (if any) will have been determined by laboratory and pilot tests. High prices of gold relative to general cost indices usually favor relatively fine crushing to improve ultimate recovery and shorten the leaching cycle. Most ores not containing excessive clay will percolate fairly well when crushed as fine as 3/8-inch (1 cm) or so. Ores presenting percolation problems may be agglomerated by USBM-suggested techniques [10], and cyanide and alkali may be added during agglomeration. A properly sized agglomerating drum might be chosen, or sufficient agglomeration might be accomplished by sprays and mixing at belt transfer points. The complexity of the crushing plant is dictated by leaching size requirements and available budget. Jaw or gyratory primary and cone crusher secondary and tertiary stages are typical if finer crushing is needed. Screening practice is standard. The selection of equipment sizes is in accordance with tonnage requirements, with suitable maintenance time available. Dust control is essential. Dust slurries from control may be pumped onto the leaching heaps.

3. *ORE SAMPLE AND WEIGHT*

Many heap leaching feeds are "spotty" in gold and silver content (contain relatively coarse values). Automated sampling to provide a large bulk sample of ore to each pad is recommended. Probably the bulk sample will typically be crushed and reduced in weight by a secondary automatic sampler at least once before going to preparation and assay. Spotty head samples should be analyzed on the basis of a 5- to 10-kilogram weighed lot, processed by gravity, cyanidation, fire and atomic absorption assay for a calculated combined head value. A weighing device should record total weight to each pad. Hand moisture samples are common. The design should not skimp on sample plant or weighing control, usually placed just before the delivery point to the pads whether a truck bin or a belt. Small properties may sample by hand.

4. *DELIVERY OF ORE TO PAD AND HEAP CONSTRUCTION*

Ore is delivered to the pad by truck or belt, depending upon individual cost analysis of capital and operating factors. Heaps up to 14 to 16 feet (4 to 5 meters) deep may be built without driving a truck over the heap; trimming is provided by a front-end loader. Trucks may be 50 ton (45 metric tons) units or larger and loaders about 8 yards (6 cubic meters). Deeper heaps may be built without running trucks over the main bulk of the ore by pushing dumped ore with a dozer or loader. Ripping of machine-compacted surfaces (if any) is recommended. Heaps may also be built by conveyor belt delivery and tripper distribution from a bridge unit. In all cases, the optimum depth of the heap should have been verified by full-height column tests.

5. *PAD DESIGN*

The choice of pad material, pad size, and design must be determined by local factors in each case. Some key elements in the selection are available area, length of the leach cycle, maximum practical height of the leaching heap, availability of local clays, local construction costs, and availability of tailing storage areas. Pads may be built for one-time or cyclic use. Environmental regulations, likewise, have a critical influence on design, as does available capital and estimated materials (ore and residue) handling costs.

Hot rolled asphalt is probably the material of choice for cyclic use. The design is based upon the heaviest load to be carried per vehicle tire, usually the front-end loader. A 2-inch (5 cm) layer of asphalt, followed by a rubberized seal coat, then 5 or 6 inches (10 to 15 cm) of hot rolled asphalt has been successful for loaders to 8 yards or so (6 cubic meters). Earthen foundations should follow best preparation practice in grading and compaction; slopes are generally at least 3 degrees in two directions to provide for solution recovery in a parallel ditch. An inter-pad curb to provide solution isolation and sampling capability from each leaching unit is a great operational help. Maximum heap depth to avoid running trucks over the heap is probably 15 to 16 feet (about 5 meters) in the case of truck/loader preparation. If trucks run over the heap, the heap should be ripped. In the case of bridge-loaded pads the height may be maximum efficient leaching depth.

Clay linings of 18 inches (46 cm) thickness or so placed on a prepared earthen foundation may be suitable for single-use pads. The particular clay proposed for use should be tested for leakage in the laboratory using the same alkalinity and water as planned for production. Even a small leakage at the present price of gold is very painful. Plastic membrane-lined drainage may be used.

Reinforced plastic membranes can be used for single-use pads. Graded, compacted bases should be covered with about 6 inches (15 cm) of rock-free cushioning sand to help avoid puncture. The membrane strips are joined in the field, then covered with an additional layer of

cushioning fine gravel or sand. Butyl plastic of 30 mil (0.8mm) thickness, reinforced with glass fiber, has been used successfully. Drainage canals may also be plastic-lined.

In all cases, bore holes should be angled under each pad to detect any possible leakage.

6. *APPLYING SOLUTION AND WASH*

Solutions are delivered to the pads by centrifugal pumps via steel or plastic pipe headers that may be buried, insulated, and/or heat-traced for freezing climates. Conservative design indicates relatively large pipes to reduce friction losses and pumping costs. Distribution to the ore heap by sub-headers is usually in plastic pipe with a gradient toward the main headers and thus drainable in case of freezing weather.

Solution application is usually sufficient at 0.002 to 0.005 gpm/sq. ft.(.0014 to .0034 liters/sec./sq. meter). Higher rates of application simply dilute the grade of the pregnant solution. Steady flow appears satisfactory in most cases. Although pregnant solutions may show increased value after a period of "rest", cumulative production usually falls on a steady flow curve, provided that solutions have enough oxygen and cyanide at all times.

Both "wigglers" (short lengths of surgical gum tubing) and various mechanical sprays have proved satisfactory for applying leach solutions. Depending on pressure, wigglers may cover a 30-foot (9 meter) radius and mechanical sprays a larger radius. Spray types providing larger drops favor conservation of cyanide, water, and heat. More work is needed to define spray drop sizes and also to apply water- and heat-conserving solution applying devices such as drip tubes.

For best washing efficiency, several fresh water washes should be applied with draining periods between if time permits. Allowable water washing is governed by evaporative and tailing-retained losses, and all makeup should be applied to the washing heaps to promote washing efficiency.

Solutions may be recycled to dumps to build up the grade, provided that soluble losses in the residues do not increase because of washing problems.

7. *SOLUTION RECOVERY AND STORAGE*

Pregnant solutions and wash draining from heaps flow by gravity in plastic, asphalt, or otherwise-lined ditches and steel or plastic pipes to storage. The storage tank or pond should be large enough to retain 24 hours of gravity drainage from saturated dumps plus a maximum estimated rainfall in the event of extended power failure. Asphalt, concrete, plastic-lined, and clay-lined storage reservoirs have been used satisfactorily.

8. *GOLD AND SILVER RECOVERY PLANT*

Alternative methods of recovering gold and silver from cyanide leach solutions include carbon adsorption, zinc dust (Merrill-Crowe) precipitation, ion exchange, direct electrowinning, soluble sulfide precipitation, and aluminum dust precipitation. Most heap leaching operations use carbon or zinc dust methods for treating the leach solutions, although the application of sodium sulfide to precipitate silver from carbon stripping liquors has received favorable attention.

The choice of carbon columns versus zinc dust precipitation should be based upon a capital and operating cost analysis plus consideration of metallurgical efficiency. Probably very large volumes of low grade solutions containing mainly gold may be treated most economically in carbon columns. Rich solutions in comparatively small volume or solutions involving a large daily production of silver should be considered for zinc dust precipitation and compared with the carbon and other systems on a cost/efficiency basis. As to other methods, any applications of direct electrolysis to recovery of gold and silver would be favored by small volumes of rich solution. Ion exchange resins, combining high precious metal loadings and ease of stripping, may be developed. The carbon and zinc dust systems are considered more fully in the following sections.

(a) Carbon System

Experience indicates that the expanded-bed, multiple stage carbon column is an efficient, low cost device for recovering gold and silver from large and small volumes of solutions. The general theory of preparing adsorption isotherms and cascade design has been treated by the U.S. Bureau of Mines [11]. In practice, countercurrent flow in five or more columns is advisable for obtaining high loadings and low barrens. Minus 10-plus 28-mesh carbon (12 x 30 in the carbon industry) provides the best compromise of flow rate and rapid adsorption. Coconut carbon is a good standard; other carbons may be judged on comparative adsorption, durability, and price. The solutions fed do not have to be clarified.

Column design is featured by orifice plate bottoms, bottom-fed upward flow of around 15 gpm/square foot (600 liters/sq. meter) and a working height of 5 feet (1.5 meter) or more above the orifice plate. Carbon when at rest occupies 24" to 30" (0.6

to 0.67 meter) of depth in the column, and weighs about 60 to 75 pounds per square foot (290 to 365 kg/sq. meter). Bed expansion is influenced considerably by solution temperature and resulting viscosity, but fortunately the carbon system is not very sensitive to the amount of expansion. Much helpful material on multiple compartment adsorption was developed by the U.S. Bureau of Mines for ion exchange adsorption of uranium [18].

Individual carbon columns may be superimposed in the form of one multi-stage vertical column. Such a design has considerable advantage if floor space is at a premium. Otherwise an open cascade design of individual columns may be considered. The open design features overflow launders on each tank leading to a feed pipe entering below the orifice plate on the following stage.

A trash screen, 20- to 35-mesh should be placed ahead of the first adsorption column. A "safety" screen with 35-mesh openings should be in circuit after the last column to catch any floating carbon.

Carbon is conveniently moved from stage to stage by hydraulic eductor. Fresh carbon is fed to the last stage, from which barren solution returns to the heap.

(b) Merrill-Crowe Zinc Dust System

Merrill-Crowe plants were once a standard item, but as a package are now available (to my knowledge) from only one U.S. firm (the State of Maine Mining Company, Tombstone, Arizona). They may be custom-designed and manufactured on the basis of a flowsheet comprising precoated clarification filter(s), deaeration unit, zinc dust feeder, precipitate filter, and accessory pumps. The solution fed to a Merrill-Crowe clarifier should not exceed 50 to 100 ppm total solids to avoid undue cleaning requirements. Manufacturer's filter design recommendations should be followed throughout.

## 9. *THE PROCESSING OF RECOVERY-PLANT PRODUCTS*

(a) Loaded Carbon

Several alternatives are available for recovering gold and silver doré from loaded carbon:

(1) Direct sale to a smelter. This has the advantage of requiring no capital investment for carbon treatment and is a viable possibility, particularly for smaller operations. Of course, the carbon itself is sacrificed.

(2) Stripping with 1 percent NaOH, 0.1 to 0.2 percent NaCN at a temperature near boiling (90 °C). Conical bottomed cylindrical tanks are used, and solutions are percolated at the rate of 10 to 12 gpm per short ton of carbon (0.7 to 0.8 liters/sec. per metric ton). Effluent solutions are usually electrowon, and the solutions returned to the stripping cycle. Some 48 to 72 hours should be allowed to strip carbon loaded typically with from 150 to 400 troy oz. (5100 to 13700 ppm) of gold plus silver down to less than 3 ounces (100 ppm).

(3) The addition of alcohol (say 10 percent of ethanol or propanol) to a 1 percent NaOH, 0.1 to 0.2 percent NaCN solution enables stripping to be accomplished in less than 8 hours at temperatures of 60 °C or so and may reduce the need for subsequent carbon regeneration. Seven to 10 bed volumes of strip solution plus wash should suffice. However, alcohol is flammable, explosive, and toxic and the system must be safely designed. At least one experienced company ("NUCON") has a packaged design for handling an alcohol system that recovers most of the alcohol for reuse.

(4) Gold and silver may be stripped from loaded carbon in less than 8 hours with 1 percent NaOH, 0.1 to 0.2 percent NaCN at temperature of 120 to 130 °C under a pressure of 70 psig (5 kg/sq. cm.) using 7 to 10 bed volumes of stripping plus wash solutions.

(5) Either the pressure or the alcohol system produces very rich solutions. They may be electrowon to recover doré. Electrolytic cells feature fiber reinforced plastic construction with stainless steel anodes and cathode frames loaded with mild steel wool.

Voltages are on the order of 2.5 to 3.5 volts per cell and amperages consistent with 30 percent current efficiency for gold, rising to 40 percent or more for high silver solutions.

Alternatively, stripping solutions may be treated with zinc dust or by chemical precipitation to provide a filterable precious metal product.

A scheduled "bleed" of stripping solution, adjusted to each operation to control impurities, is fed very slowly (to avoid inhibiting adsorption) back to the recovery plant adsorption columns.

Smelting of either gold zinc precipitates or steel wool cathodes is readily accomplished in crucible furnaces to

avoid higher dust losses in direct fired units. A No. 125 silicon carbide crucible unit should readily handle 200 ounces of doré per day. The generally larger bulk of silver operations would probably favor consideration of a small reverberatory-type furnace. Fluxes are standard; for example, a borax, niter ($NaNO_3$), silica mixture. Scrubbers on furnace gases are essential. Slag crushing, removal of metallics by screening, and shipment of accumulated slags to a smelter are common steps.

(b) Zinc Precipitates

As outlined under zinc-precipitated values in stripping liquors, zinc precipitates may be smelted in a crucible or small reverberatory unit with standard fluxes. Doré is recovered and a shipping slag produced.

In many cases, refining of doré to pure gold and silver is referred to established refining concerns, and the doré is shipped to such companies.

10. *BARREN SOLUTION STORAGE AND REAGENT FEEDING*

Barren solution effluent from the precious metal recovery plant should be stored in a reservoir of about the same size as the pregnant solution storage. Reagent makeup is conveniently done in the barren storage. Suitable storage for alkali and cyanide, and mixing and feeding facilities, should be provided nearby. For larger properties, if lime is used, a ball-mill slaker is convenient, followed by a closed loop pulsed feeder. Loop feeders are also satisfactory for caustic (if used) and cyanide. Reagent control and feeding is standard practice.

11. *INSTRUMENTATION*

For larger properties an automatic head sample and recorded weight of ore to the pads are highly desirable. If the leached residue is removed from the pads, some form of hand sampling is indicated, because automated sampling would be very costly. Residues left on pads would have to be sampled by boring.

The best and most reliable available solution flow instrumentation should be installed on leaching solutions to and from the recovery plant coupled with automatic or hand samples for grade analysis. Usual pH control is needed whether manual or automatic. Dissolved oxygen meters and/or chemical tests are required at least periodically. Titrations are made for alkali and free cyanide by standard methods.

For gold and silver solution assays, the AA (atomic absorption) method is convenient. Solution samples should be extracted into ketone for reading or else a deuterium arc light source correction should be provided for direct reading.

12. *MAINTENANCE OF CARBON ACTIVITY*

Sooner or later carbon looses activity because of contamination with organic materials (such as oils), calcium carbonate, and other impurities. Provisions should be made to wash stripped carbon with dilute nitric acid followed by several water washes. (If hydrochloric acid were used, it would be corrosive to the kiln in subsequent reactivation). The carbon should then be washed with dilute (1 percent) NaOH, drained damp, and reactivated by heating to 630 °C for ½ hour. A rotary kiln, indirectly fired, is convenient for the purpose. Carbon should be screened to remove fines and presoaked at least 12 hours before reuse.

13. *MATERIALS OF CONSTRUCTION*

- As noted before, asphalt is a good pad material and does not interfere with carbon adsorption. Local clays, if tested, may be suitable for pads. Reinforced plastics are also satisfactory if protected from punctures and tearing. Other materials such as concrete, etc., are possible alternatives.
- Non-galvanized steel, plastic, and asbestos cement may be used for piping.
- Reinforced concrete is satisfactory for pump boxes and rectangular tanks.
- Steel tanks serve for alkaline cyanide.
- Rubber lined, stainless steel, or some plastics could be used for acid washing carbon.
- Brass, bronze, zinc, copper *cannot* be used.
- Stainless steel stripping vessels, piping, and valves are recommendable (316 preferred).
- Fiber-reinforced plastics may be used for electrolytic cells.

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# HEAP LEACHING AND PILOT TESTING OF GOLD AND SILVER ORES

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Heap leaching has found considerable acceptance in the gold and silver business because it is an inexpensive method for extracting precious metals from oxidized ores. Heap leaching in this paper refers to leaching crushed ore that has been placed in unconfined piles on a prepared surface. The ore is crushed so that more rapid and predictable extraction of values can be achieved. Similar leaching of unprepared run-of-mine ore is generally called dump leaching and usually yields a smaller percentage of its values over a longer period of time.

The operator of a heap leach operation needs to be able to recover a predictable amount of gold or silver from the ore. The amount needs to be large enough to make the project profitable and predictable enough so that expense and sales committments can be met. This predictablility requires that the operator make exacting tests on the ore so that the vagaries of treating the ore by heap leaching are understood and overcome.

Heap leach testing generally begins with column tests in the laboratory. Sophisticated engineers may choose to model the column results on a computer and predict results. However, most managements want at least one final large scale confirming test which requires that a pilot heap be built in the field.

Obtaining reliable recovery data from a heap leach test program is subject to problems not usually encountered in agitation testing. The primary problems are: 1) a lack of permeability and porosity caused either by an excessive amount of fines or by improper heap building techniques which cause segregation of particle sizes or compaction, and 2), the difficulty of scaling up heap leach data to simulate commercial practice. Much testing is done on a relatively small scale because of the cost and difficulty of obtaining the large amounts of ore for proper testing and this leads to sampling errors related to the relatively large ore particle sizes.

## HEAP BUILDING TECHNIQUES

Roman[1] has commented about the variations in permeability and porosity that can exist within heaps depending upon the way they are constructed. One cause is the segregation of coarse and fine ore within the heap, and another is the pulverizing and compacting of ore by trucks and other equipment on top of the heap. There are many unfortunate stories about heaps which were not permeable. Kappes[2] made a quick survey of 22 gold and silver heap leach operations and found that eleven were failures. For example, the operator applies leach solution to the top of the heap expecting it to percolate through the ore, but it runs off the top of the heap and erodes the side instead; or it percolates part way into the heap and then comes flowing out the side of the heap at some unexpected location; or the leach solution flows down through zones of very coarse ore and does not leach the entire heap. More likely you've heard the less exciting but equally frustrating stories about operators who simply got very low recovery or took years to achieve the expected recovery.

I will now describe how three pilot heaps were built and explain a heap building technique that appears to minimize the permeability, segregation, and porosity problems.

The first heap was built at the Candelaria project in Nevada which is operated by Occidental Minerals Corporation and it contained 11,000 tons of silver ore. The next two were built at the Alligator Ridge gold project in Nevada (50/50 joint venture of Amselco Minerals, Inc. and Occidental Minerals Corporation, Amselco operator), one of 5,000 ton size and the other of 2,000 ton size.

At Candelaria the ore sample was taken from several sites and was as representative a sample as could be obtained near the surface. A portable crushing plant crushed the ore to about −2″. A leach pad was prepared and covered with a synthetic sheet. However, the heap was built improperly. The crushed ore was very dry and it was conveyed by belt to the pad where it cascaded into piles as shown in Figure 1. It segregated severely with the coarse ore going to the outside of the piles and the very fine material remaining as an inner core. Multiple piles were made, each about 20 feet high, and then the tops of the piles were bladed off with a dozer to make a flat heap about 14 feet high. As a result of the severe segregation the leach solution percolated through the coarse ore zones at a greater flow rate than it did through the fine material. This unequal distribution of solution flow through the heap caused slow recovery of silver at Candelaria. At some mines the unequal flow rate has caused the migration and redeposition of fine material which ultimately gathers at some point within the heap, as shown in Figure 2, causing a horizontal dam. Ore below this impermeable dam is not going to be leached and recovery from ore below the dam may be nil. Also, solutions reaching this impermeable dam may flow horizontally and exit through the side of the heap, sometimes with disastrous erosion of the heap. At Candelaria we did not have such problems, maybe because the ore contains only 2.5% −100 mesh; in fact, all of the ore was wetted. But it took 9 months of actual leach time under spray to achieve a recovery that normally was obtained in 2 months in much smaller tests.

Another improper heap building technique is to build heaps in layers, as shown in Figure 3, starting with a layer near the base of the heap and building upward to the final heap height. This approach, which would probably use trucks or graders or dozers to place the ore on the heap, pulverizes and compacts each layer causing a reduction of porosity

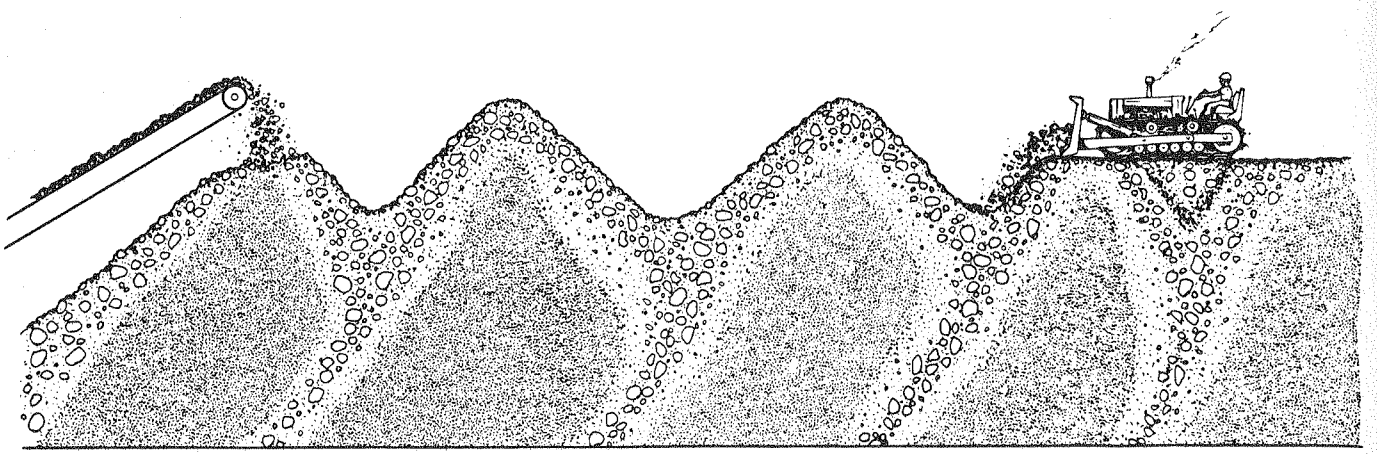

FIGURE 1. Improper heap building.

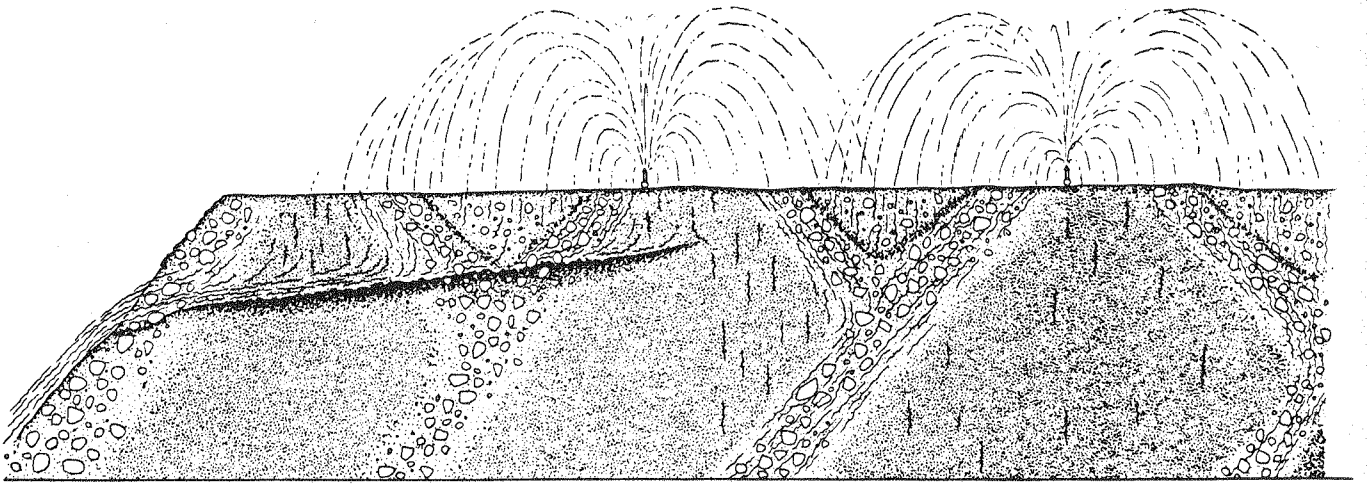

FIGURE 2. Impermeable zone in heap.

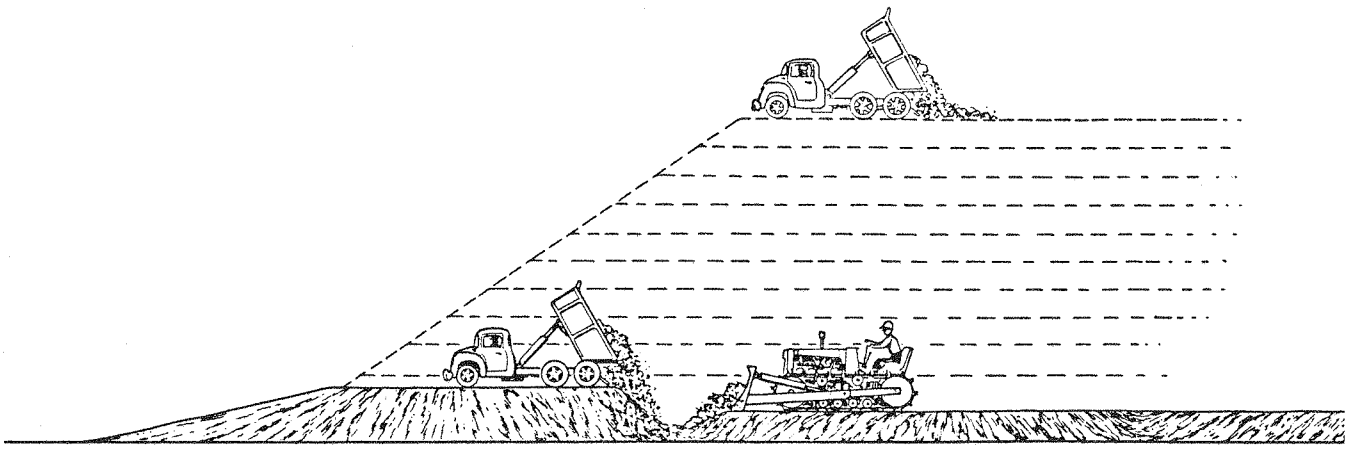

FIGURE 3. Improper heap building.

and permeability. These problems can only partially be overcome by ripping each layer with a dozer.

There is a better way to build heaps as shown in Figures 4 and 5. This technique was used at Alligator Ridge where two test heaps were built. It involves building a ramp of waste material to the full height of the heap, say 20 feet. Trucks with prepared ore will then dump their load at the top of the ramp. As heap building continues, however, the trucks will be restricted to a relatively narrow roadway across the middle of the heap because their ground presssure will be about 85 psi (loaded 50 ton Caterpillar 773). Material that they dump from the roadway will then be pushed to the sides of the heap with a dozer. The dozer's track will not compact the ore nearly as much as trucks do because its ground pressure will be 11 psi (Caterpillar D9G). Compare this to a 175 pound person who has a ground pressure as high as 15 psi when walking. The roadway is constructed two to three feet higher than the rest of the heap surface. In other words, if the heap is to be 20 feet high, the roadway would be 23 feet high. After the entire heap has been constructed, the roadway is cut down to the finished heap height and ripped. The resulting surface of the heap will have been subject to very little compaction.

It is well known that segregation occurs if a mixture of fine and coarse dry ore is allowed to cascade down the sides of a stockpile or heap. This more prevalent with coarse ore, say −4″, than fine ore, say −1″. The troublesome portion of the ore is the −100 mesh fines which cause permeability and porosity problems and not the +20 mesh ore which has a high permeability regardless of segregation. Therefore the answer to permeability problems in heap leaching is to distribute the very fine ore uniformly through the heap.

Another advantage of the heap building technique shown in Figures 4 and 5 is that the heap has been built in multiple layers at the angle of repose of the heap. It may seem that allowing the ore to cascade down the sides of the heap will lead to segregation of fines and coarse ore but this is where the second of the two techniques for eliminating permeability problems comes into play. We call it agglomeration.

Agglomeration involves pretreating the ore by wetting it to cause the attachment of fine materials on the coarse particles, Figure 6. Although we call it agglomeration, it does not result in pellets and the agglomerates are no larger than the ore particles. The bond between coarse and fine particles is not a permanent one, but it does last long enough to allow the ore to be placed on the heap and it does create a uniform distribution of fines throughout the heap. Even when the ore dries the fines continue to stick to the coarse material with sufficient tenacity to allow uniform solution flow through the heap.

The means of achieving agglomeration is simple. We adapted the technique used at the Weed Heights operation of Anaconda near Yerington, NV.[4,5] They sprayed water on the −3/8″ undersized copper ore from each of six screens all of which discharged onto a common belt conveyor. This gave the effect of a layer of ore, a layer of water, a layer of ore, and so on. It was then mixed through four conveyor transfer points which had heavy metal bars suspended in the falling ore stream. Any number of mechanical devices may be satisfactory for accomplishing the mixing. At Candelaria we will be using the multiple conveyor method with 3-inch mixing rods suspended in the transfer chutes, Figure 7, and a rotating drum will be used at Alligator Ridge, Figure 8.

It isn't absolutely necessary to have an agglomeration step in the crushing circuit. If the ore can be uniformly wetted prior to placing it on the heap, the cascading and rolling motion of coarse wet particles will bring them into contact with fines thus causing agglomeration and distribution of fines throughout the heap.

Of the three pilot heaps described only the two at Alligator Ridge were agglomerated. The Candelaria pilot heap was not agglomerated and we subse-

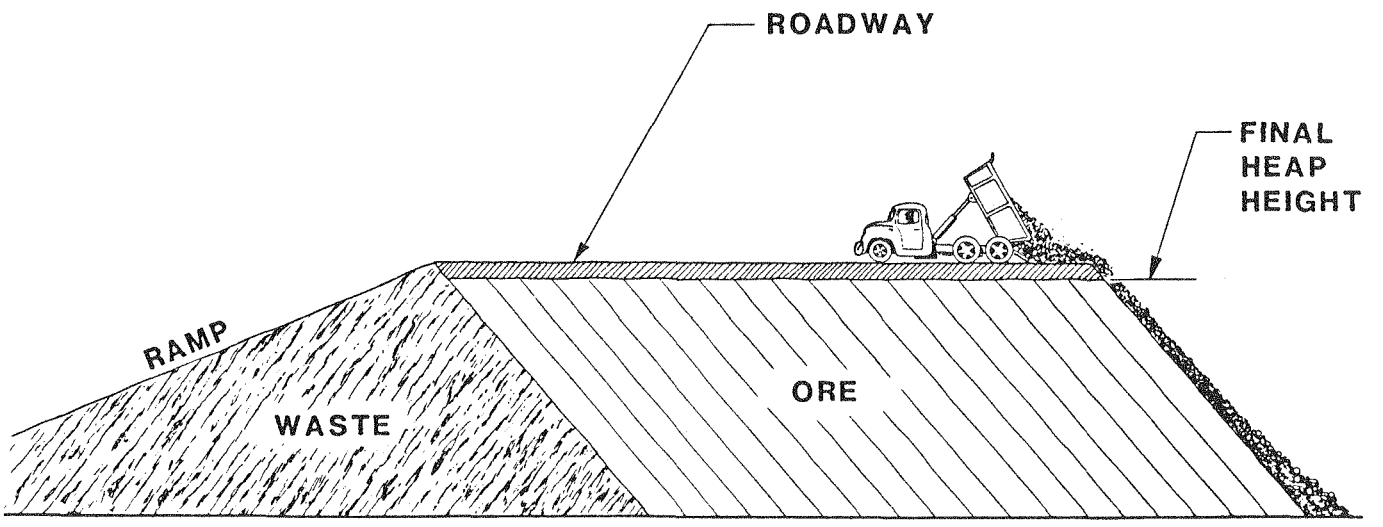


**FIGURE 4. Proper heap building.**

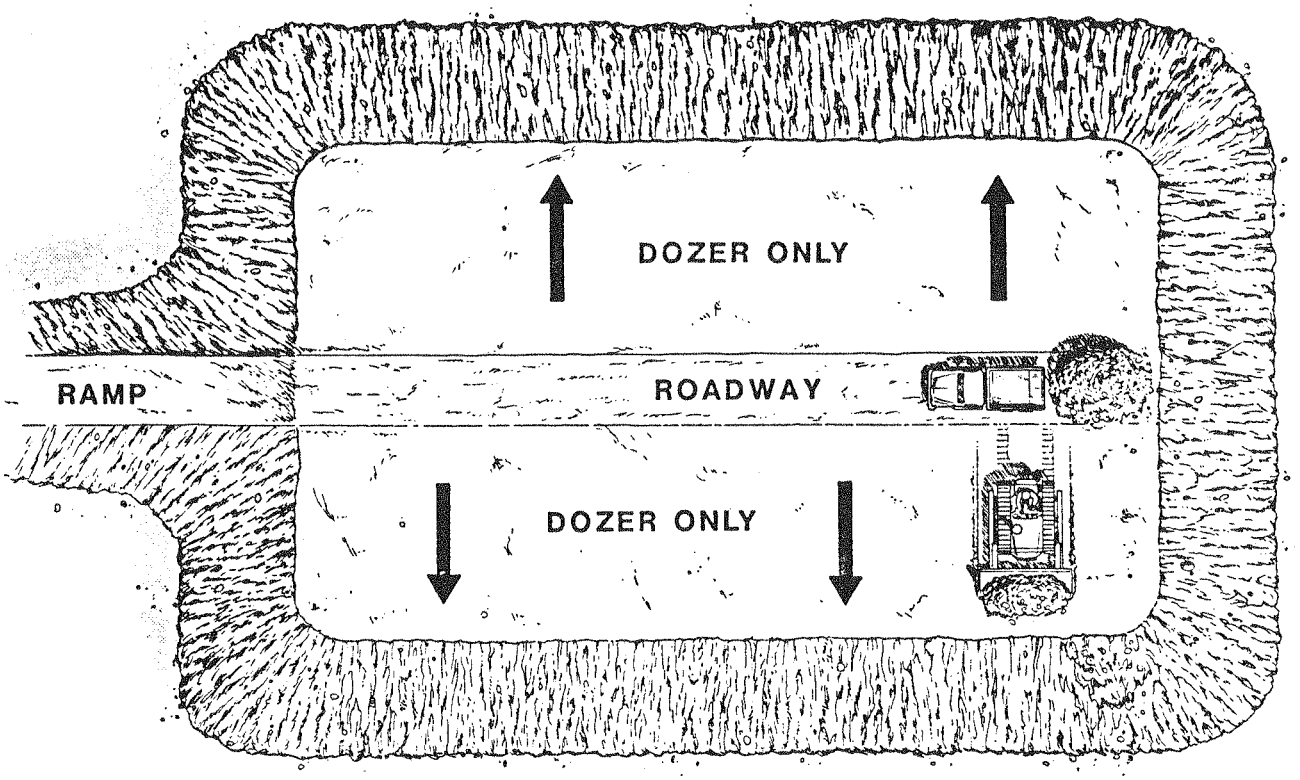


FIGURE 5. Proper heap building.

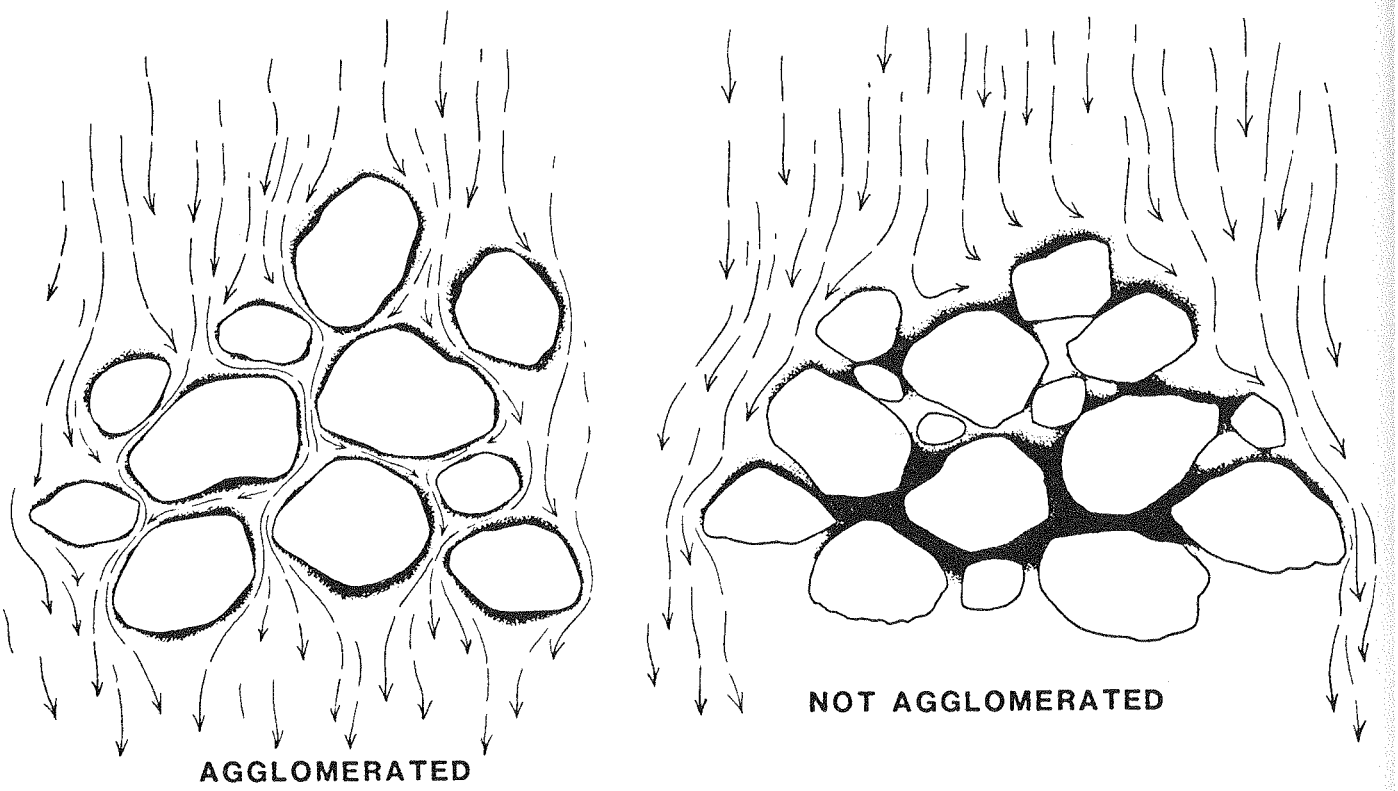


FIGURE 6. Comparison of agglomerated vs. not agglomerated.

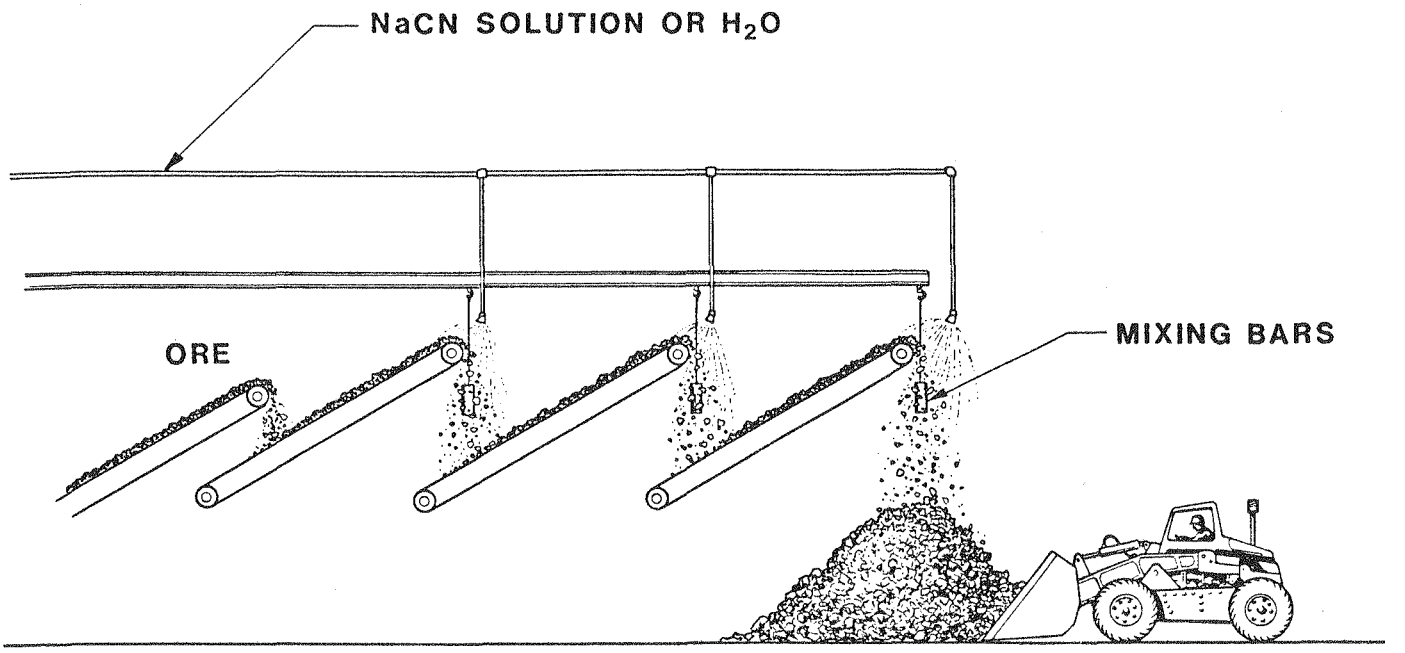


FIGURE 7. Ore agglomeration.

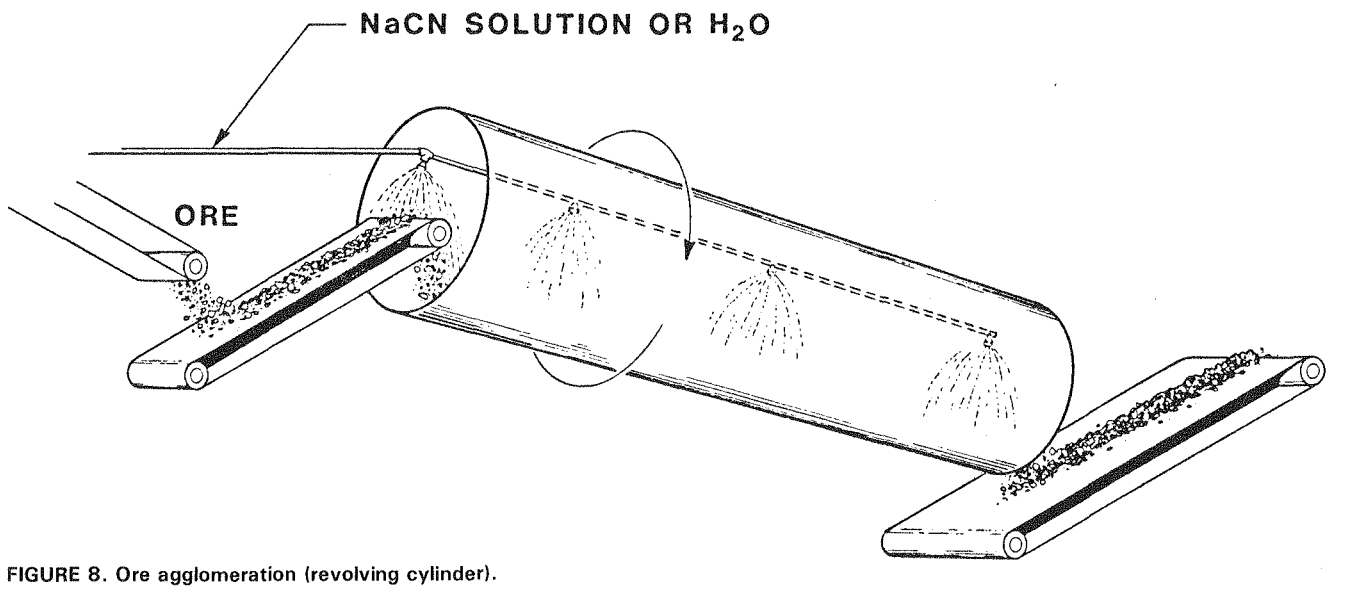


FIGURE 8. Ore agglomeration (revolving cylinder).

quently wished we had. The first heap at Alligator Ridge was 5,000 tons in size and was agglomerated with water alone. The crushing plant was a portable, compact unit with closed circuit crushing and it afforded no easy way of wetting the ore without causing the buildup of mud on return idlers or the derailment of the rotoveyor or mud accumulations on the crushers. It was not practical to add water at multiple points and have a well mixed agglomerated ore discharged from the final product conveyor. We compromised and installed a water spray bar at the discharge end of the crusher's final product conveyor and as the material fell onto a stockpile it was wetted and the coarse particles rolled down the slope of the stockpile gathering fine material. It was then picked up by a front end loader while still wet and transported to the heap which was built at its finished height as previously described. The ore was gently placed at the edge of the heap and shoved over the edge so it would cascade down the side. The cascading motion, both at the stockpile and on the heap, caused adequate agglomeration of the fines and the coarse particles. Visual examination and spot checks of the heaps indicated that a very uniform distribution of fines had been achieved throughout the entire heap. The amount of water added normally results in about a 5 to 12% moisture content depending upon the amount of fines present. The amount of water should not be so great as to make the agglomerates glisten.

The second heap at Alligator Ridge used an agglomeration technique recently developed by the

USBM[6] in which the ore is mixed with a binder and wetted with a strong cyanide solution. This can be achieved by the same methods shown in Figure 7 and 8. The amount of cement depends upon the amount of fines. Ten pounds of #2 Portland cement per ton of ore was used at Alligator Ridge and it was probably excessive; 5 pounds may have been enough. Up to 20 lbs/T has been required on ores containing as much as 50% −325 mesh material. A cyanide solution was used instead of water to get a head start on leaching—similar to the acid bake concepts—and the amount of NaCN added was equal to the expected cyanide consumption. The ore was agglomerated in concrete mixing trucks, discharged onto the ground and then picked up with a front end loader and placed on the heap. The bond between fines and coarse particles was stronger than with water alone, but this didn't yield any noticeable advantages. For example, after both heap leach tests were completed at Alligator Ridge we examined cross sections of the heap and found that the fines did not migrate very far, probably no more than three or four inches, and they migrated rather uniformly. As a result there was no massive accumulation of fines at any one point. The permeability of both heaps was excellent, and we witnessed flow rates as high as 30 gpm/ft$^2$. Compare this to the more typical flow on the heap of 0.003 to 0.005 gpm/ft$^2$.

I don't mean to imply that agglomeration with cement is not useful. It will find greatest application on ores that have a much higher percentage of −100 mesh fines than either Alligator Ridge or Candelaria. Column tests on an ore containing 50% −325 mesh have been agglomerated with cement and have had excellent permeability, porosity, and percolation rate. Unagglomerated column tests on the same ore were an absolute failure. Cement agglomeration may also create a heap that is more competent over a longer period of time, i.e. greater load bearing strength, so that good percolation rates will continue even with the addition of multiple ore lifts.

In the ideal heap each particle should be completely contacted by leach solution. Roman[1] has stated that this is definitely not the case in zones of high bulk density. Increasing the bulk density will give more particle to particle contact and individual particles will be blinded by each other. The blind areas have the effect of making the particles behave like larger particles with longer diffusion paths. As a result the leach time is increased and the potential recovery may be decreased. The agglomeration procedure has the effect of reducing the bulk density of the heap significantly particularly on ores with a large amount of −100 mesh fines. Neither Candelaria or Alligator Ridge ore contained excessive fines, 2.5% −100 mesh at Candelaria and 6% −200 mesh at Alligator Ridge, but even so the bulk density of Candelaria ore was reduced from 99 lb/ft$^3$ dry to 93 lb/ft$^3$ agglomerated and the Alligator Ridge figures were 90 lbs/ft$^3$ dry and 85 lbs/ft$^3$ agglomerated.

Both Alligator Ridge heaps gave gold extractions nearly identical to that obtained in laboratory column test. The pretreatment of the second heap with cyanide did give a much more rapid extraction of gold in the first few weeks, but it did not give a higher final extraction than the water agglomerated heap. In both cases, the extraction of gold exceeded our expectations within two months.

We feel that the Alligator Ridge experience is strong evidence that the permeability and porosity of heaps can be significantly improved by agglomeration of the ore and by careful heap building techniques. These two techniques minimize the risks of heap leach testing and production. They give greater certainty that predicted recoveries will be achieved. They give a heap with more uniform permeability and porosity because the fines are uniformly distributed through the heap. And because the ore has more porosity and less bulk density, it is possible to achieve a more rapid extraction of values, thus reducing working capital and precious metals inventory in the heap. They also allow the production rate of gold and silver to be achieved with fewer heaps under leach which reduces labor, pumping cost and maintenance.

An interesting concept that may someday be applied to heap leaching is the idea of countercurrent leaching of heaps. The fact that a heap can be leached more rapidly when it is built properly and has adequate porosity and permeability may make this possible. Such countercurrent operation of heaps would yield higher grade solutions to the precipitation circuit, higher Au/Ag loading on carbon, and could reduce the volume of solution being processed, i.e. reduce the size of the plant.

## QUANTITY OF SAMPLE NEEDED

The second significant problem with heap leach testing is the large magnitude of the undertaking, both physically and in dollars. The reason for the large undertaking is due to the large particle sizes that are used in heap leaching. Typically the ore in a heap is −1" to −3" compared to −28 mesh to −100 mesh in agitation leaching. Using the sample size guidelines in Taggart[3] for a spotty silver ore indicates that 1,000 pounds of −2" ore are needed to be representative but only 0.1 pounds of −65 mesh ore is needed—a 10,000:1 ratio and a significant difference in the magnitude and cost of testing.

Most heap leach test programs start with small scale column or barrel tests to get a "feel" for the main operating conditions and the variability of the ore types that will be encountered in the deposit. The more sophisticated laboratories may also study the kinetics of leaching individual particles and then make a computer model of the heap leach process. Roman[1] discusses computer modeling of the heap leaching operation and he clearly points out that such simulations are very difficult and take a long time if all known variables of heap leaching are taken

into account. As a practical matter, no computer program accounts for all variables and simplifying assumptions are always made. This makes the results somewhat suspect.

Our experience at Candelaria with ore that is spotty and erratic in silver content has ranged from 100 pound tests to an 11,000 ton heap. Initially we conducted a large number of barrel tests which were run in 55-gallon drums with about 100 to 200 pounds of ore. The individual results of these test were very erratic, but enough of them were run to give an approximation of the silver recovery that could be achieved.

Larger scale barrel test were performed next with 500 pounds of ore in each test. These tests were closely controlled and gave indications of the best cyanide concentration, the best pH, the best solution application rate, and the best ore size. Hindsight now shows that the results of these tests were very close to the results obtained later on the pilot heap. But at the time we weren't overly confident with the data for three reasons, 1) assays varied widely because of the spotty silver distribution in the ore, 2) the ore depth in the barrels was very shallow compared to production heaps, and 3) the amount of ore in each test did not meet the minimum sample requirements suggested in Taggart[3].

The next step was to build the 11,000 ton heap. We were fortunate to have large amounts of ore exposed at the surface and tonnage amounts of ore could be obtained for pilot testing. It was decided that evaluating a pilot heap would be preferable to computer simulation. A recovery circuit using zinc precipitation was constructed at Candelaria and preg and barren solution ponds were installed. Building the heap has been already described. The capital and operating costs for this pilot heap have totaled nearly $300,000 since September 1978.

The heap has been operated for a total of 9 months during summer and winter season. It has given fairly reliable information about the recovery of silver but not about the rate of silver recovery. Nor did it allow us the opportunity to evaluate variations of process conditions. We had to rely on the data from the 500 pound barrel tests for the leaching conditions and we weren't confident enough of that data.

What we desired was a test apparatus that was larger than the barrels but not as large and cumbersome as a pilot heap.

We learned that Anaconda had conducted vat leach simulation test in 5' diameter x 20' high tanks. They had found that the results achieved in the 5' diameter tanks were identical to the results obtained in the commercial vats. We installed two 5' diameter by 14' high columns at the field pilot plant. Each column holds about 10 tons of ore. We have since conducted many tests in these columns and have also come to the conclusion that these large column tests are good predictors of heap leach operations.

The next time we conduct a heap leach test program it will start with tests in 8"-12" diameter by 6'-20' high clear plastic columns. Both the basic process variables and the different types of ore in the mine will be tested and evaluated at this stage. Confirmation tests will be performed in the 5' diameter columns and the variations noted in the small columns will be correlated with the large scale results. If management requires further evidence, a 2500-5000 ton heap would be constructed and leached in the field. The initial column tests would require between 1.25 and 10 tons of sample based on a minimum of 12 tests. The 5' diameter columns would require 20 to 100 tons of sample and the pilot heap would require 5,000 to 10,000 tons.

## CONCLUSION

A better heap building technique and agglomeration of the ore have been demonstrated in field tests to give more uniformly permeable heaps that yield greater leaching efficiency and faster recovery of gold and silver. The agglomeration and heap building techniques cost no more than the heap building methods conventionally used.

It is difficult to compare the pilot plant performance at Alligator Ridge where the heap building and agglomeration techniques were practiced with Candelaria where they were not. The ores are completely different. Alligator Ridge is primarily an oxidized disseminated gold deposit whereas Candelaria is a "refractory" silver deposit. Perhaps the most significant comparison is that the Alligator Ridge pilot plant very closely confirmed the recovery and leach rate that laboratory column tests had predicted. The Candelaria pilot plant, on the other hand, predicted the recovery but not the leach rate; it took nine months to do what the laboratory tests did in two months. We are convinced that the application of proper heap building and agglomeration techniques at Candelaria would have enhanced the rate of silver extraction substantially. The large five foot diameter column tests have confirmed it.

Heap leach testing requires considerable amounts of ore—anywhere from 1.5 to 10,000 tons depending upon the amount of assurance that management wants.

# LEACHING PRACTICES AT SMOKY VALLEY MINE

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## LEACHING PRACTICES AT SMOKY VALLEY MINE

The Smoky Valley Mine is jointly owned by Louisiana Land and Exploration Company, Felmont Oil Corporation and Case Pomeroy Co. Copper Range Company, a subsidiary of L. L. & E. is the operator of the mine.

The mine is located about 60 miles due north of Tonopah, Nevada on Highway 376 (formerly 8A) in the Big Smoky Valley on the western flank of the Toquima Mountain Range in the shadow of Mount Jefferson.

The mine is at an elevation of about 6100 feet and is situated on the southwestern flank of Round Mountain, the hill for which the town of Round Mountain is named.

The ore is a rhyolitic ignimbrite with coarse gold occurring along fractures and with fine microscopic gold disseminated in the rock mass. Minor silver is associated with gold.

Annual mining is planned at 5.8 million tons per year with 2.2 million tons as ore (7,000 tons per day).

## MATERIAL HANDLING

After drilling and blasting the ore is loaded on fifty (50) ton trucks with rubber tired loaders or an electric power shovel and transported to an open circuit crushing system where it is reduced in size to a nominal—½ inch product. The crushed material is belt conveyed to a storage pile and is then withdrawn as required and conveyed to a truck loading bin. The crushed ore is transported to the leaching pad area by the 50 ton trucks and stacked to a height of 13 feet using loaders.

The material after leaching, washing and draining, is excavated from the leach pads with a loader and transported by trucks to an adjacent dump area.

## LEACHING PAD CONSTRUCTION

The leaching pad is constructed of Hydraulic asphalt concrete over a carefully compacted base. The build-up of the pad includes a two inch layer of asphalt which is covered with an asphalt rubber membrane and two additional 2½ inch layers of asphalt concrete. This provides an impermeable barrier to the downward percolating solutions.

The leach pad is 2525 feet long and 282 feet wide. The width includes a common drainage ditch which collects all effluent solutions and conducts them to a pregnant solution pump sump.

## APPLICATION OF LEACHING SOLUTION TO HEAPS

Current practice for application of leach solution includes use of polyethylene piping for conveying barren solution from the plant effluent sump to the leach pads and the use of removable PVC piping for the spray distribution systems on the heap surface.

The barren solution is applied to the heaps by spraying—using "Bagdad Wigglers" (12" lengths of ¼" surgical rubber tubing). The "Wigglers" are arranged in a 35 ft. x 35 ft. square pattern on the surface of the heaps with output of each spray at 5 gpm. The current application rate is .004 gpm/ft$^2$. We are planning to increase the application rate to .005 gpm/ft$^2$ to improve coverage. We are now distributing 2200 gpm of solution on the heaps but will increase this shortly to 3000 gpm.

## HEAP LEACHING METALLURGY

Metallurgical treatment consists of application of an aqueous solution of sodium cyanide and sodium hydroxide to the ore heaps to dissolve the gold and silver values.

Leaching solution concentrations to the heap are maintained nominally at 0.05% cyanide and 0.03% NaOH and solution pH is maintained at 11.0.

The leaching cycle is nominally 48 days distributed as follows:

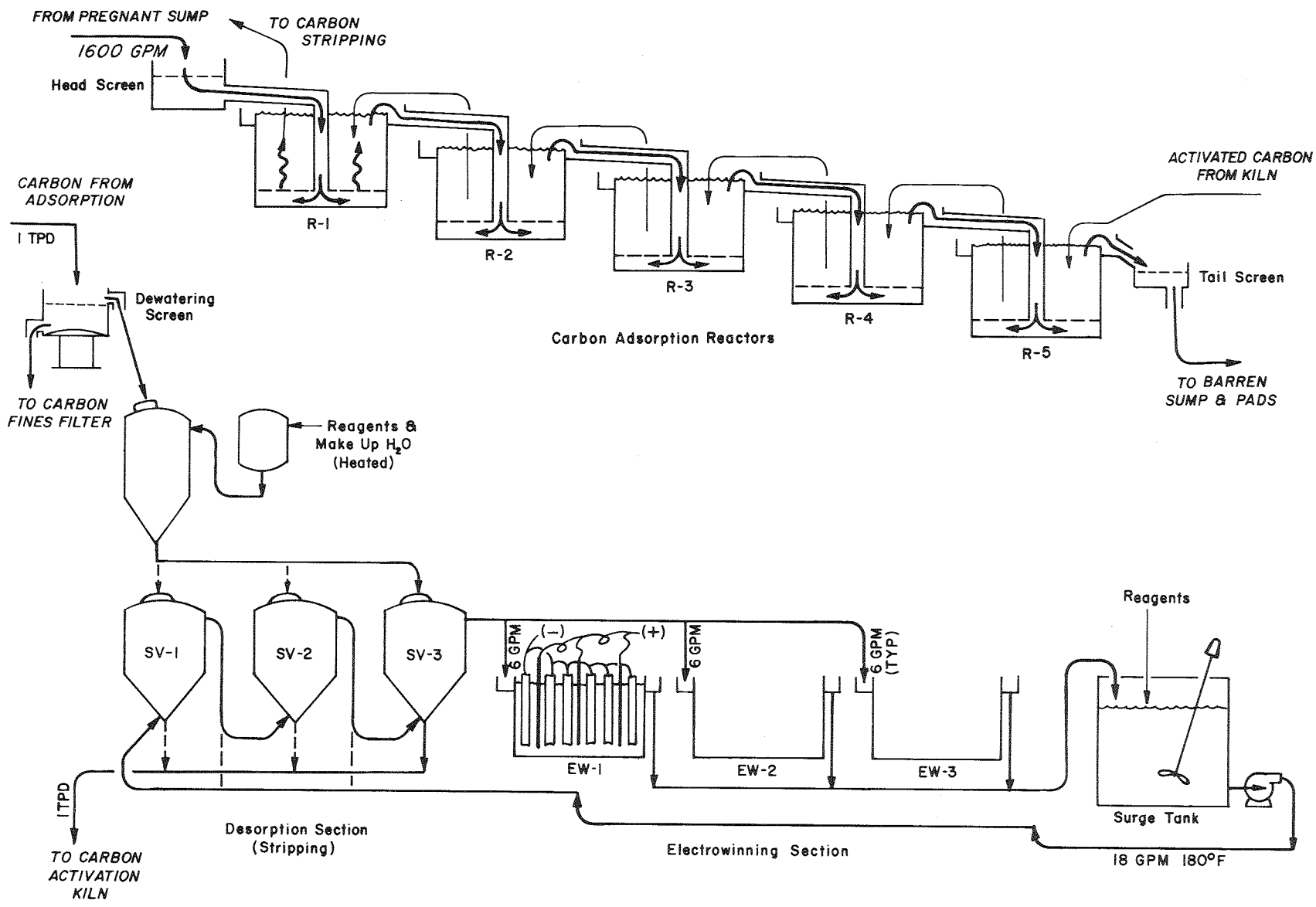
| | |
|---|---|
| LEACHING TIME | 40 days |
| WASHING TIME | 2 days |
| DRAINING TIME | 2 days |
| LOADING, UNLOADING and OPEN | 4 days |

The leaching solutions are in closed circuit and the only make-up water is for evaporation and moisture in leached tails. The usual source of make-up in the system is the wash water which essentially balances with the losses for evaporation and tailings moisture. All solution from the leach pads is collected in a common ditch and flows by gravity to a pregnant solution sump.

The composition of pregnant solution is commonly from .020 to .030 troy oz. Au/Ton of solution and from .4 to .6 lbs. cyanide/ton of solution at a pH of 9.5 to 10.0.

## PROCESS PLANT

Recovery of gold and silver from solution is by activated carbon adsorption, desorption and then electrowinning and fire refining. The final product is Doré metal typically containing 65% gold 34% silver, and minor amounts of copper, nickel and iron.




FLOW SCHEMATIC - PROCESS PLANT

SMOKEY VALLEY MINE

## ADSORPTION SECTION

Pregnant solution at the rate of about 1600 gpm is pumped from the pregnant solution sump to an elevated plant inlet screening station. The solution then flows by gravity through a series of five 12 ft. diameter x 8 ft. deep open top adsorption reactor vessels which contain activated carbon. Flow through each tank is upward through the expanded carbon bed. Each reactor contains four tons of 12 x 30 mesh coconut carbon. Carbon transfers are made counter to the solution flow in one ton batches each day into the reactor system, between reactors and from the reactor system.

The carbon transfers are accomplished by eduction using barren solution as the transfer medium. Carbon is loaded during the five stages of adsorption to approximately 200 troy oz. Au/Ton of carbon.

## DESORPTION (STRIPPING) SECTION

One ton of carbon is withdrawn from the first reactor and passed daily to the desorption or stripping section which consists of a conditioning tank and three desorption tanks. The carbon is placed in the conditioning tank and raised to about 180 °F in the stripping solution and held for one day. It is then transferred to one of the three desorption vessels and leached for three days with sodium cyanide and sodium hydroxide solutions at a temperature of 180 °F. The stripping solution at the rate of about 18 gpm is passed through the three vessels in series fashion—into the oldest and out of the newest—and the stripping effluent, containing about 2 oz. Au/Ton of solution, is fed continuously to the electrowinning section.

## ELECTROWINNING

The electrowinning section consists of three 2′ x 3′ x 4′ cells with parallel solution flows of 6 gpm. Each cell has three 2′ x 3′ stainless steel anodes and six steel wool cathodes. The cells operate at 3 volts and 140 amps or about 4 amperes per square foot. Usual pulling schedule for each cell is as follows: The two cathodes closest to input side are pulled each week, the remaining four are shifted toward the input side of the cell and the new steel wool cathodes are then put into the spaces left at the output side of the cell. The doré precipitates with residual iron from the steel wool cathodes are removed and dried to prepare for fire refining.

## REFINING

The dried precipitate from the electrowinning cells is weighed and nitre, borax and silica reagents are added in the proper proportions for the refining furnace charge. The furnace is then charged and a temperature of 2100 °F is obtained and held for 1.5 hrs. Each charge produces 200 to 400 oz. doré metal buttons. The buttons are then accumulated and melted in the casting furnace and doré bars of 1000 to 1150 oz. are cast.

Slag from button production is pulverized and then screened to remove the coarse trapped doré. The screen undersize is tabled to remove fine trapped doré. These two products are returned to subsequent melts.

## LEACH SOLUTION HEATING

Temperatures during the winter in the Big Smoky Valley commonly fall well below freezing causing shutdown of the leaching heaps for the most of December, January and February each year. In order to mitigate losses due to freeze ups, submerged combustion burners capable of providing 25 million BTU's per hour to the leaching solution have been installed and are used as necessary to allow operation of the leaching system when temperatures are as low as 15 degrees F. Even in the worst cases where shut-down cannot be prevented, resumption of production is much more rapid.

## CARBON TREATMENT

We have experienced a considerable build-up of calcium carbonate on the carbon. The effect of the build up caused a decrease in adsorption efficiency resulting in high plant tails. We are now washing the carbon with nitric acid and are operating at former levels of recovery. Tails from the plant are at .002 troy oz. per ton of solution. The technique employed is to place one ton of stripped carbon, after kiln reactivation, in a fibreglass vessel and acid solution (10%) $HNO_3$) is circulated through the carbon until a constant pH of 6 is achieved. Carbon is then washed and made basic with sodium hydroxide and reinserted into the adsorption circuit. Since we reactivate one ton of carbon per day, we wash and hold the carbon for the 24 hour period.

# PERCOLATION LEACHING OF CLAYEY GOLD-SILVER ORES

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## ABSTRACT

Heap leach cyanidation has opened marginal gold-silver reserves to economical exploitation. This processing technique has successfully produced gold dore on a commercial scale during the past decade from numerous low-grade, gold-silver deposits and mine wastes in the western United States. However, this cyanidation technique has some definite limitations. Clayey ores or ores that required fine crushing for exposure of the gold-silver mineralization to the cyanide leach solution are not amenable to heap leaching because of the lack of permeability and porosity within the prepared heaps. Recently, the U.S. Bureau of Mines developed an agglomeration technique which markedly improves the percolation flow of the cyanide leach solution through clayey ores. The technique consists of mixing the crushed ore with controlled amounts of binders such as lime and/or portland cement and water, agglomerating the resultant admixture, and then curing the agglomerated feed at ambient conditions prior to heap leaching. The use of a cyanide solution instead of water during the agglomeration step gives a head start in leaching, a more rapid gold-silver extraction, and with some ores reduces the cyanide consumption markedly.

This paper will review some of the more significant developments in minimizing permeability, segregation, and porosity problems by the application of an agglomeration pretreatment of gold-silver ores prior to heap building.

## INTRODUCTION

Heap leaching may be defined as percolation leaching of piles of ore or mine waste that have been stacked or piled on specicially prepared water tight drainage pads for pregnant liquor collection. This processing concept dates back to about 1752 when Spanish miners percolated acid solutions through large heaps of oxide copper ore on the banks of the Rio Tinto River. Since then, this process has been used extensively throughout the world to leach oxide copper values from mine wastes resulting from open pit mining of porphyry copper deposits.

Application of heap leaching for cyanidation of gold and silver ores is a comparatively recent development suggested by the U.S. Bureau of Mines metallurgists in 1967 as a low capital cost alternative to conventional agitation leaching of finely ground ore for recovery of gold and silver from sedimentary deposits that contain micron-size particles of gold in a porous host rock. The Carlin Gold Mining Co. started heap leaching mine cutoff material crushed to ¾ inch in 1971. Cortez Gold Mines installed the first large scale operation, heap leaching two million tons of submarginal ore at Cortez. The success of this operation has led to at least four significant new multi-million ton heap leach cyanidation operations in Nevada. One characteristic of the heap leach cyanidation is the wide range in plant size from one or two man operations treating 10 tons per day to plants milling 8,000 tons of ore per day.

Experience gained by the industry in conducting these heap leach operations has shown the practice to be limited to ores that contain relatively small amounts of clay or fine material, and ores from which the gold and silver values could be leached without fine crushing. There are a number of pitfalls in attempting to heap leach an ore—perhaps one of the more important of these is in obtaining uniform flow of leach solution through the heap to obtain efficient and satisfactory recovery of the gold and silver values. In this presentation I would like to discuss some of the problems in heap leaching clayey ores and the development of methods to overcome these problems.

The presence of clays or excessive ore fines in the heap hampers the percolation flow of the leach solutions, causes channeling and produces dormant or unleached areas within the heap. These effects have resulted in unreasonably long leaching periods and poor gold-silver extractions. In extreme cases the clays and/or fines have completely sealed the ore heap causing the leach solution to run off the top or out of the side of the heap, rather than percolating downward through the ore. These problems can be compounded through improper heap building techniques which results in natural sorting of coarse and fine particles during stacking. The stacking of ore in a series of cones on a leach pad results in concentration of the larger rock fragments on the lower slopes and base of the pile, as depicted in figure 1. This situation persists even after the tops of the cones have been leveled off for installation of the sprinkling system. Obviously, the leach solution will follow the path of least resistance, down through the channel of coarse rock and bypass or barely wetting areas that contain significant concentration of clays or fines. This is an effect observed in the dissection of large field test heaps in which the leach solution had actually percolated downward for some distance and then traveled laterally through the heap and emerged on the side of the heap.

Thus, research to improve the percolation flow in heap leach cyanidation practice has taken two directions. In one, in which I was involved, the U.S. Bureau of Mines investigated techniques for agglomerating the ore prior to stacking as a means of eliminating the permeability and porosity problems caused by clays and segregation of ore fines. The other direction of research, being conducted by Occidental Minerals Corp., involves the mechanical

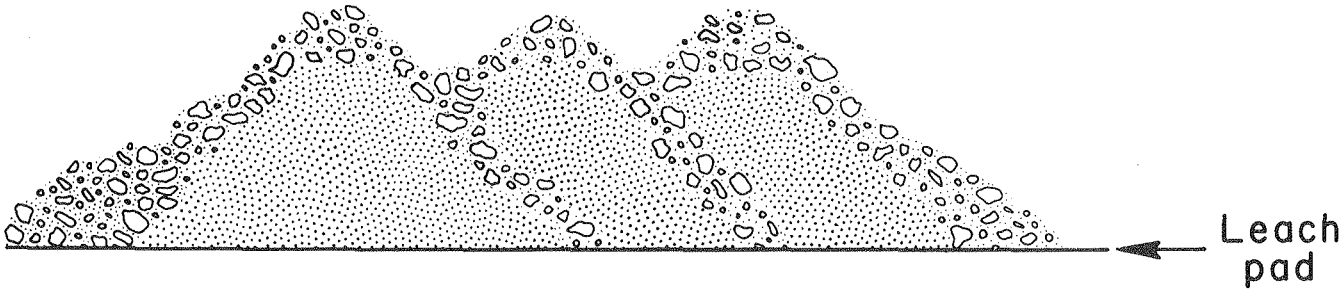


**FIGURE 1. Segregation of coarse and fine fractions resulting from improper heap building.**

construction of ore heaps with minimal compaction of the heap surface.

## EXPERIMENTAL PROCEDURE AND RESULTS

The U.S. Bureau of Mines is investigating the use of readily available binders to achieve the attachment of clays and ore fines to the coarser particles present in the feed, in such a way that the agglomerates could withstand leaching, without migration of the fines, and still remain porous and permeable. The initial study was conducted on a silver mill tailing containing 30 weight-percent minus 100-mesh fines. The binder selected for this investigation was lime because it has been used in agglomerating other materials and it is also used for protective alkalinity in conventional cyanide heap leaching. It was thought that lime might react with ore fines and partially set up with moist curing much the same as concrete does. In effect then, about the same quantity of lime would be applied to the ore as would be ordinarily used for protective alkalinity during cyanidation, but modifying the method of applying the lime so that the ore mixture is workable in a balling apparatus to produce porous agglomerates with sufficient strength and durability to enhance the percolation flow of leach solutions.

The leaching experiments were carried out on 50-pound charges of feed material. A schematic of the laboratory leach apparatus is shown in figure 2. The leach columns were 5 feet high and had an inside diameter of 5.5 inches. The agglomerated feed was placed on top of a porous support to give a bed height of about 4 feet. A three-way plastic discharge valve was employed to facilitate measurement of the flow rate, and to enable taking solution samples for assay. The pregnant effluent was pumped through three 1.5-inch-diameter columns in series, each of which contained 30 grams of 6 by 16 mesh coconut shell activated carbon which adsorbed or recovered the dissolved gold and silver. The barren solution was recycled to the top of the leach column.

Initial column leach experiments were conducted on the mill tailings to obtain baseline data for comparison with data obtained from agglomerated materials. We simply mixed various amounts of lime with the tailings— placed the mixture into the column and started flowing cyanide leach solution through the bed of feed. This procedure compared closely with present practice in heap leaching. Flow-rate measurements were made daily for seven days and were averaged to determine the flow rate. The results showed that the percolation flow rate was independent of the amount of lime added to the dry feed. The flow rate remained constant at 0.5 gal/hr/ft$^2$ with lime additions up to 30 pounds per ton of feed. The results are shown graphically as the baseline data in figure 3.

Wet agglomeration techniques were then investigated for balling the ore fines using lime as a binder. The tailings were mixed with various amounts of lime as for the baseline tests except the feed was dampened with 9 weight-percent water, balled the mixture mechanically, and resulting product was cured at ambient conditions for 24 hours prior to leaching. Figure 3 shows the data obtained using the two methods of applying lime to percolation leaching of the silver mill tailings.

A series of column leach tests were conducted to determine the effect of aging the agglomerated feed upon the subsequent percolation flow rates. The silver-bearing tailings were agglomerated using 5 pounds of lime per ton, 9 weight-percent water, and aged at ambient conditions for varying periods of time. The data presented in figure 4 show that the flow rate increases sharply with curing time, up to about 24 hours, and then starts to level off.

The next parameter investigated was the effect that the amount of water used in the agglomeration step exerted on the percolation flow rate. These tests were conducted at conditions of 5 pounds of lime per ton of feed and 24 hours of curing. The effect of moisture addition is shown in figure 5. The effect is almost startling—showing that moisture addition is a critical factor in the agglomeration procedure reaching a maximum at about 9 weight-percent. Uniform wetting of the whole charge of feed is a prerequisite for effective agglomeration.

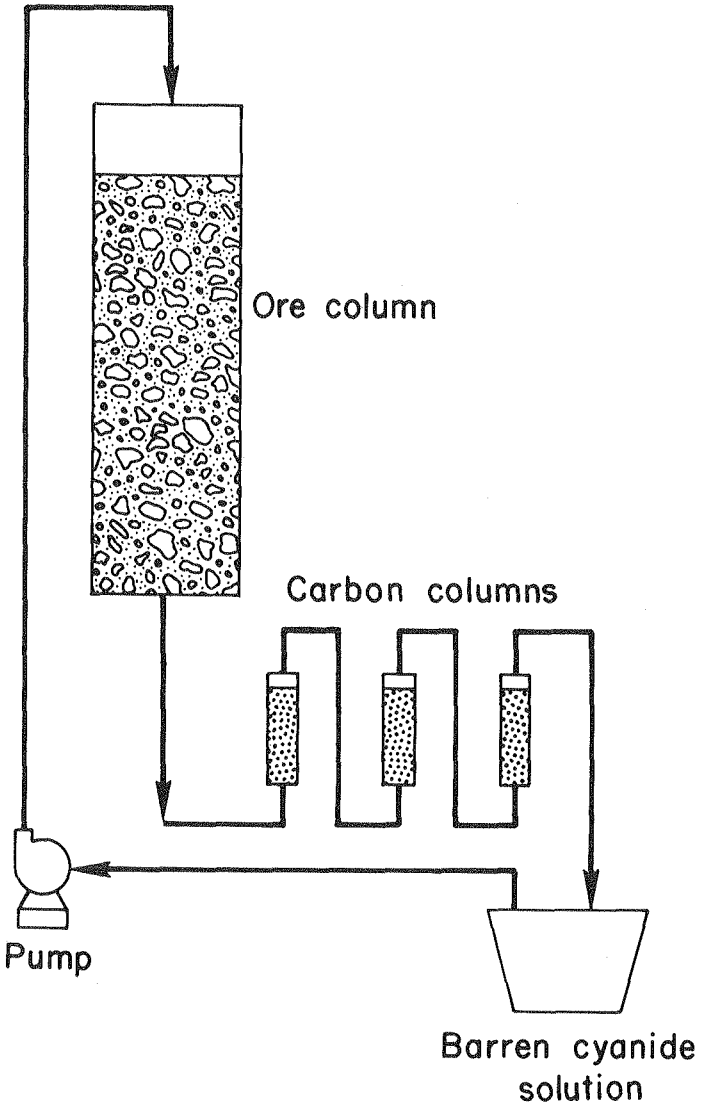


FIGURE 2. Flow diagram for simulated heap leach experiments.

The Bureau of Mines investigated the use of a number of binders for agglomerating gold-silver ores and eventually settled on the use of portland cement (Type II) instead of lime. Portland cement additions of 5 to 15 pounds per ton of clayey ore and 9 weight-percent water produces exceptionally stable agglomerates with high porosity. The bond between the clay and coarse ore fragments are several times stronger than with lime. The agglomerates produced using portland cement as a binder were capable of withstanding both spraying and flooding leach techniques with no visible migration of fines or channeling of the cyanide leach solution.

A report of Investigations describing this research using portland cement is being prepared by the Bureau of Mines for publication, and probably will be in print about July 1981. This improved technique for enhancing the percolation flow of the leach solution through ore heaps has been patented in the United States and patents are pending in the major gold-silver producing countries around the world.

## COMMERCIAL APPLICATIONS

Efforts to improve the heap leach cyanidation process and to increase the range of ores and mine wastes amenable to this process is beginning to bear fruit. Adoption of the new agglomeration technology by the minerals industry initially involved smaller mining operations conducted by entrepreneurs, who immediately recognized the advantages of the agglomeration process. For example, the Tombstone Exploration, Inc., of Tombstone, Arizona, U.S.A., in attempting to heap leach extensive mine dumps from historic old silver mining operations, found that silver from mine dump ore amounted to less than one ounce per ton. Their research showed that finer crushing to 3/8-inch greatly increased silver exposure to leaching, but produced so many ore fines that manageable percolation flow of leach solutions through the ore heap was nearly impossible. Tombstone Exploration found that by crushing to 3/8 inch and using the agglomeration process, silver recovery achieved by heap leaching could be increased to 5 ounces per ton of ore, equivalent to about 90 percent leaching recovery. They are currently processing 100,000 tons of dump ore by the agglomeration-heap leach technique. In another example,

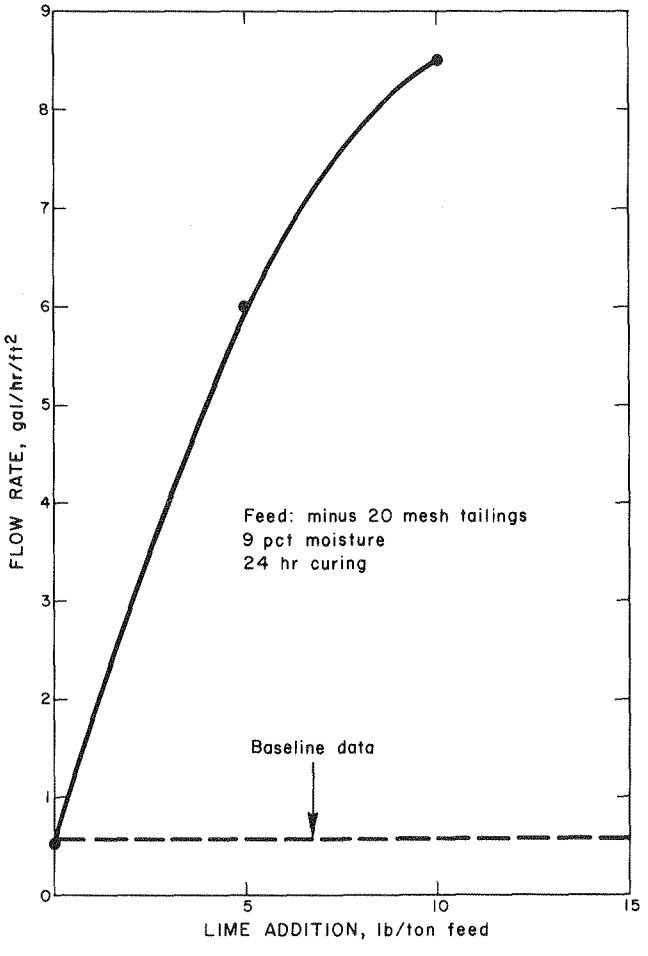


FIGURE 3. Effect of lime addition on percolation flow rate through a silver tailings.

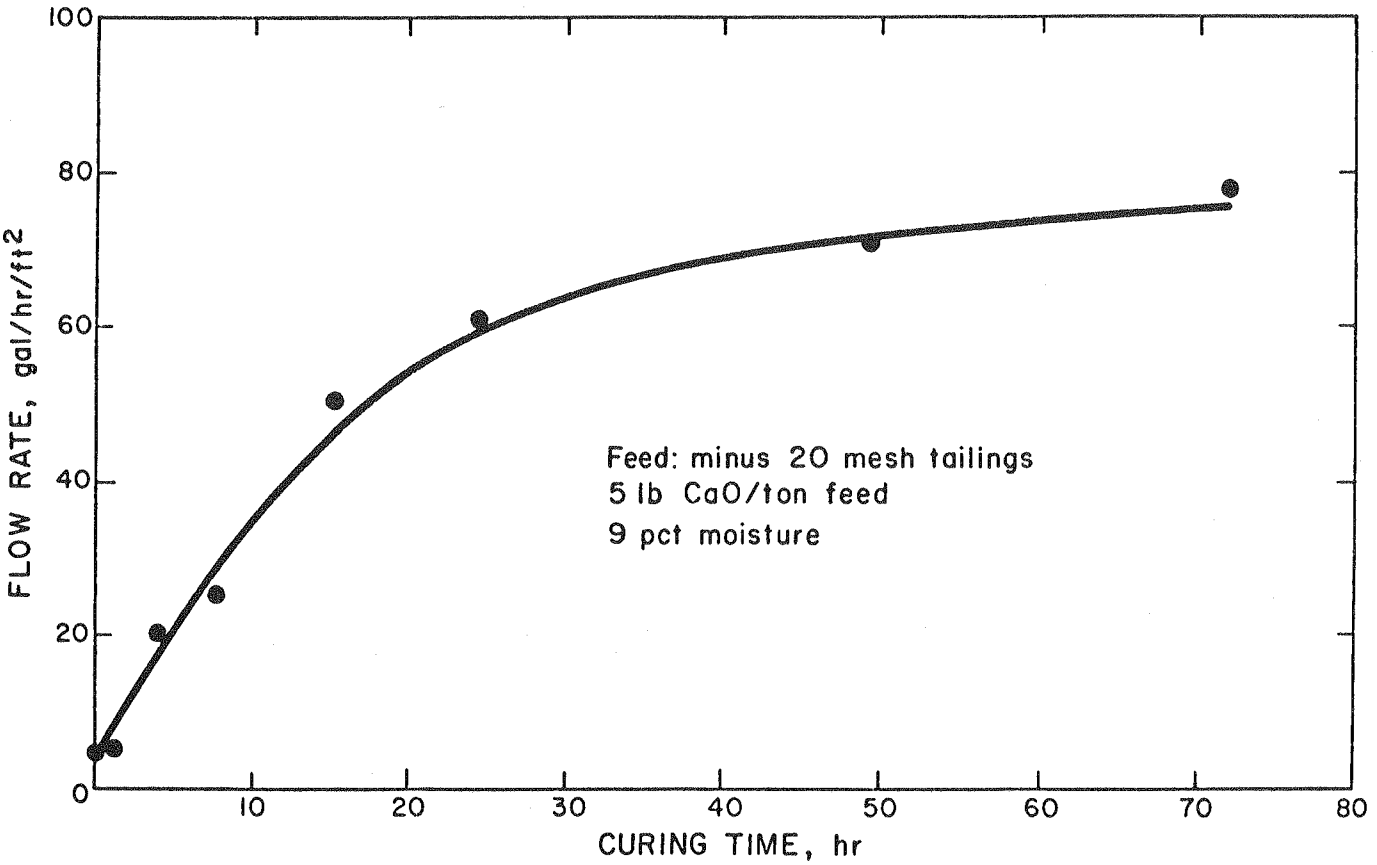


**FIGURE 4. Effect of curing time on percolation flow rate.**

Euromet, Ltd. is applying the agglomeration pretreatment step using portland cement as an aid in heap leaching their gold ore at the Adromache Mine at Proserpines, Austrailia. An example of a successful two-man heap leach operation is the one at San Marcas Gold Mine, Wenden, Arizona, operated by the Hilbrands brothers. Their feed for heap leaching consisted of minus 3/8 inch material which contained so many fines that downward percolation flow was impossible without agglomeration pretreatment. They built a rotating drum agglomerator and are using about 10 pounds of cement per ton of ore as a binder and 10 weight-percent moisture in the form of cyanide solution. They reportedly produced about 500 ounces of gold during a 2-month run last year.

Operating costs for the agglomeration step have been reported by some operators to be less than $1.00 per ton of ore processed. Other operators indicate that the cost of agglomerating may be less than that required to minimize segregation of coarse and fine ore in the building of conventional ore heaps, because the agglomerated feed simplified preparation of the ore heap. Industrial experience has shown that operating costs for heap leaching may be as low as $3.00 per ton of ore under ideal conditions. These costs would imply that cut-off ore grade to ideal heap leach operations may be less than 0.01 ounce of gold per ton of ore.

We attribute several potential benefits to particle agglomeration and cyanide pretreatment of ores prior to heap leaching. They are:

- Clayey and finely crushed ores are rendered amenable to low-cost heap leaching technique. This would allow treatment of large bodies of low-grade ores, many of which are presently considered submarginal.
- Increased overall gold-silver recovery from an ore may be obtained because additional values could be liberated or exposed by finer crushing without encountering particle segregation during the building of ore heaps. Particle segregation can cause localized accumulations of fines within the ore heap which inhibit flow of leach solutions.
- A head start on leaching and increased percolation flow rates are achieved, thus decreasing the required leaching time. This could have economic implications in that the mine capacity could be increased without significantly increasing capital cost for pad preparation, and labor, pumping and maintenance costs could be reduced.
- The highly porous nature of the agglomerates would allow the heaps to "breathe", thereby providing the oxygen necessary for the gold dissolution with cyanide. This could allow elevation of the height of the heaps so that pad preparation cost per ton of ore processed would be less and land area could be more effectively utilized.
- Favorable environmental implications may be realized in that the highly porous structure of the heap would allow more efficient washing of

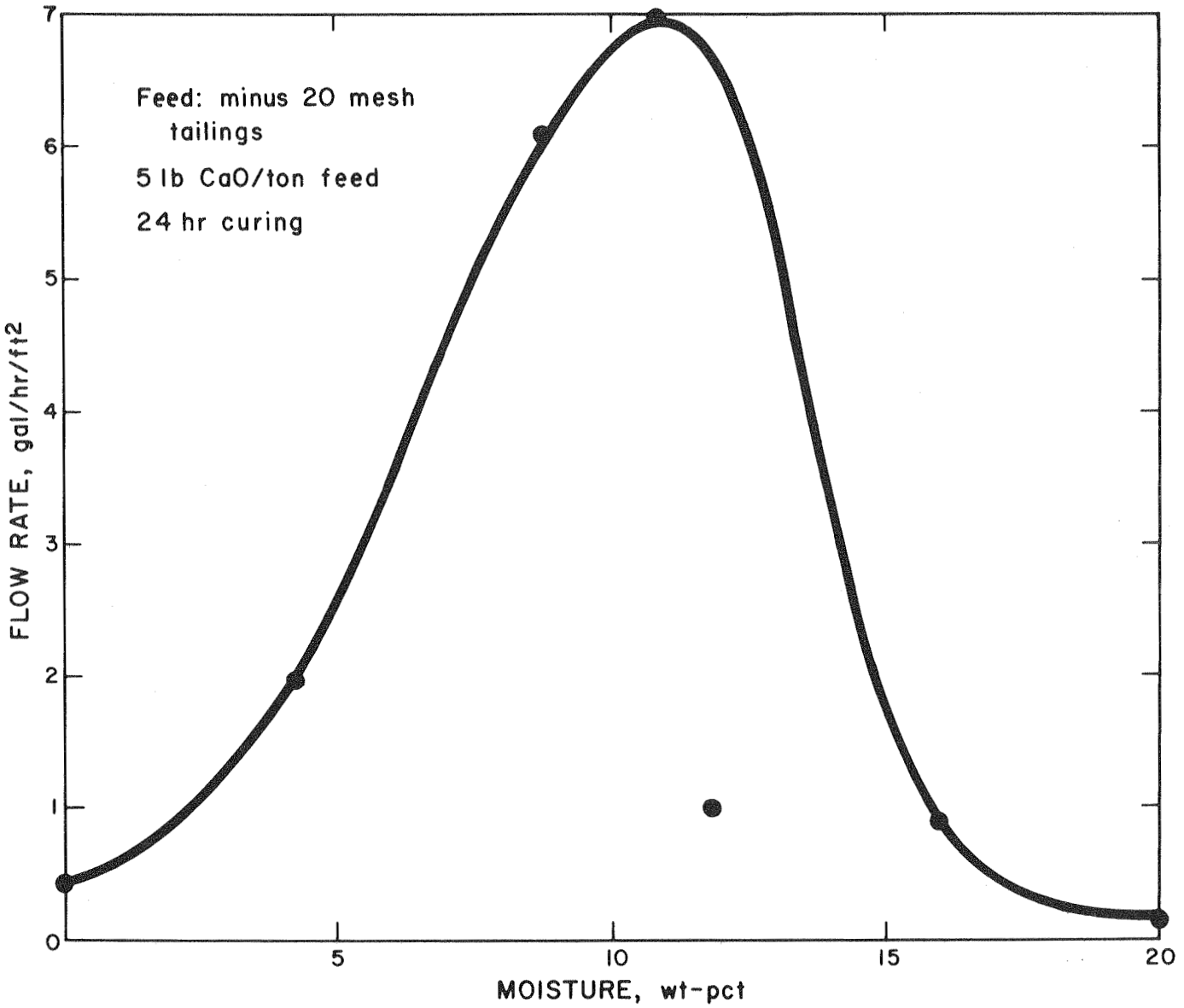


**FIGURE 5. Effect of moisture employed on percolation flow rate.**

residual cyanide from leached ore heaps, and more thorough contact and "killing" of residual cyanide if a hypochlorite wash treatment is applied. Dusting problems from abandoned ore heaps should also be minimized.

- Precious metal concentration in initial solutions exiting from the heaps should be higher, which could be expected to enhance efficiency of gold-silver recovery from these solutions by precipitation on zinc or absorption on activated carbon. The more rapid extraction of values results in a more rapid return of the capital invested.

# LEGAL PROBLEMS OF PLACER LOCATIONS

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## INTRODUCTION

The Mining Law of 1872[1] allows a person or corporation to locate 20 acres of open ground, or multiples thereof, as a placer claim.[2] An association of persons may locate up to 160 acres of placer ground.[3] In areas covered by a government survey, the claim must be described in terms of legal subdivisions[4]. For such claims there may be no requirement for corner monuments[5], or the location procedure may differ substantially from that of lode claims.[6] It must be emphasized, however, that the location requirements vary from state to state and should always be checked in advance.[7] Outside a surveyed area, placer claims are generally located in the same manner as lode claims.

There are several problems which may arise in locating and maintaining a placer claim, and these are the subject of this paper. In the following pages we will concern ourselves with four principal topics.

1. The troublesome distinction between lode and placer claims, and some guidelines for choosing the proper claim for a particular ore deposit.
2. The widespread abuse of association placer claims through the use of dummy locators.
3. Location of claims in irregular sections.
4. The "ten-acre rule" unique to placer claims, requiring a discovery of valuable minerals on each ten-acre tract embraced by the claim.

## LODE/PLACER DISTINCTION[8]

Assuming that the target area is open to location[9] and the mineral in question is locatable[10], the mining claimant is sometimes presented with the problem of classifying the deposit as a lode or placer. The distinction is important, for a placer discovery will not support a lode location, nor will a lode discovery sustain a placer claim.[11] The penalty for being wrong, in other words, is loss of the claim.

A good part of the confusion arises from the definitions set forth in the 1872 Mining Law. A lode is defined as "veins or lodes of quartz or other rock in place"[12] and all other forms of deposit are placers.[13] Thus, the definition of placer is a negative one: any deposit not a lode is a placer.

However, the statute does not define vein or lode, and the courts have struggled with the concept for over a century. Perhaps the most useful definition was formulated in a Nevada case decided in 1877:

> It is difficult to give any definition of the term as understood and used in the acts of congress, which will not be subject to criticism. A fissure in the earth's crust, an opening in its rocks and strata made by some force of nature, in which the mineral is deposited, would seem to be essential to the definition of a lode, in the judgment of geologists. But to the practical miner, the fissure and its walls are only of importance as indication the boundaries within which he may look for and reasonably expect to find the ore he seeks. A continuous body fo mineralized rock lying within any other well-defined boundaries on the earth's surface and under it, would equally constitute, in his eyes, a lode. We are of opinion, therefore, that the term as used in the acts of congress is applicable to any zone or belt of mineralized rock lying within boundaries clearly separating it from the neighboring rock.[14]

It may be helpful to think of a lode in terms of a "mineral sandwich", with the valuable ore contained between two walls of barren host rock.

The courts have also attempted various definitions of placer deposits. The Supreme Court has ruled that a placer claim consists of:

> Ground within defined boundaries which contains mineral in its earth, sand or gravel; ground that includes valuable deposits not in place, that is, not fixed in rock, but which are in a loose state, and may in most cases be collected by washing or amalgamation without milling.[15]

Generally speaking, deposits of broken, loose or scattered material shall be located as placer claims.[16]

This legal concept finds support in standard scientific definitions of a placer:

> A mass of gravel, sand or similar material resulting from the crumbling and erosion of solid rocks and containing particles or nuggets of gold, platinum, tin or other valuable minerals that have been derived from rocks or veins.[17]

These definitions were clear enough in 1872, when Senator William Stewart of Nevada wrote the general mining law. There were basically two kinds of ore deposit in production at the time: The gold-bearing gravels adjacent to California rivers, and the gold and silver quartz veins of the Comstock Lode and other Western mining districts.

Unknown to Senator Stewart were deposits which have both lode and placer characteristics. For example, the buried Tertiary gold channels of the Mother Lode would seem to be a "fossilized" placer, yet they are generally cemented and contained within well-defined boundaries of wall rock. The Commissioner of the General Land Office[18] and the California Supreme Court[19] held that these auriferous gravels were properly located as placers, yet the Secretary of the Interior later held that gold-bearing sandstone could be located only as a lode claim.[20] More recently, uranium deposits concentrated in ancient stream channels were held to be lode deposits.[21] And what is the proper way to locate a nickel laterite, in which the ore is concentrated in a loosely-accummulated but well-defined soil horizon? How does one locate tailings or waste piles abandoned on the public domain?

There are three ways of resolving those situations in which there is substantial doubt concerning the lode/placer nature of the deposit. Sometimes the laws of the United States or a particular state will create a *statutory placer*—that is, the laws dictate that a particular form of deposit shall be located as a placer. There were once several kinds of statutory placers[22] but now there is only one of importance, a Nevada law stating that tailings abandoned for ten successive years shall be located as a placer.[23]

Mining custom provides another means of resolving the dilemma. The Mining law of 1872 specifically provides that:

> The miners of each mining district may make regulations not in conflict with the laws of the United States, or with the laws of the State or Territory in which the district is situated, governing the location, manner of recording [and] amount of work necessary to hold possession of a mining claim...[24]

Thus, the blanket deposits of Leadville, Colorado, containing vein material beneath a superficial deposit of alluvium, were held to be lodes because of the prior practice of locating them as lodes:

> The fact that so many patents have been obtained under these sections, and that so many applications for patent are still pending, is a strong reason against [holding them to be placers].That which has been accepted as law and acted upon by that mining community for such a length of time, should not be adjudged wholly a mistake and put entirely aside because of difficulties in the application of some minor provisions to the peculiarities of the vein or deposit.[25]

Many of these customs have been incorporated into legal and administrative decisions summarized in Volume 1 of the *American Law of Mining.*[26]

Finally, when no other guidelines exist, it is permissible to locate the questionable ground as both a placer and a lode.[27] In doing so there are certain rules to observe. First, the placer must be located before the lode; if done in the reverse sequence, the lode may be considered abandoned.[28] Second, one should note that a lode claim with standard dimensions of 1500 x 600 feet contains 20.66 acres; therefore, a placer using the same location monuments is oversized and possibly invalid. Third, if one wishes to add millsites to the stack (i.e., "triple stake" the ground), the correct sequence of location is placer-lode-millsite.[29]

## ASSOCIATION PLACER CLAIMS

While an individual or a corporation must locate placers in 20-acre increments, two or more people can form "associations" to locate larger claims in multiples of 20 acres. For example, two people can locate 40 acres, four can locate 80 acres, and so on. However, eight or more persons cannot locate more than 160 acres.[30]

There are several advantages to the association claim. It minimizes paperwork: only one location certificate is needed to claim a quarter section of land, as opposed to eight certificates it the ground is taken in 20-acre parcels. It saves money: only one BLM filing fee ($5.00 per claim), only one recording fee (typically $3.00 per claim), and only one set of claim monuments are required. The ultimate attraction is the savings in annual assessment work: only $100.00 per year is needed to hold the entire 160-acre tract, as opposed to $800.00 per year if individual claims were used.[31]

It is small wonder, then, that an overwhelming number of people succumb to temptation and enlist their wives, children, cousins, neighbors and friends as co-locators to form an association of eight. The possibilities are mind boggling: in states which require no ground monuments, an enterprising claimant and a group of friends could "locate" a thousand square miles in the course of a busy weekend. If properly recorded, these claims would foreclose the ground to legitimate miners for at least a year. If the group then perpetuated its fraud by filing false

proofs of labor, a legitimate claimant would be forced to deal with the association or dispute their title in court.

Because of such blatant abuses, association claims are not favored by the courts. When challenged, the members of an association bear the burden of proof in showing their good faith in locating the claim(s).[32] The use of friends, relatives, or employees as "dummy" locators renders the claim invalid.[33] A company which acquires an association claim may later find itself obliged to defend a bad title after the original locators have disappeared, a nearly insuperable task.

It is this writer's opinion, shared by other mining attorneys, that an association placer claim may be considered valid only when there is a written partnership document demonstrating a bona fide relationship between the locators. Any other type of association should be considered invalid, particularly when there is a flurry of quitclaims to one or two locators.[34] (A rare exception is the transfer by eight locators to a corporation in which they are co-equal shareholders.[35])

Another problem of the association claim concerns the statutory requirement that there be a discovery of valuable minerals to validate the claim.[36] The word "discovery" has various meanings[37], but in this context a "prudent man" discovery[38] or *Coleman* discovery[39] is probably required to sustain a placer against government challenge.

The discovery rule is particularly significant when an association claim is transferred to another party. If the transfer is made before a discovery has been made, and the receiving individual or corporation then makes a valid discovery, *the claim is valid only as to the 20 acres immediately surrounding the discovery.*[40]

To avoid the problems which attend nearly all association placers, a mining claimant or company should observe two rules:

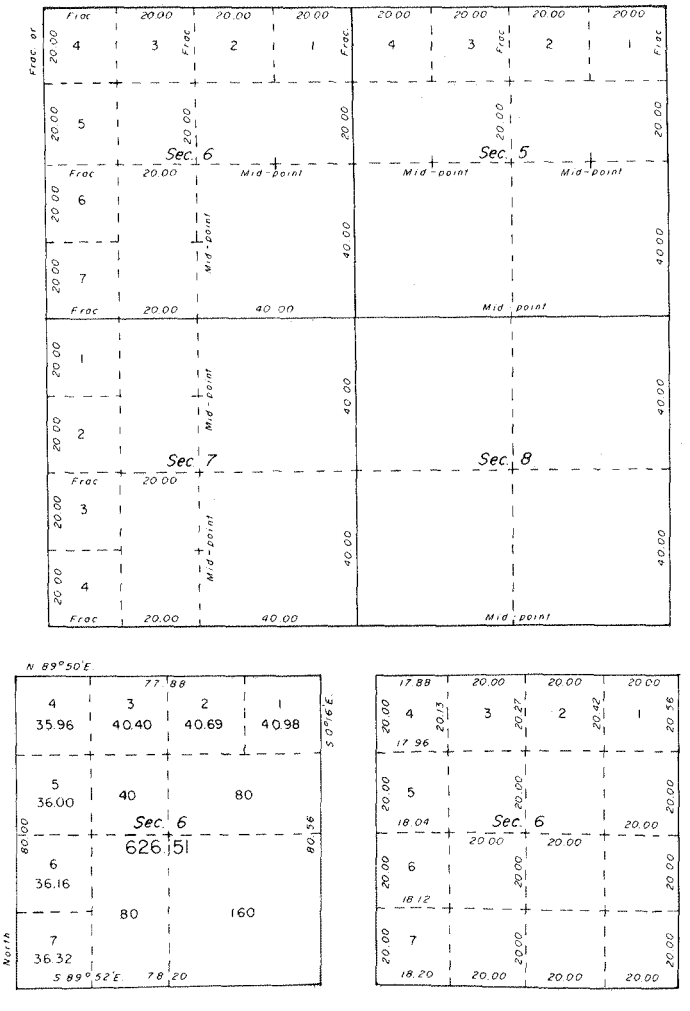
1. When locating claims in the first instance, overcome temptation and locate placer ground in 20-acre parcels.

2. Upon taking title to an association claim of dubious validity, abandon the existing claims and relocate them in 20-acre increments. A lease or purchase agreement should grant specific authority to undertake such remedial acts with or without consent of the original locators.

## LOCATING IRREGULAR SECTIONS

The 1872 Mining Law states that "where the lands have been previously surveyed by the United States, the [placer] entry in its exterior limits shall conform to the legal subdivisions of the public lands, "and" as near as practicable with the United States system of public land surveys, and the rectangular subdivision of such surveys..."[41] The regulations of the Interior Department go even further and require location by legal subdivision in *unsurveyed* areas[42]; this is accomplished through the use of protraction diagrams. If the earth were flat, this would be a perfect system; however, a rectangular survey imposed on a sphere gives rise to certain problems. Among these is the convergence of meridian lines as we near the poles. This means that 36-square-mile townships, which form the principal unit of the system, tend to narrow at the top, giving rise to odd-sized sections. These discrepancies are concentrated along the north and west boundaries of each township so that there are 25 regular sections and 11 "irregular" sections containing less than 640 acres. Please refer to Figure 1, showing the northwestern part of a hypothetical township.[43] The irregularities have been absorbed into lots along the north and west boundaries. The supplementary figures show how Section 6 can be depicted in terms of area or distance (in chains).

Let us now consider the legal description of a proposed placer claim. The SE ¼ of the NW ¼ of Section 6 can be readily subdvided into two 20-acre placer claims described as the N ½ (or W ½) of the quarter-quarter, and the corresponding S ½ (or E ½) of the 40-acre tract.




Showing areas. Showing calculated distances.

**FIGURE 1.**

The NE ¼ of the NW ¼ (Lot 3) presents somewhat of a problem, however, because it is oversize. The resulting subdivision would produce two placer claims of 20.20 acres each, greater than the statute allows. The difficulty is resolved through "the Rule of Approximation", which provides that a claim located in conformity with the rectangular subdivisions of a public land survey may encompass up to 25.0 acres.[44]

Theoretically, one should not subdivide a 40-acre government lot since the resulting boundary, cannot be precisely established.[45] The proper procedure is to request the Cadastral Engineer to subdivide the parcel into one 20-acre lot and two other lots of 10 acres, more or less. However, this process is overly precise and time-consuming for a company engaged in intial prospecting and exploration. Therefore, I recommend the following procedure for locating a government lot of 40 + acres:

1. So long as the property is merely a prospect, it is sufficient to subdivide the government lot--for example, the N ½ of Lot No. 3, Section 6, T. 20 N., R. 20 E., MDB&M. Although this method does not correspond to the exacting standards of the Cadastral Survey, it is certainly adequate to hold the ground against a rival locator.

2. When the property is deemed valuable enough for a major exploration or development program, it would be wise to stake and relocate it in terms of metes and bounds to forestall any assertion of improper location procedures.

3. Finally, if the parcel will be included in a mining operation or the locator is considering patent proceedings, absolute compliance with the laws is required. One should then ask the Cadastral Branch to prepare a supplemental plat dividing the lot of 40 + acres into three smaller government lots of 20 acres and 10 + acres. These can be appropriated by a third generation of locations.

In the absence of a survey, a placer claim may be located by metes and bounds.[46] A claimant may follow a canyon, gulch, or other topographic feature to avoid inclusion of nonmineral ground.[47] The claim may also be irregular when surrounded by prior locations.[48] However, long "shoestring" claims are not permitted unless confined to a canyon.[49] A placer, unlike a lode, is not required to overlap prior claims or patented ground in order to attain a rectangular form.[50] Noncontigous parcels may not be appropriated under a single location.[51]

If a locator inadvertently includes more than 20 acres in his location, the claim is not void, but boidable as to the excess. Until the claimant has been advised of the excess and has had a reasonable time to select the ground to which he is entitled, his possession extends to the entire claim, and one who goes on the claim to make a location of any part is a trespasser. If the locator acts fraudulently in including more ground than that to which he is entitled, the entire claim is void.[52]

A placer claim does not automatically include lode deposits within its boundaries; these must be acquired by means of long "shoestring" claims 1500 feet long and 50 feet wide.[53] Another practical deterrent is the legal requirement that the placer claimant give his consent to subsequent entry of the lode claimant; without such consent the lode location is void.

> [No] person other than the owner of the placer claim has the right to enter upon the same for the purpose of discovering [unknown] veins or lodes and locating the same, and one who attempts to do so without the owner's consent, or without his knowledge, is a trespasser, and can acquire no rights to such lode claim.[54]

However, it is my opinion that the "consent doctrine," first formulated in the 1800's, is due for reconsideration by modern courts. At its worst extreme, the doctrine allows a senior placer claimant to deny access to tens or hundreds of acres that may be more valuable for lode minerals. In areas like a desert pediment, where placer materials may form a veneer of sediment over underlying lode deposits, the prospective lode claimant should be allowed free entry to the ground for staking and exploration. If his subsequent activities (e.g., open pit mining) will disturb or destroy the placer mineralization, the lode claimant should be required to compensate the placer owner for lost values, much as a mining company must now pay damages to a surface owner. Alternatively, the parties could form a joint venture to develop and mine both deposits.

## THE "TEN-ACRE" RULE

As the result of several statutes and their interpretation by the courts[55], the locator of a placer claim must demonstrate the mineral character of each 10-acre tract within his claim. It is not necessary that there be an actual discovery or disclosure of mineral on each ten acres, as the mineral character of the tracts may be inferred from the nature of the actual discovery, the character of the deposit and formation, the surrounding geological conditions, and other facts and circumstances of the case. The unit of investigation is not necessarily a 10-acre tract produced by further subdivision of a 40-acre subdivision. Thus, where a 10-acre placer claim straddles two or more 10-acre subdivisions, a discovery on the claim is sufficient to show the mineral character of the entire claim, and not merely of the particular 10-acre tract on which it was made.

## FOOTNOTES

1. The Mineral Location Law of 1872, 17 Stat. 91 (1972), 30 U.S.C. secs. 21-64 (1970). The law is commonly referred to as the "1872 Mining Law" or the "general mining law."
2. 30 U.S.C. sec. 35.
3. 30 U.S.C. sec. 36.
4. 30 U.S.C. sec. 35. "Where the lands have been previously surveyed by the United States, the entry in its exterior limits shall conform to the legal subdivisions of the public lands."
5. Calif. Pub. Res. Code sec. 2303(b); Oregon Rev. Stat. 517.046.
6. For example, Nevada limits the marking requirement to a location monument along the north boundary of the claim. NRS 517.090.
7. Some useful references are the *Digest of Mining Claim Laws* (Second Edition),, published by the Rocky Mountain Mineral Law Foundation; and the Tables in the back of Volume 1 of *The American Law of Mining.*
8. For a comprehensive discussion of this topic, see Reeves, "Lode v. Placer," 1 American Law of Mining sec. 5.9 A.
9. Lands once open to location may have been withdrawn or reserved for Indian reservations, defense installations, national monuments, power sites, or a variety of other purposes. A *reservation* is permanent in nature, whereas a *withdrawal* is supposedly of temporary duration while the area is being considered for a possible reservation. Unfortunately, "temporary" withdrawals have tended to become permanent. One study concluded that about one-half of the public lands had been withdrawn between 1968 and 1974. Bennethum and Lee, "Is our Account Overdrawn?"
10. The Mineral Leasing Act of 1920, 30 U.S.C. secs. 181-287, withdrew from location all deposits of oil, gas, coal, potassium, sodium, and phosphate. The Common Varieties Act of 1955, 30 U.S.C. secs. 611, withdrew sand, gravel, pumice, cinders, building stone and other "common" minerals.
11. *Cole v. Ralph,* 252 U.S. 286 (1920); *U.S. v. Haskings,* 505 F. 2d 246 (9th Cir. 1974).
12. 30 U.S.C. sec. 23.
13. "Claims usually called 'placers', including all forms of deposits, excepting veins of quartz or other rock in place, shall be subject to entry and patent..." 30 U.S.C. sec. 35.
14. Eureka Consol. Min. Co. v. Richmond Min. Co., 8 F. Cas. 819 (C.C.D. Nev. 1877).
15. *U.S. v. Iron Silver Min. Co.* 128 U.S. 673, 679 (1888.)
16. *Layman v. Ellis,* 52 L.D. 714 (1929).
17. *Glossary of Geology and Related Sciences,* American Geological Institute (2d Ed. 1960).
18. Comm. Drummond to Thomas N. Stoddard, Feb. 12, 1872, Copp, U.S. Mining Decisions 78 (1874): "If it was intended to ask if the auriferous cement claims, found in what are sometimes called ancient river beds, and usually worked by the hydraulic process, properly come within the signification of the term 'rock in place', as used in the second section of the mining statute of twenty-sixty July, 1866, the answer must, undoubtedly, be in the negative; several claims of that character having already been patented under the placer mining law of July 9th, 1870, they fully coming within the meaning of the term 'placer' as defined in said act."
19. *Gregory v. Pershbaker,* 73 Cal. 109, 114-15, 14 P. 401, 403 (1887): "Whatever the origin of the subterranean channels containing gravel beds, they have long been known to exist in California and they have been generally supposed to be, and generally spoken of as, the beds of ancient rivers in which the gravel was deposited by fluvial action, and which were either from their beginning subterranean, or upon which the superincumbent earth or rock has been hurled by means of convulsion, caused by volcanic or other natural force. That the bed of gravel mentioned in the findings, to the limited extent it has been prospected by the intervenor's tunnle, 'descends or drops on an average of about eight degrees', does not of itself make the gravel deposit a lode with 'a top or apex', nor contradict the theory that the channel was the channel of a mountain stream or torrent . . . Referring to the common use of the word by miners, to the dictionaries, and to the adjudications of courts, the gravel bed with gold therein as described in the finding is a placer."
20. *E.M. Palmer,* 38 L.D. 294, 297 (1909): "From the reasoning of the authorities cited it follows that sand rock, or sedimentary sandstone formation, in the general mass of the mountain bearing gold such as is here disclosed by the evidence, is rock in place bearing mineral and constitutes a vein or lode, within the purview of the statute, and can be located and entered only under law applicable to lode deposits." In *Gregory v. Pershbaker,* 73 Cal. 109, 14 P. 401 (1887), the material of the deposit was described as "of a hard nature" which "has to be detached from its position by the use of picks and gads." In *E.M. Palmer,* the material of the deposit was described as "soft sandstone, very easily picked." One can easily agree with Lindley that the distinction between the two types of deposit "is not altogether obvious." 2 Lindley, Mines sec. 427 (3d Ed. 1914).
21. *Globe Min. Co. v. Anderson,* 78 Wyo. 17, 318 P. 2nd 373 (1957); *Western Std. Uranium Co. v. Thurston,* 355 P. 2d 377 (Wyo. 1960).
22. The Building Stone Law of 1892, 30 U.S.C. sec. 161, largely repealed by the Common Varieties Act except for stone of "distinct and special value"; the Oil Placer Act of 1897, repealed entirely by the Mining Leasing Act of 1920; the Saline Placer Act of 1901, repealed by the Mineral Leasing Act of 1920 as amended in 1928.
23. N.R.S. 517.115. Even in the absence of such a statute, tailings may generally be located as placer. *Ritter v. Lynch,* 123 F. 930 (C.C.D. Nev. 1903); *U.S. v. Grosso,* 53 I.D. 115 (1930).
24. U.S.C. sec. 28.
25. *Iron Silver Min. Co. v. Mike & Starr Gold & Silver Min. Co.,* 143 U.S. 394, 400 (1892).
26. 1 American Law of Mining sec. 2.7A-2.7U.
27. Id. at sec. 5.9A.
28. *Shoshone Min. Co. v. Rutter,* 87 F. 801 (9th Cir. 1898).
29. Harris, "The Law of Millsites," Natural Resources Lawyer, Vol. 9, No. 1 (1976) at page 120.
30. 30 U.S.C. sec. 36.
31. 30 U.S.C. sec. 28. *Reeder v. Mills,* 62 Cal. App. 581, 217 p. 562 (1923).
32. *Foster v. Seaton,* 271 F. 2d 836 (D.C. Cir. 1959).
33. *Cook v. Klonos,* 164 F. 529 (C.C.A. Alaska, 1908). See also the Notes of Decision to 30 U.S.C.A. sec. 35 and 36.
34. If an association claim based on a valid discovery is located in good faith, the locators may thereafter transfer their interests as they wish without affecting the validity of the claim. *Rooney v. Bartlett,* 200 F. 700 (9th Cir. 1912). Note, however, the twofold requirement of *discovery* and *good faith.*
35. *Borgwardt v. McKittrick Oil Co.,* 164 Cal. 650, 130 P. 417 (1913).
36. 30 U.S.C. sec. 23, as incorporated by reference into 30 U.S.C. sec. 35; see also 1 American Law of Mining sec. 4.60.
37. See the extensive chapter on mineral discovery in 1 American Law of Mining sec. 4.44-4.89.
38. The "prudent man" rule of mineral discovery requires sufficient showing of mineralization to justify a person of ordinary prudence in further expenditure of labor and money with a reasonable prospect of developing a valuable mine. *Castle v. Womble,* 19 L.D. 455, 457 (1894).
39. *U.S. v. Coleman,* 390 U.S. 599 (1968). The Supreme Court held that a valuable discovery is made when the deposit can be extracted and marketed at a profit under current economic conditions. This is also referred to as the "marketability" or "profitability" test.
40. *U.S. v. Ickes,* 98 F. 2d 271 (D.C. Cir. 1938).
41. 30 U.S.C. sec. 35.

42. 43 C.F.R. 3842.1-5(a).
43. These diagrams are taken from the excellent article "Placer Mining Claims - Selected Problems and Suggested Solutions," 23 Rocky Mtn. Min. Law Institute 385 (1977) by Earl M. Hill of Reno, Nevada, whose cooperation is greatly appreciated.
44. Henry P. Sayles, 2 L.D. 88 (1883); Hill, supra, at p. 406, 1 American Law of Mining sec. 5.23A.
45. For a detailed review of this problem, see Hill, supra note 43.
46. "Where placer claims cannot be conformed to legal subdivisions, survey and plat shall be made as on unsurveyed lands." 30 U.S.C. sec. 35.
47. *Mitchell v. Hutchinson,* 142 Cal. 404, 76 p. 55 (1904).
48. 43 C.F.R. sec. 3842.1-5(b).
49. William F. Carr, 53 I.D. 431 (1931).
50. *Dripps v. Allison's Mines Co.,* 45 Cal. App. 95, 187 p. 448 (1919).
51. *Stenfjeld v. Espe,* 171 F. 825 (9th Cir. 1909).
52. 1 American Law of Mining sec. 5.23A.
53. Id. at sec. 5.30.
54. *Campbell v.McIntyre,* 295 F. 45 (9th Cir. 1924).
55. See 1 American Law of Mining sec. 4.92 for a complete discussion of the background of the "ten-acre" rule and supporting citations.

# SOME NOTES ON PLACER INVESTIGATION

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## ABSTRACT

Today's renewed interest in gold mining is being felt in placer districts known to contain unworked gravels. This reflects recent increases in the price of gold—but gold price by itself cannot convert a long-dormant placer into a viable prospect. There are other factors.

Subject matter for this paper has been selected with the needs of exploration managers and field personnel in mind and, in part, is intended as an overview of placer valuation fundamentals. Discussions emphasize three aspects of the work. In broad terms these can be viewed as: (1) sampling; (2) sample processing and (3) factors other than mineral content to be considered.

To provide a conceptual view of placer investigation, the author explains how sampling procedures, sample processing, and the interpretation of results are affected by large particle sizes, erratic distribution of values and by the high unit value of gold. A clear understanding of these factors and of related problems is a "must" for effective placer work.

The reader will find that a competent placer investigator today must, among other things, be knowledgeable in the field of regulatory law as applied to mining. A working knowledge of placer mining methods and their limiting features is also necessary.

Many people within the mineral industry do not appreciate that compared to lode deposits, placers are difficult to sample. Also, few seem aware that drilling equipment and analytical procedures routinely used for hard-rock sampling will yield misleading results when applied to gold placers. Reasons are explained.

Within the discussion of sample processing a procedure for determining the recoverable gold content of placer samples is set out in detail.

A list of selected references is appended to this paper.

## INTRODUCTION

Placer examiners in the Western States will find that for the most part they are reappraising long-known deposits that have been intentionally bypassed, mined out or partially worked and then abandoned for one reason or another. In essence, they are sifting leftovers and looking for residual values that may merit reconsideration in light of current gold prices. But to have real meaning—the investigative process must look beyond the question of cubic yards or indicated value per yard. As late as ten years ago, obstacles to placer mining were primarily physical and economic. Today they may be statutory. Examples are found in regulatory powers exercised by federal, state and local governments which have changed our way of doing business.

In brief, we must now cope with environmental control measures, zoning ordinances or perhaps with permitting procedures and their built-in time delays and other uncertainties which add to the cost of mining and in some places preclude placer mining as a practical matter. Today, a realistic placer examination will begin with a close look at environmental factors and limits they may impose on mining in a particular area. But as a matter of fact—we generally see first efforts aimed at determining mineral value. In view of this, the fundamentals of placer sampling are given preferential treatment in the following pages.

## SAMPLING

### GENERAL

At the outset it is important to distinguish between sampling for the purpose of initial valuation and sampling to block out mining ground. It sometimes takes only a few intelligently selected samples to determine that a property has no prospective value. On the other hand, a potentially minable deposit may require a comprehensive sampling program to establish available yardage, recoverable value and other information needed to judge its worth. Sampling to block out a deposit will usually pass through a series of steps designed to give a decisive answer at the end of each step, but in any case to determine the actual recoverable value can be a difficult task. This reflects, among other things, built-in problems related to sample particle sizes, the high unit value of gold and its erratic distribution in most placer deposits. The importance of these factors in sampling, sample processing and interpretation of the results cannot be overstated.

### LARGE PARTICLE SIZE

Concerning particle size. The theory of sampling is that if enough small portions of a deposit are taken at suitable intervals, their average value will approximate that of the whole. But consider the extreme range of particle sizes to be dealt with in most placers. It is not unusual for a gravel deposit to contain large rocks or boulders weighing tens of pounds or perhaps hundreds of pounds each as well as silt, sand and cobbles in varying amounts. When taking relatively small samples from such material there is a natural tendency to avoid areas containing large-size rock or to include disproportionate amounts of fine material in samples without considering the effect on calculated values.

Consider that the gold particles in placers are found among the finer-size material, while on the other hand the large rocks or boulders contain no recoverable value within themselves. For this reason the sampler who takes too much fine material or fails

to make an allowance for the valueless boulders is likely to report an incorrect high value. In many cases the effect is the same as salting a sample, a fact which seems to escape most people.

To put it another way, where all bank-run material including boulders will be handled in a mining operation the samples should reflect this fact. A direct approach would be to cut samples large enough to include the fair proportion of boulders or large-size rock and thus obtain samples which in a physical sense represent the bank-run material. If this can't be done, the sampler should (where needed) adjust the indicated values by means of a "boulder" factor. To illustrate. If ten percent of the ground surrounding a 6"x12" (cross section) channel cut consists of rocks too large to be included in the sample, the initial calculated value should be reduced proportionately. In this case, a ten percent reduction of the indicated value would compensate for the missing barren material.

It should be pointed out that while the foregoing correction procedure is generally reliable, there are placers having boulder problems so severe they cannot be effectively sampled by any means short of mining the entire deposit.

## HIGH UNIT VALUE

Next, consider the high unit value of gold and its effect on sample calculations. In a commercial placer the ratio of gold to gravel (by volume) is commonly less than one part gold to 100 million parts of gravel[1]. Because of this ratio, a slight error in the gold content of a small sample will appear as a large error in the calculated bank-run value. The effects are illustrated by Figure 1 and Table I.

A 1-milligram particle of placer gold is about 0.7 millimeter in diameter. Four of these particles will fit on the head of a common pin. The curve in Figure 1 has been plotted to show the effect that a single 1-milligram gold particle would have on the calculated value of a 1-foot sample increment in drill holes ranging from 36 inches to 2 inches in diameter. It is seen that as hole diameters decrease, calculated values brought about by the 1-milligram gold particle increase exponentially. It clearly shows that accidental addition or loss of a single gold particle can cause serious error when relying on small-diameter drill holes for placer valuation.

A new dimension is added to the problem when uncased drill holes are used. In these, any caving or raveling will result in non-representative samples which in the long run may be worse than no sample at all. Still, attempts are being made to sample and evaluate gold placers by means of uncased holes 5 inches or smaller in diameter, in which case some sloughing of the hole is inevitable. If the source, quantity and gold content of the extraneous material (entering the sample) could be identified with certainty, corrections might be made when calculating sample values—but uncased drill holes cannot provide volume measurements within the confidence level required for placer work. In brief, where one plans to evaluate a gold placer by extrapolating data from small-diameter drill holes, volumetric measurements must be accurate.

**TABLE I. Effect of a single gold particle on placer samples of various size**

(Based on a gold price of *$100 per ounce*)

| | Size of gold particle and effect on sample, $ per cubic yard | | | |
|---|---|---|---|---|
| Channel cut size (inches) | 20-mesh (6.6 mg) | 40-mesh (0.9 mg) | 60-mesh (0.3 mg) | 100-mesh (0.1 mg) |
| 3 x 6 | $4.12 | $0.56 | $0.19 | $0.06 |
| 6 x 6 | $2.06 | $0.28 | $0.09 | $0.03 |
| 6 x 12 | $1.03 | $0.14 | $0.05 | $0.02 |
| 12 x 12 | $0.52 | $0.07 | $0.02 | $0.01 |
| 16-in. pan* | $3.44 | $0.47 | $0.16 | $0.05 |

*180 pans per cubic yard

Values shown are those which would result from one gold particle in a 1-foot sample increment and are based on 900-fine placer gold, with gold at $100 per fine ounce.

Gold weights (mg) shown in Table I were determined by sieving placer gold and finding the average weight of largest gold particles found in each size fraction.

To update values shown in table:

$$\text{Updated value } (\$) = \frac{\text{Value shown in table x new gold price}}{100}$$

## ERRATIC DISTRIBUTION OF VALUE

Obtaining satisfactory placer samples would be much simpler if the gold particles were uniformly distributed throughout the whole mass. In almost all placers, however, the heavy minerals are more or less segregated. In cases where economic values are confined to pay streaks these are likely to occur as narrow, discontinuous accumulations with little or no value in between. Where coarse gold is present it is generally very erratic in distribution and under such conditions a reliable placer valuation may depend as much on experience and judgement as on the sampling itself.

In theory, this problem can be overcome by taking samples large enough to offset the eccentricities of a deposit, but to do so, would mean taking samples measured in tons rather than pounds. In practice, taking such large samples is seldom feasible. The so-called "plant run" is a form of large-volume sample in which a portion of the deposit is mined and processed under actual working conditions. A pilot plant employing full-scale processing and recovery equipment is sometimes used. But when relying on plant runs, good judgement must be used to insure that the points or areas selected for large-scale sampling actually represent the deposit as a whole.

Some argue that if enough small samples are taken, the high values will balance the lows and when combined provide an end figure representative of the deposit. In theory this may be true, but practical considerations seldom permit taking the number of samples needed to achieve this end.

### NUMBER OR TYPE OF SAMPLES REQUIRED

There is no fixed rule for the number of samples or drill holes needed to adequately test a piece of placer ground. Speaking generally, spotty ground will require more holes than ground in which values have a somewhat general distribution. Physical conditions, tenor and size of the deposit are factors to be considered but in the end personal judgement must be relied on. The first sampling may be of a cursory nature sufficient only to determine if further interest is warranted. Where preliminary sampling is encouraging, subsequent investigation or exploration should progress through a series of carefully evaluated steps designed to give a decisive answer at the end of each step.

Placers are commonly explored by means of machine-dug trenches, test pits or by drilling. How much time and expense are needed to systematically explore a deposit is obviously a question of its size and physical characteristics. In shallow ground with little water, test pitting or trenching with a backhoe or bulldozer usually will be the cheapest and fastest method of gaining access for sampling. Where the ground stands well, suitable channels cut vertically in the exposed walls are commonly relied on for an initial evaluation. When coupled with experience, samples of this type can provide a reliable basis for estimating the size and tenor of the deposit as a whole.

In the past, large placers particularly dredge deposits, were almost always sampled by use of drills. Power-driven churn drills equipped with heavy steel drive pipe and special placer tools are employed. The fact that almost all gold dredging projects have been based on the results of churn drill sampling is significant. Details of placer drilling and related sampling procedures can be found in references at the end of this paper.

Where a property is dormant or a mere prospect, determining its prospective worth can be a difficult task. Typically, the first appraisal must rely on minimal information and perhaps a few samples taken from irregular-spaced points. The interpretation of sample results in such cases requires a large measure of common sense and an intelligent perception of what each sample means. Although such samples do not determine absolute volume or unit value, they can if properly selected, indicate the range of values within the deposit. Coupled with experience samples of this type provide a basis for estimating the size and tenor of the deposit as a whole. Where additional sampling is called for, there are no rule of thumb guides. The kind of sampling and number of samples required will depend on the type of deposit, the number of exposures available for testing and other factors. In any case, appropriate sampling and correct interpretation of data are essential, for together they provide the foundation upon which succeeding conclusions must rest.

Aside from indicating the mineral content of a deposit, a sampling program should provide other information needed for economic appraisal. The mineral value in itself may be of secondary concern where sampling or examination of the lands show unfavorable bedrock conditions, excessive amounts of sticky clay, numerous boulder or other factors that would adversely affect a placer mining operation. A good sampling program will reveal such conditions should they exist.

## SAMPLE PROCESSING

No dredge or other large-scale placer equipment saves 100 percent of the bank-run value. The manner in which placer samples are processed should reflect this fact. In other words, the sample should be processed in a way that indicates the *recoverable* value rather than total mineral content. When dealing with gold placers samples are washed in a gravity concentration device which if properly used, will indicate the commercially recoverable gold content of a sample. Suitable sample washing devices include the miners' pan, sluice box, rocker or small engine-powered trommel (screen) equipped with a riffled sluice.

After the sample has been washed and reduced to a heavy mineral (black sand) concentrate, the concentrate is placed in a pan with a globule of mercury which picks up the gold. The gold and mercury are then separated by dissolving the mercury in nitric acid. The gold content of a cubic yard is then calculated using the volume of the sample and weight of recovered gold. Details of the procedure follow.

### PROCEDURES FOR DETERMINING RECOVERABLE GOLD IN PLACER SAMPLES

a. Reduce the original sample to a black sand concentrate by panning, rocking or other suitable means. Where necessary, further reduce riffle concentrates by panning.

b. While in the pan, check for visible gold and note the number of colors, their size, shape and other characteristics. If surface features or other physical characteristics of the recovered gold are to be preserved for future reference, representative gold particles should be removed at this point. (See Note 1).

c. Add a globule of clean gold-free mercury (about the size of a small bean) and working over a safety pan reduce the black sand to a small-volume concentrate. Near the end of the panning process the mercury may tend to ride over the top of

certain fine-size or hard-packing materials, but by this time the recoverable gold should have been picked up by the mercury. If some of the mercury separates into small beads during panning, it can be reunited (in the pan) by adding a few drops of sulphuric acid.

d. Remove the mercury and place it in a 150-ml Pyrex beaker. Add 40 or 50 ml of diluted nitric acid (Note 2) and digest until the mercury globule is reduced to the size of a match head. Transfer to a #0 glazed porcelain parting cup, add fresh acid and complete the digestion using a low heat if necessary. Fine-size gold will be left in a coherent sponge-like mass if rapid digestion or boiling is avoided.

e. After decanting off the acid, carefully wash the gold three or more times with warm water. Add a drop or two of alcohol (to prevent spattering) and dry in the parting cup at low heat.

f. Anneal the gold by bringing the bottom of the parting cup to a red heat. This step will eliminate any residual mercury and is essential when working with small gold weights.

g. Transfer the annealed gold to a balance pan and weigh to the nearest 1/10 milligram.

**NOTES**

1. Placer gold may lose its original surface texture when amalgamated. In the case of fine-size particles, their end form (after removal of mercury with acid) may assume the shape of fine needle-like crystals which actually are pseudomorphs after gold amalgam crystals. When it is desired to keep placer gold in its natural form for future reference or study, some representative particles should be manually removed from the black sand concentrate before amalgamating the remainder.

2. Prepare a parting solution by diluting commercial nitric acid with water to a strength which will dissolve mercury without violent action. A 1:1 dilution is usually satisfactory.

3. Gold contained in a hard-packing or relatively large-volume black sand concentrate which is difficult or tedious to pan, can be amalgamated in a glass jar rotated about its longitudinal axis (on motor-driven rolls) at a peripheral speed of 500 inches per minute. The addition of a small amount of lye will counteract oil or grease and generally assist amalgamation. Commercially available rolls designed to drive laboratory-size grinding mills should not be used for placer work until the drive roll speed has been changed (slowed) to the 500-inch per minute peripheral speed noted above.

4. The described sample processing procedure will extract all free gold recoverable by commercial placer methods. With care, extremely low gold values can be determined. Where so-called "rusty" gold is present, provisions should be made to scour or otherwise brighten the gold prior to amalgamation. This can be dome by rubbing it on the bottom of the pan or by adding a few pebbles to the amalgamating jar. Fine grinding or pulverizing should be avoided.

## COMPUTING SAMPLE VALUES

$$¢ \text{ per cubic yard} = \frac{\text{milligrams gold} \times \text{price per mg}(¢) \times 27}{\text{volume of sample (cu. ft.)}}$$

$$¢ \text{ per cubic yard} = \frac{\text{milligrams gold} \times \text{price per mg}(¢) \times 3000^{1}}{\text{weight of sample (pounds)}}$$

[1]where 1 bank yard = 3,000 pounds.

Most placer gold falls within 40 or 50 points of 900-fine. For this reason it is customary to use a gold fineness of 900 when computing sample values. Example: with gold at $35 per fine ounce, one milligram of fine gold is worth 0.112 cent and 1 milligram of 900-fine placer gold would be worth 0.1 cent.

## BLACK SAND VALUES

Black sand is a term applied to heavy dark colored mineral grains associated with placer deposits. In the Western States the black sand content of placer gravels is commonly between 5 and 20 pounds per cubic yard of bank-run gravel. Magnetite and ilmenite are usually the most abundant minerals in black sand concentrates associated with gold placers. Common but less abundant constituents include garnet, zircon and rutile.

Over the years there have been persistent reports of black sands that allegedly contain gold and platinum in colloidal form or in a complex form not susceptible to conventional recovery methods. Many special devices and recovery schemes said to be capable of extracting these alleged "hidden" or locked-in values have been (and are still being) promoted. The fact that many have been tried but none have developed into self-sustained commercial operations should be significant.

Where black sand concentrates do contain gold —it is generally present as fine-size particles, some of which may be coated by iron or manganese oxide. This is referred to as "rusty" gold. Removal of the coating by abrasion will allow the gold to be amalgamated when brought into contact with mercury. Over the years, people within the gold dredging industry found that from an economic standpoint—the gold content of black sands that cannot be recovered by simple scouring and amalgamation in a cleanup barrel—is not worth saving.

## PROVEN RECOVERY PROCEDURES

A key factor in profitable working of low-grade placer material is *throughput.* This means operating at maximum dredge or plant capacity with minimum lost time. Effective throughput in turn calls for the use of simple, trouble-free equipment of proven design. These requirements are met by the placer-type jig and in particular by the common sluice.

Many special placer machines and recovery processes have been put on the market or proposed. Some claim the ability to extract microscopic or collodial gold from materials that show little or no value when tested or processed by conventional means. Others are intended to recover the varying amounts of fine-size gold admittedly lost in large-scale placer operations. The "inventors" or promotors of such devices seem obsessed with the idea of making a near-100% gold recovery with little consideration of added cost of the equipment or increased operating expenses that accompany such schemes. But despite the many "improved" placer machines and new gold-saving methods that have been offered, the simple riffled sluice has held its rivals have been discarded. A notable exception is the placer-type jig which has generally replaced the riffled gold-saving tables (sluices) in modern dredging practice.

Again, it should be stressed that the success of a commercial-scale placer operation will usually hinge on throughput, a high throughput being the key to low costs. In other words, the greater the throughput on a daily basis the lower the unit cost. Experience has shown that to achieve the optimum working rate some recovery must be sacrificed, or put simply, it is cheaper to lose a certain amount of gold than to save it. Operating economies made possible by the large capacity and simplicity of conventional riffled sluices and placer jigs more than offset the dollar value of the gold they may lose. On this basis, they yield the greatest operating profit.

The point to be made is that when assessing the prospective value of a gold placer, if the evaluation is to be reliable, the examiner should consider only the recoverable value that might be obtained by use of proven methods.

## DESERT PLACERS (DRY PLACERS)

Desert placers in the Southwest occur under widely varying conditions, but taken as a whole they are so different from normal stream placers they deserve special mention. When dealing with the usual desert placer the field examiner must learn to disregard some of the rules of stream desposition, or at least to apply them with caution. Desert placers are found in arid regions where erosion and transportation of debris depends largely on fast-rising streams that rush down gullies and dry washes following summer cloudbursts. During intervening periods, varying amounts of sand, gravel or side-hill detritus are carried in from sides by lighter intermittent rain wash which is sufficient to move material into the washes but not to carry it further. When the next heavy rain comes, a torrential flow may sweep up all of the accumulated detrital fill or only part of it, depending on intensity and duration of the storm and depth of fill. The intermittent flows provide little opportunity for effective sorting of the gravels and under such conditions the movement and concentration of placer gold will be extremely erratic. Where the entire bedload is not moved, any gold concentration resulting from a sudden water flow will be found at the bottom of the temporary channel existing at that time. This may well be above bedrock.

Desert miners have learned from experience that gold enrichments are sometimes found resting on caliche layers, particularly those near the ground surface. Such surface or near-surface concentrations are commonly small residual-type accumulations of gold left behind where lighter material has been removed by rain wash and wind action. In other words, such enrichments result from the removal of valueless material rather than from the concentration of gold by normal stream processes. It should be noted that in some desert placers the only minable ground was related to these superficial concentrations.

## PITFALLS

When evaluating gold placers, those involved should be aware of certain pitfalls. The most dangerous are:

1) Use of uncased or small-diameter drill holes.
2) Fire assay (or atomic absorption assay) of samples.
3) Splitting samples.

### UNCASED OR SMALL-DIAMETER DRILL HOLES

Drilling equipment routinely employed in hard-rock exploration is not suited to placer sampling and among other things it is not designed to provide short-interval sample volume measurements within the confidence level required for placer work. As noted elsewhere in this paper and shown by Figure 1, small-diameter drill holes are very sensitive to stray gold particles and to errors in measured or assumed sample volumes. Also in Figure 1, it can be seen that any sloughing of an uncased drill hole within a gold-bearing zone will tend to upgrade the calculated sample value by significant amounts. Today, any presentation in which the evidence of value is based on drilling should be viewed critically until the procedures used and all related factors have been objectively weighed and found adequate. Experience tells us that when sampling gold placers, small-diameter uncased drill holes give unreliable results. Most often the ground will be overvalued.

### FIRE ASSAYING PLACER SAMPLES

Fire assaying in essence is a minature smelting process which reports the total gold content of the

assay sample. Because of this, a fire assay is likely to report gold values that cannot be extracted by placer methods and for this reason placer samples or sample products should not be assayed. A separate reason is the fact that no amount of mixing or careful division of a gold placer sample can produce an assay charge[2] representative of the bank-run material. The practice of first concentrating a sample and then assaying the resultant black sand product does not entirely resolve this problem. Today, the atomic absorption assay procedure has largely replaced fire assaying in routine analytical work. It should be noted that use of atomic absorption procedures will create the same or worse evaluation problems so far as determining the recoverable gold content of placer material is concerned. In brief, values determined by either fire assay or atomic absorption assay, when applied to placers, will usually result in substantial over-valuation of the ground.

### SPLITTING SAMPLES

When dealing with lode deposits it is customary to reduce each sample from its original bulk to a small-volume pulp prior to assaying. This generally is done by progressive crushing and splitting. The amount finally taken for assay or other analysis may weigh only a few grams. But when dealing with gold placers the entire sample is used for analysis. Typically the sample washed to produce a heavy mineral concentrate from which the gold is extracted for weighing. In most placers gold distribution is so uneven, any attempt to proportionately divide a sample by taking alternate shovels, mechanical splitting or by other means is likely to yield erratic results. As a rule gold placer samples should not be split or otherwise divided before washing.

## MINERALS OTHER THAN GOLD

It should be mentioned that placers valuable for minerals other than gold are generally easier to sample and evaluate than gold placers. Minerals such as rutile, zircon and ilmenite have a low unit value and also they make up a relatively large part of the deposit. In such placers, extraneous particles of low-value mineral entering a sample have a relatively small effect on calculated values. Also the fact that such minerals are commonly associated with small grain size material such as beach sand makes it easier to obtain a representative sample.

Where monazite, zircon, ilmenite or other heavy minerals are of interest, the black sand concentrated must be processed in special ways which may include magnetic separation, high tension separation or other procedures designed to extract the mineral of interest and produce a commercial-grade product. Where the heavy mineral contains inclusions of foreign matter, separation into a saleable-grade product may not be feasible. Standard assay or analytical procedures may not reveal such conditions and for this reason the valuation of heavy mineral deposits or concentrates should not be based on chemical analyses alone.

## ENVIRONMENTAL FACTORS

In addition to the usual problems and uncertainties that are part of any mining venture, today's placer miner must cope with a new and often perplexing problem. This is the ability of regulatory agencies to in effect, grant or withold permission to mine. Their discretionary power has for the most part been brought about by environmental legislation generated in response to the ecology movement of the 1970's. During a ten-year period beginning in 1969, more laws aimed at protecting the environment were passed than in the preceding two centuries. Most are adverse to mining generally and to surface mining in particular. Commercial-scale placer mining falls within the purview of most surface protection legislation. In addition to causing surface disturbance, placer mining normally generates large quantities of muddy water which must be contained or treated to meet stringent point source effluent standards imposed by state or federal law. The record shows that environmental factors, particularly in populated areas can outweigh the mineral value of an otherwise economic placer deposit.

In today's world an environmental assessment of any proposed sampling or mining activity should be made early in a placer investigation. If adverse environmental factors preclude mining, the question of sample values becomes moot.

## POINTS TO REMEMBER

Where the reappraisal of a long dormant placer property or prospect begins with reconnaissance-type sampling, the interpretation of sample results will require a large measure of common sense. The number and size of samples needed for an adequate investigation must be determined by judgement after considering the type of deposit, available exposures and purpose of the investigation. Although they do not determine absolute volume or unit value, the initial samples if properly selected, should indicate a range of value which coupled with experience, can provide a basis for estimating the possible size and tenor of the deposit.

The person planning a placer investigation should be aware that compared to lode deposits, gold placers are difficult to sample and evaluate. Also it should be recognized that drilling equipment and assaying procedures routinely used for hard-rock exploration can cause serious error when applied to placers. In particular, placer samples obtained from small-diameter uncased drill holes are likely to give spurious results and experience tells us they usually lead to an over-valuation of the ground. Today, any placer presentation having evidence of value based on drilling should be viewed critically until the procedures used and related matters have been looked into and found adequate.

If fire assay or atomic absorption procedures are in any way used in a gold placer evaluation, the results should be rejected.

Equipment for the recovery of placer gold has changed very little over the years. Most devices rely on the ability of heavy minerals to resist the action of moving water while the lighter materials are carried away. Although many have tried, no one has devised a gold saving device or system that can replace the ordinary riffled sluice or the placer jigs used on today's dredges or comparable large-scale placer operations. It is true that conventional recovery equipment may lose some of the fine-size gold but this is normally offset by its high unit capacity and low operating costs which combine to return the greatest dollar profit. With no exceptions—a gold placer evaluation should include only values that are recoverable by the use of standard recovery methods.

The price of gold will not be the deciding factor in reopening long dormant placer mines in the Western States. Overriding problems are found in today's environmental laws and regulations, particularly those pertaining to water quality. For example, if a commercial-scale placer miner cannot impound his muddy water or otherwise comply with effluent limitations established by state or federal law, the property has no value as a mine. As a practical matter—an assessment of environmental factors should be the first step in today's placer investigations.

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## FOOTNOTES

1. Ground having a gold-gravel ratio of 1:100,000,000 (by volume) is worth approximately $3.00 per cubic yard (gold @ $660/oz.)
2. The amount of sample used in a fire assay crucible charge may be 29.166 grams or more commonly, 14.583 grams. These amounts represent 1 assay ton and ½ assay ton respectively. Atomic absorption techniques for routine gold analyses utilize 10 grams of sample material.

# APPENDIX

## WHAT IS AN ADEQUATE PLACER INVESTIGATION

An adequate placer investigation will develop a considerable body of information in addition to the obtainable from samples alone, some of which will be found vital when assessing the actual worth of a placer property or prospect. For example:

Sometimes most of the value in a commercial placer is found on or in the bedrock and perhaps several feet must be taken up to recover the pay. In such cases its hardness and irregularities must be known and failure to consider this has proved fatal to more than one mining project.

Because boulders can be disastrous to a placer operation their maximum size, number and distribution should always be of prime concern during the investigative process.

Failure to recognize or properly assess tailings or muddy water disposal problems can prove expensive. This problem, by itself, can preclude working an otherwise profitable mine.

In brief, physical details often determine the success or failure of a placer as much or more than the mineral content itself. The importance of considering *all* factors and their possible effect on a mining operation cannot be overemphasized.

Because no two deposits are alike, no rule can be made as to what should be included in a placer investigation. Also, the degree of inquiry will vary depending on the purpose of the investigation. Where it is clearly evident that a property has no value or prospective value a detailed field investigation may not be needed, but here, sufficient data for a well-informed report should be gathered.

The following check list is intended first as a field guide and second, to show the range of inquiry which may be necessary for an adequate placer investigation. The user should, of course, tailor this list to suit a particular need.

### FIELD GUIDE AND CHECK LIST FOR PLACER INVESTIGATIONS

1. Date of examination _____
2. Name of claim(s) or property _____
3. State _____, county _____ District _____
4. Township _____, range _____, section(s) _____
5. Reason for examination _____
6. Examined by _____
7. Assisted by _____
8. Others present _____
9. Number of claims or acres _____
10. Names of locators and present owner _____
11. Owner's address _____
12. Type of deposit (stream, bench, desert, etc.) _____
13. Terrain _____
14. Gradient of deposit: less than 5% ( ), more than 5% ( ), remarks: _____
15. Is the deposit dissected by deep washes or old workings? Yes ( ), no ( ), remarks: _____
16. Type of extent of overburden _____
17. Depth to permanent water table _____
18. Depth to bedrock _____
19. Kind of bedrock (rock type) _____
20. Hardness of bedrock _____
21. Bedrock slope or contour to be expected _____
22. Are high bedrock pinnacles or reefs in evidence? Yes ( ), no ( ), remarks: _____
23. Gravel is: well-rounded ( ), sub-rounded or sub-angular ( ), angular ( ), remarks: _____
24. Does gravel contain rocks over 10-inch ring size? Yes ( ), no ( ), remarks: _____
25. Boulders (max. size, number, distribution, etc.) _____
26. Rock types noted in gravel _____
27. Predominent rock type (if any) _____
28. Sand (kind, amount, distribution, etc.) _____
29. Sorting or bedding patterns (if apparent) _____
30. Sticky clay? Yes ( ), no ( ), remarks: _____
31. Cemented gravel? Yes ( ), no ( ), remarks: _____
32. Caliche? Yes ( ), no ( ), remarks: _____
33. Permafrost? Yes ( ), no ( ), remarks: _____
34. Buried timber? Yes ( ), no ( ), remarks: _____
35. Hard or abrasive digging conditions? Yes ( ), no ( ), remarks: _____
36. Character of gold: coarse ( ), flaky ( ), fine ( ), rough ( ), shotty ( ), smooth ( ), bright ( ), stained or coated ( ), remarks: _____
37. Can good recovery be expected by use of riffles or jigs? Yes ( ), no ( ), remarks: _____
38. Is recovery said to depend on secret process or special equipment? Yes ( ), no ( ), remarks: _____
39. Are black sands said to contain locked gold values? Yes ( ), no ( ), remarks: _____
40. Have black sands been checked for valuable minerals other than gold? Yes ( ), no ( ), remarks: _____
41. Distribution of values in deposit (if known) _____
42. Record or evidence of previous sampling _____
43. Results of prior sampling (if known) _____
44. Are old workings in evidence? Yes ( ), no ( ), remarks: _____
45. Past production (if known) _____
46. Date of last production or work _____
47. Reason for quitting _____
48. Present work (if any) _____
49. Applicable mining method _____
50. Possible cost to bring property into production _____
51. Possible mining cost _____
52. Dimensions of (physically) minable ground _____
53. Possible extensions _____
54. Maximum yardage indicated to date _____
55. Mining equipment on ground _____
56. Accessory equipment or improvements on ground _____
57. Water supply _____
58. Power supply _____
59. Does property have adequate tailings dump room? Yes ( ), no ( ), remarks: _____
60. Would mining in this area come under county, state or federal water quality control regulations? Yes ( ), no ( ), remarks: _____
61. Fish and game regulations? Yes ( ), no ( ), remarks: _____

62. Can settling ponds be built to effectively retain or clarify the muddy water? Yes ( ), no ( ), remarks: _____
63. Is property subject to resoiling or other surface restoration regulations? Yes ( ), no ( ), remarks: _____
64. Elevation of property _____
65. Climate _____
66. Working season _____
67. Season governed by _____
68. Surface cover and its effect on mining _____
69. Merchantable timber or other surface values _____
70. Nearest town _____
71. Access _____
72. Reference maps _____
73. Aerial photos (USGS, Forest Service, etc.) _____
74. Reference literature _____
75. Previous examinations or reports _____
76. Other reference sources _____
77. Sampling (describe or attach notes) _____
78. Additional information and remarks _____
79. Attach suitable map or sketches (if needed) _____
80. Attach photographs of pertinent features (if available).

# EVALUATION OF TERTIARY GRAVELS IN THE SIERRA NEVADA

**JAMES J. HODOS** Placer Service Corp., 12050 Charles Drive, Suite A, Grass Valley, California 95945

The middle of the last century saw the Sierra Nevadas as the scene of a fantastic gold rush. Fortunes were made (and lost) overnight as little more than a shovel, a pan, and a stream were needed to start mining. As miners flooded the area, the more easily recovered gold was taken from the streams and methods requiring more sophistication and capital were introduced. Mining techniques such as ground sluicing and hydraulicking were applied to the immense gravel deposits discovered by the ever-swarming miners. In less than 30 years, virtually every exposed gravel deposit in the Sierras had been located, tested, and usually mined to some extent. The industry of these late nineteenth century miners was such that over 5,000 miles of water-bearing flumes and ditches were constructed to supply the enormously consumptive habits of this type of mining. Mine tailings were discharged into the streams and eventually they found their way to the valleys below, oftentimes burying some of the richest farmland in the world.

As the surface deposits were exploited, drifting was employed to follow the course of the Tertiary rivers under the rhyolitic and andesitic lava caps. Finally, in the 1880s, the first environmental restrictions (to my knowledge) in the west—and the nation—were ordered. The decision of Judge Sawyer in 1884 dramatically curtailed mining in the Sierra Nevada Tertiary gravels as it prohibited discharge of mining debris into the rivers and streams of California. Mining town populations dwindled, bars closed, and production came mostly from underground drift mines and self-contained surface mines. Tailings disposal had become a very serious problem to gravel miners. As subsequent costs rose and the commodity price remained static, for all practical purposes, mining gravel in the Sierra Nevadas had severely slowed by the early 1900s. By the 1930s, only token production from selected mines (mostly drift mines) remained. Dredging in the Quaternary alluvium was then in its heyday and not much interest remained in the high cost, hard-to-develop gravels in the mountains and foothills in eastern California.

Today the price of gold is approximately 20 times more than what it was in 1934, and costs have also gone up significantly; but with more efficient mining methods, the remaining Tertiary deposits are once again the subject of great interest. So here we are in the Sierra Nevadas of California, an area obviously extensively mined and one with, we believe, a promise of significant mineable reserves.

The question now facing us is, "How do we find and properly evaluate a Tertiary gravel deposit?"

First, we must try to locate the deposit in question. Our first clue may be found by looking around today's streams. Although the stream may be full of suction dredgers, swimmers, and white-water enthusiasts, we may also be able to view some exposures of Tertiary gravels. There are some old workings at or near current river levels that may give us a base point for our search.

Old hydraulic workings in general are reasonable places to look, as the hydraulic pits often stopped where the gravel dips under a lava cap. Attempts may have been made to follow the channel by underground methods; while some were very successful, many such attempts stopped far short of exhausting the deposit.

Since the rather dynamic life of the typical Tertiary river was rudely interrupted by a sequence of block uplift and westward rotation, volcanic eruptions, and various periods of low to moderate displacement faulting, the channels do not always occur where you think they should. Even such respected experts as Lindgren, Turner, and Whitney were not always in accord about the downstream whereabouts of a particular stretch of channel.

For example, the rivers may have been disrupted by burial under volcanic flow, uplift by the Sierran orogeny, and subsequent erosion by younger rivers. It is not at all unusual to trace a channel along a ridge only to have it vanish at the edge of a canyon, 1000 feet above the bottom, and not be able to see it anywhere on the other side. Nor will it appear in the canyon wall upstream or downstream, as later rivers may have cut their channels directly over a Tertiary channel obliterating it in the process.

Surface expressions we find of merit include the previously mentioned old workings, exposed gravel, or a volcanic cap. Some volcanic caps show the results of differential weathering, and current topography may bear little resemblance to the pre-volcanic erosional surface.

The deposits themselves have a wide range of possible characteristics.

In general, the lowest lying Tertiary gravels are darker, coarser, and richer than overlying gravels. They usually contain more metamorphic and intrusive rocks with dark green, blue, or black clays and coarse sands. Boulders up to three feet or more in diameter may be found. Quite often, this layer has been subject to post-depositional cementation.

The richer pay streaks usually occur in the bottom of the channel or on a false bedrock. False bedrock has been described by Jenkins as a " . . . superficial layer of clay, volcanic tuff, or some such impervious material overlying previously deposited gravel . . . ." Some pay streaks on false bedrock are actually richer than those of the Tertiary gravels lying below them.

Overlying these lower Tertiary deposits are substantial layers of generally more quartzose finer gravels that may contain large lenses of light-colored sands and clays. These lighter colored gravels may also be extensively iron-stained and occasionally

have a limonitic cement. White volcanic clays, locally known as "pipe clays," are common in large lenses. In my experience, the deposits are usually water-laden, but the permeabilities are quite varied. Their gold content is also much lower.

As the volcanic flows filled the valleys and covered the dividing hills, much interaction took place. In some areas the upper gravels have been mixed with the flows, resulting in mudflows over several hundreds of feet thick.

Successive periods of erosion and lava flow have deposited interbedded volcanic gravels and laminar flow volcanics. Andesites up to 2000 feet thick can be underlain by a few hundred feet of rhyolite.

The intervolcanic channels and mudflows are characterized by more extrusive cobbles and often a matrix of volcanic cement. They are usually quite poor in gold content, with some (very) notable exceptions.

Occasionally, long periods of erosion may have recut whole sections of the gravels, reconcentrating them and introducing later materials into the lower sections. While this process can remove the original Tertiary deposit in its entirety, the resulting enrichment can be quite remarkable.

Glaciation has also occurred in some areas, but is generally confined to the more eastern sections.

As we might expect, a single deposit can contain almost any combination of the above.

Since most often we find ourselves standing on a surface of lava or a gravel bed, how can we find out what is below our feet? Before we go any further on the ground without contacting the landowner, a good deal of background work should be considered necessary. In spite of an abundance of mines in the Sierra's gravels, there is not an abundance of definitive literature on these mines. Pinpointing a prospect—other than an outright submittal—is not simply a matter of plotting all the mines on a map and going after what is left. While a comprehensive literature search is essential, it often leaves us with many conflicts regarding a particular area. As most of the literature is descriptive rather than analytical, it is of limited use in providing the information necessary to resolve these conflicts. Field reviews, mapping, and photo interpretation are necessary complements to this literature examination. Combine them all and add a little imagination and a lot of legwork and we may be able to come up with a property or two suitable for further investigation.

About now we should want to know what the ownership of this suitable property is. If the land is government held, the procedure has been well explained elsewhere and I will not discuss it here. Should the land be patented or privately held, then we have some talking and/or negotiating to do.

Let's say that we are interested in a particular piece of (we hope) potentially rich Tertiary gravel and the private owner is wiling to let us have a closer look. No particular legal problems exist on our theoretical property, so we can begin.

We want to be able to determine the sequence and composition of the various layers that comprise our deposit. The distribution and nature of the gold is our prime concern. It will also be to our benefit to know the bedrock configuration as well as the location and extent of the various depositional sequences and buried items, such as boulder swarms, clay lenses, perched aquifers, cemented layers, and log jams, among other things.

Our approach must be as comprehensive and as cost-effective as we can make it.

We may then choose from a rather limited array of tools to do the examining:

In attempting to define subsurface conditions, geophysical techniques have been, in my experience, limited in application. Seismic refraction studies have uses in determining velocity layering of the deposits for such things as rippability studies, blasting configurations, and so on, but refraction must be carefully applied as the cost of the additional drilling required to calibrate the equipment and improve resolution to a usable degree may come quite close to the cost of obtaining the desired information by just drilling the holes. Seismic reflection for bedrock configuration investigations shows some promise as the drill holes required do not appear to be so numerous. A new-to-me technique is subsurface radar which is said to have a high resolution of subsurface conditions, but capability only to 80-100 feet below the surface. EM has been applied to the Tertiary gravels but results that I have seen have yet to be substantiated.

A tool that I have found most useful is a good quality contour map, preferably on a scale such as 1" = 200' with 5 foot contour intervals. These are not inexpensive to have made, but should be very seriously considered as early as possible in your investigation.

Sampling the property is our next step and is an extremely critical one. We may choose to pan the surface materials—although this only confirms the presence of gold at the surface and not the grade of the underlying deposit. As it is generally done with a shovel and a stream, we are not really getting much information about the lower gravels. Also, if the surface is covered with a volcanic cap, sampling problems are intensified.

The same holds true for trenching or pitting, as a volcanic cap of any but the least thickness will preclude these methods. Properly sampled trenches and pits can give reliable results, but these methods are limited in depth. Generally, pits and trenches can be dug to water level and safely sampled. Below the water level, elaborate methods of support are required. While the floor of a pit may be accurately sampled below water level, it requires caissons, pumps, and other devices. The face of a backhoe- or cat-dug trench is generally sampled by a wall cut. Sampling done here below water level is extremely susceptible to "salting" by inflowing water and should not be considered valid. The amount of land disturbed by these methods should also be considered. In many areas of the California Sierra, you may not leave your sample pits or trenches open for more than a few days at most. This requires you to

be doubly sure that you have accurately taken a sample as you will not be able to return to that trench should you mishandle your sample. The U.S. Forest Service, as well as most landowners, requires you to reclaim your testing area and a small backhoe-dug trench or pit requires less work to reclaim than does a bulldozed trench. A rubber-tired backhow can generally dig a trench more easily, more quickly, and more cheaply than a bulldozer and its maneuverability makes it a very serviceable choice as a sampling tool. Backhoes of this type are generally limited to less than a 20-foot digging depth.

But what if our deposit is estimated to be greater than 20 feet thick? Actually, the Tertiary gravel deposits can be up to 100 feet thick and be buried under a few hundred feet of later gravels. Some deposits have over 500 feet of gravels between surface and bedrock. If you add a few hundred feet of volcanic capping, the distance from surface to bedrock may exceed 750 feet in extreme cases.

At this point we are forced to drill. The "tried and true" method of placer drilling is the churn or cable tool drill. This technique is slow but reliable. Given a trained sampling crew—not a water well driller—samples can be taken to great depths. A churn drill can drill through water saturated formations as well as drier varieties. However, boulders and tight or cemented ground will take its toll in time and material.

The churn drill is not well suited to drilling through a tough overlying lava cap, so other methods must be used to open a hole down to the gravels. Air or mud rotary drills, down-the-hole hammers, and other percussion rigs have all been used here. Particular application of a certain machine is a decision best based on local availability, formations present, and the driller's experience. In this instance, we are looking for the cheapest, fastest way to get a true, serviceable hole through the cap.

Once we are through the cap, the churn drill replaces the other rig and we can begin our sampling.

Good practice is to lay out drill lines approximately perpendicular to the channel length on 1000 foot to 3000 foot intervals for long, narrow channels. Hole spacing is 100 feet to 200 feet across the width of the deposit. Broader deposits may receive closer line spacing and slightly wider hole spacing. Should the deposit prove interesting, we may tighten the grid with relative ease.

Recently we have been using a rig that, when properly applied, we believe is superior in many ways to the churn drill. This rig is known as a hydraulic casing oscillator. It permits us to drill larger diameter holes (36 inches in diameter) which yield a very reliable sample. The machine can be supplied in diameters of approximately one-half meter to over two meters. The casing oscillator requires quite a bit more equipment and manpower than a churn drill. We have been using this machine for over a year on one property and the results have been technically very satisfactory.

If our sampling/drilling indicates a grade that may be economic, we then have to gather some other information. A very important part of the property evaluation is what I call the "nondeposit considerations."

This group consists of elements necessary to evaluate the property as a potential mine, such as:

1. Land ownership and surrounding land ownership: It can make a great difference to your prospect if the property is controlled by one person or by many individuals. The surrounding landowners—upstream and downstream, as well as those owners not on the channel—can also aid or hinder your eventual operations.

2. Land usage designation: The best property in the Sierras may not be mineable if it has been subdivided and designated as urban or residential. Or your prospect may be inside the next state park. Preferable land use classifications are grazing lands, timber production in certain locales, and unclassified general forest.

3. Available labor force: Very few people experienced in placer sampling and mining are around today. You must train your own drillers and panners on the spot or import them. As mentioned earlier, a water well driller has been trained in the methods of making a hole quickly and not in terms of careful sampling. Should you want to use a well driller and his churn rig, some of his tools—such as the dart valve bailer—should be replaced with those more suitable for placer sampling. If your property is deep, you may have to mine it by underground methods. Providing underground miners today is a task that usually involves training local labor from scratch.

4. Environmental considerations: In California today, there is a plethora of regulations with which you must contend. Of these, the dominant law is the California Environmental Quality Act, known as "CEQA." Another strong group with which to be reckoned is the State Water Quality Control Board. These and other administrative groups and statutes have the final say on what your mine (and sampling project occasionally) may or may not do. For example, no contaminated water may be discharged—period! Your processing water must be impounded and treated before being released. Streams flowing across your property must remain as clean as they were before you stepped on site. Nor may you interfere with the rights of a downstream user in many cases. Other topics of note in CEQA include biotics, noise, air quality, and archaeology.

5. Do not forget our good friends from MSHA. Their requirements add to the cost of the project and must be properly evaluated. Even exploration may be subject to MSHA standards if you are exploring in a previously mined pit.

6. Should you pursue your gravel deposit to the mining stage, you must also prepare a reclamation plan. The California Surface Mining and Reclamation Act (SMARA) is another powerful document. It requires you to have an accepted reclamation plan for the project or your project—no matter how worthy—will be disapproved. Not all counties in California enforce CEQA and SMARA with the same strength, but they are all becoming increasingly aware of how the environmental laws apply to your property. An underground mine will draw most attention to its surface effects and to the hydrological balance. A surface mine will be very closely scrutinized at every step for any real or potential environmental damage. A surface operation may also be objectionable to some people in the local community and require special methods to alleviate their concerns. This is where an attractive, well-considered reclamation goal can be useful. *We* know that mining is an interim use of the land. When we are gone and the mine is finished, the local residents remain. They may not raise as many objections if they know that they will have an attractively reclaimed site next door when all is done.

It is obvious that what was once a rather straightforward sampling program to evaluate a property has now evolved into a rather complex process that may hinder all but the best properties' chances of success. However, as we become more familiar with the laws and as we become more efficient evaluators *and miners,* we can become capable of successfully operating a mine in the environmentally tough woods of the Sierra Nevadas in California.

Some of these remarks made in this talk are based on my own observations and may not be universally accepted.

Thank you for your attention. Good prospecting!

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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

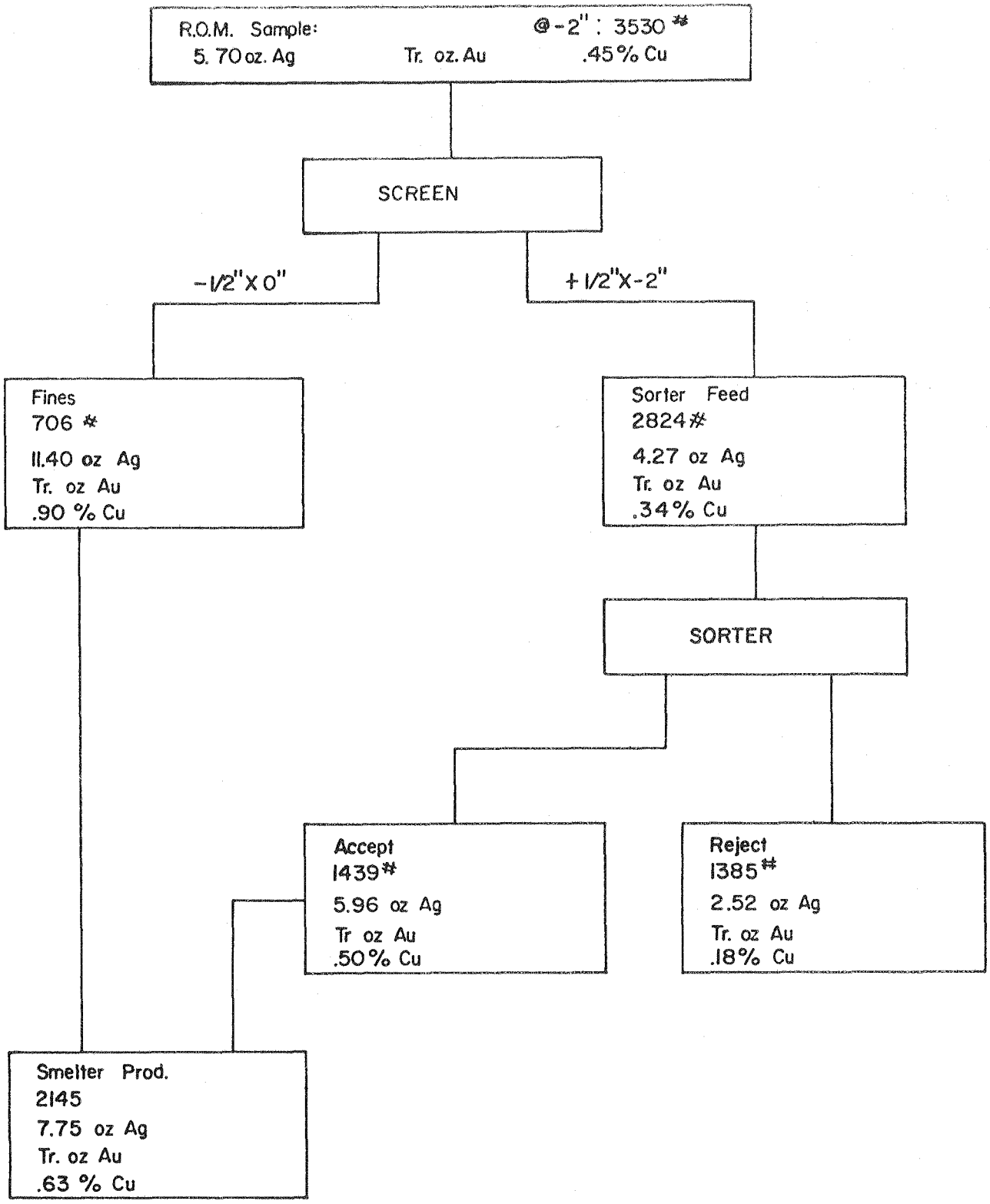


FIGURE 1. Schematic first test, ore sorters assay bulk test.

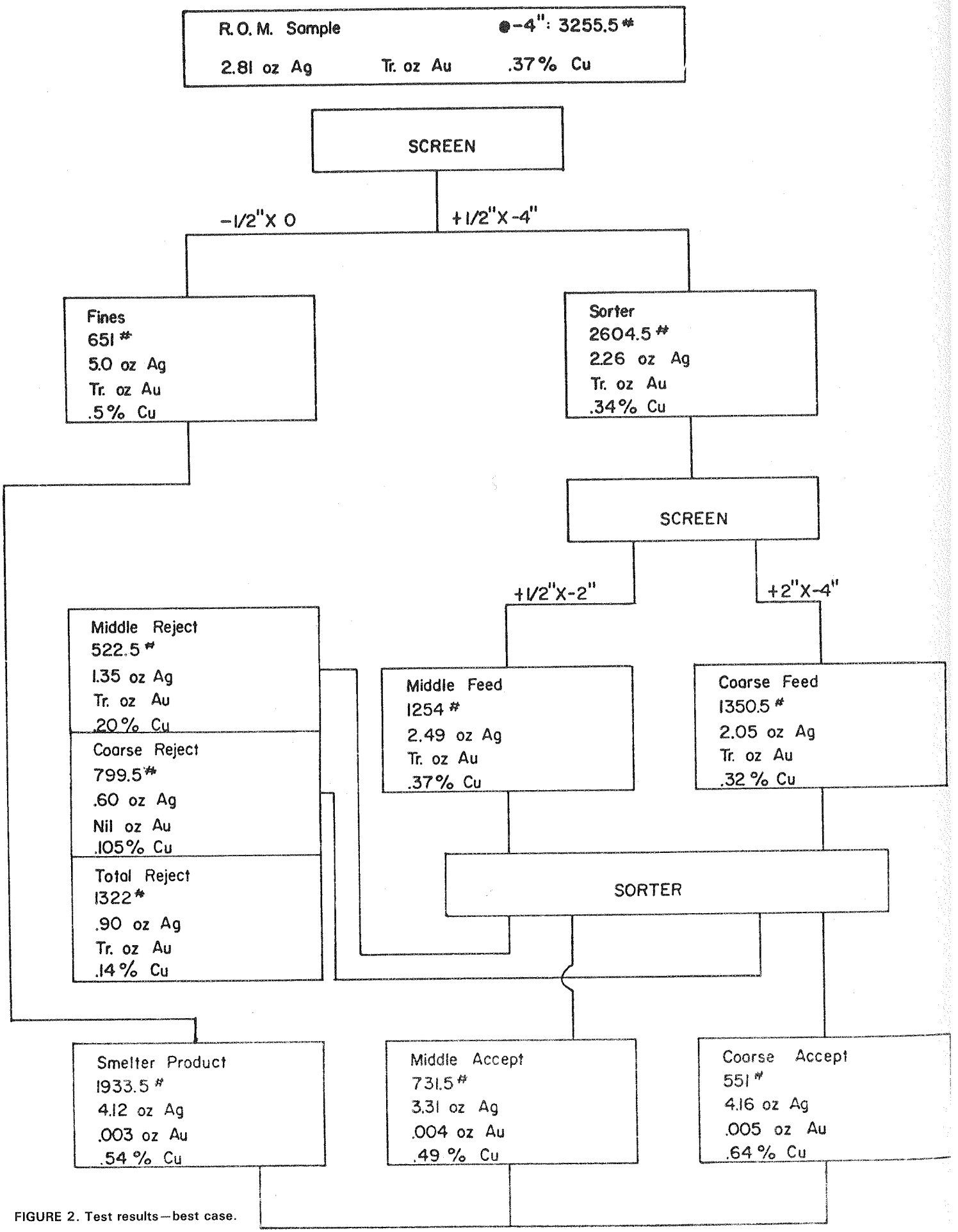


FIGURE 2. Test results—best case.

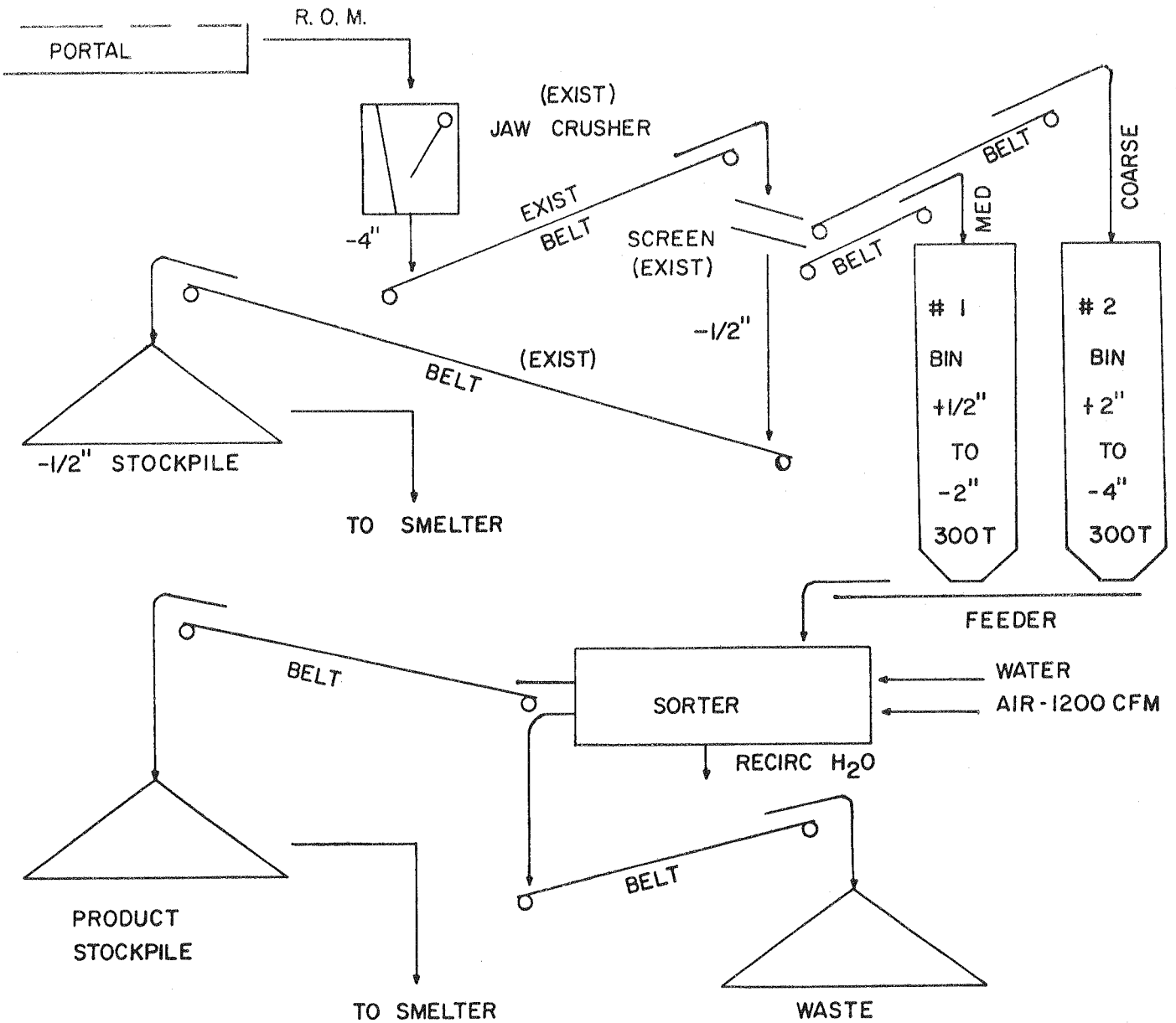


FIGURE 3. Schematic of sorter plant.

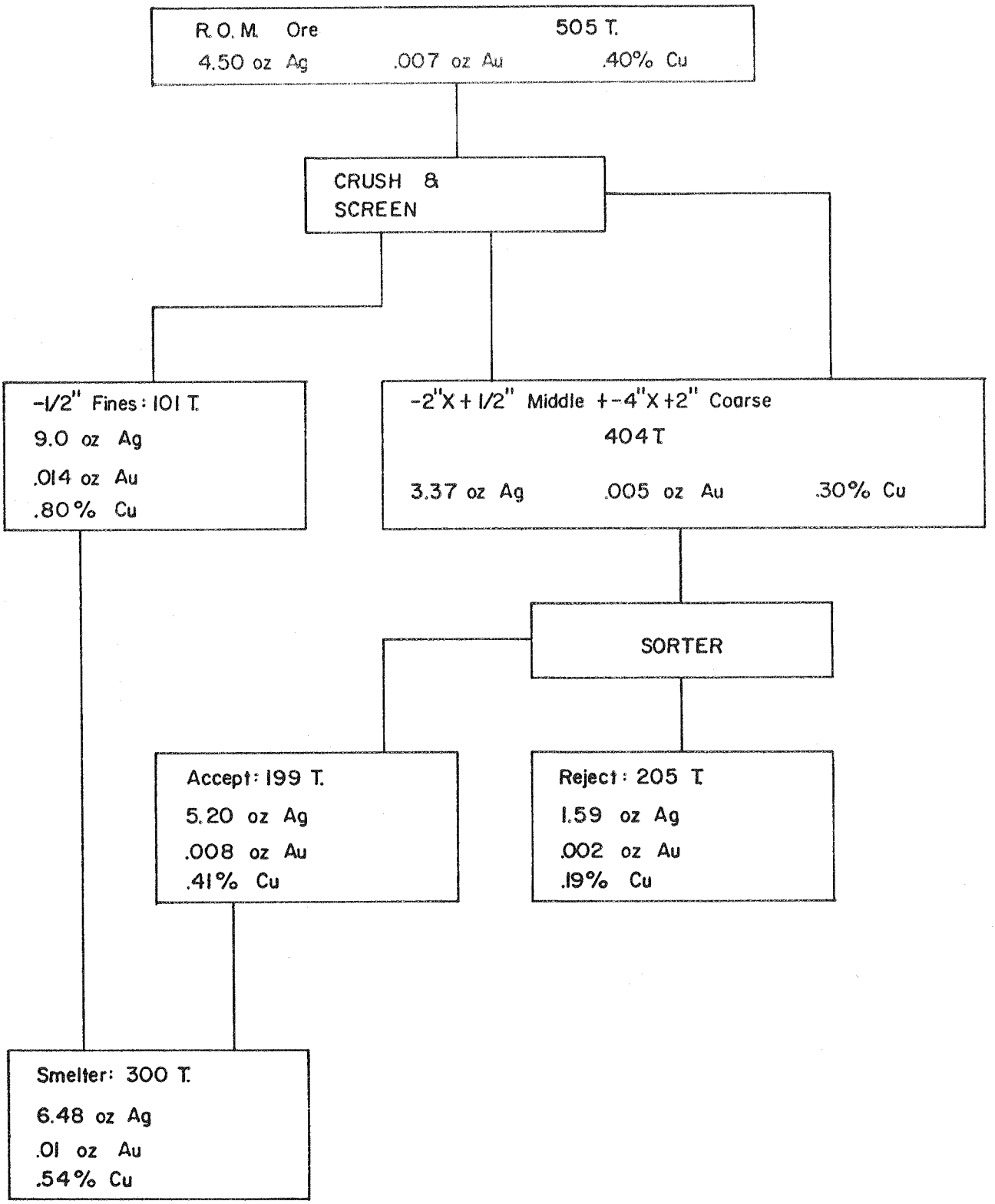


FIGURE 4. Projected production.

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."

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# APPLICATION OF THE LAMELLA™ GRAVITY SETTLER/THICKENER IN PRECIOUS METAL RECOVERY PLANTS

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## INTRODUCTION

The Lamella™ Gravity Settler is an inclined, shallow-depth, solids/liquid separation device. It performs the same function as a conventional, raked bottom thickener or settling basin, but it occupies only a fraction of the space. Thickening area is provided by parallel, inclined plates. The close spacing of these plates yields a compact thickener which requires much less plant area than conventional thickening basins.

The Lamella™ Gravity Settler/Thickener (LGST) combines the plate thickening section of the standard LGS unit with a raked bottom to provide optimum sludge concentration. Included in the sludge tank section is a picket fence type thickener which optimizes sludge concentration.

To date, over 600 such LGS and LGST units have been sold for various applications. They have been successfully applied to such varied applications as metal hydroxide removal, steel mill clarification, surface and potable water clarification, and mineral thickening.

This paper will present basic theory for inclined plate, solids/liquid separation and the design features for the Lamella™ Gravity Settler/Thickener. Preliminary sizing tests for a gold cyanidation plant will be discussed. Finally, full-scale experience in a similar plant will be reviewed with operating results included.

## THEORY

As stated above, the LGST provides the necessary area for clarification and thickening by utilizing a series of inclined plates. For an initial understanding of the LGST theory, we will limit ourselves to articles which exhibit "free settling". This phenomenon occurs when the concentration of particles is low enough that the individual particles or flocs settle independently of one another and follow Stokes' Law. Examples of free settling applications would be mine water clarification and pregnant liquor final clarification before precipitation. At higher concentrations, the settling particles interfere with one another and "hindered settling" is encountered. It is characterized by a clearly defined interface between the suspension and the clarified liquor. Examples of hindered settling are typically primary thickening and countercurrent decantation.

The basic equations for sizing settling basins were formulated over seventy years ago. Consider for the moment an ideal settling basin (Figure 1). The thickener feed enters at one end of the basin, flows uniformly along its length at velocity $V_L$, and exits at the other end. The particles settle towards the bottom at velocity $V_S$. The trajectory of the particles is indicated by the vector $V_p$. If this trajectory takes the particles to the bottom of the basin before they reach the far end, it is assumed that they are removed from the liquid. Therefore, a particle starting at the top must settle through the distance H at velocity $V_S$ in the same time (or less) than the liquid is in the basin. Thus,

$$\frac{H\ (ft)}{V_S\ (ft/min)} = \frac{L \times W \times H\ (ft^3)}{Q\ (f^3/min)}$$

Simplifying,

$$\frac{Q}{L \times W} = \frac{Q\ (ft^3/min}{A\ (ft^2)} = V_S\ (ft/min)$$

where A is the settling area of the basin and Q/A is known as the "overflow rate" or "surface loading" (and is usually expressed in GPM/ft²). From this relationship it can be seen that all particles are removed which have a settling rate equal to or greater than the overflow rate, and that the height (or detention time) of the basin is not one of the main parameters effecting the separation efficiency.

The fact that this is true can be illustrated another way. Compare Figures 1 and 2, the only difference

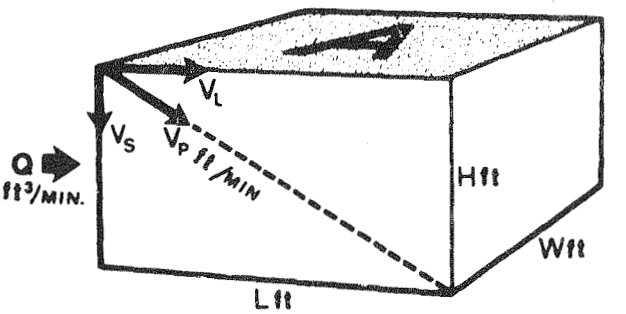


FIGURE 1.

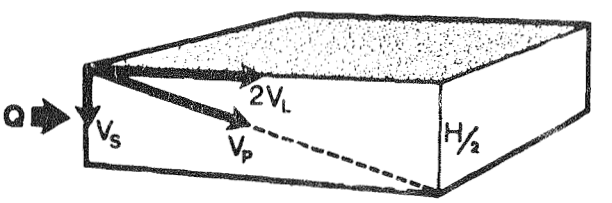


FIGURE 2.

being that the second basin is half the height of the first. At the same flow rates, the detention time is only half as much and the suspension moves through the basin at twice the velocity $V_L$. The trajectory of the particles has only half the slope, but since the basin is only half as deep, the particles are still removed.

If the height of the basin is reduced to a few inches and a number of such units are stacked on top of each other, the result is a primitive shallow depth sedimentation device. Figure 3 shows a unit containing ten parallel compartments. Theoretically it can handle ten times the flow rate as could the same basin without any plates. The liquid detention time is one-tenth as long. However, the same separation efficiency is achieved since the overflow rate is still the same (10Q/10A = Q/A). Note that the settling area now includes the area of all the plates.

The shallow depth sedimentation device shown in Figure 3 is utilized in some mineral processing applications. It is typically called a tray thickener and would have up to four or five individual trays (or plates) utilized. This would therefore give approximately a four-five fold increase in the feed rate for a given area compared to a normal thickener. The number of trays is limited because of the necessity to remove solids from the individual trays. This is usually accomplished by installing individual mechanical scrapers (with a single center shaft and drive) for each tray. Thus, the number of trays which can be applied is limited by the mechanical complexity of the scraper system and by the height of the total vessel.

A more general solution to the problem is to incline the plates so that the sludge will be self draining. Figure 4 shows such an arrangement containing ten plates set at an angle of 60° above the horizontal. The total thickening area is thus the number of plates times the area of the plates times the cosine of the angle of the plate. This is due to the fact that only the horizontal projection of each plate is utilized. In the example, the total settling area is 10 A cosine 60° = 5 A and the capacity of the unit is 5 Q.

The device shown in figure 4 is exemplified by the present-day tube settler. Tubes are used instead of plates but the basic principle is the same. Such devices are usually utilized for low suspended solids applications and are generally not applied to classical thickening applications. Such a design suffers from two serious limitations. First, there is no means provided to ensure that the flow is uniformly distributed

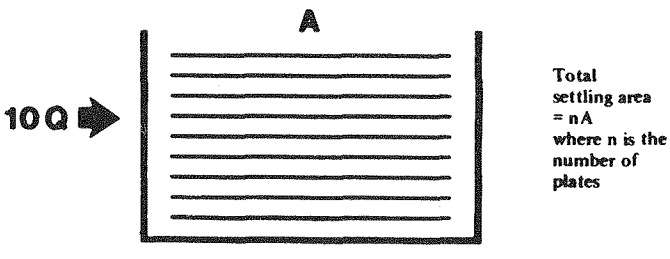


FIGURE 3.

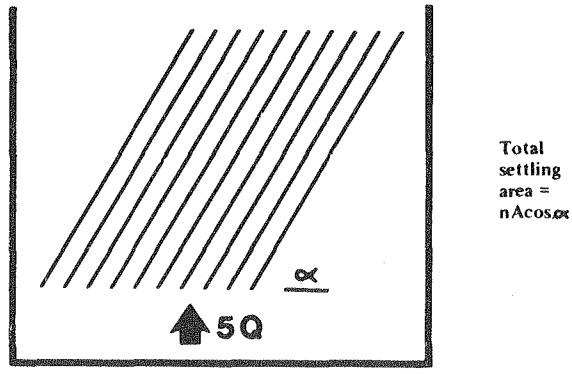


FIGURE 4.

throughout the unit. Thus, some parts of the thickener may be overloaded while other parts are underloaded. Secondly, the sludge which settles on a tube must fall through the incoming feed to reach the bottom of the thickener and be removed. As a result, some of the solids are re-entrained by the feed. Due to these limitations, the thickener is typically operated at a loading of only 25-50% of its theoretical loading rate.

The LGST design successfully overcomes these limitations. To assure uniform flow distribution to each individual plate, a small pressure drop (usually 2-4 inches of water) is taken over each plate. Effluent throttling orifices are provided to assure this pressure drop. The effluent is throttled rather than the influent so that the flocs are not destroyed. A second feature is that feed is into the side of the plates rather than at the bottom. Thus, the thickened sludge can slide off the bottoms of the plates into the quiescent sludge thickening hopper without any re-entrainment by the feed.

The throttling holes range from about ½ to 1 inch diameter, large enough to preclude plugging. Each individual plate is provided with one orifice. Thus, all of the projected area can be successfully utilized and no downgrading of surface loading rates, required for tube settlers, is necessary.

## DESIGN FEATURES

The LGST product line really encompasses two distinct products, the Lamella™ Gravity Settler (LGS) and Lamella™ Gravity Settler/Thickener (LGST). The plate section of both units is the same. The main difference is in the way that the sludge is handled within the unit. The LGS is fitted with a vibrator in the sludge hopper. Vibrations are transmitted from an external vibrator motor to a nest of plates in the hopper. The two are connected by a shaft which passes through a flexible rubber seal on a tank wall. The purpose of the vibrator is to vibrate the sludge not the tank. The vibrations, which are low frequency (60 Hz) and low amplitude (about 0.2 mm), promote thickening of this sludge and compression and produce a lower apparent viscosity which facilitates sludge removal from the hopper.

# SOLIDS FLUX CURVES

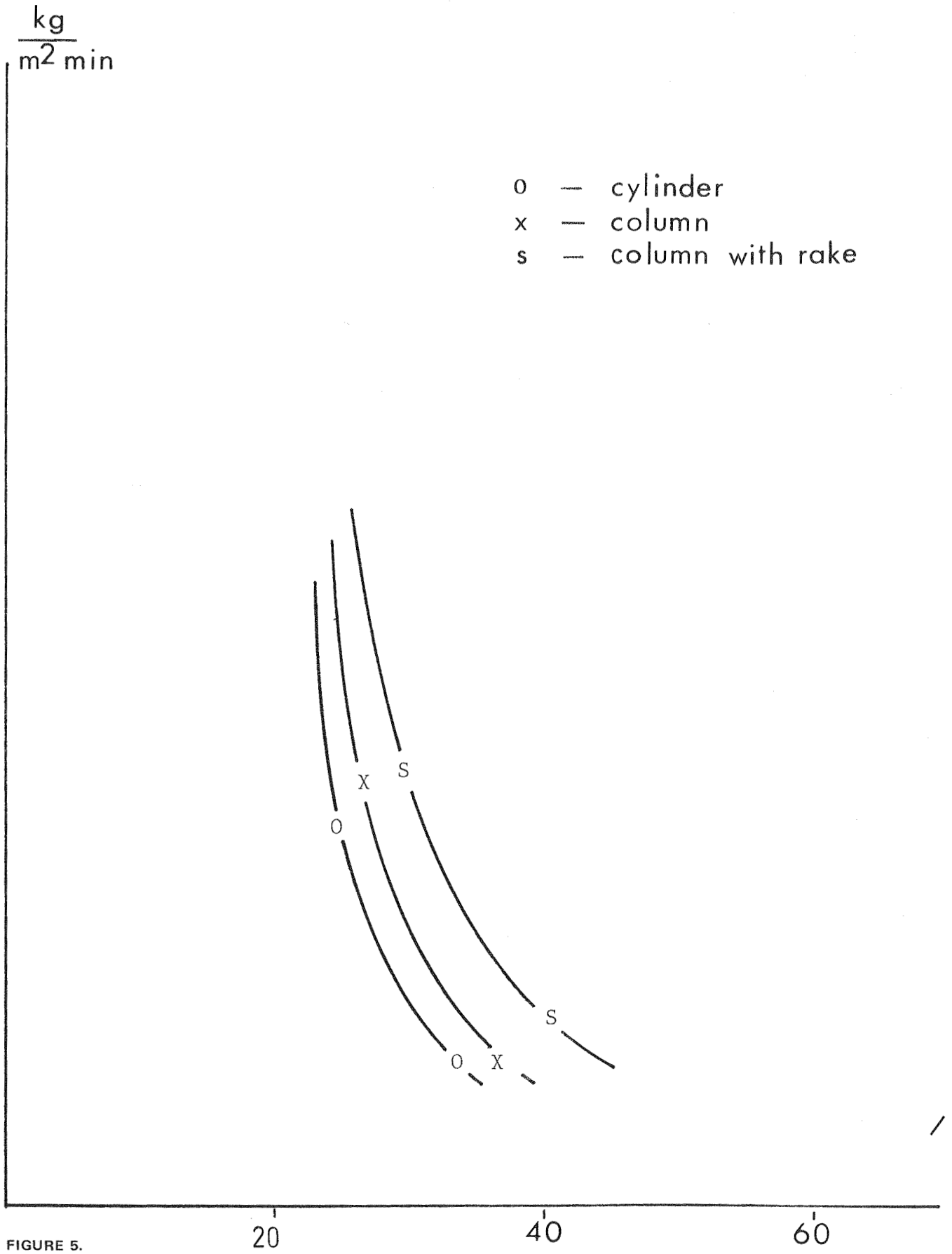


FIGURE 5.

TABLE I

| Type of Test | Solids Wt % | Loading TPD/ft² | Overflow ppm | Predicted Sludge, Wt% | Polymer Dosage, #/ton |
|---|---|---|---|---|---|
| 1. Cylinder | 23.7 | 1.84 | 53 | 32.0 | 0.29 |
| 2. Column, no rake | 22.9 | 1.69 | 37 | 33.0 | 0.18 |
| 3. Column, no rake | 22.9 | 0.84 | — | 37.8 | 0.19 |
| 4. Column, no rake | 22.9 | 0.42 | — | 45.5 | 0.17 |
| 5. Column, with rake | 21.1 | 1.52 | — | 38.0 | 0.19 |
| 6. Column, with rake | 21.1 | 0.76 | — | 51.9 | 0.20 |
| 7. Column, with rake | 21.1 | 0.38 | — | 64.0 | 0.21 |
| 8. Cylinder, with recycle | 23.7 | — | 43 | 62.0 | 0.14 |

The LGST is fitted with a cylindrical sludge storage tank with a conventional-type sludge rake. Also included is the picket fence mechanism which promotes reorientation of the flocs and helps promote thickening. Excellent sludge concentrations, equal to those produced by larger rake bottom thickeners, are produced for several reasons. First, the picket fence mechanism noted above helps promote thickening. Second, because of the orifices, not only is the liquid flow to each plate equal but also the solids flow is equal. Thus, the thickener portion receives sludge throughout its bottom area at equal loading and without solids classification. Thus, all of the volume is equally used and certain areas do not receive only the coarse fraction while others receive the slimes. Thus, utilization is equal. Finally, the height characteristics of the thickener tank allow for excellent compression characteristics. Typically, a sludge height of up to ten feet is utilized. Thus, compression characteristics are utilized to the fullest producing high sludge concentrations.

The LGST plates are typically 2-ft. wide by 10-ft. long with a spacing of 2 inches between adjacent plates. Plates are inclined at an angle of 45 or 55° depending on the application. The 10-ft. plate length is divided into two sections, one above and one below the feed point. The section above is called "clarification area" and the section below the feed point is called "thickening area". In unit sizing, testing is done to determine the amount of both areas needed and each is independently provided in the unit by proper placement of the plate inlet. Where needed, large thickening areas are provided or large clarification areas are provided.

Polyelectrolyte flash mixing and/or flocculation may be required ahead of the LGST (as with all settlers). Independent tanks sized based on retention time needs are provided with the LGST.

The unit may be constructed of various materials. Standard materials of construction are usually epoxy-painted mild steel with FRP plates. For corrosive applications, various grades of stainless steel or rubber-lined wetted surfaces can be provided.

Sizing of the LGST is provided by several manners:

1) Graduated cylinder type settling tests,

2) Cylinder type compression & thickening tests,

3) Pilot-scale testing,

4) Experience.

Laboratory testing is conducted on representative samples to determine loading rates and polyelectrolyte dosage requirements. Pilot equipment is available on a rental basis to determine actual performance in plant. Pilot equipment is sized to typically operate at 20-50 GPM to reduce scale-up factors as much as possible.

## INITIAL TESTING

In order to determine initial design parameters, samples were requested from operating installations. Some testing was conducted on a sample of cyanide leach CCD circuit pulp. Various polyelectrolytes were screened and American Cyanamid Superfloc 127, a high molecular weight, non-ionic material, was found to produce the best result. Other types of polyelectrolyte produced good flocculation but this material was the most cost-effective.

A large series of tests was conducted on one large sample. The sample which we received was sludge from an intermediate CCD thickener and contained 39.6wt% suspended solids at pH 10.6. The sample was diluted to a customer's instructions to about 25%. Tests were conducted in a 500 ml graduated cylinder. Various dosages of the Superfloc 127 were tested to determine the optimum. All settling tests exhibited hindered settling so the loading rate for clarification was determined by the bulk settling velocity. Height versus time readings were taken for the sludge and flux calculations were completed to determine the required thickening area. The results and predictions from this series of tests are shown in Table I.

As can be seen from the results in Table I, effluent quality and surface loading rates were acceptable but the sludge concentration was more dilute than predicted. This is expected when utilizing such a small sample because there is little chance for compression of the sludge. Therefore, further testing was conducted in a tall cylinder to optimize sludge thickening. A plexiglas cylinder approximately 3" in diameter by 6' tall was utilized for compression tests. That cylinder includes a picket fence type rake mechanism to detemine the effect of the pickets on sludge thickening capability. Tests were run with and without the pickets in service. As can be seen from the results in Table I, the larger sludge height allowed better compression and a better sludge concentration prediction was achieved. Of particular interest are the results utilizing the picket fence. Much

better sludge compression was exhibited in these tests versus even the other tall column tests.

From these results, the following conclusions and preliminary design were determined:

1) Because of the necessary sludge thickening capability, the raked bottom LGST should be chosen for this type of application. The LGST would allow optimum sludge concentration while providing a smaller unit size because the thickening area can be minimized.

2) Surface loading rates for clarification at optimum polyelectrolyte dosage were 0.3 GPM/sq. ft. For sludge concentrations in excess of 50wt%, a loading of 0.76 tons/day/sq. ft. was needed.

3) Polyelectrolyte dosages of 0.17-0.21 #/ton were utilized for optimum results. It was felt that this addition rate was acceptable but further testing was planned to see if this figure could be reduced.

A final series of tests was conducted utilizing clarified liquor recycled to the feed to optimize polyelectrolyte dosage rate. This type treatment has been utilized in other applications to minimize polymer dosage and it was felt that the high suspended solids level of the feed would lend itself well to this type treatment. For testing purposes, one part of overflow was added to one part of fresh feed. The lower solids level allows better dispersion into the feed with resulting better floc formation.

The tests were conducted at a loading rate of 0.6 GPM/ft$^2$ so that the clarification area needed was not effected. Column tests, as above, were conducted to determine if thickening area requirements were effected. Various polyelectrolyte dosages were tried to determine the optimum. The selected loading rate was acceptable for clarification. The area needed for thickening and the sludge predictions were uneffected by the overflow recycle testing. However, reductions in polymer dosage were proven capable. Thus, a polymer dosage of 0.11-0.14 #/ton of dry solids were deemed necessary to provide the expected results.

## FULL-SCALE EQUIPMENT EXPERIENCE & DESIGN

During the course of our work, we were presented with the opportunity to supply an LGST for a gold cyanidation circuit fines thickening application. Settling tests were conducted which showed the material to behave according to the above guidelines. The material did, however, settle at a somewhat higher surface loading rate and clarification area requirements dictated a 0.65 GPM/sq. ft. loading. For the maximum flow rate of 800 GPM, one Model 2500/45 LGST was sold. The design parameters were to produce an overflow of 50 ppm suspended solids or less and a sludge of approximately 60wt%, if pumpable. The influent contained up to 30wt% solids at pH 10.5-11.5. The liquor was at ambient temperature.

Laboratory testing had shown that a polymer dosage of 5-7 ppm (0.10-0.15 #/ton) was necessary. To adequately disperse the flocculant into the feed, a 15-second detention time flash mixing tank was included in the package. The projected area was split 50/50 between clarification and thickening. The unit was an LGST and thus employed the picket fence type, raked bottom sludge thickening tank.

Some initial operating problems were experienced because of poor polymer dispersion. This caused an excessive amount of solids in the effluent and a somewhat dilute sludge. Utilizing the overflow dilution technique mentioned above, the feed was reduced to 15-20wt% solids. At this concentration, excellent polymer dispersion was achieved with a dosage rate of 0.1 #/ton. Because more rapid settling was achieved, the equipment was then able to handle the full design flow rate. The sludge concentration exceeds 60wt% and, in fact, is withdrawn somewhat more dilute to assure that enough can be pumped by the existing pump. The equipment as supplied now meets all design and performance criteria.

**TABLE II. Commercial installation design & performance gold cyanide circuit**

| | | |
|---|---|---|
| I. | **DESIGN** | |
| | Area | : 1250 ft$^2$ clarification<br>1250 ft$^2$ thickening |
| | Type | : 2500/45 LGST |
| | Feed | : 800 gpm |
| | Solids | : ~25 wt% pH: 10.5-11.5 |
| | Overflow | : less than 50 ppm |
| | Underflow | : 50-60 wt% |
| | Loading | : 1.74 TPD/ft$^2$ |
| II. | **PERFORMANCE** | |
| | Feed | : 800 gpm |
| | Overflow | : 5-10 ppm |
| | Underflow | : 52-64 wt% |
| | * Feed | : 15-20 wt% |
| | Polymer Dosage | : 0.10-0.15 #/ton |

*after dilution with overflow recycle.

## SUMMARY

The LGST represents a new alternative to conventional raked bottom thickeners for precious metals recovery plants. Based upon prior applications in various other similar applications, testing was conducted to determine design parameters specifically for gold and silver recovery. These design parameters have been achieved in full-scale installation.

The LGST offers several design advantages over conventional-type equipment and these include:

1) Compact size of the units allows indoor installation, reducing effect of ambient conditions.

2) The small relative size of the equipment reduces the necessary amount of liquid and chemicals stored in the circuit. Also, the amount of precious metals which are contained in the circuit are reduced considerably, thus reducing the investment necessary to place the circuit in operation.

3) The LGST is fully factory-preassembled with only minor disassembly for shipment. Thus, installation costs are quite low. Combined with competitive selling prices, significant capital equipment investment savings can be realized utilizing the LGST.

4) Due to the preassembled package nature of the equipment, deliveries can be made in very short time frames. Thus, the equipment can be placed in service earlier than with conventional equipment.

5) The LGST can be relocated with little inconvenience and very little loss in site preparation. Thus, it lends itself to utilization in relatively small mines and in mines with small ore bodies. Also, the equipment may be skid-mounted for movement between several areas. Thus, it may be included in portable circuits which could work off stockpiled ore from several mines.

6) The relatively small equipment can be insulated. Even without insulation, temperature changes are small. Thus, where temperature of the solution may effect dissolution and recovery efficiencies, the circuit temperature can be easily maintained. Also, where elevated temperatures are necessary for optimum recovery, considerable savings can be realized in total plant fuel costs.

Based on testing and full-scale installations and on the above-mentioned advantages, the LGST provides an excellent alternative choice to conventional thickening. With this experience, design and equipment selection can be made for similar full-scale installations. We will be happy to combine our knowledge with your plant to provide a cost-effective system.

# METALLURGICAL PRACTICES AT HOMESTAKE, LEAD OPERATION

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## METALLURGICAL PRACTICES AT HOMESTAKE LEAD OPERATION

The Homestake Ledge or Lead was discovered by Fred and Moses Manuel on April 9, 1876. They recovered a small amount of gold from the quartz outcrop, but unfortunately they were prospectors not hardrock miners. Their claim was sold to a group of experienced California mining men in 1877. These mining men incorporated their holdings as the Homestake Mining Co. This, therefore, was the beginning of one of the greater gold mining enterprises of the western world. Metallurgical operations progressed over the years from an eighty stamp grinding circuit using amalgamation for recovery, to the more complex circuit of today.

The Homestake Mine today, is an underground mine with the deepest winze reaching the 8000 level. The ore is hoisted to the surface by the two primary shafts, the Ross Shaft and the Yates Shaft. It is crushed in three stage crushing plants located in the headframes. From here, it is trammed to the mill fine ore bins by electric locomotives.

The Lead grinding plant milling circuit, known as the South Mill, receives ore from the crushing plant at minus ½ inch. This material is dumped into overhead fine ore bins from five ton rail cars. The fine ore bins have a live capacity of about 3,700 tons. The South Mill bins can also be fed by a conveyor system which reclaims ore from an 80,000 ton capacity stockpile. This stockpile is built up from peak mining production periods and available surface low grade material. It is then reclaimed during low mine production periods.

The grinding circuit at the South Mill consists of four rod mills in open circuit followed by four ball mills in closed circuit. Three of the rod mills are 6 feet by 12 feet Marcy's. The other is a 7½ feet by 15 feet Marcy. All the ball mills are 9 feet by 11 feet Allis Chalmers.

Ore is fed to the rod mills by means of automatic Jeffrey-Traylor vibrating feeders at the bottom of the ore bins. The feeders are in turn controlled by Transportoweighers. The ore is mixed with water at the rod mill feed spouts. Rod milling is done at 77 to 79 percent solids. No chemicals are added in the milling circuit. The rod mills are in open circuit and the ore passes through and goes directly to the ball mill scoop. Rod and ball mills are set up as units. Number one rod mill feeds number one ball mill, number two rod mill feeds number two ball mill and so on down the line. Size reduction of the rod mill may be noted on the attached sizing sheet. Rod usage in the rod mills is about 0.5 pounds per ton of ore. New rod size is $2^{15}/_{16}$ inch diameter.

Ball milling takes place in closed circuit. Primary classification on the first three units is accomplished with Dorr rake classifiers. The fourth unit primary classification uses cyclones. Circulating loads average 500 percent. Balls used are 35 percent 1½ inch and 65 percent 2 inch diameter. Ball mills are run at 80 percent of critical speed. Ball usage is about one pound per ton of ore ground. Primary classifier overflow from the ball mills is 65 percent minus 200 mesh. This material then goes to secondary classification where it is separated into sand and slime fraction. Secondary classification is done by nine D-20B Krebs cyclones. Sizings of these fractions may be noted on the attached sizing sheet. The cyclone underflow, or sand fraction, goes directly by gravity to the sand leach plants and the cyclone overflow or slime fraction goes to thickening and then to the carbon-in-pulp plant. Lead ore contains a certain amount of gold which is liberated by grinding. Until 1970 about 65 percent of the total gold was recovered by amalgamation. When it became necessary to discontinue amalgamation the total recovery declined. Larger particles of gold were reaching the cyanide circuit and did not have enough contact time to totally dissolve. It was evident that some type of gravity separation was necessary for recovery of these larger particles.

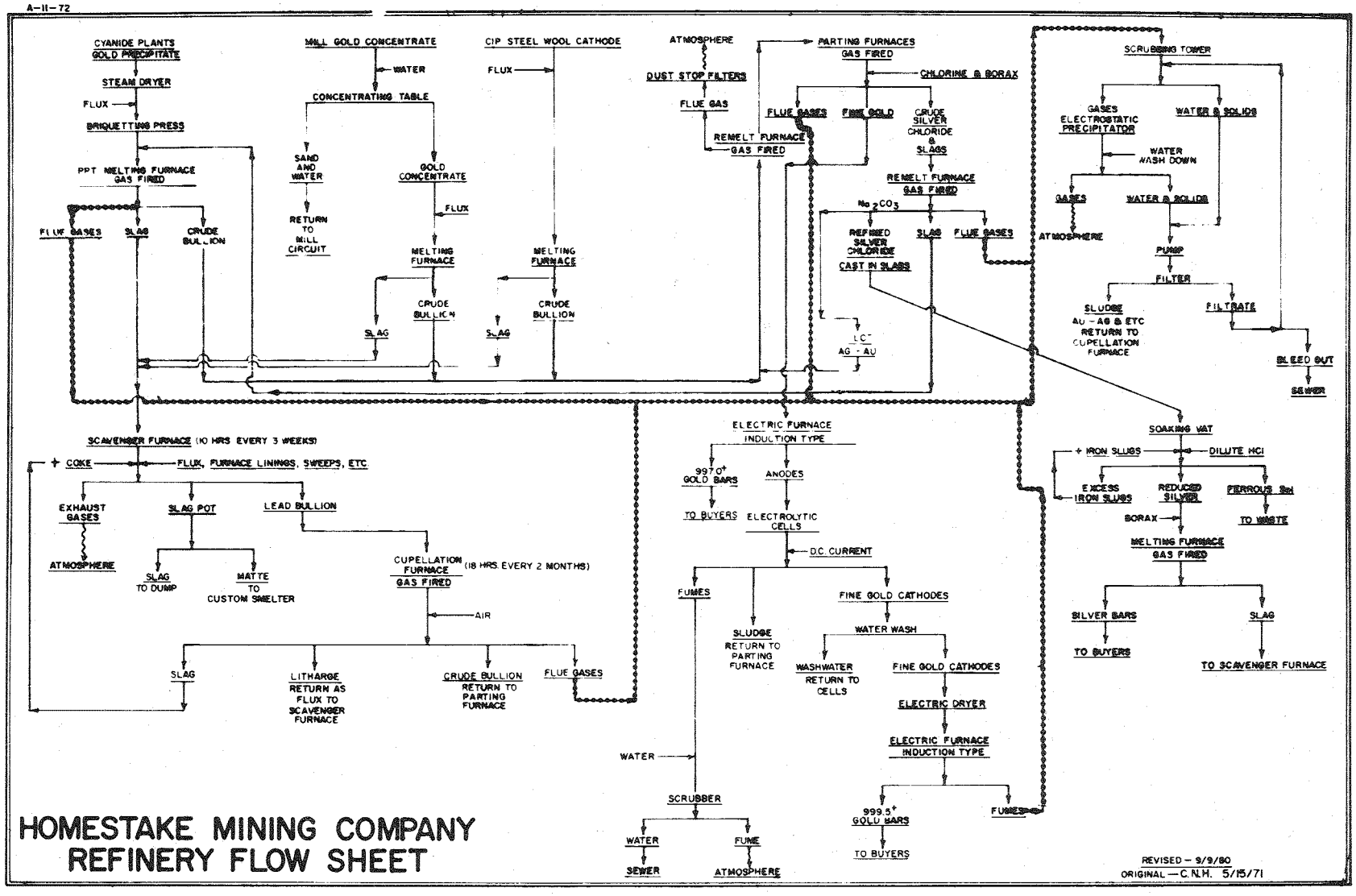
The gravity recovery system in use today consists of a series of sumps and riffles after the ball mill discharge. Riffles are constructed of expanded metal at different heights from the bottom of the launders. Once a week the mill is shut down and a cleanup made. This high grade sand is then run over a half size gravity table whose concentrate will run about ten ounces doré per pound. Gravity separation now accounts for 28 to 30 percent of our total recovery. Testing is continuing for a means of improving gravity recovery.

## CYANIDATION

Processes used to remove gold and silver from the precious metal-bearing ore and from precious metal-bearing solutions are referred to as cyanidation processes. Sand vat leaching, Carbon-In-Pulp and carbon cones constitute the cyanidation processes at Homestake.

The sand vat leaching process has been used at Homestake since 1901. We recover approximately 55% of our gold from this process. The process involves treatment of the coarse fractions, called sands, from secondary classification with cyanide-bearing solution in a leaching vat. There are thirty-five leaching vats in two plants. We have twelve leaching vats in the West Sand Plant that treat 800 tons of sand per vat. The East Sand Plant contains twenty-three leaching vats with approximately 775 tons of sand per vat.

Each sand vat has a cocoa matting and canvas bottom stretched over two inch by four inch notched support lumber to make a false bottom. The false


A-11-72
CYANIDE PLANTS GOLD PRECIPITATE
STEAM DRYER
FLUX
BRIQUETTING PRESS
PPT MELTING FURNACE GAS FIRED
FLUE GASES
SLAG
CRUDE BULLION
MILL GOLD CONCENTRATE
WATER
CONCENTRATING TABLE
SAND AND WATER
RETURN TO MILL CIRCUIT
GOLD CONCENTRATE
FLUX
MELTING FURNACE
CRUDE BULLION
SLAG
CIP STEEL WOOL CATHODE
FLUX
MELTING FURNACE
CRUDE BULLION
SLAG
ATMOSPHERE
DUST STOP FILTERS
FLUE GAS
REMELT FURNACE GAS FIRED
PARTING FURNACES GAS FIRED
CHLORINE & BORAX
FLUE GASES
FINE GOLD
CRUDE SILVER CHLORIDE & SLAGS
REMELT FURNACE GAS FIRED
Na2CO3
REFINED SILVER CHLORIDE CAST IN SLABS
SLAG
FLUE GASES
LC AG - AU
SCRUBBING TOWER
GASES ELECTROSTATIC PRECIPITATOR
WATER & SOLIDS
WATER WASH DOWN
GASES
WATER & SOLIDS
ATMOSPHERE
PUMP
FILTER
SLUDGE AU - AG & ETC RETURN TO CUPELLATION FURNACE
FILTRATE
BLEED OUT
SEWER
SCAVENGER FURNACE (10 HRS EVERY 3 WEEKS)
+ COKE
FLUX, FURNACE LININGS, SWEEPS, ETC
EXHAUST GASES
ATMOSPHERE
SLAG POT
SLAG TO DUMP
MATTE TO CUSTOM SMELTER
LEAD BULLION
CUPELLATION FURNACE GAS FIRED (18 HRS. EVERY 2 MONTHS)
AIR
SLAG
LITHARGE RETURN AS FLUX TO SCAVENGER FURNACE
CRUDE BULLION RETURN TO PARTING FURNACE
FLUE GASES
ELECTRIC FURNACE INDUCTION TYPE
997.0+ GOLD BARS
TO BUYERS
ANODES
ELECTROLYTIC CELLS
D.C. CURRENT
FUMES
SLUDGE RETURN TO PARTING FURNACE
FINE GOLD CATHODES
WATER WASH
WASHWATER RETURN TO CELLS
FINE GOLD CATHODES
ELECTRIC DRYER
ELECTRIC FURNACE INDUCTION TYPE
999.5+ GOLD BARS
TO BUYERS
FUMES
WATER
SCRUBBER
WATER
SEWER
FUME
ATMOSPHERE
SOAKING VAT
+ IRON SLUGS
DILUTE HCl
EXCESS IRON SLUGS
REDUCED SILVER
FERROUS SO4
TO WASTE
BORAX
MELTING FURNACE GAS FIRED
SILVER BARS
TO BUYERS
SLAG
TO SCAVENGER FURNACE
HOMESTAKE MINING COMPANY
REFINERY FLOW SHEET
REVISED - 9/9/80
ORIGINAL - C.N.H. 5/15/71

bottom is needed to allow the percolation of solution through the sand bed and through the cocoa matting and canvas bottom to collection sumps.

The sand plant's leaching cycle consists of filling, draining, aeration, strong solution leaching, washing and sluicing of each leach vat. The total treatment cycle will take approximately eight days.

After a vat is filled with sand most of the water included in the filling is drained off through the bottom. This draining is then followed by forcing compressed air upward through the sand to saturate the vat with oxygen needed in the gold dissolution reaction. Cyanide solution at approximately .05% strength is then added to the top of the sand. The treatment cycle consists of four drain, air and cyanide solution periods. After the cyanidation period the sand vat is sampled and then sluiced to the sand dam where the sand can be put back underground for backfilling stopes or sent to the Grizzly Gulch tailings disposal dam. A sand plant treatment cycle is explained in detail in a table.

The solution collected during the cyanidation periods of the leach cycle are collected in 250 ton pregnant solution storage tanks. The solution is then passed through a Merrill-Crowe zinc precipitation process. The zinc precipitation process consists of adding fine particles of zinc dust to a deaerated gold-bearing solution and precipitating the gold. The gold-bearing precipitate is filtered from the solution in plate and frame filters. The solution is then returned to storage tanks where the cyanide solution strength is raised back to the .05% starting strength and reused in the leaching cycles. The precipitate is then removed twice a week from the presses and treated in the refinery to remove the gold.

The second phase used in the cyanidation process is the Carbon-In-Pulp process which treats the fine fraction, or slime, from the secondary classification.

For many years the slime fraction was treated in the plant in Deadwood, South Dakota. It was pressure leached with cyanide solutions in thirty Merrill filter presses. Gold contained in the pregnant solutions was precipitated with zinc dust. The precipitate was then sent to the refinery.

In August of 1973 the Carbon-In-Pulp process replaced the seventy year old slime plant. Operation of the Carbon-In-Pulp plant is on a continuous 24-hour seven day per week basis. Slimes from the grinding circuits which average .06 ounces of gold per ton of ore are thickened in two 125' Dorr thickeners to approximately 45% solids and then pumped to the Carbon-In-Pulp plant.

The Carbon-In-Pulp plant consists of conditioning, dissolution, adsorption, desorption, and reactivation circuits. In the conditioning circuit of the Carbon-In-Pulp plant slimes are passed over two feet by 4 feet Denver-Dillon vibrating screens with 28 mesh stainless steel cloth to remove wood chips from the slimes prior to treatment. The screen undersize flows to one 18 foot diameter by 20 foot high tank with propeller agitation to condition the slime with air and lime. Lime is added at a rate of about two pounds per ton of ore treated.

Following the conditioning, the slime pulp flows to a dissolution feed sump where sodium cyanide is added to a cyanide strength of .04% to .07%. From the sump the slime pulp is pumped to one of seven 30 feet diameter by 22 feet depth Denver rake type agitators. The slime pulp flows by gravity from tank to tank and is agitated and aerated by sparge rings and center air lift columns. Dissolved oxygen is measured and controlled to satisfy the oxygen demand required in the dissolution of gold. Recovery is attained in approximately 20-hours with closely controlled cyanide strength and pH adjustments. Undissolved gold in the pulp from the circuit averages .005 ounces per ton. At this point the gold in solution will average .04 ounce per ton of solution.

From the last tank in the dissolution circuit the pulp is pumped to the first of five adsorption tanks where it comes in contact with activated coconut carbon.

The adsorption circuit at Homestake was modified in recent months from the original design, because of adsorption problems and higher tonnages. Homestake converted one of the conditioning tanks to an adsorption tank.

The adsorption problems have been associated with the recycle water system. Homestake experimentation has determined that calcium carbonate loading on the carbon caused gold adsorption problems. Either periodic acid treatment of the carbon or phosphate additions to the slime pulp has minimized the calcuim carbonate problem. Preliminary metallurgical testing has also indicated recycle water temperature could effect carbon loading.

Also, with higher tonnage of lower grade ore being treated at Homestake, the use of the additional adsorption tank makes it possible to do preventive maintenance on the circuit without gold losses and tonnage losses.

The adsorption circuit consists of one adsorption tank containing propeller and air sparging agitation. The remaining four tanks contain center column airlift and air sparge agitation. Slime pulp is moved through the tanks from No. 1 to No. 5 and the activated carbon is moved countercurrent to the slime pulp flow. The slime pulp and activated carbon are elevated, by outside airlifts from each tank, to Derrick double deck vibrating screens which have 22 mesh stainless steel cloth. The slime pulp is separated from the carbon when it flows through the screens and the carbon is retained on the screens as oversize and circulated back to the adsorption tank. Carbon is advanced on a batch basis from tank to tank when carbon is pulled from the No. 1 tank to the desorption section. Reactivated carbon is returned to the circuit in tank No. 5.

A typical profile from the adsorption circuit is as follows:

**Adsorption Profile**

| | Loaded Carbon (oz gold/ton Carbon) | Solution Values Tank Outflows (MO Au/Ton soln) |
|---|---|---|
| Adsorption Plant Feed | | 30.0 |
| No. 1 Adsorption Tank | 150 | 12.0 |
| No. 2 Adsorption Tank | 90 | 6.5 |
| No. 3 Adsorption Tank | 55 | 4.5 |
| No. 4 Adsorption Tank | 20 | 1.5 |
| No. 5 Adsorption Tank | 10 | 0.5 |

Total carbon losses from the plant average .04 pounds of carbon per ton of slime treated. The carbon losses are composed of carbon that passed the 22 mesh screens in the adsorption section and carbon removed at the reactivated sizing screen. The fine carbon accumulated over several months is sent to a smelter for recovery of gold that is on the carbon.

After the carbon has been loaded to 150 ounces per ton it is educted to a 1½ foot by 3 feet vibrating wash screen to remove the slimes that coat the surface of the carbon and prevent the desorption of the gold.

Desorption of the carbon consists of passing 190 °F sodium hydroxide (1%)—sodium cyanide (.2%) solution at 10-12 gallons per minute through three cylindrical stainless steel desorption vessels. The vessels are cone shaped on the bottom and have a capacity of about one ton of carbon. Two cone vessels are used during stripping and the third vessel is being filled with carbon for later stripping.

Once the solution has passed through the carbon beds the solution will then pass through three stages of Zadra electrolytic cells. The gold is deposited on cathodes containing 8 to 10 pounds of medium grade steel wool per cathode.

After the solution passes through the electrolytic cells it is returned to the boiler and reheated for reuse in the stripping step. About 50 hours of time is required to strip the carbon from 150 ounces per ton to less than five ounces per ton.

About 600 to 700 ounces of gold is removed from the No. 1 Zadra cell each week. The steel wool in the No. 2 and No. 3 Zadra cells are moved up to the No. 1 and No. 2 Zadra cells. A new cathode of steel wool is then installed in the No. 3 Zadra cell.

During the time the activated carbon is in the process the carbon adsorbs materials which are not stripped off during the desorption steps. Therefore a loss of gold adsorption activity occurs if the carbon is sent back to the adsorption step without going to reactivation.

Reactivation is accomplished by heating the carbon in the absence of air to a temperature of 1100 °F. Each batch of carbon is reactivated in a gas fired rotary kiln. The kiln consists of a rotary horizontal stainless steel tube 20-inches in diameter and 12 feet 7-inches in length, which is heated externally by a gas-fired, rectangular-shaped firebox. About seventeen hours are required to reactivate one ton of carbon.

Once the carbon has been air cooled, it is passed over a sizing screen of 16 mesh screen cloth. The undersize is collected and sent to a smelter. The oversize is educted back to the No. 5 adsorption tank for reuse.

Carbon cone tanks are being used as the third form of gold recovery system. Homestake is recovering gold from decant water from our Grizzly Gulch impoundment. We have additional gold being dissolved in small amounts during the pumping of carbon-in-pulp tailings and sand plant tailings to Grizzly Gulch. Homestake presently has 8 four-ton carbon cones in use and four additional carbon cones are in design stage. We are loading the carbon to about 40 ounces per ton before removing it from the circuit. The forty ounces per ton on the carbon is about maximum loading because iron and copper cyanide complexes also occupy carbon adsorption sites. The carbon is presently being shipped to a smelter. Homestake is in the process of designing a carbon strip system to handle the water treatment carbon.

The carbon cone tanks are 8 feet in diameter and 17 feet in height. Each tank has a screen box to dewater the carbon. The flow of water through the carbon is maintained at a rate to very slightly fluidize the carbon bed.

## SMELTING AND REFINING

Feed to the Lead Refinery consists of gold-zinc precipitate from the sand leach plants, gold plated steel wool from the carbon-in-pulp plant and high grade concentrate from the south mill gravity circuit. These three sources are treated as separate entities through the refining operation. This is done for calculations of efficiencies and recoveries of each plant.

Gold-zinc precipitate comprises about 55 percent of the refinery feed. It is cleaned from filter presses and transported to the refinery in locked containers. This precipitate contains about four ounces of gold per pound. After being sampled, the precipitate is dried to around ten percent moisture and mixed with flux. Homestake does not acid treat or roast the precipitate, but elects to flux off impurities with a strongly oxidizing acid flux. The flux used consists of soda ash, silica, sodium borate and sodium nitrate.

The fluxed precipitate is compressed into ten pound briquets and charged into a natural gas fired 26 inch Monarch-Rockwell single chambered furnace. Total capacity of the furnace is 500 pounds. Melting temperatures are 2000 to 2200 degrees F°. Time for each melt is approximately two hours. During this time the base metals have been oxidized and taken up in the slag. When the melt is complete, the slag is poured off and the doré bullion collected. This bullion, assaying 800 fine gold and 170 fine silver, is weighed, sampled, and stored for further refining.

Steel wool cathodes from the carbon-in-pulp plant are treated in much the same manner as the gold-zinc precipitate. The cathode consists of eight to ten pounds of medium grade steel wool which is loaded with 600 to 700 ounces of doré bullion. The plated

steel wool is fluxed with sodium nitrate, silica and sodium borate and melted in a Monarch tilting furnace. Melting time is one hour at 2000 to 2200 degree F. The melted charge is poured into a cone mold, allowed to cool, and the bullion recovered. The resulting bullion will assay 815 fine gold and 170 fine silver. This bullion is weighed, sampled and stored for further refining.

Concentrate from the south mill gravity circuit averages ten ounces doré per pound. Primary impurities are metal sulfides present in the ore. These metal sulfides are fluxed off in the same manner as the steel wool. The concentrate is also melted in a Monarch gas fired tilting furnace. The resulting bullion assays 780 fine gold and 160 fine silver. This is again stored for further refining. Slags from each operation contain about 20 ounces gold per ton and are eventually run through a scavanger furnace.

All crude doré bullion, from the three prementioned plants, is upgraded by the Miller-Process or chlorine parting. The Miller Process consists of forcing chlorine through the molten bullion and changing all but the gold to chlorides.

The Miller Process is done in 500 ounce lots in small gas fired furnaces. The bullion is contained in a clay crucible protected by a larger graphite crucible. A clay tube, with chlorine passing through it, is inserted into the molten bullion. Silver and base metals quickly react with the chlorine and form chlorides which float to the top. The chlorides are skimmed off until no more are formed. At this time sodium chloride and borax are placed on the surface of the molten bullion, the crucible covered to prevent volitization and the chlorine flow continued at a reduced rate for one hour. At the end of an hour the borax and salt are skimmed off with any other chlorides which may have formed. The refined gold, assaying 997 to 999 fine, is then poured into bars. These bars are cooled, cleaned and charged into a 4,500 ounce capacity induction furnace. Here they are poured into either anodes for electrolytic refining or good delivery bars of desired weights. About 40 percent of bars sold come directly from the Miller Process. Presently about 60 percent of the gold from the Miller Process is cast into 200 ounce anodes for further refining by the Wohlwill Process or electrolytic refining.

The electrolytic gold refining section has eight cells in series, each with a capacity of 800 ounces. Anodes from the Miller Process are placed in polypropylene tanks containing 90 to 100 grams per liter of gold chloride. The solution also contains 15 percent excess HCl for proper dissolution and deposition. The HCl reacts with the silver, which is the main impurity in the anode, and forms silver chloride, which being insoluble, falls to the bottom of the cell. Cathode starter strips are rolled from previously produced fine gold. Direct current is supplied by a rectifier at 0.5 volts and 100 amps. A small amount of air is bubbled through the cells to provide a mixing action. Gold deposited on the cathode is washed, dried, and melted into good delivery bars. Bar fineness is 999.9 plus.

A lead blast furnace is used to recover values remaining in slags, broken crucibles, used cupels, floor sweeping, and any other source which contains a small amount of gold. The blast furnace is 24-inches in diameter and is operated every two weeks for 12 to 14 hours. Coke is used as a heat source and also acts as a reducing agent for the litharge.

Products recovered from the blast furnace are a high grade matte, lead bullion and the final slag. The matte is shipped to a smelter once a year, the lead bullion cupelled to recover precious metals and the slag returned to the milling circuit.

Lead bullion from the blast furnace and sludge collected from flue gases are charged into a 50-inch by 40-inch by 9-inch elipitical cement cupel. The cupel is preheated, before charging, in a natural gas fired Homestake designed furnace. Melting of the sludge takes place in a reducing atmosphere. When this charge is completely fused, an oxidizing atmosphere is used. A stream of air is blown onto the surface of the lead bullion, oxidizing the lead to litharge. The litharge produced is skimmed continuously. Eventually, after eight to ten hours, the lead has been oxidized and the doré bullion remains. This bullion is recovered and goes to the Miller Process.

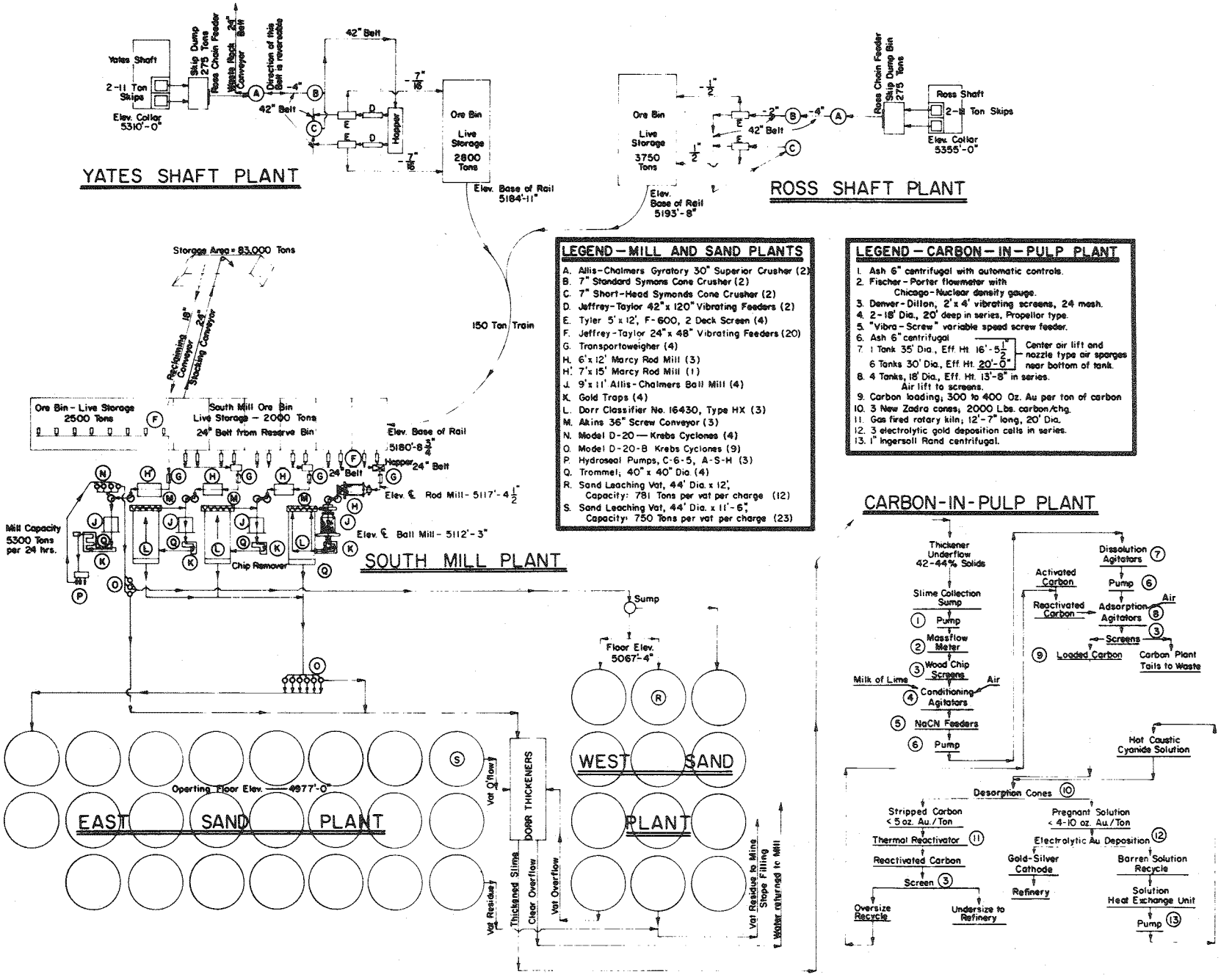
Silver chloride from the parting operation is remelted and a small amount of sodium carbonate is added to the molten chloride. This reduces any gold chloride which may have been formed during parting and also a small amount of the silver chloride. The charge is allowed to cool to a dull red heat and the chloride poured into ½ inch slabs six inches square. The reduced gold and silver is solidified and is held for the next parting run.

The silver chloride slabs are reduced to metalic silver in acid proof stoneware trays. These trays hold 75 pounds of chloride cakes alternated with a layer of iron slugs made from scrap steel. The chloride-iron layers are covered with dilute muriatic acid. The silver is reduced to silver metal and the iron is oxidized to iron chloride. This reaction is complete in four to five days.

When the silver chloride has been reduced to metallic silver it is melted, with a small amount of borax, and poured into 400 to 500 ounce bars. These bars assay 920 to 960 fine with copper the primary impurity. The crude silver is then refined electrolytically using Thumb cells. Final silver purity from electrolytic refining is 999.0 plus. This is poured into forms to suit customer needs.

## MINE BACKFILL AND TAILINGS DISPOSAL

Sand residue from the sand leach plants is sluiced from the vats and runs by gravity to two holding areas called sand dams. Dam capacities are 6,000 tons. Directly below the sand dams is a raise which bottoms at the 1,100 foot level of the mine. The sand, in these holding areas, is hydraulically moved into a pipeline running down the raise. Normal density of sand slurry is 55 to 60 percent solids. Pipe connections on the 1,100 foot level direct the sand to


YATES SHAFT PLANT
Yates Shaft
2-11 Ton Skips
Elev. Collar 5310'-0"
Skip Dump 275 Tons
Ross Chain Feeder
Width Hopper Conveyor
24" Belt
Direction of this Belt is reversible
42" Belt
Hopper
Ore Bin Live Storage 2800 Tons
Elev. Base of Rail 5184'-11"
ROSS SHAFT PLANT
Ore Bin Live Storage 3750 Tons
Elev. Base of Rail 5193'-8"
Ross Chain Feeder
Skip Dump Bin 275 Tons
Ross Shaft
2-11 Ton Skips
Elev. Collar 5355'-0"
Storage Area = 83,000 Tons
Reclaiming Conveyor
Stocking Conveyor
150 Ton Train
Ore Bin - Live Storage 2500 Tons
South Mill Ore Bin Live Storage - 2000 Tons
24" Belt from Reserve Bin
Elev. Base of Rail 5180'-8 3/4"
Hopper
24" Belt
Elev. ℄ Rod Mill - 5117'-4 1/2"
Elev. ℄ Ball Mill - 5112'-3"
Mill Capacity 5300 Tons per 24 hrs.
Chip Remover
SOUTH MILL PLANT
Sump
Floor Elev. 5067'-4"
Operting Floor Elev. 4977'-0"
EAST SAND PLANT
DORR THICKENERS
WEST SAND PLANT
Vat O'flow
Vat Residue
Thickened Slime
Clear Overflow
Vat Overflow
Vat Residue to Mine Stope Filling
Water returned to Mill
LEGEND — MILL AND SAND PLANTS
A. Allis-Chalmers Gyratory 30" Superior Crusher (2)
B. 7" Standard Symons Cone Crusher (2)
C. 7" Short-Head Symonds Cone Crusher (2)
D. Jeffrey-Taylor 42"x 120" Vibrating Feeders (2)
E. Tyler 5' x 12', F-600, 2 Deck Screen (4)
F. Jeffrey-Taylor 24"x 48" Vibrating Feeders (20)
G. Transportoweigher (4)
H. 6'x 12' Marcy Rod Mill (3)
H'. 7'x 15' Marcy Rod Mill (1)
J. 9'x 11' Allis-Chalmers Ball Mill (4)
K. Gold Traps (4)
L. Dorr Classifier No. 16430, Type HX (3)
M. Akins 36" Screw Conveyor (3)
N. Model D-20 — Krebs Cyclones (4)
O. Model D-20-B Krebs Cyclones (9)
P. Hydroseal Pumps, C-6-5, A-S-H (3)
Q. Trommel; 40" x 40" Dia (4)
R. Sand Leaching Vat, 44' Dia. x 12', Capacity: 781 Tons per vat per charge (12)
S. Sand Leaching Vat, 44' Dia. x 11'-6", Capacity: 750 Tons per vat per charge (23)
LEGEND — CARBON — IN — PULP PLANT
1. Ash 6" centrifugal with automatic controls.
2. Fischer - Porter flowmeter with Chicago - Nuclear density gauge.
3. Denver - Dillon, 2' x 4' vibrating screens, 24 mesh.
4. 2 - 18' Dia., 20' deep in series, Propellor type.
5. "Vibra - Screw" variable speed screw feeder.
6. Ash 6" centrifugal
7. 1 Tank 35' Dia., Eff. Ht. 16'-5 1/2"
6 Tanks 30' Dia., Eff. Ht. 20'-0"
Center air lift and nozzle type air sparges near bottom of tank.
8. 4 Tanks, 18' Dia., Eff. Ht. 13'-8" in series. Air lift to screens.
9. Carbon loading; 300 to 400 Oz. Au per ton of carbon
10. 3 New Zadra cones, 2000 Lbs. carbon/chg.
11. Gas fired rotary kiln; 12'-7" long, 20' Dia.
12. 3 electrolytic gold deposition cells in series.
13. 1" Ingersoll Rand centrifugal.
CARBON-IN-PULP PLANT
Thickener Underflow 42-44% Solids
Slime Collection Sump
1 Pump
2 Massflow Meter
3 Wood Chip Screens
Milk of Lime
Air
4 Conditioning Agitators
5 NaCN Feeders
6 Pump
Dissolution Agitators 7
Pump 6
Activated Carbon
Reactivated Carbon
Adsorption Agitators 8
Air
Screens 3
9 Loaded Carbon
Carbon Plant Tails to Waste
Hot Caustic Cyanide Solution
Desorption Cones 10
Stripped Carbon < 5 oz. Au./Ton
Pregnant Solution < 4-10 oz. Au./Ton
Thermal Reactivator 11
Electrolytic Au Deposition 12
Reactivated Carbon
Gold-Silver Cathode
Refinery
Barren Solution Recycle
Screen 3
Oversize Recycle
Undersize to Refinery
Solution Heat Exchange Unit
Pump 13

the level and stope to be backfilled. During the final six to eight inches of stope filling, dry cement at a one to ten ratio is added to the sand slurry. This forms a cement cap on the floor of the stope on which slushing of broken rock can be done. At the present time about 50 percent of the residue sand is used for mine backfill.

All tailings from the carbon-in-pulp plant and sand tailings not used for mine backfill are pumped to a tailings pond located in Grizzly Gulch which is located three miles from Lead. These tailings are pumped, at 35 to 40 percent solids, by a bank of eight BC-6-6C ASH rubber lined pumps in series. The system has two banks of pumps and two eight inch rubber lined tailings lines. Vertical lift of the tailings system from the tailings pump station to the impoundment area is 487 feet. Tailings line length is 15,651 feet. The line is unusual in that it goes up a steep hillside, down a deep valley and back up another steep 15 percent grade to the tailings dam area. The lines are equipped with vent valves to relieve entrapped air and with dump valves for draining. The dump valves, located in Kirk Canyon, drain the pipelines to a holding area on the 1,100 foot level of the mine. The lines must be drained when not in use in the winter and also during emergency shutdowns. Pump discharge pressure from the final pump stage range from 400 to 500 pounds per square inch. Normal tonnage pumped is 4,000 tons per day.

## WASTE WATER TREATMENT

Although we do not yet have a method of meeting our cyanide and heavy metal standards, we have installed a Baker pressure sand filter for suspended solids removal and an 80 feet by 150 feet cooling reservoir. A layout drawing is located in the appendix.

The Baker sand filter began 24-hour per day operation in August of 1979 with minimal start-up and operating problems. Discharge to date has been averaging less than our NPDES limit of 10 ppm suspended solids.

Homestake has been involved in extensive research to develop technology that would solve our cyanide and heavy metal problems in our waste water.

**Sand Plant Treatment Cycle**

| Operation | Duration In Hours | Remarks |
|---|---|---|
| (1) Filling | 7–8 | Filling time of East Plant 23 vats 7-8 hours; filling time of West Plant—12 vats—17-18 hours. Pulp at 25% solids with minus 16 mesh lime slurry (2.00 lbs/ton of sand) charged to water filled vat thru Butters-Mein Distributor. Vat overflow pulp deslimed and water reused. Charge 0.160 oz. gold/ton. Extraction 92.0 to 94.0. |
| (2) First Drain | 19 | Effluent to waste. |
| (3) First Aeration | 18–32 | Gauge pres. 9 lbs. Effluent valve closed. Aeration period varies according to alkalinity determination of sample taken 8 hours after vat filling. |
| (3a) Alkaline Wash | 2–4 | Clarified filling of water pH 11.5 to 12. Effluent to waste. |
| (4) First Strong Solution Leach | 20 | Leaching rate, 2 3/8'' per hour<br>Ongoing solution strength .0475 to .06% NaCN. Effluent to waste. |
| (5) Second Drain | 14 | Effluent to waste until trace of cyanide, then to solution collector tanks followed by conventional zinc dust precipitation. Barren solution used as 3rd period leach solution. |
| (6) Second Aeration | 6 | Gauge pres. 9 lbs. Effluent valve closed. |
| (7) Second Strong Solution Leach | 16 | Leaching rate 2 3/8'' per hour.<br>Ongoing solution strength .0475 to .06% NaCN. Effluent solution to collector tanks and precipitation. Barren solution reused. |
| (8) Third Drain | 12 | Effluent solution to collector tanks and precipitation. Barren solution reused. |
| (9) Third Aeration | 4 | Gauge pres. 8 lbs. Effluent valve closed. |
| (10) Third Strong Solution Leach | 16 | Leaching rate 2 3/8'' per hour<br>Ongoing solution strength .0475 to .06% NaCN. Effluent solution to collector tanks and precipitation. Barren solution reused. |
| (11) Fourth Drain | 12 | Effluent solution to collector tanks and precipitation. Barren solution reused. |
| (12) Fourth Aeration | 4 | Gauge pres. 8 lbs. Effluent valve closed. |
| (13) West (Barren) | 20 west<br>16 east | Solution strength .02 to .03% NaCN. Effluent solution to collector tanks, standardized to .0475 to .06% NaCN and recirculated for ongoing strong leach solution. Period duration varies according to amount of solution used. |
| (14) Washing | 19–22 | Leaching rate 2½'' per hour. Effluent solution to collector tanks, standardized to .0475 to .06% NaCN and recirculated for ongoing strong leach solution. Normal requirements are 48'' to 55'' of wash water to maintain solution balance. |
| (15) Sluicing | 7 | With clarified water thru Butters-Mein Distributor. Residue sands returned underground for stope fill. |
| (16) Refilling of Vat with water | 2 | Clarified water. (Limed when available.) |

## TYPICAL SAND PLANT CHARGE

| | |
|---|---|
| Dry Weight | — 775-800 tons per leach vat |
| Assay | — .150 oz. Gold/Ton |
| Extraction | — 92.0 to 94.0 |

## SCREEN ANALYSIS

| Mesh | +48 | +80 | +100 | +150 | +200 | −200 |
|---|---|---|---|---|---|---|
| Per Cent | 0.0 | 6.0 | 8.3 | 17.4 | 22.4 | 45.9 |

Total Treatment Cycle—196 hours.

Chemicals Used:

| | |
|---|---|
| Flotation Cyanide | — .30 lbs. per ton of sand treated |
| Lime | — 1.35 lbs. per ton of sand treated |
| Zinc Dust | — 0.035 to 0.040 lbs. per ton of sand treated |

The waste water is comprised of fresh water taken into the water system from mine cooling, gland water for pumps, rain water and springs. The fresh water cannot be separated from internal water. This creates an imbalance in the water system, which requires a discharge to keep the water system in balance.

The major heavy metal problem in the waste water is tied directly to complex cyanide formations of copper, iron, nickel, and zinc. Free cyanide also exists in the waste water at various concentrations based on temperature.

A few experimental technologies that are being used on treating Homestake's waste water for cyanide removal, without success so far, are Prussian Blue Precipitation, Ion Exchange, Ion Flotation, Ozonation, Alkaline Chlorination, Air Sparging, and Hydrogen Peroxide.

Homestake is on a very strict schedule to develop technology to treat cyanide. A consent decree by EPA, the State of South Dakota and Homestake states that Homestake will have a cyanide treating plant started by September 1981.

**Sizing of Homestake Milling Products**

| Mesh | Feed to Open Circuit Rod Mills | Rod Mill Discharge Units 1—2—3—4 | Rod Mill Discharge Unit 1—2—3—4 | Ball Mill Discharge Units 1—2—3—4 | Ball Mill Discharge Unit 1—2—3—4 | Ball Mill Class. & Pri. Cyclone O'Flo Feed to Sec. Class No. 1—2—3 | Ball Mill Class. & Pri. Cyclone O'Flo Feed to Sec. Class No. 4 | Secondary Class Products Sand E. Plant | Secondary Class Products Slime E. Plant | Secondary Class Products Slime W. Plant |
|---|---|---|---|---|---|---|---|---|---|---|
| | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. |
| +¾" | 2.0 | | | | | | | | | |
| +½" | 16.0 | | | | | | | | | |
| +¼" | 28.5 | | | | | | | | | |
| + 8 M | 19.3 | 9.3 | 0.6 | | | | | | | |
| + 10" | 4.2 | 7.7 | 1.4 | 2.0 | 0.6 | | | | | |
| + 28" | 8.8 | 27.6 | 20.9 | 12.2 | 7.5 | | | | | |
| + 48" | 3.2 | 11.2 | 15.8 | 16.0 | 18.5 | 0.0 | 0.0 | 0.0 | 0.1 | |
| + 80" | 3.4 | 6.4 | 9.6 | 22.0 | 25.9 | 3.1 | 4.0 | 1.5 | 4.9 | |
| +100" | 1.1 | 2.3 | 3.5 | 9.8 | 9.6 | 5.0 | 5.5 | 3.7 | 5.2 | |
| +150" | 2.0 | 4.3 | 5.3 | 10.8 | 11.8 | 11.6 | 11.7 | 14.6 | 15.0 | |
| +200" | 2.6 | 5.9 | 8.4 | 7.8 | 10.0 | 16.0 | 15.3 | 25.9 | 24.7 | 0.3 |
| −200" | 8.9 | 25.3 | 34.5 | 19.4 | 16.1 | 64.3 | 63.5 | 63.5 | 50.1 | 99.7 |
| Total | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

Milling Rate = 5,350 Tons/Day (yearly average)
Cyanidation of Sand = 59.3% of Tonnage—3173 Tons/Day
Cyanidation of Slime = 40.7% of Tonnage—2177 Tons/Day
Rod Mill Solids = 74—76%
Ball Mill Solids = 67—70%
B/M Class. O'flo Solids = 17—18%
B/M Circulation Load = 500%

# CALCULATION OF SAMPLE REQUIREMENTS AND THE DEVELOPMENT OF PREPARATION PROCEDURES FOR RELIABLE ANALYSES OF PARTICULATE MATERIALS

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## SYNOPSIS

Sampling of particulate materials and the subsequent preparation and analysis is most important in the mining and metals industries in exploration, mine development, ore production control and ore processing also in the marketing and/or smelting of ores and concentrates.

The objectives of sampling are to obtain reliable measurements, usually chemical analysis, of a bulk sample, drill core, etc., from which the analyzed samples were taken and prepared. While analytical methods are generally accurate, the initial sampling practices and preparation of those samples, to yield small representative portions for analysis, are frequently inadequate. This is especially true for ores and products containing precious metals because of the low concentrations of those values and frequently their erratic distribution. These give rise, consequently, to serious errors in the analytical values reported. The analytical data obtained thus may often be incorrect or seriously misleading.

Dr. Pierre Gy has developed a general equation to calculate sample requirements for particulate materials based on the physical and mineralogical characteristics of the material to be sampled. That equation is described and discussed, and its uses illustrated. The calculation and formulation of sample preparation procedures, to give the required or specified accuracy in the final sample assays are also described. The features and practical uses of Gy's slide rule, which incorporates Gy's equation, are also included. The requirements to be met, and the limitations, in the use of Gy's equation and slide rule are also given.

Examples are given for a number of ore types including those containing precious metals, which require modifications to Gy's general equation.

The special characteristics of precious metal ores and products, especially those containing coarse values, result in the need for preparation procedures which are markedly different from those for base metals and iron ores, etc. Some special procedures for preparation of precious metal ores and products are discussed. The checking of any preparation procedure is important to confirm its accuracy and to determine what changes may be required for its improvement.

## INTRODUCTION

In exploration projects core, chip or channel samples are taken to determine from the assays and related tonnages the ore reserves within a mineralized deposit. Those samples require preparation culminating in the removal of small portions for analysis. Clearly it is important that the assay values obtained represent the metals and minerals contents of the original samples within an acceptable and known accuracy.

Similarly in mine development, drift, cross-cut and raise samples are collected for analysis to permit calculation of the reserves and the tonnages, grades and ore categories which may be mined.

Mine development after start-up of an operation, as well as for control of ore production, also requires extensive sampling and assaying of working stopes and development headings, etc., to ensure continuity of production and the supply of the required tonnages, grades and ore types to the processing plant to meet production objectives.

The overall operation and effective metallurgical control of an ore processing plant involves sampling of the feed and all end products as well as key intermediate products to obtain an overall metallurgical balance and measurements of productions and inventories of concentrates etc. Sampling, weighing and analysis of concentrates shipped to the smelter or sold to a consumer are also important for process control and for calculations of the payments to be made by the smelter or purchaser.

Differences between mineable ore reserves and actual tonnages and grades of ore mined often arise and can be attributable, in part at least, to inaccurate analysis which may be caused by inadequate sampling or sample preparation procedures.

Differences between grades of ore reported by the mine and those received by the concentrator also might be explained by poor sampling in the mine or mill and/or unsatisfactory sample preparation practices. Unaccounted for differences in the concentrator feed and output and between concentrate shipments and smelter receipts often are caused by the sampling and preparation methods employed. Obviously there may be many other reasons for those differences, including analytical methods, weight and moisture determinations, contamination and shipping losses.

Differences in assays between seller and buyer of concentrates may represent substantial amounts of money. Errors in sample assays may often give rise to serious mistakes in investment analyses or in the projection of cash flows, etc., occasionally with disastrous consequences.

## GENERAL SAMPLE PREPARATION AND ANALYTICAL PRACTICES

Analytical laboratories (in-house, at the smelter or independent) are staffed usually with trained and experienced personnel and the procedures or techniques used for sample analysis are generally, if not always, accurate and yield reproducible results. When differences arise in the analysis reported be-

tween shipper and buyer of concentrates, for example, assays are made by an umpire or referee on a duplicate split of the final pulverized sample. The sampling and preparation procedures that were actually employed, however, are usually not disputed or checked.

The personnel employed for sampling and sample preparation, however, are too often not adequately trained and/or are poorly supervised. While great importance is usually given to the assay results obtained, the preparation of those samples may have been carried out using inaccurate and unacceptable procedures. This, consequently, may have invalidated the analyses, resulting in incorrect calculations or wrong decisions being made from them.

A sample preparation procedure which may have been found suitable for a copper ore or lead-zinc ore to give reliable analyses of copper, lead and zinc with an acceptable accuracy is often applied arbitrarily to other ores, especially of precious metals, usually with disastrous results. Only rarely is the sample preparation procedure actually designed to give the accuracy required for a specific ore or particulate product or material.

To ensure that the analytical results obtained from the assay of a few grams of pulverized sample reflect the contents of the primary sample taken of a bulk sample, core split or shipment of concentrate, etc., it is obvious that the initial sampling step and each stage of the preparation procedure must be given careful attention and be designed and carried out with the same high level of precision and control as is given to the analytical methods. The important physical and mineralogical characteristics of each material must be considered carefully in the design of a good sample preparation system.

Reference to Figure 1 shows a not uncommon procedure for preparation of a channel or drill core sample from a copper deposit. In this example, the procedure involves two stages of crushing of the total sample from about minus one inch size followed by three stages of riffle splitting of the 100 percent minus 1/8 inch material to reduce the sample weight to about 1¼ kg. That sample is dried at ±105 °C and pulverized to 100 percent minus 150 mesh, followed by two stages of further weight reduction using the method of rolling and quartering. Four final assay samples are obtained, two for analysis and two for reference or check analysis, as appropriate.

## STATISTICAL ASPECTS OF SAMPLE PREPARATION

An important point to introduce here is that of "statistical error" or the random error which is inevitably introduced into the analyses of the samples during the taking of the sample, its preparation and analysis. Other nonrandom errors, such as those incurred by salting, dusting losses or biases in the preparation, such as in riffle splitting or rolling and quartering, may also be significant. These can be avoided by use of good equipment, practices, training and supervision but are too extensive and numerous to consider in this paper. Reference to these aspects can be found elsewhere[1,5,8]. The statistical error cannot be eliminated, but it can be controlled to within acceptable limits by the sampling and preparation procedures employed. In all sampling, sample preparation and analytical steps, random errors are introduced which cause the final assay to differ from the true assay of the original material. The extent of the random errors depend upon the physical and mineralogical characteristics of the sample at each step in the process, and the preparation procedures and systems used.

The preparation procedures in figure 1 shows six major steps in the process of sampling including analysis at each of which a statistical or random error is introduced. If $S_1$ is the statistical error incurred at the first riffling step and $S_2$ is the error incurred at the second riffling stage, etc., then the total error ($S_T$) would be given by:

$$S_T^2 = S_1^2 + S_2^2 \text{ ---- } S_6^2$$

$S_T^2$ is the variance and $S_T$ is the standard deviation of the overall statistical sampling error. The step or stage which introduces the largest error will have the greatest effect on the total error ($S_T$) because it is affected by the square of that error. Larger errors usually occur in the initial sampling stage and at coarser particle sizes. This is discussed and illustrated later.

It will be evident that the expected standard deviation ($S_T$) in the final assay of a sample must be controlled to within limits set by the requirements for which the assays are to be used, such as for geochemical analyses, ore reserve calculations, concentrator metallurgical balances or concentrate sales.

A basic concept to appreciate is that any analytical result, or assay value obtained is not an accurate measure of the true assay of the original particulate material but will fall, within predictable limits, of the true assay for a given confidence level, usually as defined by Normal or Gaussian distribution curves as illustrated in Figure 2. It is evident that curve A shows a much narrower distribution than B and consequently an overall preparation procedure illustrated by curve A will give much more accurate and reproducible assay results.

Important characteristics of Normal distribution curves are, for examples, that 68% of the assay values of a sample will lie between $\pm S_T$ (one standard deviation) of the mean assay; 95% of the assay values of a sample will lie between $\pm 2S_T$ of the mean assay; and 99% will fall between $\pm 3S_T$.

Thus, if we require that 95% of the assays of a single sample are to fall between plus or minus 10% of the mean assay then $2S_T$ must be 10% or $S_T$ must be within 5% of the mean assay. If the mean assay of a sample is 5.00% Cu then for $2S_T$ to be ±10%, (0.5% Cu) then $S_T$ is ±0.25% Cu. That is 95 times out of 100 sample assays will be between 4.50% and 5.50% Cu. The preparation and analytical pro-

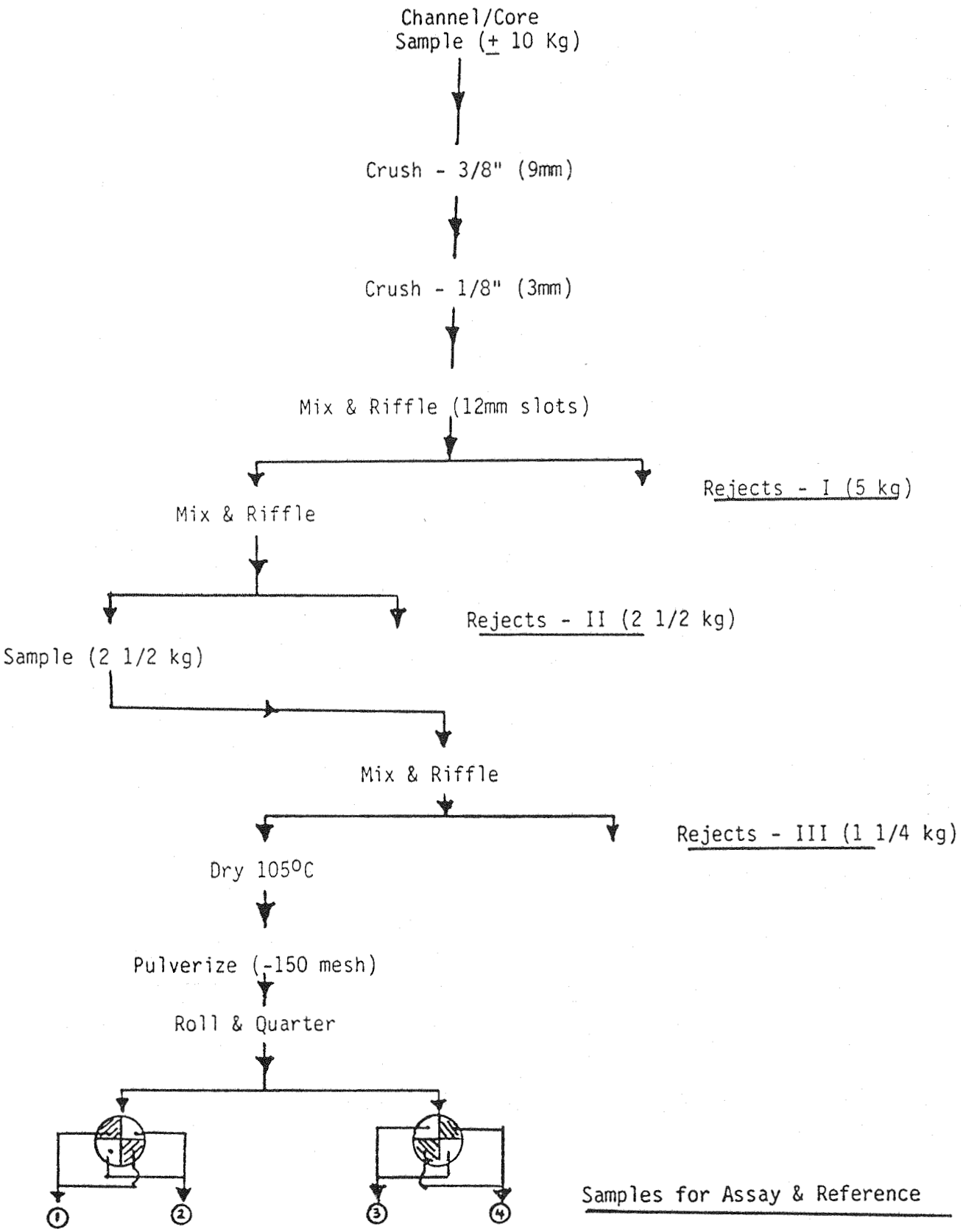


**FIGURE 1.** Typical sample preparation procedure for a porphyry copper ore.

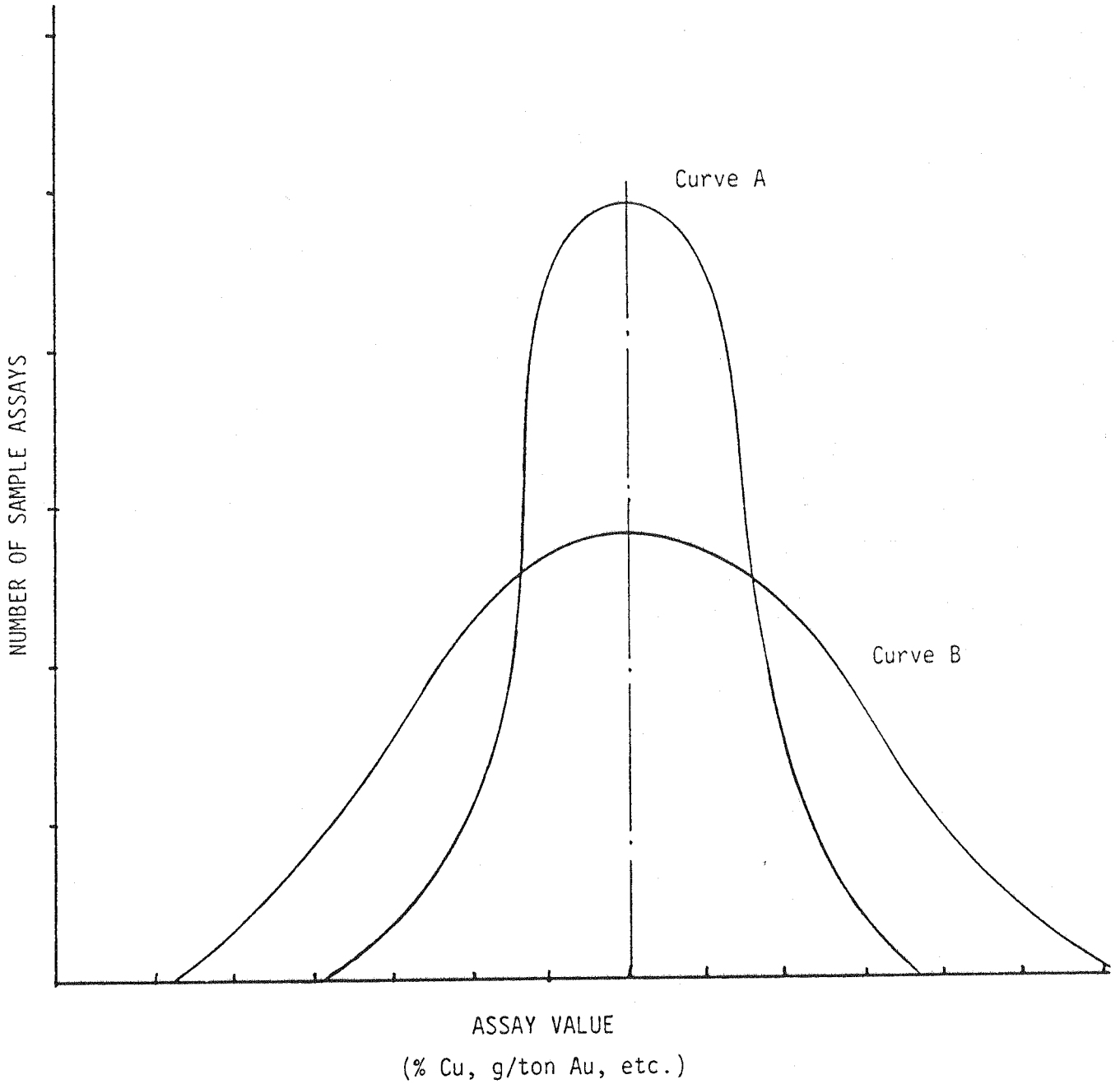


**FIGURE 2. Distribution surves illustrating the effects of sample preparation procedure on assay error or range.**

cedures used must then be designed and carried out to give that accuracy and reproducibility of assay values.

For a given preparation procedure, such as that illustrated in Figure I, we must thus know what is the overall statistical error ($S_T$) and what each step contributes to the total error. We must also know how to design a preparation procedure which will ensure that the error in the final assay value is within acceptable limits of $S_T$.

## GY'S SAMPLING EQUATION

The basic equation developed by Gy is:

$$M = \frac{Cd^3}{s^2}$$

where:

M = weight of sample required in grams

C = a sampling constant for the material being sampled. This is composed of four parameters which are characteristic of the material - f, g, l and m, wherein C = f × g × l × m

- f = shape factor, which is equal to 0.5 in all practical cases, (except for alluvial gold ores where f = 0.2)
- g = particle size distribution factor which usually has a value of 0.25 (except for closely sized materials where g = 0.5)
- l = liberation factor which has values between 0 and 1. For completely homogeneous material l = 0 and for completely heterogenous material l = 1. Values of 'l' are determined from a table worked out from a large number of practical studies relating the 95% passing size of the sample (d) and the practical liberation size of the mineral or minerals (L) where d/L = top particle size/liberation size. ('l' values are shown in Table I relating d/L values and 'l' values)
- m = mineralogical composition factor which can be calculated from the equation $m = \frac{1-a}{a}[(l-a)r + at]$ where 'r' and 't' are the mean densities of the valuable mineral and gangue minerals respectively; 'a' is the average mineral content (not metal equivalent) of the material expressed as a decimal part of 1.0. For an ore of a given size distribution f, g and m remain essentially constant. The sampling constant C is thus an intrinsic characteristic of that material. C will change if the material is crushed because the 'l' value will change.

d = dimension of the largest pieces in the lot to be sampled. For practical purposes this is taken as the 95% passing size of the sample, as determined by screening on a square mesh opening and is expressed in centimeters.

s = standard deviation of the fundamental error committed in random sampling of a material which assumes a normal distribution of the error for a large number of sample assays taken from a parcel of a particulate material. A random error is committed at each sampling stage such that the variance of the total error ($s^2$) is the sum of the variances incurred at each sampling step, as already discussed. Usually the error is determined from the value of 2s for 95 out of 100 assay values of a sample.

## USES OF GY'S EQUATION

Gy's equation can be used for three basic types of sampling calculations:

1. Calculation of the weight of sample (M) to be taken from a larger mass, from a knowledge of the approximate assay, the 95% passing size (d), the accuracy required in the sample assays (2s) and the approximate liberation size (1) of the valuable mineral or minerals. The equation would be in the form:

$$M = \frac{Cd^3}{s^2}$$

2. Determination of the sampling error which would be incurred if a given weight of sample is taken, knowing the top size of the particles, the liberation mesh, the approximate assay and other mineral characteristics. The equation would be in the form:

$$s^2 = \frac{Cd^3}{M}$$

3. Calculation of the particle size to which a sample should be crushed so that a required weight of sample can be removed, knowing the acceptable limits of the sampling error and the mineralogical/assay characteristics of the material to determine the value of the constant C. The equation would be in the form:

$$d^3 = \frac{Ms^2}{C}$$

It should be realized that nonrandom errors or biases which can occur in sampling and preparation of particulate materials are not provided for by the Gy equation. Thus in using this equation it is assumed that the techniques and equipment used will not introduce any nonrandom errors or biases. All possible efforts must be taken in practice to eliminate or minimize bias by the use of properly designed and installed equipment and the use of sound sampling techniques with good supervision and control.

**TABLE I**

| | | | | | | |
|---|---|---|---|---|---|---|
| Liberation factor 'l' | 0.8 | 0.4 | 0.2 | 0.1 | 0.05 | 0.02 |
| Feed and Middlings — $\frac{d}{L}$ values | 1 | 4 | 10 | 40 | 100 | |
| Concentrates | hetero-geneous | | | homo-geneous | | |
| Tailings | hetero-geneous | | | homogeneous | | |

A few calculations are given to illustrate the use of Gy's equation:

1. What is the weight of sample to be taken from a 1.0% Cu ore (mineral chalcopyrite - 34.5% Cu) crushed to about 95% minus 2.0 cm (d), where the practical liberation size (L), of the copper mineral is 150 mesh (0.1 mm or 0.01 cm) and the accuracy of the assay is required to be ±10%?

% Mineral = 1.0/0.345 = 2.9% (approx.) or *0.029*. From Table I for a value of d/L = 2.0/0.01 = *200*, the value of 'l' (liberation factor) is about *0.04*. The mineralogical factor (m) is calculated to be 139.2 using mean density values of 4.2 and 2.8 respectively for the chalcopyrite and the gangue minerals. Using average values of 'f' and 'g' of 0.5 and 0.25 respectively, the value of 'C' (sampling constant) is thus:

$$C = 0.5 \times 0.25 \times 0.04 \times 139.2 = 0.7 \textit{ approx.}$$

The tolerable assay error of 10% (0.10) gives an 's' value of 0.05 ($5.0 \times 10^{-2}$) and thence $s^2 = 2.5 \times 10^{-3}$.

$$M \text{ (sample weight)} = 0.7 \times (2.0)^3/2.5 \times 10^{-3} = \textit{2240 grams}$$

Comparable sample weights have been calculated similarly for this ore for other situations:

2. - for 95% minus 15 cm top size (-6 inch)
3. - for 95% minus 48 mesh top size (0.3 mm)
4. - for 95% minus 2 cm top size but with an accuracy of assay within 5%.

The results are summarized in Table II.

Example 2 illustrates the difficulty of obtaining a reliable sample of crushed ore from a stope or drift in the mine because of the large weight of sample required at a typical run-of-mine top size.

The expected higher accuracy of a sample taken in the concentrator of the mill feed is demonstrated by the small sample weight requirements for example 3. There are many sources and causes of errors, however, in the cutting, preparation and assaying of a head sample taken in the plant, which must be carefully considered in designing, installing and operating that sampling system.

**TABLE II**

| Top Size (95% passing) | 'l' Liberation Factor | C Sampling Constant | $s^2$ (Variance) | M Sample Weight |
|---|---|---|---|---|
| 1. 2 cm | 0.04 | 0.07 | $2.5 \times 10^{-3}$ | 2240 g |
| 2. 15 cm | 0.01 | 0.17 | $2.5 \times 10^{-3}$ | 574 kg |
| 3. 48 mesh | 0.50 | 8.7 | $2.5 \times 10^{-3}$ | 94 mg |
| 4. 2 cm | 0.04 | 0.07 | $6.25 \times 10^{-4}$ | 8960 g |

To reduce the error in the assay by half (10% to 5%) requires, as expected, a sample of about 4 times the weight as shown by example 4.

Gy's equation assumes that all particles in a bulk material have an equal opportunity to be sampled. This can be achieved by cutting the sample from a conveyor belt or pulp stream flow, using a suitable design and installation of cutter. It is not applicable to sampling of stockpiles or other materials where this requirement cannot be met.

The equation does not provide any information as to how the sample should be taken nor, in the case of a moving stream of ore on a conveyor or a pulp flow in a concentrator pipeline, how frequently to cut the stream nor the size of each increment. This is a complex subject and beyond the scope of this general paper. Reference should be made to the literature for additional information.[1,6]

Gy has developed a slide rule to facilitate the use of his equation in the laboratory, plant or field. The description and use of the slide rule have been given in several papers (2, 3, 4) and is summarized in Appendix A. Modules have now been developed by Gy for use with various solid state programmable calculators.

The Gy equation is applicable only to sampling for assay. There is another Gy relationship for sampling for size analysis.[1]

## DEVELOPMENT OF SAMPLE PREPARATION PROCEDURES USING GY'S EQUATION

Gy's equation and slide rule can be used also to work out the statistical error which can be tolerated at each step in a sampling procedure and to be within the total error given by the Gy equation. Usually the largest errors are introduced at the initial sampling steps and this has the greatest effect on the overall error as previously discussed. For convenience and simplicity, the total variance ($S_T^2$) can be assumed to be equally divided among the various sampling steps involved in the preparation of a specific sample. Thus from Figure 1, with 5 sampling stages, the variance of the error at each step $S_1^2$, $S_2^2$, $S_3^2$, etc., would thus be taken to be not more than $\frac{S_T^2}{5}$.

As an illustration of the use of Gy's equation for the design of a continuous sample preparation system for a 1500 TPD concentrator, we will consider a four stage sample preparation procedure to handle an ore crushed to minus 2.5 cm top size (minus 1 inch), containing 10% ZnS (sphalerite).

Four sampling stages are involved at 1 inch (2.5 cm), ¼ inch (0.64 cm), 10 mesh (1.65 mm) and 100 mesh (0.15 mm) and three crushing stages. We have determined that the sphalerite has a liberation size of 0.2 mm and an accuracy in the final sample for assay of 0.2% ZnS (95 times in 100) is required. This continuous sampling system is illustrated in Figure 3.

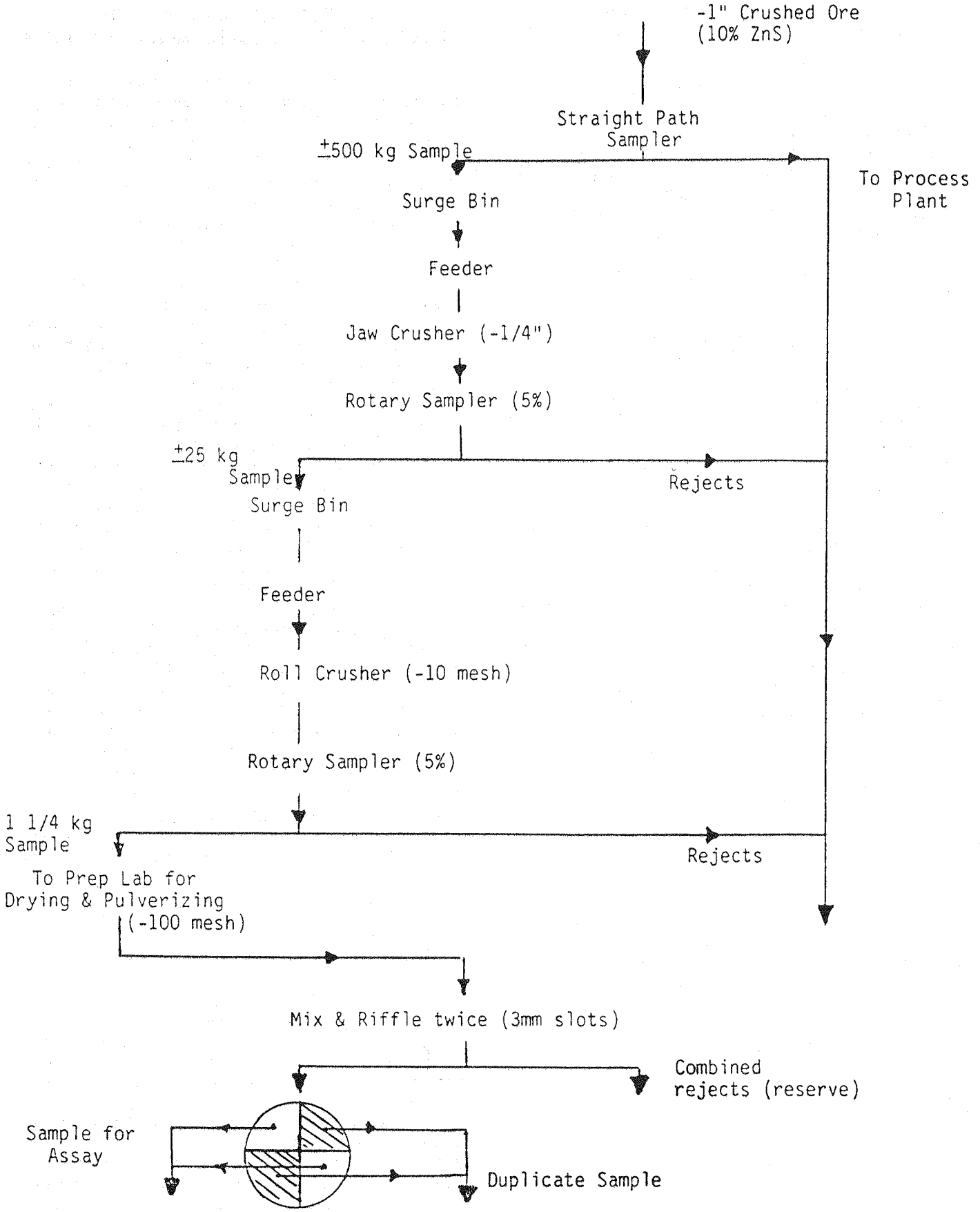


**FIGURE 3. Outline flowsheet for a continuous sampling plant for a zinc ore.**

The total error $S_T$ incurred, assuming no biases or distortion but completely random sampling, would thus be:

$$S_T = S_1^2 + S_2^2 + S_3^2 + S_4^2 + S_A^2$$

Where $S_1$, $S_2$, $S_3$ and $S_4$ are the errors incurred respectively at each sampling stage and $S_A$ is the overall error incurred in the analytical procedure which we may assume is negligible and thus can be ignored in this example. It is also assumed for simplicity that no distortions or biases occur during the sampling processes.

If we assume that an equal error(s) is to occur at each stage then:

$$S_T^2 = 4S_1^2 \text{ or } S = \frac{S_T^2}{4}$$

A 2% error on the head assay of 10% ZnS, 95 times in 100, thus gives a range of mineral assay values of 10.0 ±0.2 or between 9.8 to 10.2% ZnS (6.71 ± 0.13% Zn). With $2S_T = 2 \times 10^{-2}$ or $S_T = 1 \times 10^{-2}$ therefore $S_1^2 = 1 \times 10^{-4}/4 = 0.25 \times 10^{-4}$ or $S = 0.5 \times 10^{-2}$. This same value of S will apply to $S_2$, $S_3$ and $S_4$.

The minimum weights of sample required and pertinent parameters used in applying Gy's equation are given in Table III. These are the minumum sample weight requirements and usually larger samples are taken as suggested in the table to compensate for possible biases and poor preparation practices. Logically, the sample weights taken must be convenient and practical to collect and prepare for assay. Other combinations of error values incurred at each stage should, of course, be used to meet actual circumstances.

The overall system must be designed in as practical and realistic a way as possible to give the required accuracy but with minimal investment, handling time and operating costs.

**TABLE III. Key parameters and sample weights for the preparation of a specific zinc ore**

| Parameter | Sampling State 1 (-1 inch) | 2 (-¼ inch) | 3 (-10 mesh) | 4 (-100 mesh) |
|---|---|---|---|---|
| Liberation Factor (f) | 0.05 | 0.10 | 0.20 | 0.80 |
| Constant (C) | 0.27 | 0.52 | 1.10 | 4.20 |
| Particle Size (d) | 25mm | 6.5mm | 1.65mm | 150 microns |
| Sample Weight (M) | 170kg | 5.5kg | 200g | 0.5g |
| Actual Weight Taken (approx.) | 250kg | 16kg | 1kg | 150-250g |

## SAMPLE PREPARATION OF PRECIOUS METAL ORES AND PRODUCTS

For ores containing free gold (alluvial or lode ores ground to liberation size) Gy recommends the use of values for 'f' (shape factor) of 0.2 and 'g' (size distribution factor of 0.2) with a liberation factor of 1.0. The lesser values of 'f' and 'g' are used because free gold particles do not grind easily and tend to be flattened during preparation.

For the mineralogical parameter (m) the value to be used is 20/a, where 'a' is the gold assay in grams/metric ton.

The special gold scale shown on the slide rule was developed for use with alluvial gold ores but has been used successfully by the author for sample calculations for a number of lode gold ores and ores containing silver and platinum group metals of different types associations and characteristics.

The Gy formula was used effectively, for example, to develop a sampling and preparation procedure for channel samples for a Cu-Au-Ag ore and this is illustrated in Figure 5. This should be compared with Figure 1 and the principal differences noted. For the gold-silver-copper ore the total 10 kg sample required finer crushing to 100 percent minus 35 mesh (0.4mm) before any splitting down by riffling, whereas for the copper ore the first riffling stage was carried out at -3mm (⅛ inch) and the sample weight reduced at that particle size by further riffling to about 1¼ kg before pulverizing to -150 mesh.

In Figure 4 the 1¼ kg sample of -35 mesh Cu-Au ore material was pulverized to 100% minus 200 mesh before rolling and quartering for production of four final samples for transfer to the analytical laboratory for reference purposes and check assays.

Ores, plant products and concentrate shipments containing precious metals require special attention in sampling because of:

a. the low concentrations of values (1-2 grams/ton of Au or less is not uncommon in ores profitably mined and processed today).

b. high accuracy needed in the assays (because of high metal values and prices).

c. variable distribution and particle sizes of the gold, silver or platinum bearing particles.

For these reasons the Gy formula for alluvial gold and other precious metal ores may often be unsuitable or require major modifications.

An excellent account of the range and type of occurrences of gold in ores has been given by Henley[9]. Clearly the mineralogical associations of the gold (also of Ag and Pt group metals, if present) and the range, association and distributions of particle sizes of the precious metals are most important to the design of an accurate and acceptable sampling and sample preparation procedure. In the sampling examples considered previously in this paper, as illustrated in Figures 1 and 4, only the copper and zinc mineralization, respectively, were considered. The procedure may require significant modification

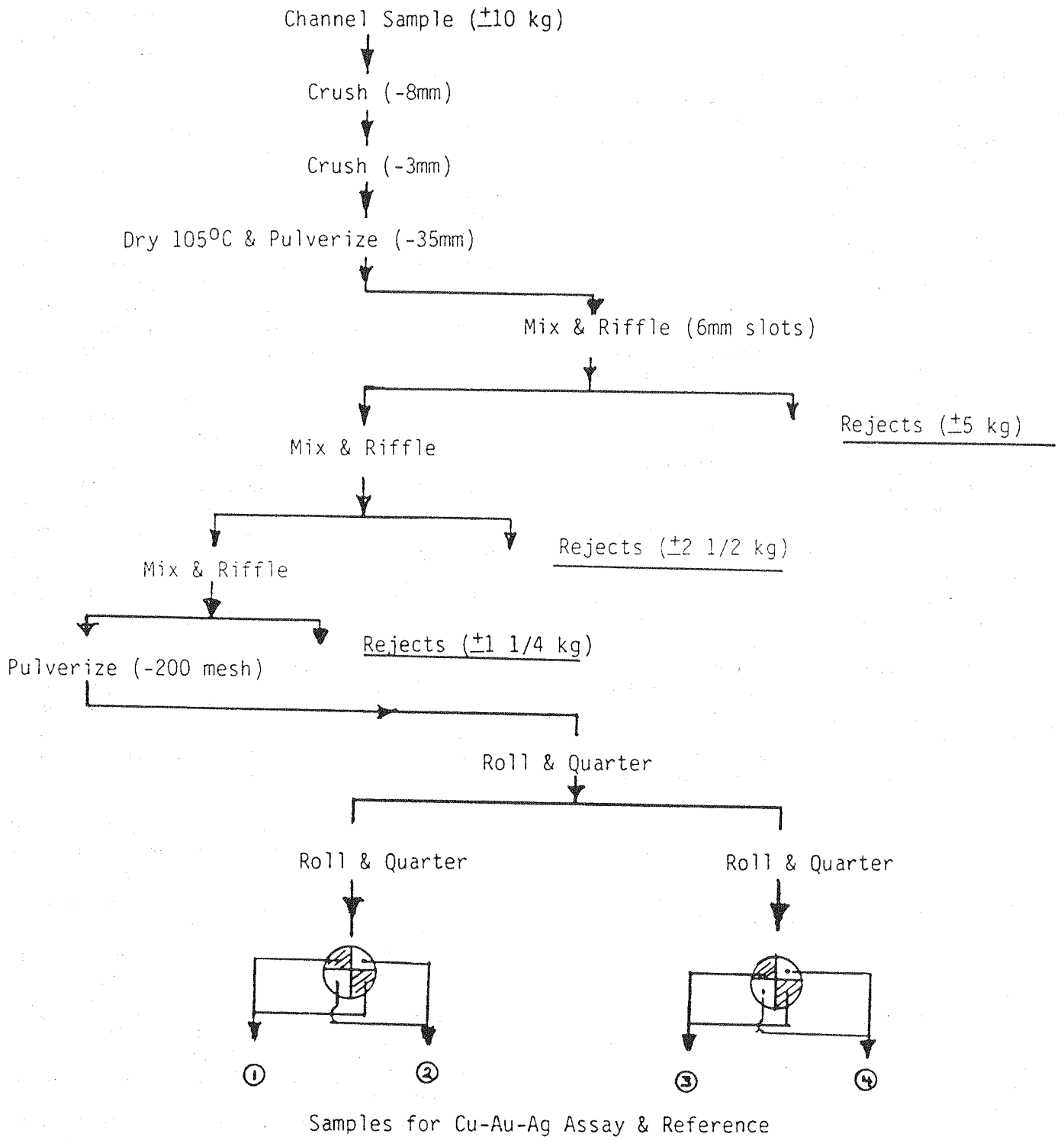


**FIGURE 4.** Sample preparation procedure for a copper-gold-silver ore.

where a high accuracy in the precious metal assays is important. This was evident in comparing the sample preparation procedures shown in Figures 1 and 4 and is further illustrated in Figure 5. Figure 5 shows an acceptable sampling procedure developed for the preparation of bulk samples for analysis and metallurgical testing in the development phase of a copper-gold-silver deposit containing significant amounts of fine gold values of essentially minus 200 mesh size particles, occurring partly as free gold and partly in association with the copper minerals.

The factor of greatest importance in the development of an acceptable and accurate sample preparation procedure for ores containing precious metal values is the particle sizes of any gold, silver and platinum present, especially free metal particles of plus 200 mesh size. A knowledge of the range of gold or silver particle sizes, especially the maximum particle size, thus should be determined or estimated by appropriate mineralogical studies of a number of representative samples of ore or concentrate.

Where plus 200 mesh free gold particles occur, the sample preparation procedure must be modified to include a screen to remove coarse metallics at one or more preparation stages. The metallics are then weighed and assayed separately. A proportion of that gold content is added to the assay value of the pulverized sample, from which the metallics were removed, to obtain the total assay value.

For alluvial ores wherein very coarse gold particles may occur, it is common practice to take large bulk samples often of several tons and to process them in their entirety, by wet screening to reject the coarse barren oversize followed by some form of mechanical gravity concentration (jigging, panning, or tabling) to remove the "recoverable" precious metal values, and any other heavy minerals present. The assay of each pit or bulk sample is then computed from the bulk sample volume, its moisture content and density and thus, dry tonnage if needed using the weights and assays of the heavy fractions recovered. Careful selection of the locations of the pits and their number are important to the reliable evaluation of deposits of this type. This requires the close cooperation of the geologists, mining engineers and minerals engineers and possible other project personnel.

The simplified data in Table IV illustrates the problem of sampling and assaying of precious metal ores particularly where the free metal particles are coarser than about 200 mesh. The gold particles were assumed to be of spherical shape and all of one size which would not occur in practice, of course.

At 8 mesh, gold particles would each weigh about 0.13 grams, thus there would be roughly only 8 particles in a ton of ore containing 1 gram/ton. At 65 mesh size there would be about 1250 gold particles in a ton of ore of that same grade and for 400 mesh size about 1.7 million particles of gold in a ton of that ore.

In the preparation of these ores for assay, it would be usual to take several 30 gram samples of the pulverized material for assay and the average Au value determined from them or from combining the lead buttons for cupellation.

**TABLE IV. Weights and distributions of gold particles for an ore containing one gram per ton of gold as metallics**

| Gold Particle Size-mesh Tyler (mm) | Weight Per Particle Gold (spheres) | No. Particles Per Gram Gold | No. Gold Particles in 30g Assay Sample |
|---|---|---|---|
| 8 (2.362) | 0.13 | 8 | 0? |
| 10 (1.651) | 0.045 | 22 | 0? |
| 65 (0.208) | 0.0008 | 1,250 | 0? |
| 150 (0.104) | 0.00001 | 100,000 | 3 |
| 200 (0.074) | 0.000005 | 200,000 | 6 |
| 400 (0.037) | 0.0000006 | 1,667,000 | 50 |

For an ore assaying 1 g/ton of gold, containing 8 mesh size gold particles a 30 gram sample would thus be expected usually to contain no gold particles and hence a zero gold content would be reported. Even for a material containing 200 mesh gold particles there would be, on average, only six gold particles in that sample. In the preparation it might be expected that five, six or seven gold particles might report in the final sample taken for assay. The assay would thus be between 0.83 and 1.17 g/ton Au, a difference of 34% and an error of ±17%. The importance of thorough mixing between sampling stages and the removal and separate assaying of coarse "metallics" are well illustrated by this simple data.

Two rules of thumb that have been widely practiced[11,14] are that i) there should be at least 20 particles of the valuable mineral or metal in the final sample taken for assay, or ii) no particle should represent more than 1/20,000 of the weight of the sample.

A simple method for determining the assays of precious metal ore samples is to subject a number of samples of one or two kilos or more to metallurgical separation in the laboratory.

For example, a 2 kg sample of minus 8 to 10 mesh material split out from the preparation system may be ground and subjected to flotation and/or cyanidation. All or a portion of the weighed flotation concentrate and/or a known volume of the cyanide solution would be assayed; also a representative part of the tailing, after drying, weighing and careful preparation. The original sample assay would be calculated from the weights and assays of the products obtained. Any significant differences between the calculated assay from the metallurgical treatment and the direct assay of the ore should be investigated and explained. This procedure would be particularly useful where the ore contained erratic distribution of precious metal or other coarse high value minerals, such as tin, tungsten and tantalum.

Where high grade precious metals concentrates are produced, especially those containing coarse values, it may be necessary to take a number of samples for analysis and to average the assay results

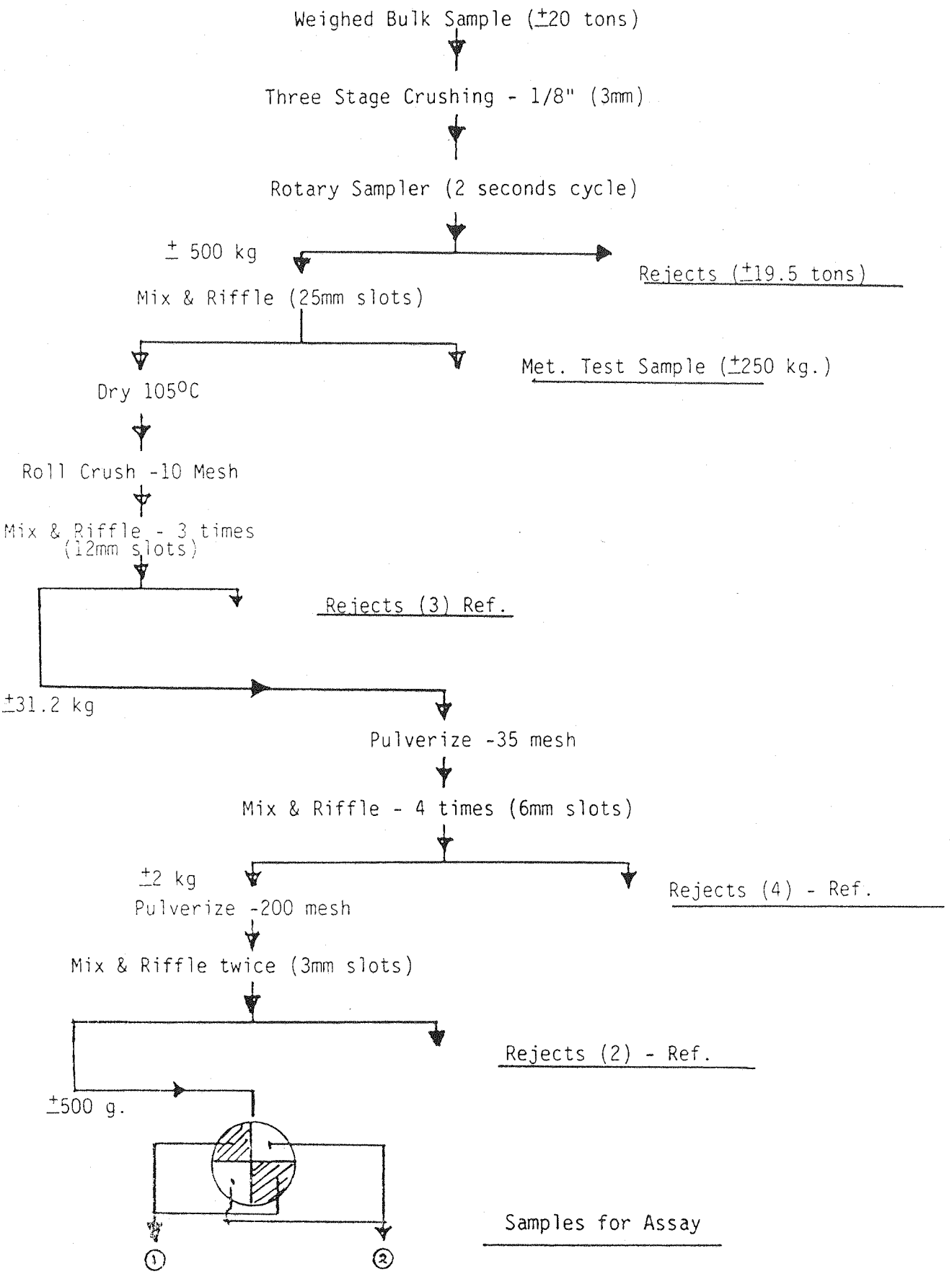


**FIGURE 5. Bulk sample preparation procedure for copper-gold-silver ore.**

obtained. An example is a jig concentrate, shipped to a smelter, containing 2000-3000 oz/ton of silver, partly as free metallic particles, and partly in association with base metal sulphide minerals, of up to ⅛ inch in size. The metallics were removed by screening at several stages in the preparation and each weighed and assayed separately. Sets of 8 samples were taken of the final -200 mesh material and assayed separately. The average of the 8 values was taken, provided all were within ±10 oz/ton Ag of each other. Proportional values were added to that assay for each of the metallic portions removed.

## CHECKING OF SAMPLE PREPARATION AND ANALYTICAL PROCEDURES

The sample preparation procedure developed for any ore, especially, those containing precious metals, using Gy's or other equations may not, for many reasons, give assay results of the required reproducibility or accuracy. It is, therefore, essential that the accuracy of the method or procedure used be established or confirmed for every material.

This can be done by taking the reject portions from the first step of the preparation procedure and splitting them down in the same way and assaying those end samples. The results from the rejects can be compared with those of the original split. This should be done for several ore samples, because different types and grades of ore may exist in a deposit. These checks can be extended further, if the accuracy of individual preparation steps are to be determined, by splitting down all or any of the separate reject samples from the various sampling steps and assaying all of the end products. Analysis of the results will show the overall accuracy of the preparation procedure including the analytical method used, and also of the individual steps. It will reveal where changes may be needed to the procedure to improve the accuracy.

For example, with reference to Figure 4, the 5 kg, 2½ kg and 1¼ kg rejects from the three riffle splits would be prepared separately for assay and the results for each compared with the assays from the initial preparation. Any changes that are needed would be introduced and the preparation system rechecked in the same way. It will be realized that where several ore types occur in a deposit that if only one preparation procedure is used it must satisfy the accuracy requirements for all ore types. This may be more practical than employing several different preparation procedures for each distinct ore type in one laboratory.

## CONCLUSIONS

From the data and discussions presented in this paper it is evident that adequate emphasis must be given to sample preparation of particulate materials in all areas of the primary minerals and metals industry.

The use of arbitrarily selected sample preparation procedures is dangerous and methods designed to give an acceptable and known accuracy and reproducibility in the assay results must be worked out for each ore or mineral product. The same care and supervision usually given to analytical practices must be given also to sample preparation.

Ores and products containing precious metals must be given particular attention because of their low concentrations, high unit values and frequently, the irregular distribution of those metals or metal bearing minerals.

Gy's equation (and slide rule) can be a useful basis for the design of sample preparation systems for most ores, provided the pertinent mineralogical characteristics are known or can be determined. The modified Gy equation for alluvial gold ores (and other precious metals) also serves as a good base for the design of sample preparation systems for many of those ores and treatment products. The particle size and distribution of the precious metals, especially if present as plus 200 mesh metallics, will exert substantial effects on the accuracy of a preparation procedure. Those types of ores require the development of special procedures and modifications to ensure reproducible assay results of an acceptable accuracy. This involves the removal by screening and/or metallurgical treatment of the coarse metallics for separate analysis. Acceptable preparation procedures for precious metal ores and products can thus be expected to be very different from those for base metals, iron ores and most non-metallic minerals.

This somewhat simplistic review of the importance of, and requirements for, good sample preparation may hopefully provide sufficient encouragement and interest in this subject to seek more details and information.

A selection of useful references are given which should provide some insights into the many important basic and practical aspects of the sampling and preparation of particulate materials.

## ACKNOWLEDGMENTS

The suggestions and constructive comments of many friends and colleagues in the mining and minerals industry given during the preparation of this paper are greatly appreciated. The permission of Mineral Systems Inc. to prepare this paper is gratefully acknowledged.

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# APPENDIX A

## HOW TO USE THE GUY SLIDE RULE

A. The value of the sampling constant C must first be determined.

1. Place the red line of the cursor on the line representing the % *mineral* content of the sample using the top red scale. In the case of alluvial gold ores and coal respectively, the bottom red scales are used.

2. Determine the liberation factor 'l' from the table on the reverse of the slide rule (Table I in paper).

3. Read off the values of 'C' against the corresponding value of 'l'.

B. Determination of the unknown factors 's', 'd', or 'M'. Assume that the weight of sample is to be calculated.

1. Line up the value found for 'C' on the fixed scale with the required value of 's' on the sliding scale (not $S^2$).

2. Read off the 'd' scale the value of 'M' that corresponds to this value. This is the theoretical minimum weight of sample to be taken.

3. 'd' and 's' can be calculated in a similar manner, knowing 'M' and 's' and 'M' and 'd', respectively, and the value of 'C'.

### Definitions

s = the statistical error in the assay of the sample taken ($\pm 2s$ is usually the error value most commonly used which, for a normal distribution, will result in 95 out of 100 values falling within that limit of the mean assay value)

d = 95% passing size of the material being sampled

M = weight of sample required or to be taken

C = sampling constant

Values of 'f' (0.5) and 'g' (0.25) are built into the slide rule. Similarly, densities of 5.0 and 2.6 are assumed for the valuable mineral and gangue, respectively.

# PRACTICAL POINTS OF BULK MATERIAL SAMPLING

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Bulk sampling of ores containing precious metals has a long history in the earlier Nevada mining eras. Taggart's "Handbook of Mineral Dressing" on sampling cites as references sampling practices at a variety of mining operations including many turn of the century gold and silver properties.

Discussions of sampling directed specifically to gold and silver ores require a qualification by pointing out the distinctive nature of these ores. Precious metal ores are characterized by occurrence of randomly dispersed flakes and discrete grains of valued metallics. Particle distributions are widely varying and populations are sparse. There are notable exceptions of ores occurring with uniformly disseminated microscopic minerals, but these are also quite lean in values. Because of these characteristics, precious metal ore sampling typically requires large proportions of sample to ore quantity. Noting the generally abrasive nature of the ore, and the necessity to maintain particularly good reliability of the sampling procedure to assure owners and operators of integrity in milling operations, the net result is a requirement for design of sampling equipment to meet high standards of accuracy and equipment reliability. A good design can often be quite costly. Sampling on the practical level may be a compromise relative to cost-effective objectives, relating to both mechanical and technical reliability.

## APPLICATION OF THE THEORY

Sampling is a probabilistic process, and the probability of taking a representative sample of an ore lot depends on statistical parameters characterizing the ore. It is possible to define a sampling procedure to achieve a stated level of accuracy. Sampling accuracy in practice responds to the degree that theoretically valid sampling procedures can be employed.

A sampling procedure may be designed to meet specified accuracy standards by reference to the theory such as that developed by Gy (ref. 2) outlined in the presentation by Ottley at this symposium (ref. 3). A theoretical recommendation for the minimum quantity of primary sample needed from a bulk precious metal ore stock can be substantial, particularly as the crushed product top size (95 percent passing) becomes larger. Fig. 1 is a guide taken from early studies of sampling requirements for dry ore materials as published in Taggart (ref. 1). This representation of sampling data may be used to make quick—though relatively inaccurate—estimates of the minimum sample needed for various mineralized ore classifications vs. top size based on 95 percent passing. Precious metal ore sampling guidelines are the uppermost curves requiring a high ratio of sample to ore, and their occurrence as discrete grains.

It is also to be noted that the early date (circa 1900) used in making recommendations of adequate sample size in Fig. 1 were drawn from experience with higher assay grades than is presently the norm for exploitable precious metal ore deposits. Therefore, it is unfortunately the case that substantially greater minimum sample weights would be recommended to assure adequate representation for the leaner grades now of commerical interest. A sampling ratio for conservative practice from dry ore on a conveyor would be a minimum of one percent if crushed to a minus one-inch, and no less than five percent at 4-inch.

To sample at these high ratios, a multiple stage sampling system is employed with a continuous first stage (primary) sample cut. Particle size of first stage sample would be typically reduced through roll crushing (or other suitable size reduction equipment) followed by second stage, and if necessary third stage sampling. It is important to assure final sample integrity through a multistage sampling procedure by maintaining weight reduction limitations according to the top size or crushing stage as described by Ottley according to the Gy guidelines. The same guidelines apply, by the way, to drill core sample reduction (ref. 4), or other laboratory scale splitting and reduction procedures for assays.

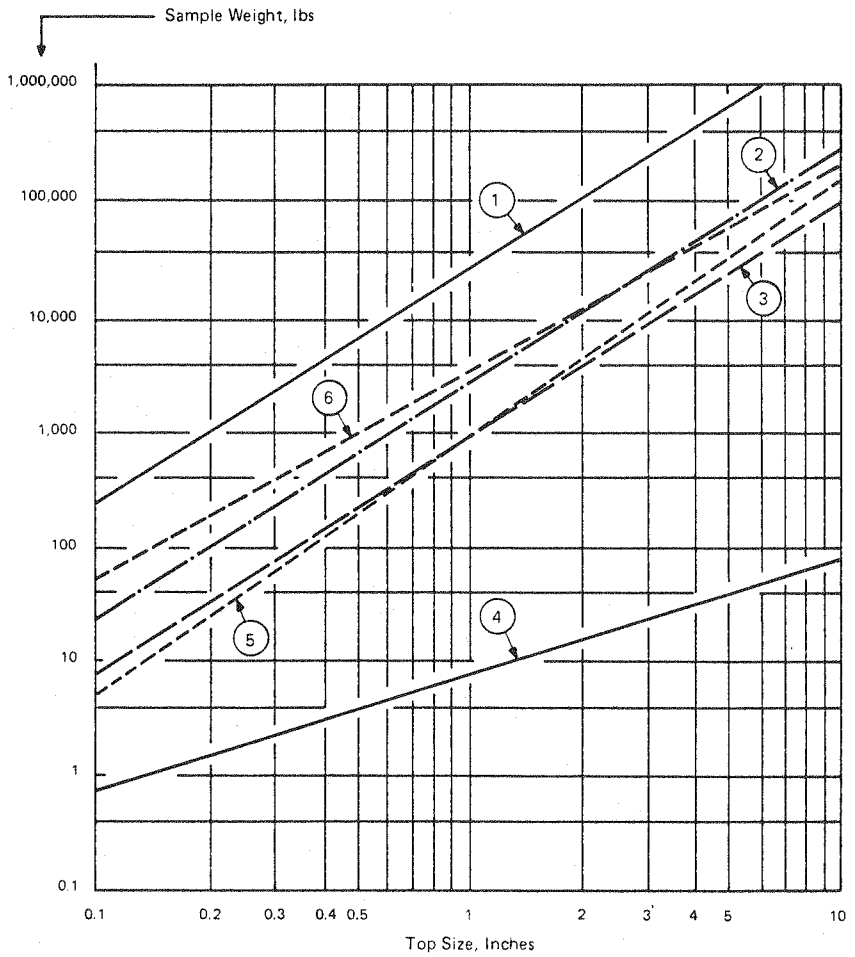
The focus of this presentation is practical factors in sampling, and discussion of theory will be concluded on a further practical point. To reiterate the basic theory of bulk material sampling, a representative sample is taken from the flowing stream by cutting through the stream uniformly and completely over a short time interval. Stated in another way, each sample increment is the entire stream for a short interval, in contrast to a portion of the stream over a longer interval. Fig. 2 is a diagram showing a traversing cutter with an opening of L inch moving through a falling stream and collecting the volume $w_c$ (abcd) according to the concepts of correct sampling. The cutter moves a constant speed, $u_c$ (in. per sec.).

The weight of sample in an increment taken in one traverse is

$$m = \frac{ML}{u_c}$$

where M is the lb./min. of feed material. Fig. 3 provides the equivalent diagram for a rotating scoop sampler. (Note the diameter of the cutter circle must be greater than the width of the falling stream by a factor of 1.3 or greater.) The sample increment taken with a rotating cutter at rotating rate $\omega$ and angle of scoop $\beta$, is

$$m = \frac{\beta M}{360\omega}$$




| NO. | ORE TYPE APPLICATION | REFERENCE |
|---|---|---|
| 1 | GOLD | RICHARDS |
| 2 | SILVER | RICHARDS |
| 3 | LOW GRADE, UNIFORM DISTRIBUTION | RICHARDS |
| 4 | BASE METALS, HIGH GRADE | DEMOND AND HALFERDAHL |
| 5 | BASE METALS, LOW GRADE AND HIGHLY VARIABLE COMPOSITION | DEMOND AND HALFERDAHL |
| 6 | AVERAGE GRADE, VARIABLE DISTRIBUTION | WOODBRIDGE |

**FIGURE 1. Sample weights for top size of stream.**

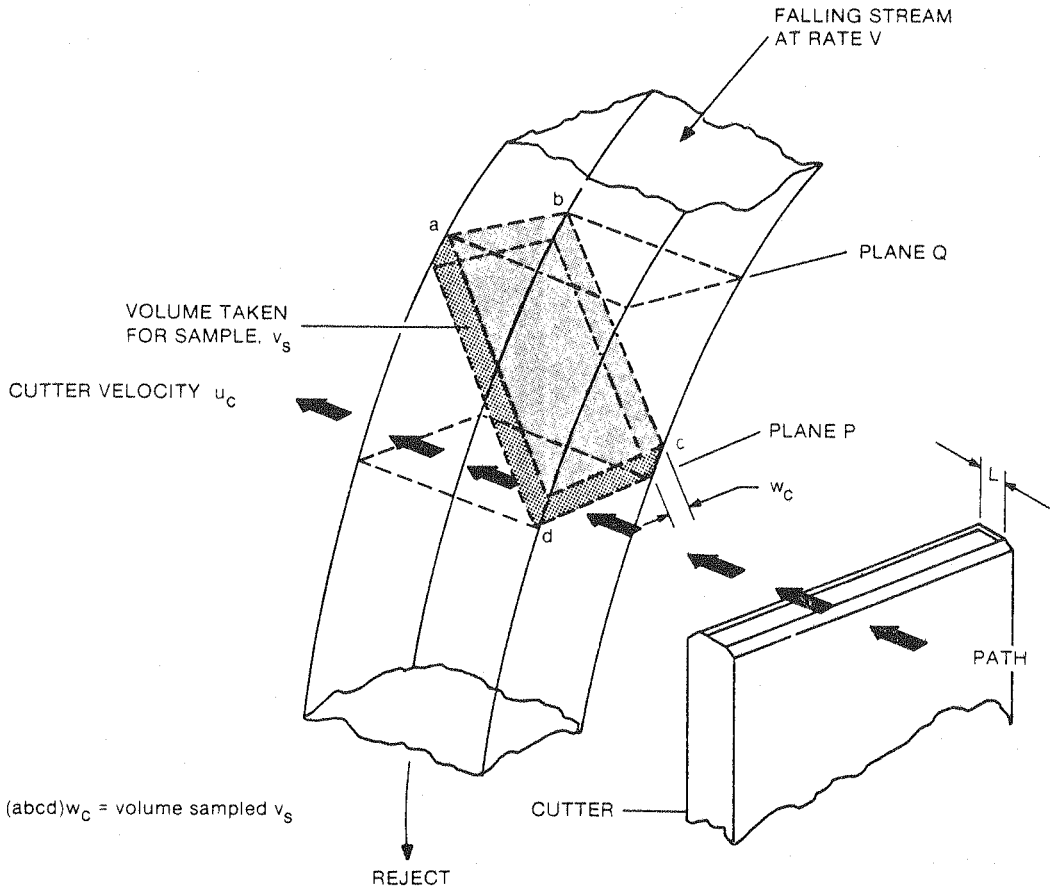



**FIGURE 2. Diagram of traversing cut.**

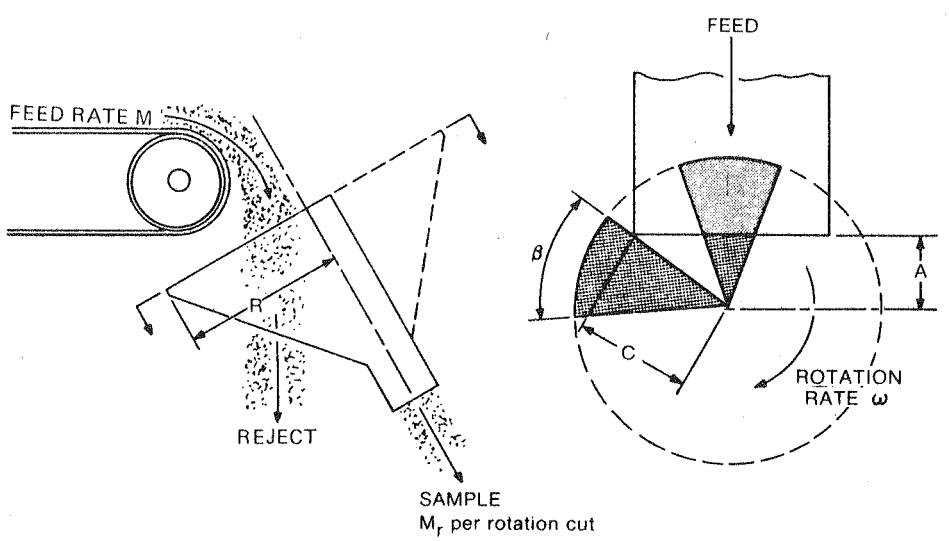



**FIGURE 3. Diagram of rotating cut.**

There are many other requirements for correct sampling to meet standards of the theory. Cutter opening is one important point—it should be at least 2.5 times the particle diameter for 95 percent passing, but no less than 3/8-inch. The higher the cutter speed the greater the ratio of particle size to cutter opening. A further theoretical factor is how frequently should a sample be taken from a falling stream of material. Precise determination of this parameter is an exacting task employing autocorrelation statistics which culminate in the "variogram" technique. Qualified experience indicates frequencies are optimum in the time range of 15-30 minutes. That is, no improvement in results on a statistical basis would occur with sampling at greater frequencies.

Usually cutting frequency is of academic concern in sampling precious metal ores, because increments are needed as rapidly as possible to collect the necessary statistical weight for a primary sample in the range of one to five percent of the stream, depending on the size of the crushed ore. A rotating scoop cutter for one percent sampling ratio would require 3.6 degree cutter opening rotating continuously. Remember that it is necessary to check the effective cutter opening at ⅔ of the cutter radius as shown in Fig. 3, to insure the effective width is suitable for the particle size of the stream.

## WHAT WILL INCREASE EQUIPMENT OPERATING RELIABILITY

The need for reliable operation is the primary practical consideration in sampling. Equipment reliability is as important as technical reliability of the sampling procedures. Selection of equipment to achieve high reliability for precious metal ore sampling poses a challenge because of high sampling ratios. Because conveyor belt speeds are usually low, a good choice of cutter design for the circumstances is the rotating scoop (PRL) sampler. The PRL unit illustrated in Fig. 4 has the attributes of simplicity, good discharge characteristics, and heavy duty construction. It operates continuously with great ease and accuracy for a belt narrower than 48 inches and speeds less than 300 ft. per minute. A further advantage is that it can be installed at a conveyor discharge with a low headroom layout, providing considerable elevation savings, by comparison to a traversing cutter arrangement where clearance below the belt must be provided for the cutter and collection hopper. A final point is that the PRL sampler can be economically enclosed in a compact dust sealed enclosure. On the basis of these advantages, a practical recommendation is to examine the PRL rotating cutter sampler as a first step in developing a reliable sampling configuration.

A traversing sample cutter is frequently dictated by operating factors for an ore sampling station. The cutter design must accommodate gravity discharge of solids by maintaining slopes of minimum 50-60 degrees, and should be provided with wear resistant sliding surfaces. The drive must be suitable for effectively continuous service in a dusty, abrasive environment. If continuous operation is employed, up to ten traverses per minute or about 5,000,000 operations per year will occur. Good results have been achieved with Poweroll™ belt and pulley cutter drive mechanism, illustrated in Fig. 5, for this demanding service. Chain and sprocket drives are subject to severe wear from dust abrasion at this operating frequency, resulting in erratic drive speed and incorrect sampling. Pneumatic drives suffer from similar wear difficulties, and are found to be incapable of driving the cutter at constant speeds in forward and reverse traverse strokes.

A multistage sampling system adds several parameters of complexity. Attention is needed to the details of design and equipment selection to maintain high standards of reliability for operation and sampling accuracy. With good design advice backed by experience, nearly any application can be handled in a satisfactory, economical manner.

## HOW NOT TO TAKE A SAMPLE

Practical points in sampling might suggest that short cuts can be offered in taking samples. Unfortunately, use of short cuts and misguided simplifications are frequently encountered, and while these short-cut procedures may have attractive features to reduce costs or make the equipment easier to use, the actual cost to the user in poor material accounting can far outweigh any apparent savings.

Errors in sampling usually are traced to failure to collect samples according to the precepts of Fig. 2 and Fig. 3. Grab samples, either in a manual or automatic mode clearly fail the test. However, for those cases of homogeneous material of fine, uniform grind, a sample taken anywhere from the flow will often be acceptably representative. There is no significant classification or segregation in terms of density or particulates in such circumstances. This is exceptional, noting that finely ground precious metal ores will categorically be outside the realm of this ideal because they are not homogeneous due to the density difference between gold or silver fines and rock matrix fines. Thererfore grab sampling for precious metal ores immediately falls into the category of bad sampling practice even for finely ground rock.

The wrong way to take a representative sample will be illustrated by an example of a poor technique that can be encountered from time to time in the field. The moving gate method is the bad practice to be described. As the moving gate passes through material falling from the belt, the cutter first enters the stream through the material pitched from the top of the load on the belt. This is the coarsest size because of classification of the fines to the bottom of the load. The cutter progresses through remainder of the load and is immediately reversed to move back to its original position. The sample taken by this method is rich in the coarse material and lean in the

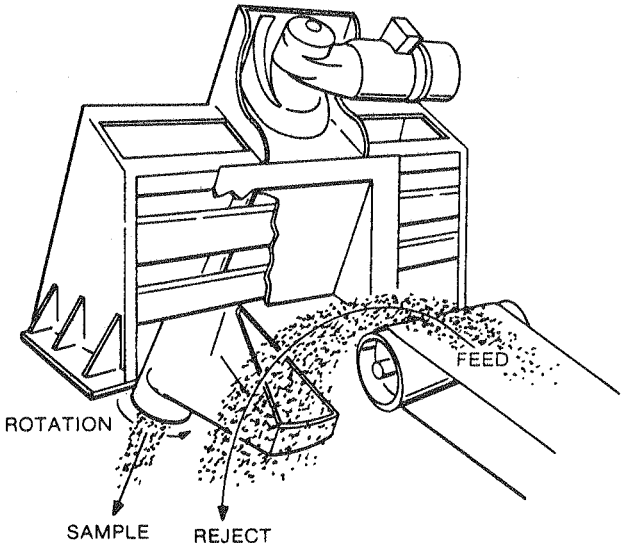



**FIGURE 4. Rotating scoop (PRL) sampler.**

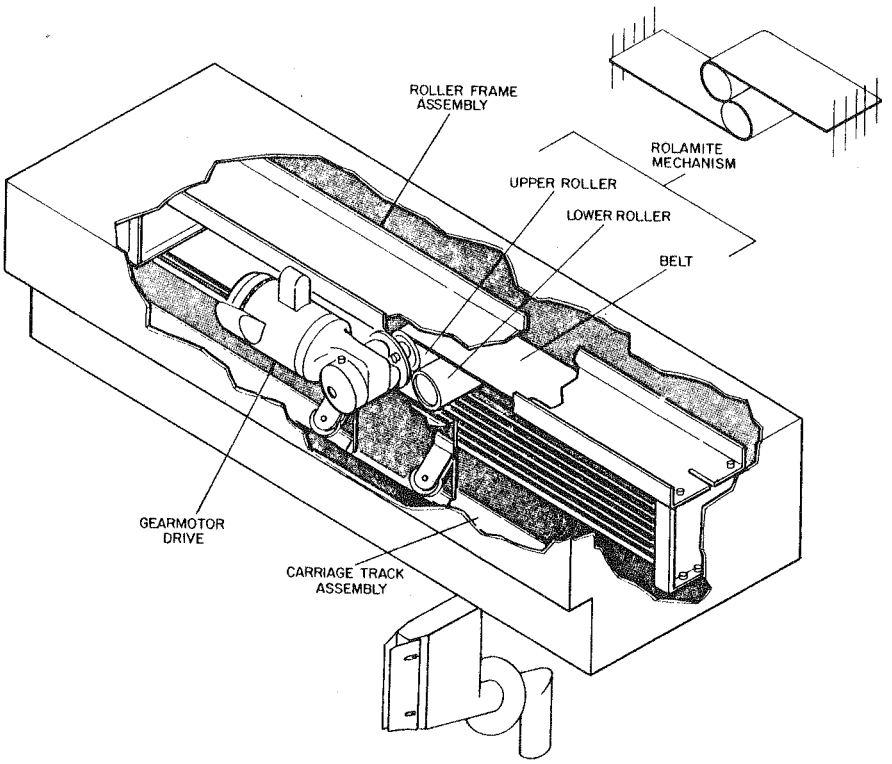
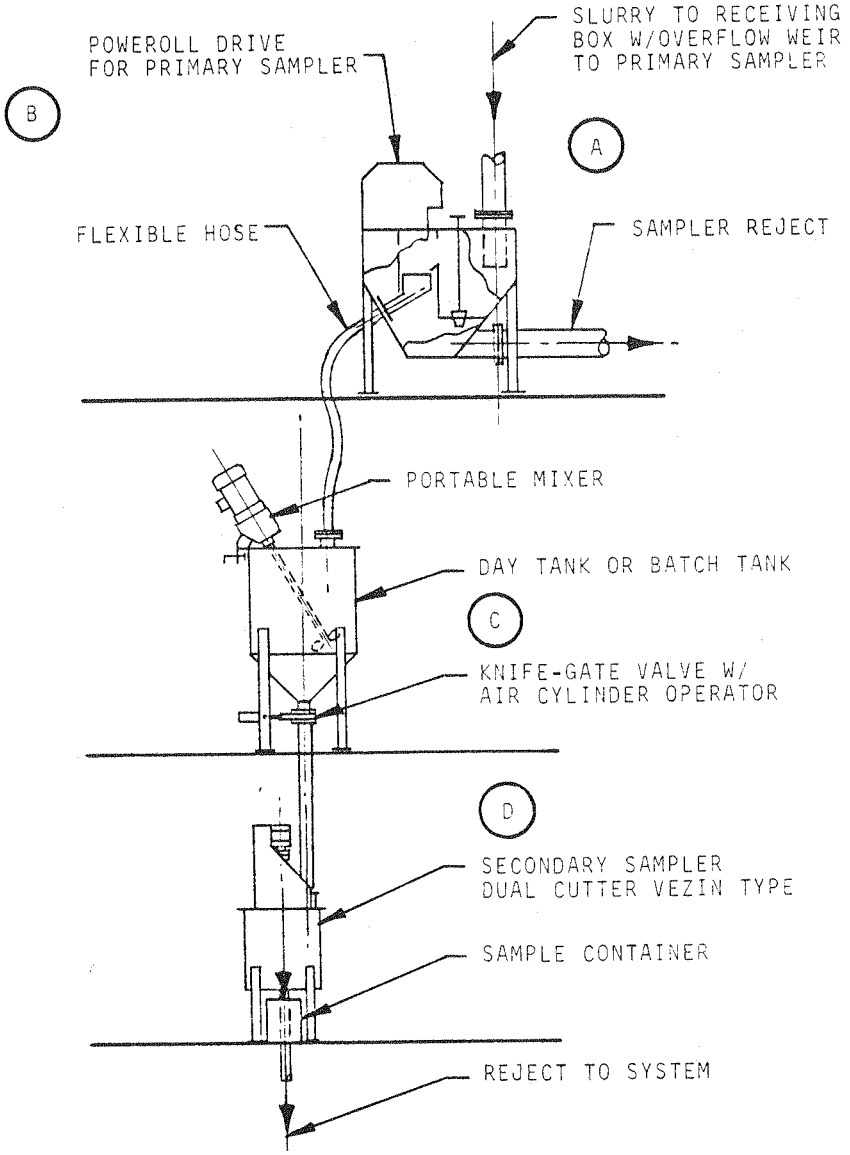



**FIGURE 5. Poweoll™ traversing sampler.**




SLURRY FLOW RATE IN LAUNDER A IS NOMINAL 1350 GALLONS PER MINUTE (5.11 CU.M./MIN.), BUT IS VARIABLE TO UNKNOWN DEGREE. THE CUTTER IS SET AT 1/4-IN. OPENING (6.3 MM). THE TRAVERSING SAMPLER CUTTER B MOVES AT 18-IN./SEC. (45.7 CM/SEC.) FOR 36-IN. (91.4 CM) COMPLETELY THROUGH THE STREAM, THEN PAUSES FOR 5 SEC. BEFORE BEGINNING THE NEXT STROKE. SLURRY TAKEN BY THE SAMPLER IS COLLECTED IN MIXED TANK C AND STORED FOR 8 HOURS. THE MEASURED VOLUME AT THE END OF THE PERIOD IS 1250 GAL. (4.73 CU.M.) AND THE CONTENTS ARE SAMPLED BY ROTARY SAMPLER D FOR LABORATORY ANALYSIS OF THE COMPOSITE. THE LABORATORY DETERMINATION OF PULP DENSITY GAVE 31.6 PERCENT, AND A SPECIFIC GRAVITY OF 1.22. THE MASS FLOW OF DRY SOLIDS DURING THE SPECIFIED PERIOD IS CALCULATED AS FOLLOWS:

$$\text{tons dry solids} = \left[\frac{(\text{gal. of sample})(\text{wt. fraction pulp density})(\text{sp. gr.})(\text{water lb./gal})}{\left(\frac{\text{cutter opening}}{\text{cutter speed}}\right)\quad(\text{lb./ton})}\right]\left(\frac{\text{traverse distance + dwell time}}{\text{cutter speed}}\right)$$

$$= \left[\frac{(1250)(.316)(1.22)(8.33)}{\frac{0.50}{18}\quad(2000)}\right]\left(\frac{36}{18}+5\right) = 505.8 \text{ tons}$$

**FIGURE 6. Proportional sampling.**

fines because the time of sampling the top material is greater than the time of sampling the fines portion at the far end of the stroke. Typically, crushed precious metal ores will generate fines richer in precious metal grains and flakes than the bulk, average material.

Another "automatic" sampling method subject to even larger errors is the crossbelt sampling method involving a scoop cutting through the load while carried on the conveyor belt. The fines left on the belt are not taken into the sample, of course. Cross-belt sampling leads to errors even more pronounced than the gate method.

Incorrect sampling procedures can be encounterd in various other embodiments, often times in the form of home designed units. Such may be mechanically viable, but suffer from fundamental problems such as inconsistent cutter speed, failure to take the entire stream, and assorted other failings.

## PROPORTIONAL SAMPLING— A PRACTICAL METHOD FOR MATERIAL BALANCES

The *Handbook of Mineral Dressing* section on sampling has a brief note on a practice employed by Homestake Mining Company termed "proportional sampling". It is applicable when the plant feed is ground to a water-ore pulp for extraction. The reference describes use of a constant speed rotating cutter to collect a slurry sample in a tank for a specified time period, which is then measured for volume. The accumulated slurry volume will be proportional to the total volume of flow if the sample extraction is accurate and reproducible throughout the sampling time period. Subsequently, the sample is split to a laboratory size sample for analysis of pulp density, specific gravity, and metal values. The measurements allow computation of tons of dry solids. Fig. 6 provides a flow diagram and sample calculation of the tonnage represented by a proportional sample, as applied to high volume flow of plant feed slurry and tails slurry, to compute material balances.

Proportional sampling is practical when using a reliable traversing sampling drive capable of good speed control and timing. The drive must be suitable for operating cycles as frequent as 30 seconds. It must be assumed that cutter is provided with an accurately set opening and good edges. By comparison to a dry solids sampling system and belt weigh scale systems, the cost of proportional sampling is very favorable. It would be noted that accuracy of material balances using proportional sampling is also excellent, considering the potential for substantial weight errors due to the weigh belt instrumental and mechanical factors, and the possible variation of moisture content of the ore.

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# SAMPLING AND ORE RESERVE ESTIMATION FOR THE ORTIZ GOLD DEPOSIT, NEW MEXICO

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## INTRODUCTION

The Ortiz gold deposit was brought to production as an open pit/heap leach operation early in 1980 by Gold Fields Mining Corporation. The geology of the deposit has been described by Lindqvist (1980) and by Wright (In press). The deposit is located in a pipe-like body of brecciated Mesaverde quartzite probably formed by explosive volcanic activity in Oligocene times. Permeable areas of the breccia formed pathways for late volcanogenic hydrothermal fluids which caused alteration of feldspars in the arkosic quartzite to clay minerals, sericite and carbonates and precipitated calcite, pyrite and iron oxides together with minor gold, scheelite and copper sulfides. The auriferous zone is roughly elliptical in plan with a long axis of approximately 700 feet and a short axis of 400 feet.

## SOURCES OF SAMPLE DATA

The Ortiz gold deposit has been sampled by Gold Fields Mining Corporation (GFMC) and by previous operators. The types of sampling that have been used are summarized in Table 1. GFMC has used diamond drilling and underground drifting, cross-cutting and slabbing of pre-existing tunnels and, to a lesser extent, surface trenching. Previous operators have used diamond drilling, wagon drilling, and churn drilling to provide samples. GFMC drilling amounts to only one-half of the total footage drilled on the deposit and the reappraisal of the old data was an important step in the evaluation of the deposit. A typical section (Figure 1) shows the irregular spacing and directions of the drill holes.

**TABLE 1. Sampling methods used on the Ortiz deposit**

| Operator | Sample Type | No. of Holes | Feet | % of Total |
|---|---|---|---|---|
| Previous No. 1 | Vertical Diamond Drill | 14 | 3,213 | 11% |
| Previous No. 1 | Churn Drill | 24 | 5,502 | 19% |
| Previous No. 1 | Wagon Drill | 12 | 2,889 | 10% |
| Previous No. 2 | Vertical Diamond Drill | 2 | 1,404 | 5% |
| GFMC | Vertical & Inclined Diamond | 22 | 13,230 | 48% |
| GFMC | Tunnel Rounds | | 1,346 | 4.5% |
| GFMC | Slabbed Tunnels | | 695 | 2.5% |
| Total Drill Footage | | | | 26,328 |
| Total Underground Development | | | | 2,041 |
| Total Sampled Footage | | | | 28,369 |

## SAMPLE ACQUISITION, PREPARATION AND ASSAYING

The problems of sampling low grade gold deposits have long been recognized. With the recent increases in gold value and the introduction of low-cost heap leach technology there has been a marked reduction in the gold grade requirements for a viable mine. This interest in low grades has re-emphasized the tripartite problem of gold sampling. The sampling process can be considered in three distinct stages: firstly, sample acquisition or the means by which the sample is obtained from the deposit (diamond drilling, for example), secondly, sample preparation or the process used to reduce the acquired sample bulk down to a quantity suitable for analysis, and thirdly, the analytical process itself. At each of these stages errors are incurred and these errors can be monitored and controlled independently. The procedure used for each stage is determined as an exploration management decision.

## SAMPLE ACQUISITION

The methods used to obtain samples at Ortiz included core drilling, non-core drilling and tunnelling. The relative merits of one method over another are often difficult to determine and the overriding importance of obtaining a sample that is unbiased has tended to make operators prefer diamond core drilling over rotary drilling methods. The multiple uses to which cores may be put give it a further advantage over non-core drilling methods and certainly in the earlier stages of evaluation of a deposit the importance of obtaining geological information from core, as well as a sample for assay, can not be overemphasized. However, it is apparent from the interpretation of nugget variances that the larger diameter open hole methods gave greatly improved sampling performance relative to split NQ drill core. This improved sampling performance is also seen in the rotary blast hole samples obtained for production purposes. An early appraisal of different methods of sample acquisition can lead to more cost-effective evaluation sampling of a gold deposit such as Ortiz.

## SAMPLE PREPARATION

The process of reducing the acquired sample down to a quantity that is suitable for assay is part of the overall sampling process that is most easily controlled by project management. However, it is an often underrated source of a major part of the total sampling error. Adequate sample preparation routines are a prerequisite to reliable assay values.

The two principal types of samples obtained by GFMC at Ortiz were the ten foot split NQ drill core

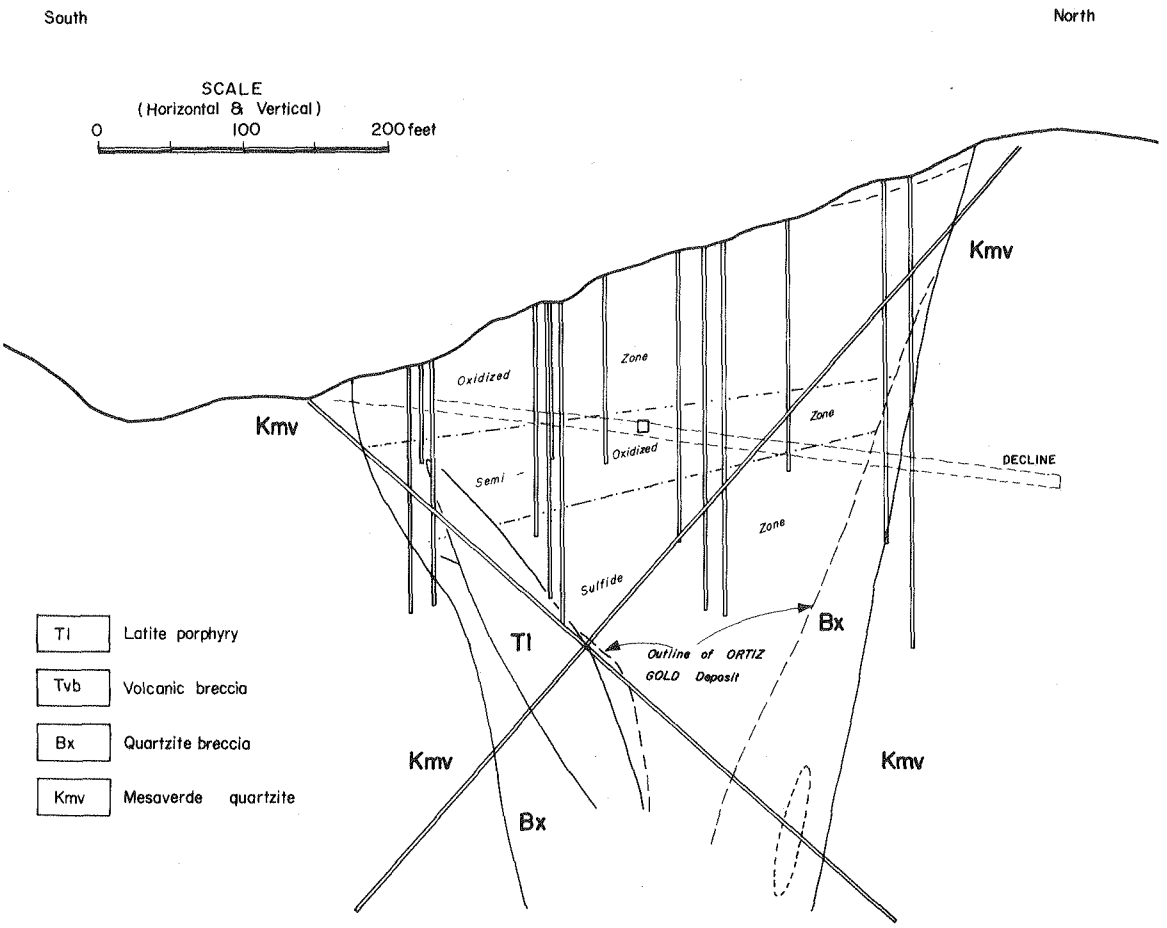



FIGURE 1.

samples consisting of about 10 pounds of material and the two types of sample obtained from each tunnel round, firstly a grab sample taken from the mucked round and secondly the 'confirmatory' sample taken by processing the entire thirty ton round. The drill core sample preparation routine is shown in flow sheet form (Figure 2). The additional processing steps required to reduce the thirty ton round to approximately 10 pounds for treatment by the same process as the core samples are shown as Figure 3. A combined sample reduction diagram (Figure 4) shows the entire process on a log-log plot with a comminution stage shown as a horizontal line and a split or subsampling stage shown as a vertical line. The aliquot weights actually assayed are shown for several different assay techniques.

Gy's equation, (Gy, 1978) was used to ensure that a reasonably equitable distribution of fundamental relative variances was incurred through the process. Gy's equation can be shown to give apparently incorrect results in the case of gold ores. The reason for this lies in the selection of the 'liberation size' of the gold. It was known from microscope studies of the ore that gold in excess of 100 microns occured. Using 100 microns as the liberation size gives an overestimate of the sub-sampling variances due to the presence of a large proportion of much smaller gold particles. Figure 5 shows the theoretical relative variances for the core sample preparation process calculated by Gy's equation. Clearly there is an overestimation of the errors incurred when compared with the nugget variances (Table 2). In practice it is very difficult to assess the 'effective' liberation size of gold in a deposit, or to determine the gold particle size distribution. Gy's formula was used firstly to ensure equitable distribution of errors through the process and secondly to give a rough 'order of magnitude' sense of how significant these errors were. Clearly there are practical constraints on what is economically feasible by way of establishing a preparation routine and it is important not to set up too elaborate a plant due to wrongly estimating the liberation factor. Finally, after some

**TABLE 2. Nugget variances from different sampling methods**

| Method of Sample Acquisition | Nugget Variances | Relative Standard Deviation |
|---|---|---|
| Bulk Sampled Tunnel Round | 0.020 | 14% |
| Grab Sampled Tunnel Round | 0.260 | 51% |
| GFMC Diamond Drilling | 0.270 | 52% |
| Previous Diamond Drilling | 0.260 | 51% |
| Churn Drilling | 0.050 | 22% |
| Percussion (Wagon Drilling) | 0.135 | 37% |
| Production Blast Holes | 0.170 | 41% |

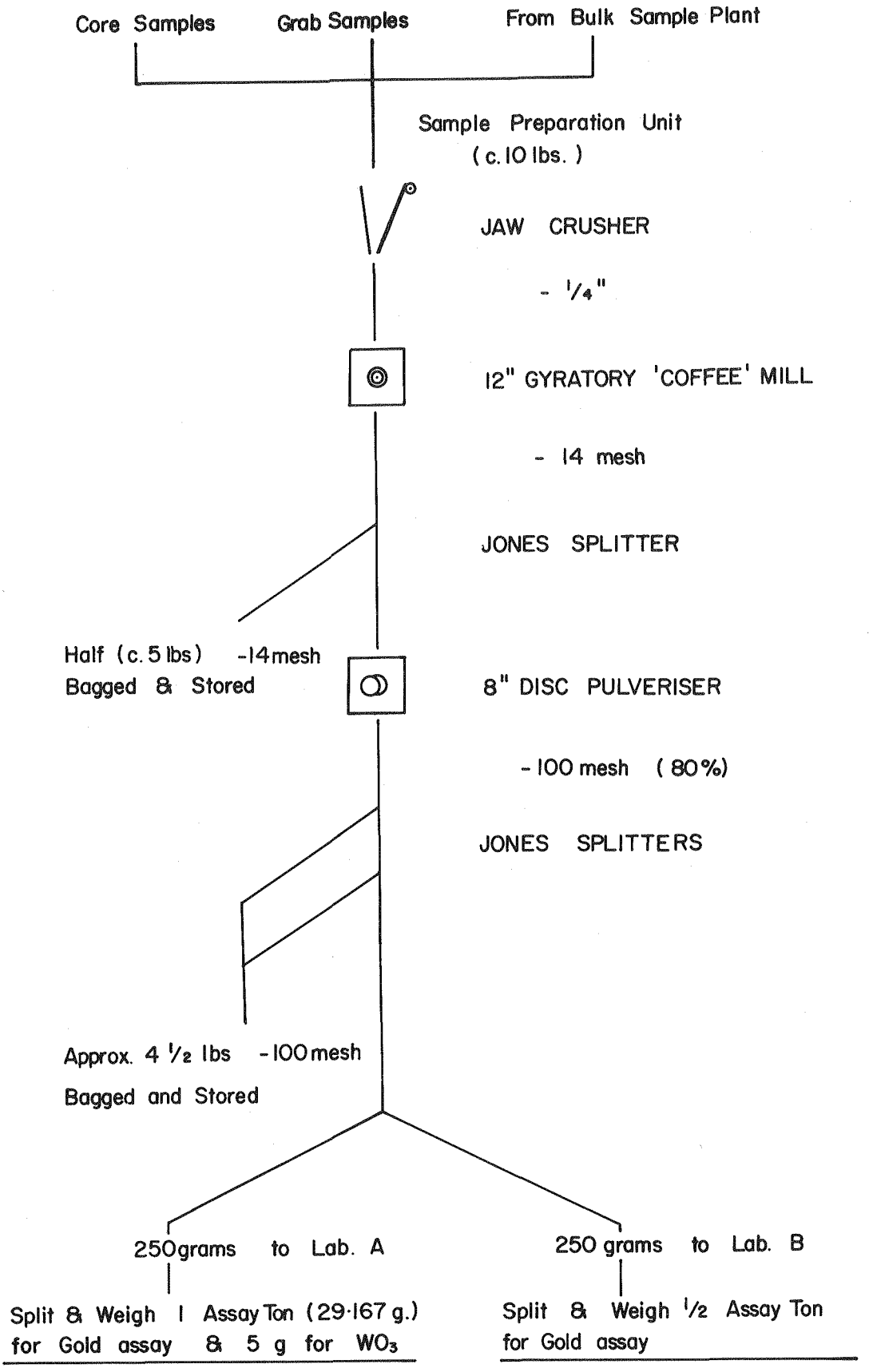



FIGURE 2. Flowsheet of sample preparation from cores, grab samples and final stage of bulk sampling.

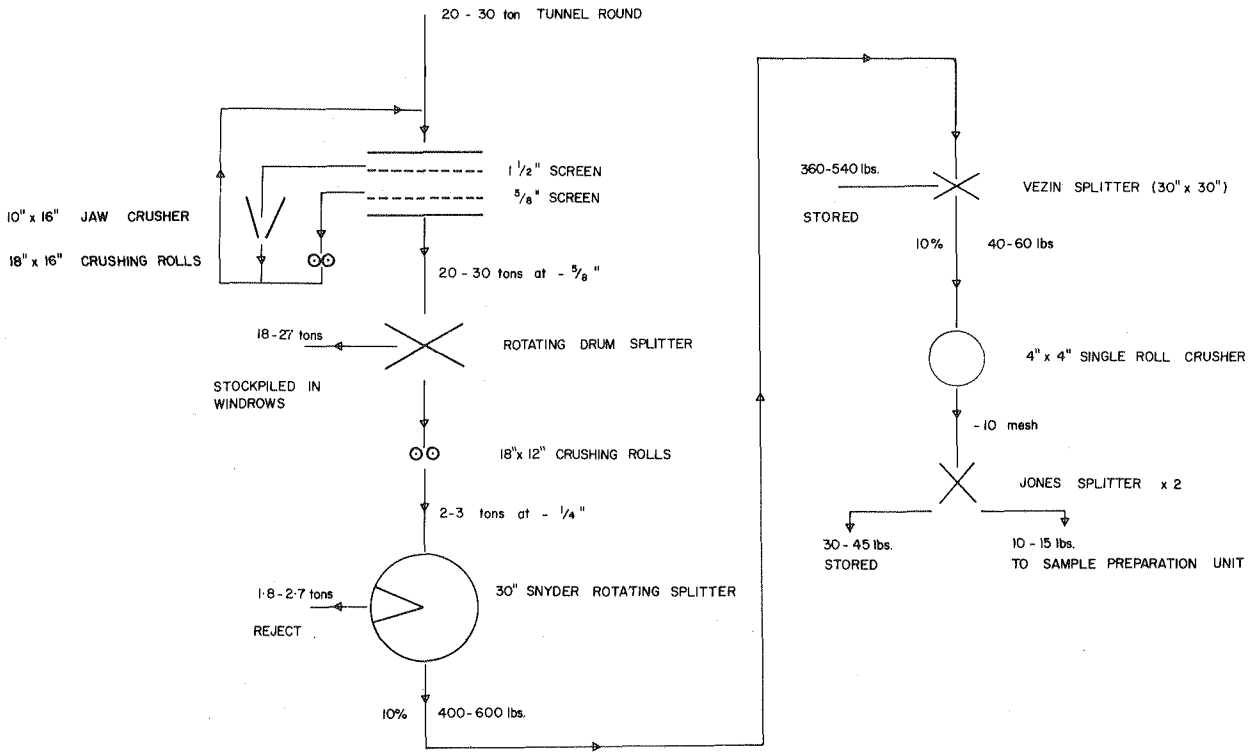



**FIGURE 3. Flowsheet of bulk sample preparation.**

exhaustive test work, it is possible to estimate an 'effective' liberation size that causes the formula to provide estimates of relative variance that are in good accord with the results obtained in practice. The problem lies not with the equation itself, but with the estimation of an empirical parameter that attempts in one number to summarize the complex gold particle size and shape distributions.

Sample preparation errors are often neglected, and in the case of low grade deposits, as much attention should be given to devising an adequate preparation routine as to selecting a drilling method or an assay laboratory.

## ASSAYING

The third and final stage in order to estimate a value for the sample is the analytical process itself. Several techniques are available for gold analysis; however, only two are widely accepted and atomic absorbtion and fire assay are likely to remain the predominant techniques for the next few years. Neutron activation on large samples is not available as a routine method in North America. Complex techniques on large samples, such as bottle amalgamation and cyanidation, are costly and time consuming, although they allow an accurate and precise determination of gold content on a large (for example, 10 lb.) sample.

During the Ortiz evaluation, fire assay was carried out routinely on two separate splits at two laboratories on every sample. Laboratory 'A' used a one assay ton (29.167g) charge and Laboratory 'B' used a one-half assay ton charge. Tungsten was known to occur as scheelite in significant quantities and this was determined only at Laboratory 'A' using the gravimetric cinchonine technique.

Analytical errors should be bias-free and can be expected to have a smaller random error component than the random error due to subsampling if reliable laboratories are used. Performance at different laboratories can vary markedly as can be seen from Table 3, where results on 25 replicate pulps assayed at 5 different laboratories are shown. The material had been very carefully pulverized to pass a 150 mesh screen, blended and split into replicates and presented to the laboratories with other samples. The actual value of the lot is 0.080 oz/T. The standard of reproducibility set by laboratory 'E' should be attainable by any good laboratory on samples as meticulously prepared as this standard batch. Laboratory 'D' represents an exceptionally good performance attained at a proprietory laboratory using replicated assays on aliquots in excess of 1 assay ton.

Scattergrams of replicate paired assays on 226 tunnel round samples are shown as Figures 6 and 7.

**TABLE 3. Replicate sample assays from five laboratories**

| Laboratory | Replicates | | | | | Mean | C.V. |
|---|---|---|---|---|---|---|---|
| C | 0.060 | 0.070 | 0.010 | 0.090 | 0.085 | 0.063 | 0.51 |
| D | 0.082 | 0.081 | 0.071 | 0.078 | 0.079 | 0.080 | 0.02 |
| E | 0.072 | 0.073 | 0.069 | 0.077 | 0.073 | 0.073 | 0.04 |
| F | 0.128 | 0.177 | 0.111 | 0.117 | 0.111 | 0.117 | 0.06 |
| G | 0.066 | 0.068 | 0.071 | 0.061 | 0.099 | 0.073 | 0.21 |

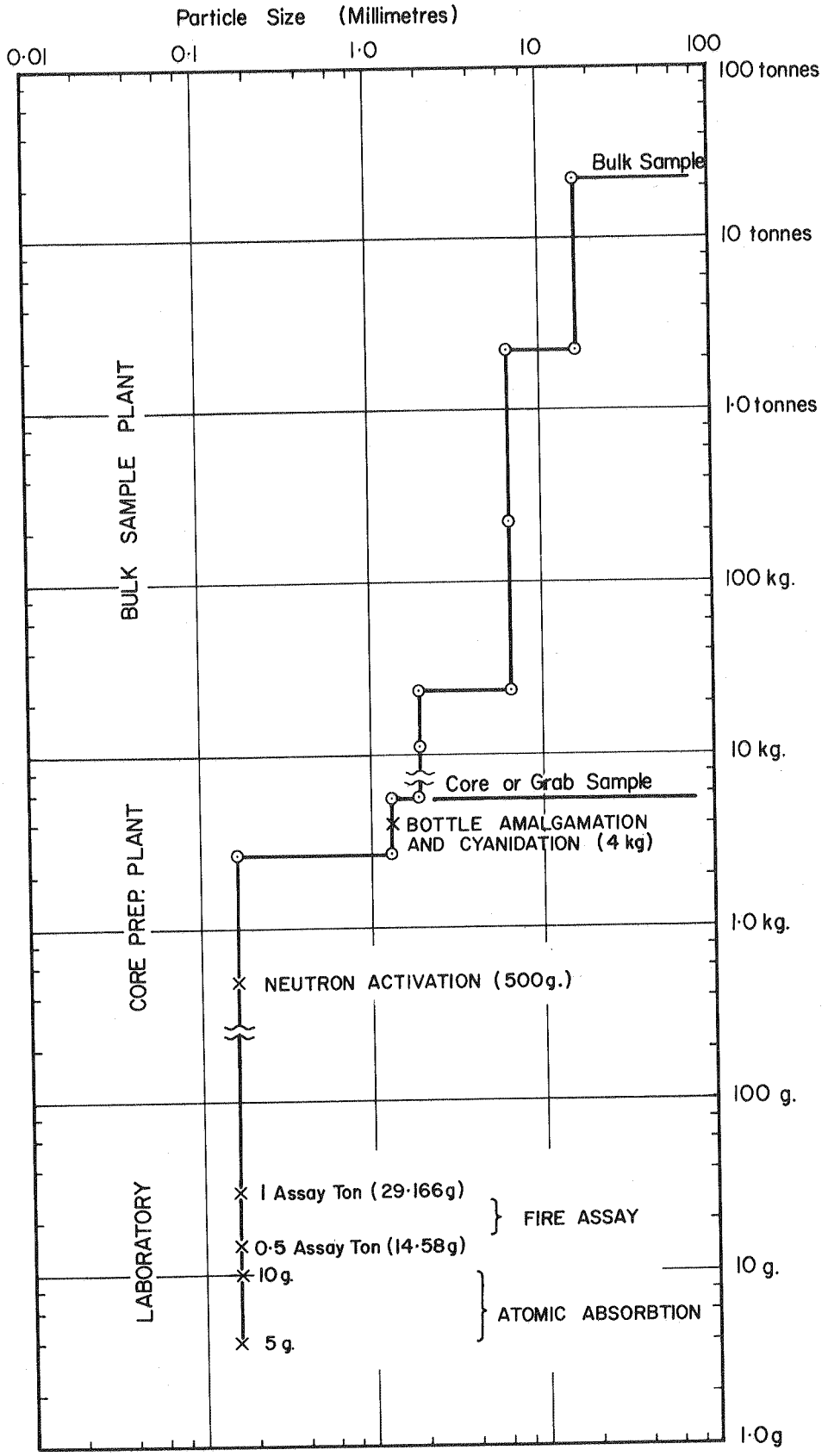

Particle Size (Millimetres)
0·01
0·1
1·0
10
100
100 tonnes
10 tonnes
1·0 tonnes
100 kg.
10 kg.
1·0 kg.
100 g.
10 g.
1·0 g
BULK SAMPLE PLANT
CORE PREP. PLANT
LABORATORY
Bulk Sample
Core or Grab Sample
BOTTLE AMALGAMATION AND CYANIDATION (4 kg)
NEUTRON ACTIVATION (500g.)
1 Assay Ton (29·166g)
0·5 Assay Ton (14·58g)
10g.
5 g.
FIRE ASSAY
ATOMIC ABSORBTION

FIGURE 4.

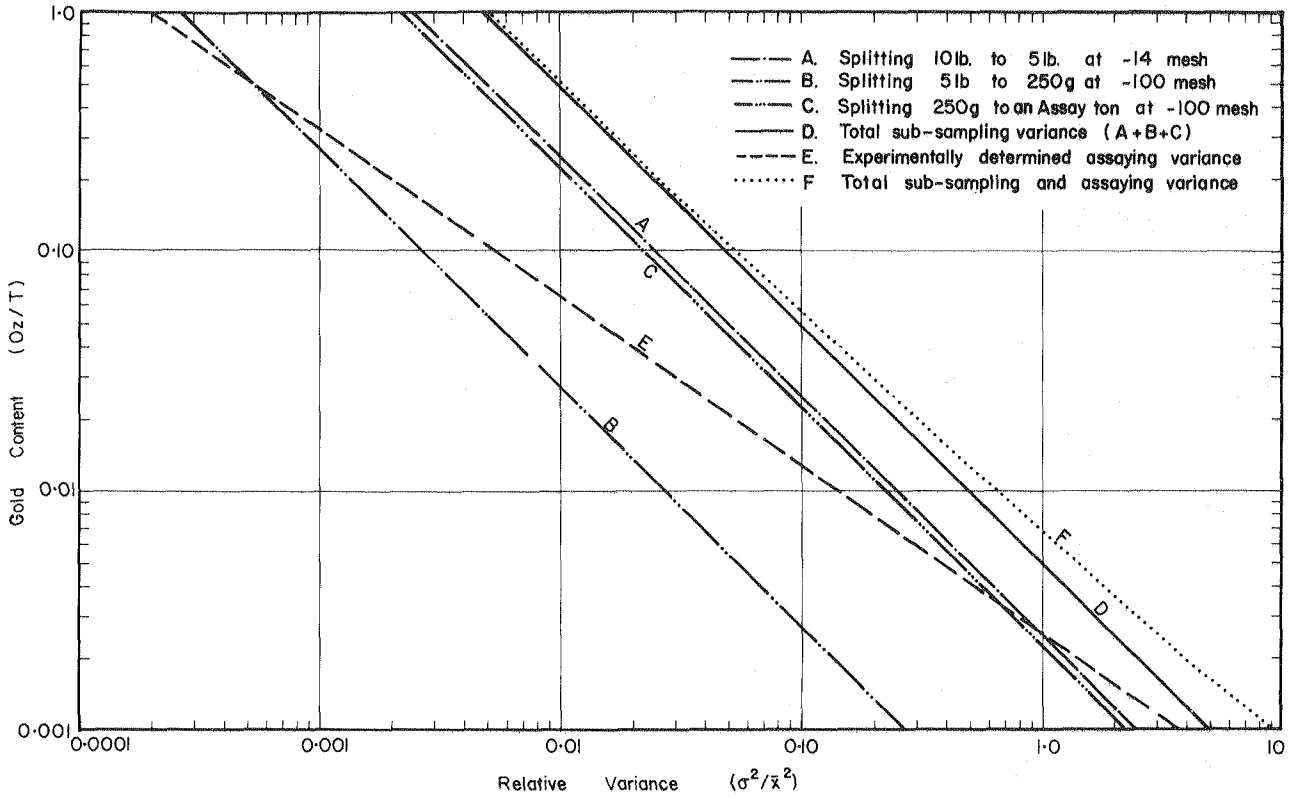



**FIGURE 5. Plot of the relative variances due to assaying and to each sub-sampling stage for a range of gold values.**

A scattergram of 'confirmatory' assays against the grab values for each round shows both a very substantial random error and a bias component. Figure 7, which shows the paired replicate assays carried out on each round confirmatory sample at the two laboratories, shows no significant bias component and a considerable improvement in the reproducibility with the correlation coefficient at 0.87 in contrast to 0.74.

The 15 paired tungsten assays shown on Figure 8 show excellent precision with a correlation coefficient of 0.99 but are very seriously biased due to the use of an inappropriate assay technique at Laboratory 'A' which effectively failed to report the lowest 0.02% $WO_3$ thereby introducing a significant negative bias. With as good precision as is shown in this case it is possible to apply a correction using the regression equation for laboratory 'C' given laboratory 'A' (Y on X) assuming that 0.02% is below any critical level as any value reported by laboratory 'A' as zero is in fact indeterminate in the range 0 to 0.02% $WO_3$.

The need for continual monitoring of laboratory performance by both internal and interlaboratory checks should be apparent. It is clear from the results shown in Table 3 and from the scattergram shown in Figure 8, that assaying can provide serious sources of both random and bias errors.

## GEOSTATISTICAL APPRAISAL OF THE DATA

The problems discussed in the previous sections are generally of a nature that can be resolved by classical statistical techniques in conjuction with a thorough understanding of the sampling, sub-sampling and analytical methods employed, and of the geology of the deposit. In order to fully appraise the sample data it is necessary to examine the distribution of values and to assess the effects of, for example, sample volume on distribution shape and to compare and contrast the data obtained by different sampling methods to determine whether any particular technique was subject to bias and should therefore either not be used, or be modified in some way prior to use.

Histograms are shown (Figure 9) for some of the 10 foot split NQ drill core samples and for the tunnel round confirmatory assays. The very highly skewed nature of the core histogram with a very low mode and extended range is immediately apparent. The tunnel round histogram shows a much less skewed form, a smaller range and generally rather different shape. It is apparent that assumptions regarding the permanence of the approximate log-normality shown in the core histogram would lead to erroneous conclusions if extrapolated even to thirty ton units.

Variograms were constructed for each sampling method employed and for several discrete directions through the deposit. The experimental variograms were subject to a marked proportional effect and use of the relative variogram was found to give tractible variograms that could be modelled on a consistent basis.

The variogram values are one-half of the mean squared differences between sample pair values grouped according to distance and direction. The

## SCATTERGRAMS

## SUMMARY STATISTICS

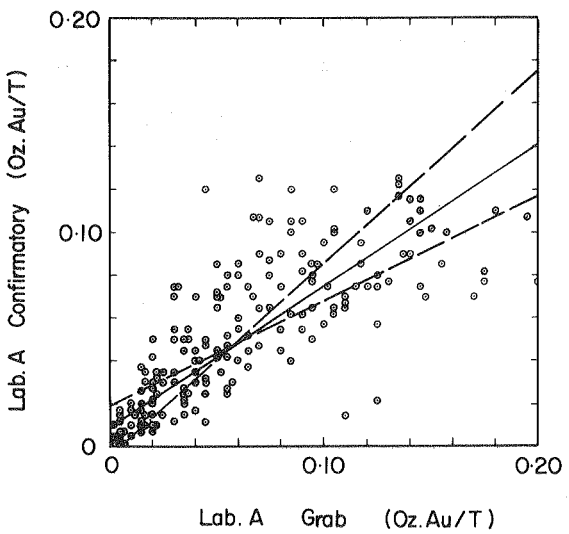



FIGURE 6.

| | | | | | | |
|---|---|---|---|---|---|---|
| n | = | 226 | | | | |
| r | = | 0·74 | | | | |
| $r^2$ | = | 55% | | | | |
| $\bar{y}$ | = | 0·0470 | | $C_Y$ | = | 0·76 |
| $\bar{x}$ | = | 0·0577 | | $C_X$ | = | 0·93 |
| Y on X : | Y = | 0·492 | X | + | | 0·019 |
| X on Y : | Y = | 0·898 | X | + | | 0·006 |
| RM. Axis : | Y = | 0·666 | X | + | | 0·009 |

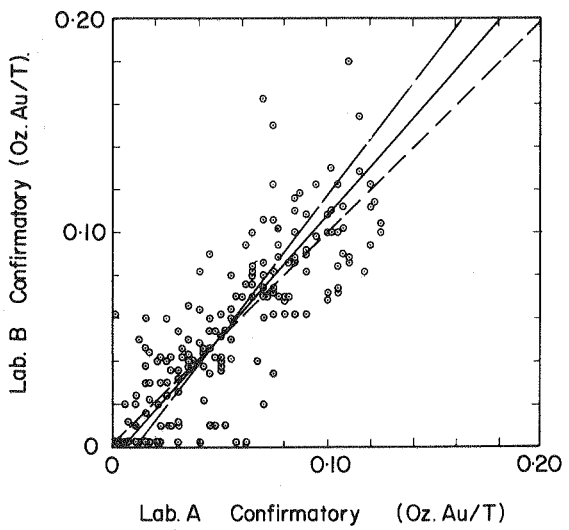



FIGURE 7.

| | | | | | | |
|---|---|---|---|---|---|---|
| n | = | 226 | | | | |
| r | = | 0·87 | | | | |
| $r^2$ | = | 75% | | | | |
| $\bar{y}$ | = | 0·0483 | | $C_Y$ | = | 0·84 |
| $\bar{x}$ | = | 0·0470 | | $C_X$ | = | 0·76 |
| Y on X : | Y = | 0·982 | X | + | | 0·002 |
| X on Y : | Y = | 1·311 | X | - | | 0 013 |
| RM. Axis | Y = | 1·135 | X | - | | 0·005 |

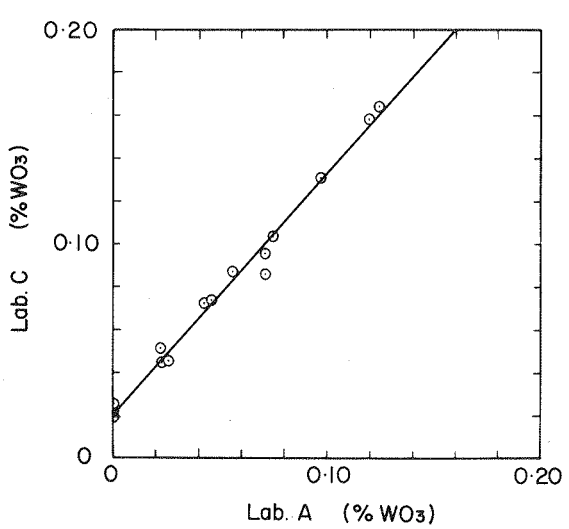
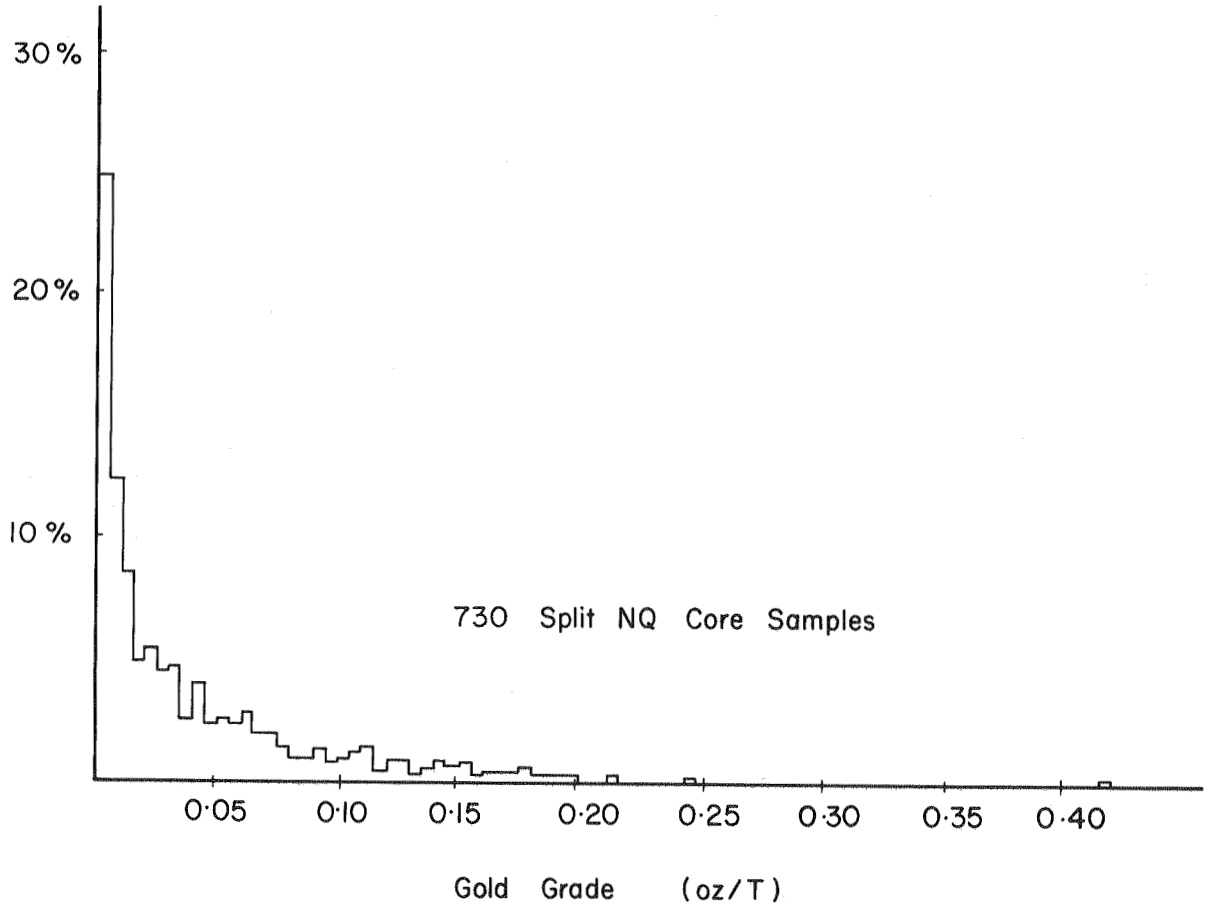



FIGURE 8.

| | | | | | | |
|---|---|---|---|---|---|---|
| n | = | 15 | | | | |
| r | = | 0·99 | | | | |
| $r^2$ | = | 99% | | | | |
| $\bar{y}$ | = | 0·078 | | $C_Y$ | = | 0·59 |
| $\bar{x}$ | = | 0·052 | | $C_X$ | = | 0·80 |
| Y on X : | Y = | 1·121 | X | + | | 0·021 |
| X on Y : | Y = | 1·136 | X | + | | 0·019 |
| RM. Axis | Y = | 1·128 | X | + | | 0 020 |


30 %
20 %
10 %
730 Split NQ Core Samples
0·05
0·10
0·15
0·20
0·25
0·30
0·35
0·40
Gold Grade (oz/T)

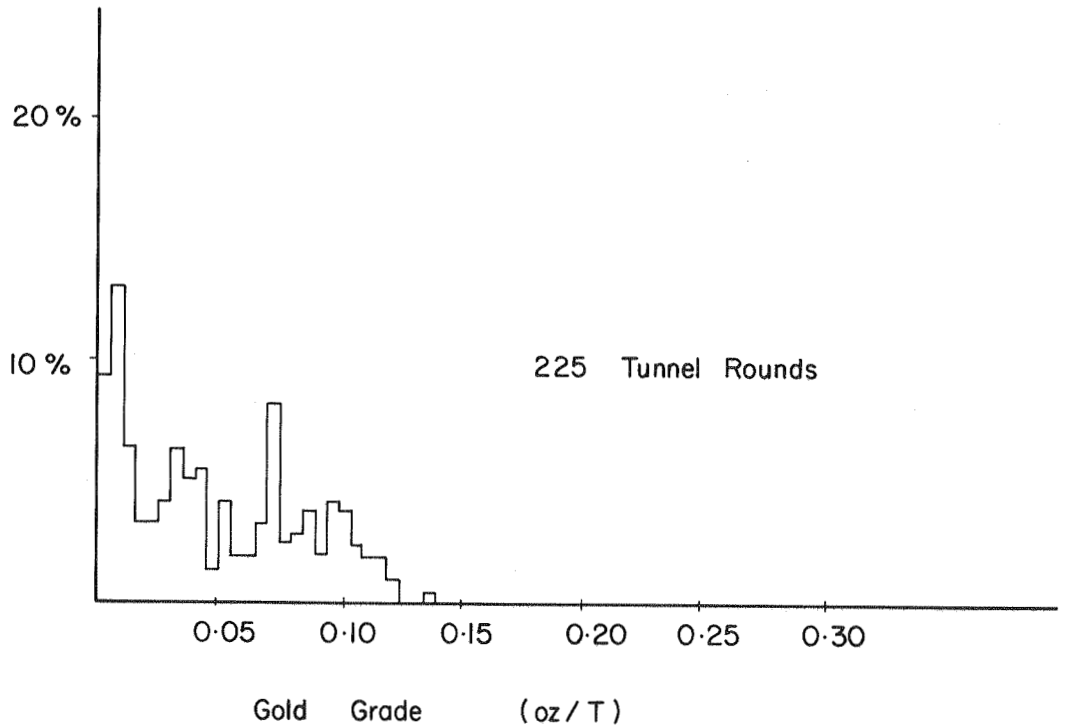

20 %
10 %
225 Tunnel Rounds
0·05
0·10
0·15
0·20
0·25
0·30
Gold Grade (oz/T)

FIGURE 9.

values are plotted graphically with the separation (termed 'lag' and symbolized 'h') as the abscissa and one-half the mean-square successive difference (termed 'gamma (h)' and symbolized '$\gamma$ (h)') as the ordinate. The resulting graph provides an experimental variogram that is a spatially related analogue of the variance of classical statistics. The variogram quantifies the traditional idea of 'area of influence' and embodies the widely believed (and usually correct) concept that two geological samples taken close together are more likely to be similar than two that are widely separated.

## VARIOGRAPHY

Generalized relative variograms are shown for each of the major sampling techniques employed on the property (Figures 10 and 11), including those of the previous operators. The variograms all show consistent forms with increasing gamma (h) as the lag increases, with the slope of the curve flattening to a sill value. The 'y' axis intercept obtained by projecting back to give an estimated ordinate value at lag zero is termed the nugget variance and is a measure of the sum of the relative variances due to analysis, sample preparation and acquisition of the sample from the deposit. In the case of Ortiz, the nugget variances estimated from the variograms provide an excellent means of assessing overall sampling performance of each sampling method. The nugget variances and the equivalent relative standard deviations (as a percentage) are given in Table 2 for different techniques including the rotary drill hole sampling being done for production purposes.

The variograms are well modelled by three nested spherical schemes using different nugget variance for each method. The variogram is almost isotropic as is shown by the cross-cut (approx E-W) and the decline (approx N-S) variograms. The model parameters are given in Table 4. The significance of the nugget variance can be more fully appreciated by considering the three cross-cut variograms (Figure 11). In each case the parent sample lot is the same—a thirty ton tunnel round—only the sample preparation process differs. The grab samples show a poor performance (relative standard deviation of 51%), whereas the fully processed confirmatory samples from the same rounds give the best samples obtained (relative standard deviation 16%). The split assayed at laboratory 'B' using one-half assay ton as opposed to one assay ton shows a very slightly poorer performance (relative standard deviation 20%). In practice the mean value for the paired confirmatory assays was used and the variogram for the meaned cross-cut pairs is shown with the fitted nested spherical model as Figure 12. As additional confirmation of the quality of the model, Figure 13 shows the experimental variogram derived from forty production blast hole samples with the same model plotted, differing from the cross-cut model only in the nugget variance.

**TABLE 4. Parameters of the three nested spherical schemes of the modelled variogram**

| Scheme | Range (a) | Sill (c) |
|---|---|---|
| 1 | 15 | 0.050 |
| 2 | 55 | 0.170 |
| 3 | 250 | 0.250 |

## ORE RESERVE ESTIMATION

At the conclusion of the exploration phase gold reserves were estimated by a plan polygon method, a cross-sectional method and by a rolling mean method. Results were computed for a range of cut-offs and the methods were generally in good agreement although the rolling mean technique was consistently lower in its contained gold estimate than the other two methods. At this time GFMC did not carry out a geostatistical ore reserve estimation, but the estimation errors applicable to the rolling mean reserve estimate were determined and found to give a relative percentage standard deviation of 7% for grade and 6% for tonnage.

Following these preliminary reserve estimates further geostatistical analysis was carried out (GFMC had access to a computer only after the preliminary estimates had been made) and the variograms described above were developed. A number of pit design parameters, such as bench height, had been provisionally determined and a reserve estimate was required that could be used for more detailed planning and that was more closely tied to such measures as bench height.

As the geostatistical characteristics of the mineralization were well established it was decided to use a block kriging technique using basic block units of 50′ x 50′ in plan and 25′ vertically (corresponding to the 25′ bench height proposed). These 5000 ton units would form the final estimate about which the pit would be designed and on which long term (for example, one year) scheduling would be planned.

The data was organized into bench height composites in the case of drill holes and into rather smaller composites giving a similar nugget variance as the bench height composite variogram in the case of tunnel rounds. These composites were assigned the weighted (by sample length) average grade and given the cartesian coordinates of the composite centroid to form the sample data base for the kriging estimate.

Kriging is the general name for the geostatistical reserve estimation technique that assigns weighting factors to selected samples in order to minimize the estimation variance incurred in estimating the expected grade of a specific point or block. There are now many forms of kringing and the more straightforward kriging techniques are well documented and the validity of the method is supported by published case histories (Journel and Huijbregts, 1978).

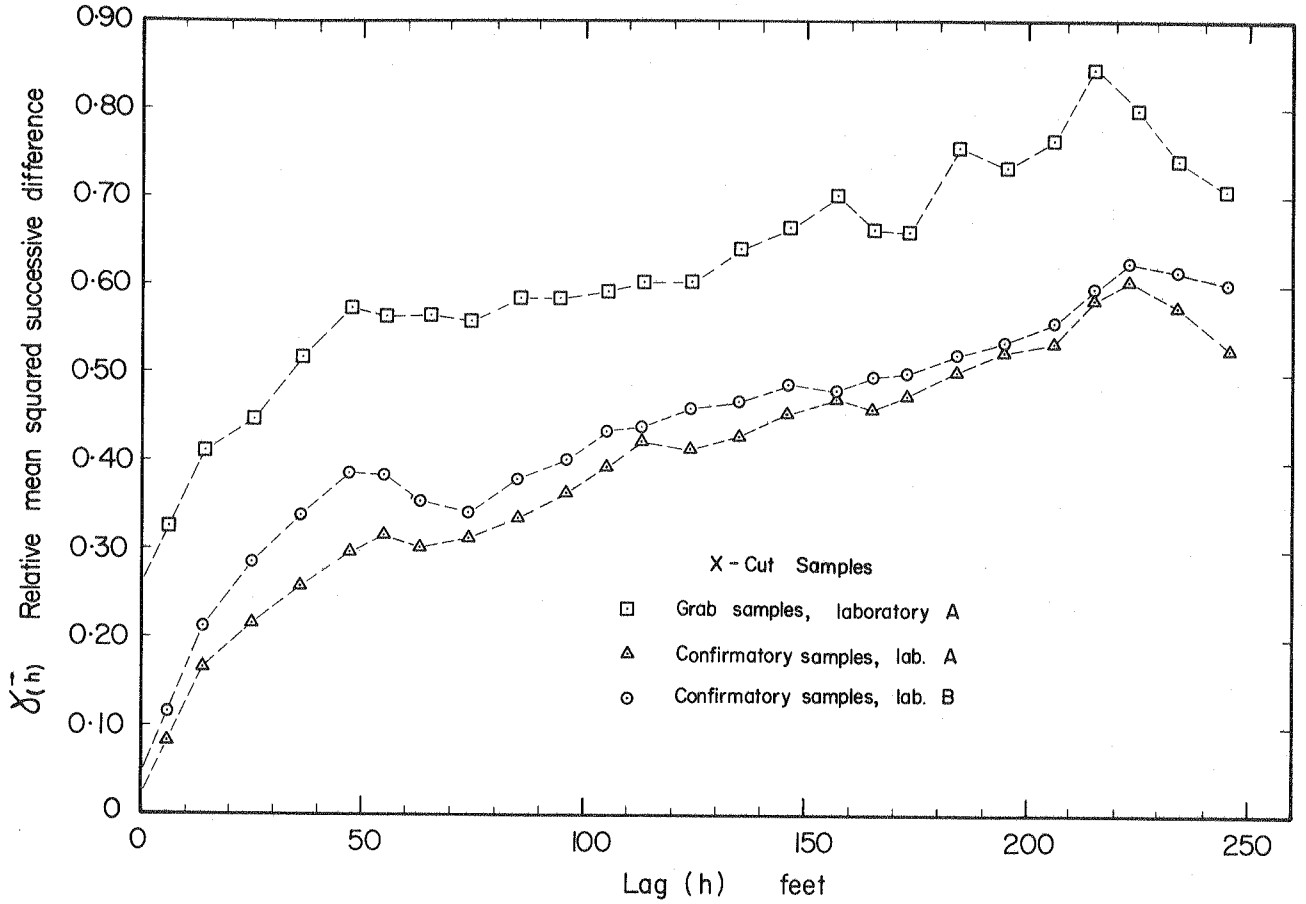



FIGURE 10.

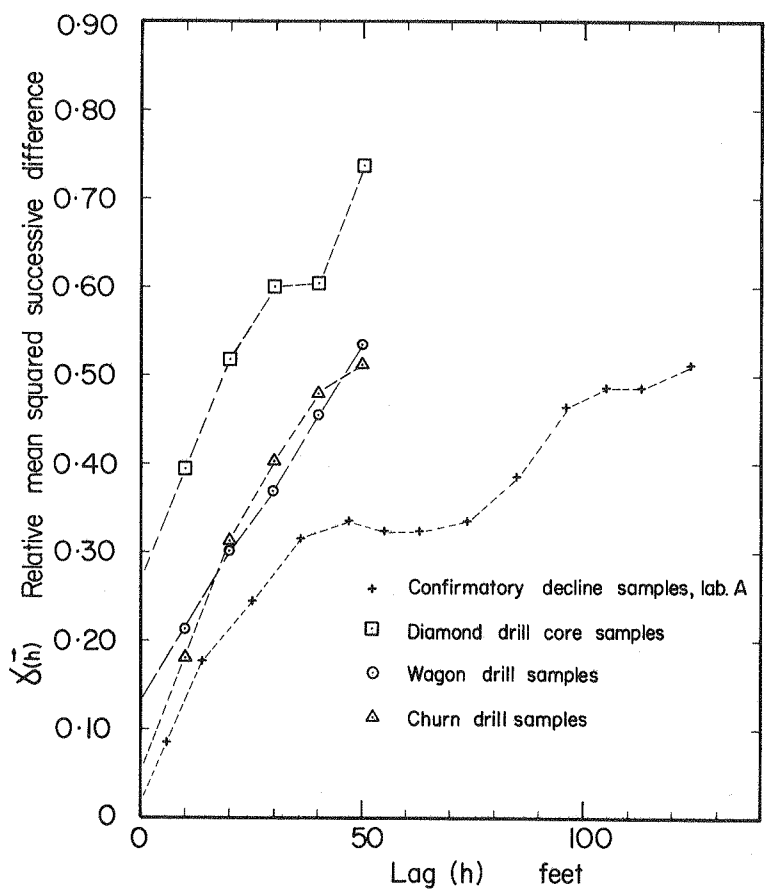



FIGURE 11. Relative variograms for different sampling methods and different directions.

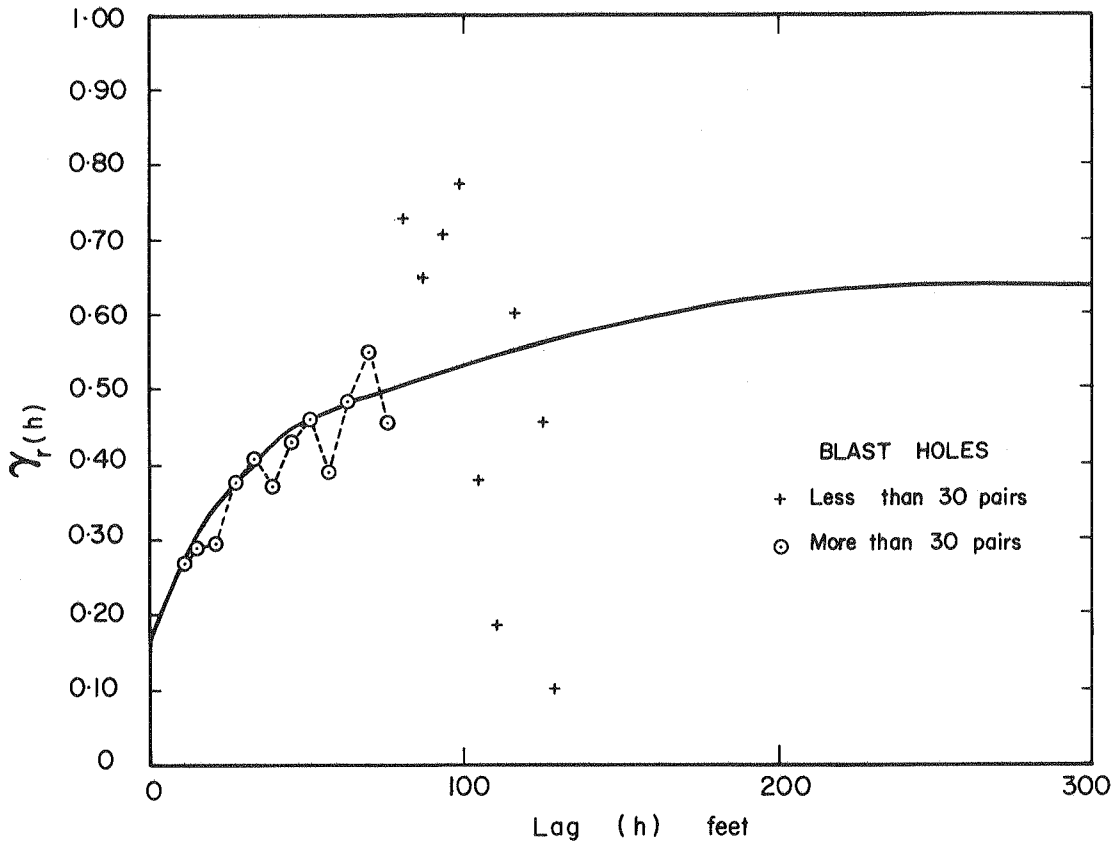



FIGURE 12.

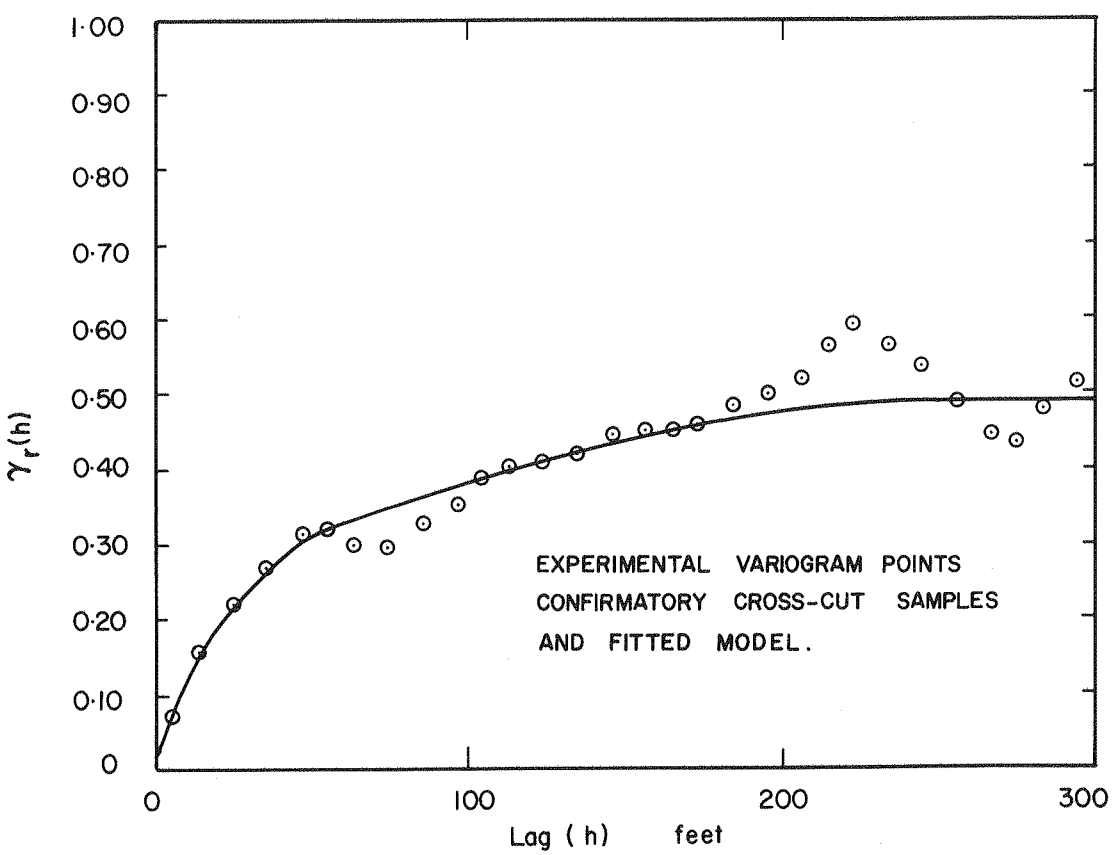



FIGURE 13.

Block kriging as used in the Ortiz evaluation is a form of 'ordinary kriging'. It has the advantages of taking into full account the known spatial correlations within the deposit, as portrayed in the variogram, of making use of all of the irregularly spaced data and of providing estimates on units that have the same geometry and that are of useful size to the engineers designing the pit. In addition the estimates have, by definition, minimized estimation variances and the variance is calculated for each block in addition to grade.

In essence, for each block to be estimated the block is 'simulated' by a series of discrete points within the block and those sample composites within a certain maximum distance of the centroid of the block are selected to be used in the calculation. The spatial relationship between each selected composite and every other selected composite is calculated from the variogram function and a matrix of sample to sample covariances is formed. Likewise, the matrix of selected composite-to-block average covariances is established, thus every spatial relationship between each selected sample and every other selected sample, and between the block and all of the selected samples, is included in the system. Constraints are added to the system to ensure that the weighting factors to be calculated firstly sum to unity and secondly provide minimum variance. The system of simultaneous equations so formed is solved for the unknown kriging weights to be applied to the sample values to give the grade estimate and to the sample to block covariances in order to drive the estimation variance. The covariance of a point within the block (the 'F' function) is also required in order to determine the actual estimation variance and this was determined once for all the 5000 ton blocks. Examples of basic kriging problems can be found in Rendu (1978) or David (1977). A computer is required in order to construct the kriging equations and to solve them. The model variogram function is essential to the establishment of the covariances that form the equations.

Block grade estimation variances were generally high, ranging upward from relative standard deviations of 16.7%. The estimation variances are used in forward planning to determine the degree of grade fluctuation that might be anticipated. Comparison between averaged production blast hole sample grade and corresponding kriged block grade estimates appear to be unbiased and within predicted limits.

Comparisons between the preliminary estimates by more traditional methods is not straightforward as some further drilling had added significant low grade tonnage which is included in the kriged estimate but not in the others. In general, at low grade cut-offs (substantially less than the overall mean grade of the deposit), the kriged grade is very slightly lower and the tonnage is slightly higher, but as the cut-off grade approaches the deposit mean grade then the kriged estimate tends to show a lesser tonnage (15% less than plan polygon) at a similar (3% higher) grade. As cut-off grade increases the discrepancies become more and more marked.

## CONCLUSIONS

Geostatistics has proved to be an important tool in both the evaluation and the operation of the Ortiz Mine. The Ortiz deposit is characterized by a variogram model so consistent that features of the different sampling techniques are revealed by the nugget variances. Sampling programs for deposits such as Ortiz must be carefully designed with particular attention to adequate sample preparation. Analytical performance at any laboratory requires constant checking for both reproducibility and particularly for bias. Large diameter open hole drilling methods would provide an excellent means of cost-effective drilling in a deposit with the same characteristics as Ortiz. However, no two deposits are the same and there are real dangers in extrapolating from lessons learned at Ortiz to another deposit without thorough verification and testing.

## APPENDIX

### GY'S EQUATION:

$$\sigma^2 = (1 - \frac{m}{M})\frac{d^3}{m}(1 - \alpha)[\frac{(1-\alpha)}{\alpha}\varrho_o + \varrho_\omega]\ f.g.k.$$

Where $k \cong \sqrt{\frac{\ell}{d}}$ liberation factor

And

| | | |
|---|---|---|
| $\sigma^2$ | Relative Variance | |
| m | Mass of sub-sample | (g) |
| M | Mass of material sub-sampled | (g) |
| d | Screen size passing 90% of material | (cm) |
| f | Shape factor | |
| g | Sizing factor | |
| $\ell$ | Liberation size of mineral of interest | (cm) |
| $\varrho_o$ | Density of ore mineral | (g/cm³) |
| $\varrho_\omega$ | Density of gangue | (g/cm³) |
| $\alpha$ | Proportional content of mineral of interest | |

### POINTS OF EXPERIMENTAL VARIOGRAM:

$$\gamma_{\vec{(h)}} = \sum_{i=1}^{N_{\vec{(h)}}} \frac{(Z_{x_i} - Z_{x_{i+\vec{h}}})^2}{2N_{\vec{(h)}}}$$

Where

| | |
|---|---|
| $Z_{x_i}$ | is the grade at location $x_i$ |
| $Z_{x_{i+}+h}$ | is the grade at vectorial distance $\vec{h}$ from $x_i$ |
| $N^{(h)}$ | is the number of grade pairs separated by vector h |

**SPHERICAL VARIOGRAM FUNCTION:**

$$\gamma_{(h)} = C_o + C \quad [1.5\,h/a - 0.5h^3/a^3] \quad 0<h<a$$
$$\gamma_{(h)} = C_o + C \quad h \geq a$$
$$\gamma_{(h)} = C_o \quad h=0$$

Where

| | |
|---|---|
| $C_o$ | Nugget variance |
| C | Sill |
| h | Vectorial distance |
| a | Range |

## ACKNOWLEDGEMENTS

The author thanks Gold Fields Mining Corporation for permission to present this paper.

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# GEOSTATISTICS AT THE FEASIBILITY STAGE GOLDEN SUNLIGHT DEPOSIT, MONTANA

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## ABSTRACT

Geostatistical procedures applied to ore reserve estimation for the Golden Sunlight deposit, southwestern Montana, are discussed in some detail. Block estimates prove particularly useful because of an accompanying error estimate. The principle difficulty encountered relates to confident development of a correct semi-variogram model in the face of much apparent instability of semi-variograms for individual data sets of each data (support) type. This problem is minimized by (1) ommission of obvious outlier data in estimating semi-variograms, (2) careful editing of data (3) use of composites of 10 feet instead of raw data based on samples of 5 feet (4) use of relative semi-variograms, and (5) the recognition that geology (rock type) has a tremendous influence on the form of the semi-variograms.

An appreciation of geostatistical structure can be obtained from the most comprehensive data set of constant support. Details of the structure so determined, provide a useful basis for interpreting semi-variograms derived from less comprehensive data sets involving other supports, limited orientations and various rock groups.

A method is described for use with a computerized kriging program, that accounts for systematic variations in structural orientation relative to block orientations across the total field in which estimates are to be made. Such methods are necessary where an anisotropy parallels the changing geological structural trend.

Calculations show that sampling methods based on large support provide much better block estimates than small support samples. Consequently, rotary drilling may be more desireable than diamond drilling where confidence exists that serious sloughing of sidewall material does not occur in rotary holes where sampling methods are dependable and where an adequate geological base exists.

## INTRODUCTION

Golden Sunlight deposit is a large tonnage, low grade gold deposit in Jefferson County, southwestern, Montana, about 26 miles east of Butte (Figure 1). The part of concern to us here, called the Mineral Hill zone, has been known for many years and produced small tonnages on several occasions in the past. Placer Development Limited has held the property for about 20 years and our geostatistical study deals mainly with the abundant exploration data generated during this interval. Some of the data, specifically channel sample and AX diamond drill core (and sludge) assays predate involvement by Placer Development Ltd. but are included in our study.

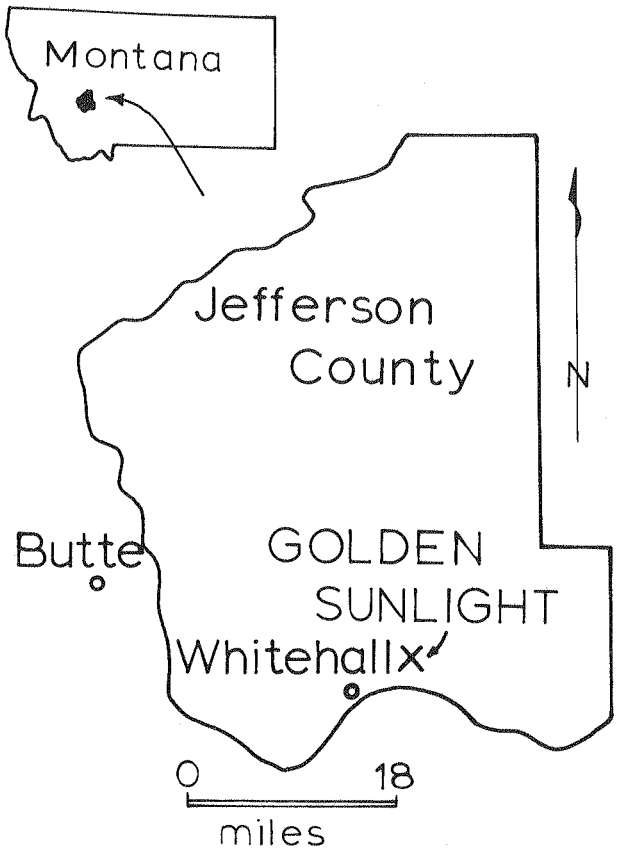


**FIGURE 1. Location of Golden Sunlight gold deposit.**

Geology plays an important part in our geostatistical study so the essential geological elements of the property are described briefly. The Mineral Hill area is underlain by a gently dipping sequence of Precambrian clastic sedimentary rocks, mainly shales (Newland Shale, Greyson Shale and Lahood Formation) (Figure 2). These shale units are cut by a near sill-like body of latite porphyry. A zone of brecciation developed above the latite porphyry. All these units (shales, latite and breccia) are cut by a series of subparallel, near vertical, quartz veins whose strikes change gradually from about 090° on the west to 030° on the east. Pyrite is the principal sulphide in the veins, which are gold-bearing. Gold is both within pyrite and in quartz. An important aspect of the mineralization is the extent to which it extends outward from veins into the wallrock. Mineralization is more-or-less pervasive in the breccia, penetrates somewhat less into fractured latite porphyry and extends only slightly and erratically into the various shale units.

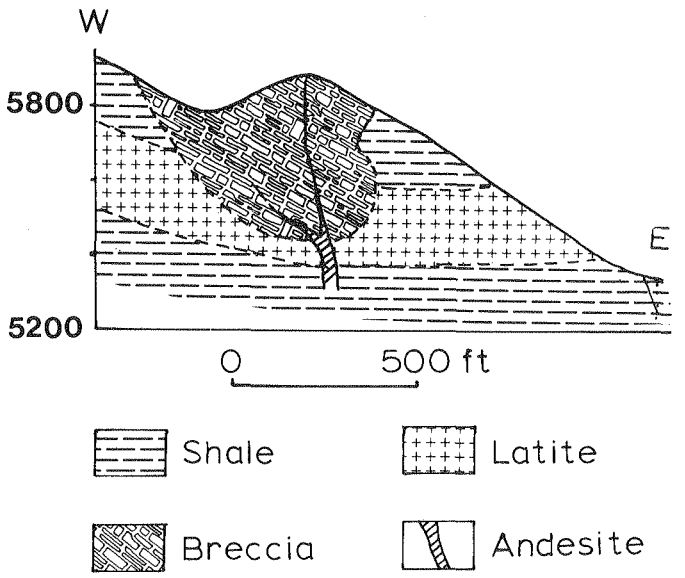


**FIGURE 2. East-west section through the Mineral Hill zone of Golden Sunlight deposit. Principal rock units are shown in somewhat simplified form to give some idea of their geometries. The section is roughly parallel to quartz veins, particularly in the western part. Veins are not shown.**

The purpose of our geostatisitical study was to provide block estimates throughout the Mineral Hill zone as part of a comprehensive feasibility study. Our initial goal was to obtain mean grade and error estimates for blocks measuring 100 x 100 x 25 $ft^3$ where 25 feet was a bench height imposed by the anticipated mining procedure. Estimates were desired for more than 5000 such blocks in the general area of a proposed open pit. In addition, it was hoped that the study would provide guidance on how much additional drilling would be necessary to provide adequate sample control to give sufficiently high quality estimates for all 100 x 100 x 25 $ft^3$ blocks and to investigate how much additional drilling is necessary to give high quality estimates for 50 x 50 x 25 $ft^3$ blocks.

Consequently, our study followed a fairly traditional course for geostatistical estimations including (1) date evaluation, (2) generation of experimental semi-variograms, (3) development of semi-variogram models, (4) kriging, and (5) evaluating the effect of various sampling patterns on error estimates. Although it is not our intention to describe all the details of our study, an indication of the nature of our results for all the forgoing subject categories will be presented.

It is important to bear in mind that a study such as this involves an enormous amount of data and an equally large quantity of repetitive calculations. These can be done only in the computer and involves considerable attention to data organization and programming. We do not attempt to deal with these aspects of a geostatistical study and the interested reader is referred to Journel and Huijbregts (1978).

## NATURE OF AVAILABLE DATA

Golden Sunlight deposit is like many other deposits that have been developed to the feasibility stage—a variety of data types have been generated intermittently during exploration of the deposit (Figure 3). In such cases it is not uncommon that some of the earliest data suffer in acceptability relative to more recent data. The reasons for this are not necessarily because early data are poorer quality, but stem from the fact that information on *quality control* (duplicate assays, accuracy, precise description of sampling techniques, etc.) is missing. Such was the case for channel samples and AX drill core and sludge assays for Golden Sunlight.
Sunlight.

Our first efforts were to examine density distributions for various data types, to evaluate replicate sampling and replicate analyses where available, and to examine these data for expected trends based on variable sample support. In order to check for general consistency in the data we settled on a uniform sample length of 10 feet. This meant that cross-sectional area for individual sample types was proportional to sample support (Figure 3). An added advantage to later studies was that semi-variograms were more stable than for 5 feet sample lengths.

Two sample histograms are shown in Figure 4 and indicate a close approach to lognormal distributions.

Most assay data available to us had been provided by fire assay from 5 independent laboratories. Pairs of these laboratories had in a number of cases analyzed duplicate samples and we were able to use the student t-test to compare such paired data. We were not able to recognize biases in data supplied by any of the 5 laboratories.

## EXPERIMENTAL SEMI-VARIOGRAMS

In generating experimental semi-variograms it is important to combine data of only a single support for each semi-variogram. A bit of trial and error shows that for Golden Sunlight we obtain

| Data types | | Area (sq. in.) |
|---|---|---|
| Percussion | | 28.3 |
| Rotary | | 17.7 |
| Diamond | AX | 0.6 |
| | NQ | 1.4 |
| | HQ | 2.5 |
| Channel | | 9.0 |

**FIGURE 3. Principal data types with cross-sectional areas (square inches) of sample. For NQ core, for example, the indicated cross-sectional area is for split core and is therefore only half the cross-sectional area of NQ core ($\pi D^2/8$). By contrast, channels were 6 inches by 1½ inches and all the material was taken as sample. Of course, supports varied in length as well as cross-sectional area.**

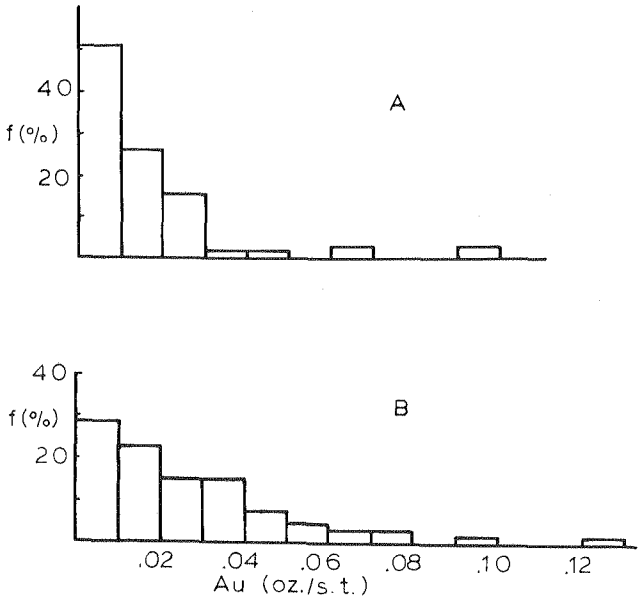


**FIGURE 4. Histograms for 10-foot samples from (a) rotary drill holes, (b) HQ split drill core.**

dramatically different semi-variograms for data of uniform support but from different rock types. Consequently, we will have to examine semi-variograms for each data type within each rock type. What is more, in order to test for anisotropy we require experimental semi-variograms in several different directions. In practice it is commonly impossible to meet all these requirements for a variety of reasons including (1) preferred orientations of data types, (2) concentrations of data in particular geographic zones, (3) geological data may not been available for some assay information, (4) precision of location may be inadequate, and so on.

Despite these limitations it is essential to group or organize data in such a way that the effects of various potential controls such as rock type and orientation can be evaluated in as thorough a manner as possible. To achieve this end it is most sensible to begin with the most comprehensive data set first and work progressively through those of decreasing applicability.

For Golden Sunlight the most comprehensive data set of a single support are blasthole assays representing blastholes of length 20 feet and diameter 6 inches, from two benches mined to provide material for bulk leaching tests. Trial and error with these and other data indicated that a significant proportional effect existed for Golden Sunlight data that could be taken into account adequately by use of the relative semi-variogram. Thus, we accepted the relative semi-variogram as a method of cutting down on "fluctuations" in experimental semi-variograms and thereby improving ease of model-fitting. Two plots (Figure 5) of standard deviation versus mean grade for 10 feet samples from surface channels and HQ diamond drill hole samples have linear trends which are characteristic of other data types and justify the use of relative semi-variograms as a means of improving the apparent instability of semi-variograms. Four directions of relative semi-variograms are

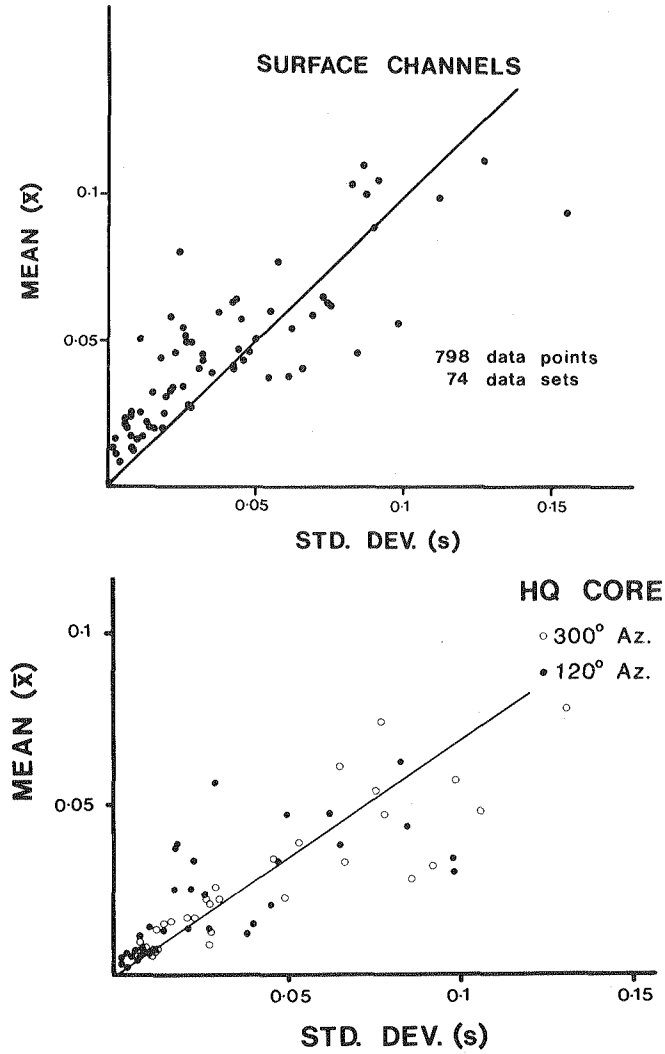


**FIGURE 5. Plot of means vs. standard deviations for 10-foot samples from (a) surface channels and (b) HQ (split core) drill holes. Each plotted point represents parameters for a single drill hole.**

shown for one level of blast hole data in Figure 6. Experimental semi-variograms for directions 2 and 4 (south and northwest respectively) are similar but differ significantly from those for directions 1 and 3 (east and northeast respectively).

Figure 7 is a plot of relative semi-variograms for combined NQ and HQ diamond drill core assays. Trial and error had shown that although these two types of sample have different supports because of different sample cross-sectional area (split-core) the individual semi-variograms could not be distinguished even where hundreds of data points were used to establish each. Consequently, the two were combined to develop a single set of models for all rock types. The particular relative semi-variograms pictured in Figure 7 do not represent correct ranges because they are a preferred orientation relative to the structural trend in the area and as will be demonstrated in a subsequent section a strong anisotropy is present. Never-the-less, the individual relative semi-variograms clearly indicate the differences in nugget effect and sill level from one rock type to another.

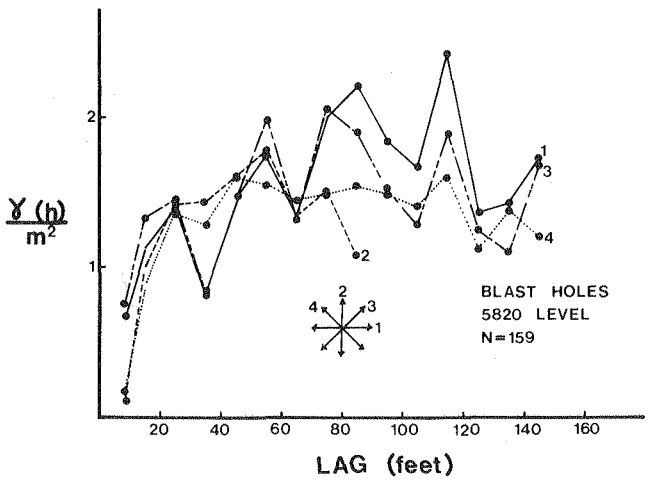


FIGURE 6. Relative semi-variograms for four directions of 20-foot vertical blast holes for the 5820 bench Mineral Hill zone, Golden Sunlight deposit. Direction 2 is north-south.

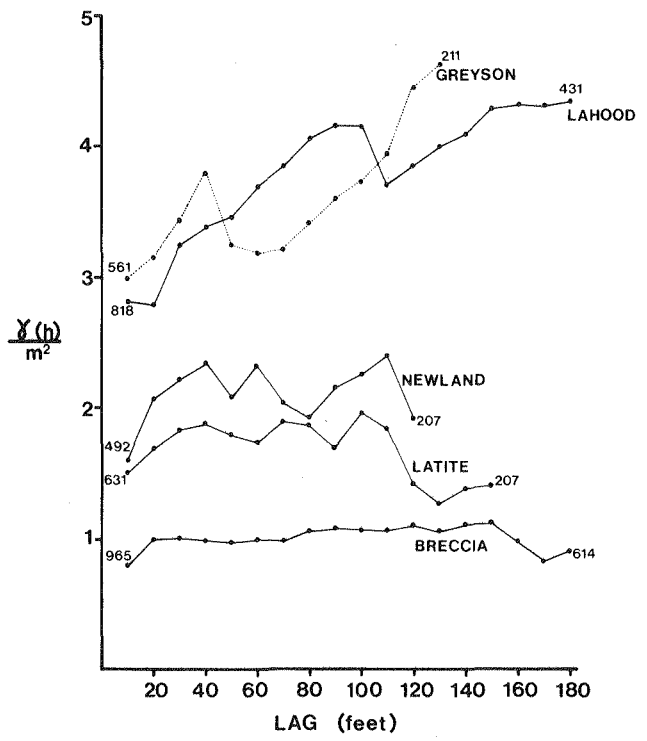


FIGURE 7. Semi-variograms for 10-foot samples from HQ-NQ drill core as a function of rock type. Numbers at each end of each semi-variogram represents number of sample pairs on which first and last points of the semi-variogram are based.

## FITTING MODELS TO RELATIVE SEMI-VARIOGRAMS

Where data are incomplete for any reason it is essential to utilize the most comprehensive sets to establish semi-variogram models with confidence, and where possible extrapolate to aid in developing models for less comprehensive data.

The blast hole experimental semi-variograms of Figure 6 clearly demonstrate a structure that can be closely approximated by two nested spherical models both of which appear to be anisotropic. A short range model is difficult to define accurately because its ranges in various horizontal directions are about 1 to 2 lags (10 to 20 feet). This structure is not of particular consequence for estimates that we must eventually make because our sample spacing is commonly 50 to 200 feet and the short range structure will be included in nugget effect. The longer range structure however is important and is shown diagramatically in Figure 8. Here a maximum range of 105 feet occurs in a 067° direction which is parallel to gold-bearing quartz-pyrite veins in the

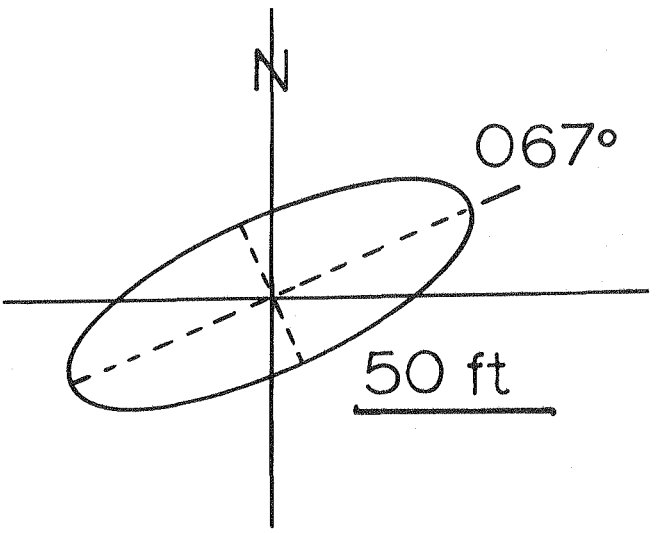


FIGURE 8. Summary of average long range structure indicated by semi-variograms for blast hole data (Figure 6).

area. A shorter range of 35 feet occurs across the structural trend. These results clearly define the average 2-dimensional structure in breccia. The third dimension is slightly more tenuous to obtain. On geological grounds the third dimension which is parallel to the vein orientation should be the same as the horizontal direction parallel to the vein (i.e. range of 105 feet). This geological isotropy parallel to structural trend is supported by data from -45° drill holes (HQ-NQ core) which are consistent with geostatistical isotropy parallel to veins. Consequently, we arrive at a picture of the geostatistical structure within breccia. To generate models for other data types in breccia we use each data type as shown in Figure 7 to define sill level and nugget effect, impose the ranges determined by analysis of the blast hole experimental semi-variograms, and develop an appropriate point spherical model.

A similar approach can be used for other rock types except that certain assumptions must be made in each case. We have no detailed control on the horizontal two-dimensional structure of latite and shales as we had with breccia. However, geologically the veins have the same character and appearance regardless of rock type. On this basis we might expect the same range parallel to structure for all rock types. Although this is an assumption it is consistent with the apparent ranges determined from drill holes in several orientations. Furthermore, these apparent

ranges are consistent with the geological requirement that the true ranges in a direction perpendicular to veins be less in latite and shales relative to breccia. In fact, the cross-vein range for latite can be estimated at about 25 feet whereas those for shales are more uncertain but are about 10 to 15 feet. Once these ranges have been estimated in a manner consistent both with geological information and experimental semi-variograms for a few preferred orientations, it is possible to calculate point spherical models in the same way as was done for breccia. This dependence of geostatistical structure (range) on rock type is a reflection of the distance mineralization penetrates into wallrock i.e. extent of penetration of mineralization perpendicular to the vein controls the range in that direction. This correspondance of geological and geostatistical structures is also seen in the sill level of the experimental semi-variograms (Figure 7). As the sill level rises the extent of penetration of mineralization becomes less but is more irregular. Breccia is permeated, produces relatively uniform mineralization and therefore generates low sill levels to experimental semi-variograms. At the other extreme, shales are irregularly and only slightly fractured and mineralized outward from veins: thus the sill level is higher with the resulting more erratic mineralization.

## KRIGING

Once relative semi-variogram models have been established for all data types and rock types individual blocks can be kriged, level by level in a systematic manner. Individual assays are combined to form composites that extend from the upper to lower bounding planes of each bench. Each such bench composite is stored with the block in which the composite's center is located.

For practical kriging we must select a search zone from which we will accept all available composite assays. Here we chose to search one aureole of blocks (3-dimensions) and the central block to be estimated. Thus, in the kriging of a single ideal block a total of 27 blocks are searched for assay data. However, we must confine ourselves to a single data type and only those data from the same rock type as the block to be kriged. We accomplished this for Golden Sunlight by use of a computer and by running our kriging program 3 times to produce, potentially, three separate estimates for each block, one estimate for each of the three data types. These three sets of estimates for block grade and error were then combined as follows: mean grades by the three methods were weighted inversely to their respective estimation errors

$$m = \frac{\frac{\chi 1}{\sigma 1} + \frac{\chi 2}{\sigma 2} + \frac{\chi 3}{\sigma 3}}{\frac{1}{\sigma 1} + \frac{1}{\sigma 2} + \frac{1}{\sigma 3}}$$

where m is the final mean estimate of a block, $\chi 1$ is the kriged estimate by the 1 the data type and $\sigma 1$ is the corresponding kriged standard error. A final error estimate was also determined for each block by weighing individual error variances by the squares of the number of composites used for individual estimates. Thus,

$$\sigma_e^2 = \frac{n_1^2\sigma_1^2 + n_2^2\sigma_2^2 + n_3^2\sigma_3^2}{(n_1 + n_2 + n_3)^2}$$

where $\sigma_e^2$ is the final estimation variance, $n_1$ is number of composites used to estimate $\chi 1$, and $\sigma_1^2$ is the kriging variance for $\chi 1$.

Using these procedures we obtained kriged estimates for 26 different bench levels. For each level we attempted to krige 360 blocks measuring 100 x 100 x 25 ft arranged in a rectangular array measuring 20 blocks by 18 blocks i.e. 360 blocks per level. Kriging was attempted only if more than three composites were located within the search volume, or if there was at least one composite within the block being kriged. Otherwise, no estimate was made. In order to improve estimates in the outer fringes of the deposit where data are sparse we expanded the search area to include two aureoles of blocks. For this larger filter more than five composites were required to krige a block. This restriction, as with the smaller filter, requires there to be at least two drill holes (near vertical) within the filter volume, or kriging would not be performed. Consequently, if no additional restrictions are imposed on kriging we will provide a kriged estimate in areas of sparse data at the margin of the deposit, for up to 200 feet and perhaps even 300 feet from the nearest control point. Such an extrapolation may not be desired. The easiest way to overcome the problem is to edit out undesired blocks when kriging is completed. An alternative approach would be to incorporate more rigid restrictions in the initial kriging—for example, we could require at least 9 composites in the search area in order to krige a block.

One innovative aspect of our kriging procedure arises from the geometric anisotropy in association with a curved structural trend over the Mineral Hill zone. Horizontal traces of near vertical veins cutting the Mineral Hill area are curvilinear and can be approximated closely by arcs of a circle drawn from a center located by trial and error (Figure 9). The problem we face is that all our semi-variograms models are described in terms of two ranges one of which *parallels* geological structure and the other which *crosses* geological structure. Therefore, the orientation of our models change relative to the property grid (N-S and E-W) used to locate samples. In order to estimate mean gamma value between any two points for use in kriging equations it is essential to determine an "effective distance" between them that takes into account the anisotropy ratio. In order to obtain this "effective distance" we must know relative coordinates of the two samples in terms of a

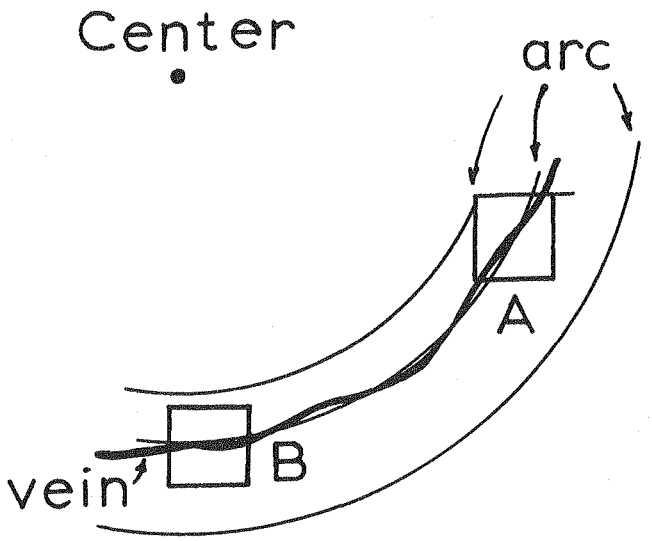


**FIGURE 9. Schematic diagram indicating a center located by trial and error such that arcs about the center closely describe the horizontal trace of veins in the Mineral Hill Zone, Golden Sunlight deposit. Note the different orientation of structure relative to blocks A and B to be estimated.**

component that parallels structure and a component that is perpendicular to structure. (One of these two directions is multiplied by the anisotropy ratio and the resulting perpendicular components are then used to determine the "effective distance" between the samples as the corresponding hypotenuse).

This problem must be taken into account for each of the tens of thousands of mean gamma values to be calculated and an approximate solution has been found that can be incorporated into the kriging program as a simple subroutine. Fortunately, the horizontal trace of the *general geological structure* can be closely approximated by the arc of a circle constructed from a center located by trial and error, as illustrated diagrammatically in Figure 9. To determine new relative coordinates appropriate for determining the "effective distance" between the two points one procedure is to determine the difference in lengths of radii to the two points from our center. This provides us with a *cross-structure* component. In order to estimate a *parallel-structure* component we accept the true difference between the two points as the hypotenuse of a right angled triangle (Figure 10) and solve for the third side.

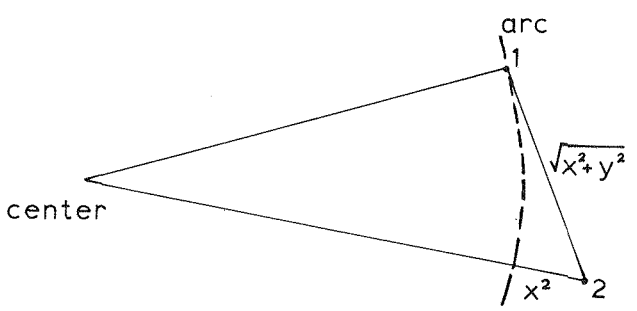


**FIGURE 10. Graphical representation of procedure for determining distances parallel to structure (y) and cross structure (x) between two points 1 and 2.**

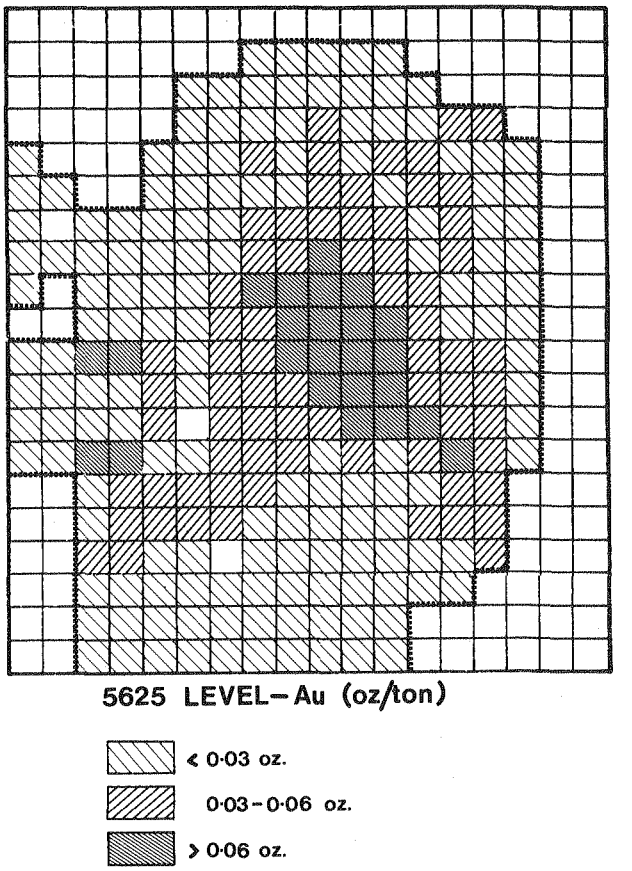


**FIGURE 11. Coded block gold grades (oz/ton) for 5625 bench, Golden Sunlight deposit. Blank blocks are those for which insufficient data were available to make kriged estimates. Blocks are 100 x 100 x 25 ft.**

In our case this procedure was particularly easy because the veins are nearly vertical and we did not have to concern ourselves with the vertical coordinate. A more complicated trigonometric solution would be necesssary if the veins were moderately dipping for example.

An example of "kriged output is illustrated in Figures 11 and 12. Kriged block grades have been generalized somewhat but indicate a relatively high grade zone bordered by moderate grades which in turn are surrounded by lower grades. Two internal blocks and numerous marginal blocks are blank indicating that sufficient data were not found within the search aureole to proceed to kriging. Corresponding error estimates are generalized in Figure 12. These values are related inversely to density of data and directly to rock type distributions. The central, relatively high grade zone corresponds to the greatest density of data and the breccia zone. Drill data are sparse towards the margins with sample separations approaching 200 feet.

## EFFECT OF ADDITIONAL WORK

We have made a preliminary investigation of the errors produced by several ideal sample distribution patterns in various rock types as one means of looking into the need for additional sampling. About 70

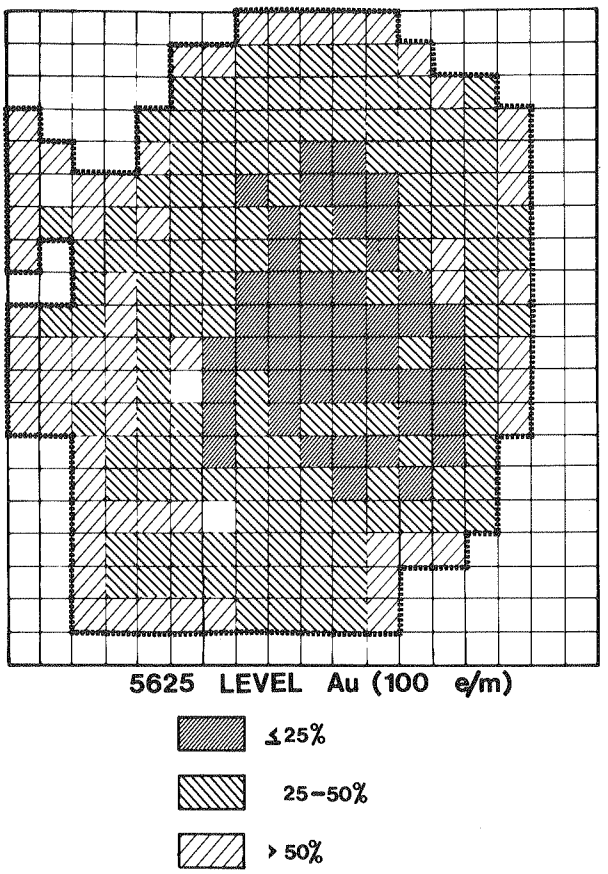


**FIGURE 12. Coded relative kriging errors (one standard error as a percentage of the kriged block mean) for 5625 bench, Golden Sunlight deposit. Compare with Figure 11.**

percent of higher grade ore is within breccia and latite prophyry and some of our results from these two rock types will be used to illustrate the procedure. In all cases we assume a block of data extending to the limits of a cube 200 x 200 x 75 ft, equivalent to data points at the centers of 26 blocks surrounding a block to be estimated and thus easy to compare without initial kriging results.

First, consider the effect of making estimations using different data bases. Kriging errors produced with 27 ideally located samples used to estimate a 100 x 100 x 25 ft block by 3 different data types are shown in Table I. Channel samples and rotary drill samples are significantly better than diamond drill samples (split NQ-HQ core) providing one is confident that rotary drill samples are truly representative of their stated intervals (i.e. no sloughing has occurred). Given this latter assurance and the added fact that sufficient geological information is available, rotary drilling is to be preferred over diamond drilling for fill-in sampling.

We then used NQ-HQ semi-variogram models to krige using various sample densities and examine the effect of sample densities and examine the effect of sample densities on kriging errors. Results for two sample spacings and three rock types are listed in Table II. In all cases a sample spacing of 50 feet reduces the relative kriging error by about 40 percent or slightly more.

**TABLE I. Error estimates in block kriging of breccia using different data supports***

| SUPPORT | RELATIVE KRIGING ERROR (%) |
|---|---|
| Channel samples | 23 |
| Rotary drill samples | 25 |
| NQ-HQ Split Core | 30 |

*Blocks estimated are 100 x 100 x 25 ft. One complete aureole of block-centered data was used in making error estimates.

Suppose we are interested in error estimates for blocks measuring 50 x 50 x 25 ft and considering only a centrally located point plus one aureole of composites (i.e. 27 composites). For breccia the relative error is 33 percent for a sample spacing of 100 feet, and 27 percent for a sample spacing of 50 feet. This is not a substantial improvement considering that the 50-foot spacing represents an addition of 3 times as many drill holes as present in a 100-foot spacing pattern.

## DISCUSSION

In extensive geostatistical studies of this sort data must be grouped and examined in categories that are meaningful in terms of both geology and sample support so that a valid interpretation can result. Obviously efficient use of computer files and efficient programming are essential because of the vast amount of data that has to be treated. In generating our models we examined a wide variety of data supports and attempted to krige 9360 blocks. Of course we actually kriged only slightly more than half of these because of the disposition of data. Even so, the effort is prodigious and computer costs can be high. Because of these high costs it is essential that correct data be utilized. Consequently, it is important to thoroughly edit data entered into the computer. Incorrect values can lead to much fluctuation in semi-variograms and result in difficulty in establishing a model.

Gold deposits such as the Golden Sunlight are noted for local variability in assays. This leads to considerable variability or fluctuation of sills and nugget constants in individual semi-variograms. Some useful means of cutting down on such variability include (1) rejection of poor quality data, (2) thorough editing of data, (3) use of large sup-

**TABLE II. Relative kriging errors in estimation of centrally located block measuring 100 x 100 x 25 ft**

| Rock Type | Sample Spacing 100 ft* | 50 ft** |
|---|---|---|
| Breccia | 30 | 16 |
| Latite | 30 | 17 |
| Lahood Shale | 40 | 25 |

*27 data points
**75 data points

ports (large volume samples) as a basis for obtaining semi-variogram models, (4) use of relative semi-variograms to take account of proportional affect, and (5) rejection of obvious outlier values.

Appropriate semi-variogram models are essential for best quality block estimates. If an "average" semi-variogram that combined data from all rock types were used to make estimates for latite and breccia blocks at Golden Sunlight, the calculated errors would have been more than 50 percent higher than estimated. Obviously, it is not *optimal* to krige if geology is ignored.

In any kriging program it is essential to be aware of the nature and effects of filters used to select data for kriging a single block. If data are selected from *two* aureoles of blocks surrounding the block to be estimated, then for blocks of 100 feet we will produce estimates for a distance of about 200 feet outward from the outer most samples. This may or may not be a desirable extrapolation of the data and such effects can be avoided or minimized in a variety of ways including (1) edit out such estimations after kriging, perhaps based on accepting kriged results only within specified outer boundaries, (2) kriging margins of deposits with a narrower data selection filter, or (3) kriging a block only if the data selection filter locates at least a specified minimum number of composite samples. Any or a combination of these procedures are relatively simple to apply. Of course, the same problem arises in kriging blocks *within* the sample field but between widely spaced data. Such interpolations may be treated by any of the procedures mentioned above. An added refinement may be to accept as valid only those kriged blocks whose relative errors are less than a specific value, say 40 percent.

## CONCLUSIONS

This preliminary geostatistical study provides a useful example of the complexities that can arise in considering grade estimation of large, low grade gold deposits. Geology has been shown to be of paramount importance in establishing semi-variogram models. Problems in determining semi-variogram models for various sample supports and rock types arise principally from the tremendous variability that exist for semi-variograms derived from small subsets of data. These problems can be overcome by a variety of simple procedures that help decrease fluctuations in sill level, including (1) thorough editing of data, (2) omission of outliers, (3) combining data into composite samples longer than the sample interval for raw data, (4) use of relative semi-variograms to account for a proportional effect, and (5) use of at least 200 sample composites with which to generate each semi-variogram. A log transform recommended by some authors might have been used but was unnecessary in view of our use of the relative semi-variogram.

A simple but novel approach has been developed to deal with geometric anisotropy where orientation of geological structure changes regularly and systematically cross the field in which blocks are to be kriged. For Golden Sunlight a two-dimensional solution was adequate for incorporation with an automatic kriging program. The procedure can be extended easily to three dimensions.

All other things being equal, the largest sample support provides minimum estimation variance for blocks in a given rock type. Thus, rotary drilling might be considered preferable to diamond drilling as an exploration tool if one can be satisfied that sloughing has not occurred in rotary holes, that sampling problems do not exist, and that geological information such as would be obtained from drill cores is not required.

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For information concerning the geology and mineral resources of Nevada, contact: Director/State Geologist, Nevada Bureau of Mines and Geology, University of Nevada Reno, Reno, NV 89557-0088. A publication list will be sent upon request.