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THE METALLURGICAL STORY AT INSPIRATION

BY

C. B. KETTERING and K. L. POWER

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C. B. KETTERING* and K. L. POWER**

In the early operations of the Inspiration Consolidated Copper Company, which extended from 1914 to 1926, the ore as mined contained principally sulphide copper minerals, mainly chalcocite (Cu_2S). This ore was milled in a 20,000-ton concentrator and the copper was recovered as a flotation concentrate, which was smelted at the nearby plant of the International Smelting and Refining Company. In those days no attempt was made to recover copper from any oxide minerals present.

As mining operations progressed it soon became evident that the proportion of oxide minerals present would steadily increase, and it was realized that some method of treatment would have to be developed, whereby copper could be recovered from both oxide and sulphide minerals contained in the ore. A research program was instituted and after some years of experimental and pilot plant work a new hydro-metallurgical process was developed, whereby it became possible to extract copper from both the oxide and sulphide minerals in a single leaching operation. The solvent adopted consisted of a mixture of sulphuric acid and ferric sulphate. Following dissolution, the dissolved copper would be recovered by electrolysis. Also in the process of electrolyzing, spent solutions of sulphuric acid, H_2SO_4, and ferric sulphate, $Fe_2(SO_4)_3$, would be regenerated and so made available for continuous re-use. The process appeared to be a success and the decision was made to build a full-sized leaching plant for the future treatment of Inspiration mixed ore. Construction was started and the leaching plant was completed in 1926. With the start of ferric-sulphate leaching the Inspiration Concentrator was shut down and, with the exception of a few short-time runs, has not operated since that time.

The ferric sulphate leaching process served Inspiration well for some 30 years and during that time won a name for itself as a classic example of hydro-metallurgical treatment of mixed sulphide-oxide ores. Originally the plant was designed to process some 7,500 tons per day of a mixed ore containing about 1.2% copper, of which 60% was present as oxide and 40% as sulphide. Down through the years various improvements were made for the purpose of increasing plant capacity, and as well to change the limiting factor of sulphide-oxide ratio. The final result was a leaching capacity of 9,600 tons per day when treating a 1% copper ore with a reversal of the sulphide-Oxide ratio to 60% sulphide and 40% oxide.

SLIMES SEPARATION

Very early it was recognized that satisfactory leaching could only be carried on if free percolation of solutions through the bedded ore was maintained. Such free percolation was seriously interfered with by the presence of colloidal fines. Accordingly, additional plant was constructed to "de-slime" the ore prior to bedding. Wet classification of the primary screen undersize

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accomplished this purpose and resulted in a separation of about 7.5% of the total feed. This classified product contained about 85% minus 200 mesh material. The "slimes" so separated are given a dual process treatment in the "Slimes Plant", wherein the sulphide minerals are recovered by flotation and oxide minerals are dissolved in an agitation leach, the copper being subsequently precipitated as cement copper in a suitable series of iron launders.

Present-day operations at increased tonnage rates have called for expansion of both screening and classifying plants. Rake classifier capacity has been increased and, at the same time, "cyclone classifiers" are being added to the circuit to further physically improve the products going to both leaching and slimes plant circuits.

De-sliming of the ore to be leached was the first great forward step to be taken in the working out of this ferric sulphate leaching process. This has been followed by many another change. Such changes made down through the years involved methods of bedding ore, physical and chemical control of leaching solutions, improvement of contact time, methods of washing and draining leached ore. All such steps were to lead to improved metallurgy and copper extraction, gained in spite of a steady and constant change in the character of the ore, both in grade and in sulphide-oxide ratio.

CEMENT COPPER LEACHING

Increase of sulphide ratio in the ore posed a serious problem, which called for heroic measures. Successful leaching of the sulphide mineral chalcocite (Cu_2S) calls for the presence of ample ferric sulphate in the leaching solvent. In this process ferric sulphate is manufactured, or regenerated, by the oxidation of ferrous salts contained in the electrolyte. However, the rate of such regeneration is strictly a function of the rate of copper precipitation in the electrolytic cell. In due time it became obvious that if the necessary concentration of ferric sulphate for the dissolution of increased amounts of chalcocite in the ore was to be maintained, more copper than was currently contained in leaching solutions, coming off the ore, would have to be supplied to the tank house electrolyte.

In the leaching process the final wash waters contain copper in solution. The concentration is too low to permit electrolytic recovery, so these wash waters are sent to an iron launder system, where the dissolved copper is recovered as cement copper. The quantity of cement copper produced is quite material and amounts to better than 1,000,000 pounds per month, or around 20% of the copper input to leaching. Such cement copper for years has been subsequently treated by smelting. Now it was realized that if cement copper could be re-dissolved and the copper delivered in concentrated solution to the tank house electrolyte, then the electrolysis of this additional amount of copper would furnish all the ferric sulphate necessary for the expanded leaching of sulphide copper. After a considerable period of testing and experimental work, a successful method of dissolving cement copper was developed. A cement copper leaching unit was built and successfully operated. Through this means it became possible to maintain the ferric iron balance, and so leach increasing amounts of chalcocite. In recent years, at times up to 70% of the copper content of the ore has been present as chalcocite.

At the present time, having electrolytic capacity available, the Cement Copper Leaching Plant is being expanded so that in the future all cement copper produced from the several sources of Leaching Plant wash waters, Slime Plant

cement, and Leaching in Place cement, may be dissolved and re-precipitated as cathode copper directly, thus eliminating the expensive steps of smelting and anode refining.

THE DUAL PROCESS

In 1947 Inspiration began the change-over from underground block caving to Open Pit mining. This change made possible the future mining of low-grade fringe ore. It was also known that in the bottom levels of the mine there would be an increasing amount of copper present as chalcopyrite. Chalcopyrite cannot be recovered by leaching and so a new problem presented itself.

In leaching sulphide ores the time element is all-important and a satisfactory extraction cannot be obtained without ample contact time. This in itself would limit the tonnage treatment rate and the rate of copper production would be seriously affected when the time came to treat the lower grade ores. Also, there was the before-mentioned complication of the increasing presence of the mineral chalcopyrite, from which copper could not be recovered by leaching.

Thus, after extensive studies initiated in 1954, it was decided to abandon the well-tried ferric sulphate leach and to convert Inspiration's metallurgical operations to the "Dual Process". In this process ore is first acid leached to recover the oxide copper and the residue from this leach is delivered to the Concentrator, where, after grinding, the sulphide minerals are recovered by flotation. Such a process would enable a treatment rate up to 16,000 tons per day, would insure the maximum recovery of all sulphide mineral, and would assure the maintenance of an adequate increased copper production rate.

NEW CONCENTRATOR BUILT

The year 1956 was devoted to re-building of the old concentrator. The rehabilitation was complete and the new mill was equipped with the most modern of milling equipment. To point this up, it is notable that in the new plant seven 10-1/2' x 14' ball mills replaced 40 of the original 8' x 6' mills for approximately the same grinding capacity. Modern mechanical flotation machines replaced the obsolete canvas-bottom Inspiration type of flotation cells. Within the limits of the mill building there remained ample floor space for repair areas, a complete self-contained electrical substation, together with sufficient room for future possible expansions or modifications. Only the main building, the ore bins, modified to suit, and existing cranes remained from the old plant.

Layout and design of the new plant was done under the direction and supervision of the Inspiration mechanical and metallurgical staffs. Construction was by contract.

The September, 1957, issue of The Mining World contained a very excellent article on the completed concentrator, written by Mr. Stanley Dayton, Associate Editor of that paper. In the interest of brevity many details of plant description mentioned in that article will be omitted in this paper. Incidentally, a complete description of the original 20,000-ton mill was published in the AIME Transactions of 1916, Volume 55. The original Leaching Plant was described in Volume 106, published in 1933.

CHANGE TO DUAL PROCESS

Early in January, 1957, the change to Dual Process operations was made. The change-over involved the stripping of copper from all ferric sulphate leaching solutions, with a final discard of stripped solutions. At the same time, it was necessary to build up fresh iron-free acid solutions for subsequent oxide leaching. This change-over was accomplished without material difficulties and the Dual Process was "ready to roll". Within a short time operations were stepped up to a rate of 15,000 tons per day.

In this Dual Process operation, after leaching the bedded ore, the leached residue is excavated and transported to the Concentrator in 60-ton bottom dump railroad cars. As would be expected the increased rate of operations called for a complete revision of crushing, bedding, excavation, and railroad haul schedules.

LEACHING OPERATIONS

Leaching operations for the recovery of oxide copper have proceeded in a satisfactory manner. From preliminary studies made it had been estimated that iron would be dissolved from the ore and thus solutions would contain some ferrous iron, which, during electrolysis, would be converted to ferric sulphate. It was anticipated that, as a result, the leaching operation, in spite of low ferric iron concentration and short contact time, would recover about 20% of the sulphide copper. However, surprisingly enough, it was found the iron concentration of solution gradually worked up to as high as 7 grams per liter, and, of this total, as much as 4 G/L was oxidized to the ferric state. Thus, it turned out that sulphide extraction at times has run as high as 50%. In the beginning oxide extraction was not as good as had been hoped for. Lately this has improved and it is possible to make an oxide tail not to exceed 0.03% Cu.

The result of this somewhat surprising and mixed-up metallurgy is that an extraordinarily low "head" is sent to the Concentrator. However, economic studies made have demonstrated that, within reason, the more copper extracted by leaching the better will be the overall outcome.

Typical leaching data are as follows:

(October, 1957)	Tons	% Cu Total	% Cu Oxide	% Cu Sulphide
Ore as mined	407,426	0.857	0.465	0.392
Slimes removed	32,571	1.298	0.835	0.463
Ore bedded	374,855	0.816	0.433	0.383
Leached Residue	374,200	0.228	0.033	0.195
Extracted		0.588	0.400	0.188
% Extraction		72.059	92.379	49.086

Solution Analyses

	Grams per liter					
	* Copper		Acid	Iron	Ferric Iron	
	In	Out	Out	Total	In	Out
Leaching solvent and Tank House Electrolyte	25	20	17	6.5	0.7	3.7

*In or out figures refer to solution flow to and from the Tank House.

ELECTROLYTIC PRECIPITATION

Original Dual Process planning indicated that the electrolysis of leaching solutions would require only about 50% of the cell capacity of the Tank House. With this in mind, plans were made to utilize the remaining capacity for anode refining. In this way it was hoped it would be possible to turn out Inspiration's entire production as refined cathode copper. To further this purpose a new anode casting plant was built, which went into service in October of 1957.

As things have turned out, because of the higher extraction of copper by leaching, only about 35% of Tank House capacity has so far been made available for refining. The remaining capacity is still required to precipitate copper from leaching, or, as they are known, "commercial solutions". This fact, in itself, has an important bearing on, and, in fact, becomes the control of Tank House operations.

COMMERCIAL SECTION OPERATION

Commercial solutions, the analyses of which have been previously mentioned, are distributed from a center launder to the required number of electrolytic cells. In these cells use is made of 8% antimonial lead anodes. Starting sheets produced in the stripper section weigh 13 pounds. The final seven-day cathode produced weighs 125 pounds.

Because of the nature of the solutions and the use of lead anodes, cathode efficiency is low and power consumption is relatively high. The electrolytic cells are operated with a 4" electrode spacing. Mechanics of the operation have to be very carefully watched and controlled. Special additives are made use of to control "sprouting" and short circuiting in the cells. In spite of all such precautions, current density must be held down and it is seldom that a figure of 13.5 amps per square foot can be exceeded.

TYPICAL COMMERCIAL SECTION DATA
(October, 1957)

Rate of solution flow	100 GPM/Cell
Average weight cathode	125 Lbs.
Average voltage between anode and cathode . . .	2.51
Average current density	13.6
Average ampere efficiency	76.30
KWH per Pound Copper Precipitated	1.386
Number cells in service	96

REFINING SECTION OPERATION

The anode refining operation has proved to be a most satisfactory one. However, since the Tank House has one single electrical circuit, so far it has not been possible to separate commercial cell operation from refining cell operation. Thus, since current density on the commercial section is more or less fixed, it is not possible to increase the current on the refining cells. This, to some extent, limits the refined cathode output.

The copper anodes are very large and weigh 1,300 pounds. They are placed in the cells with a 6" spacing. Starting sheets are the same as those used in the commercial section and come from the stripper section of the Tank House.

Electrolytic efficiency averages well over 90% and a very satisfactory cathode, weighing 220 pounds, is produced. To the maximum extent possible, all handling of anodes and cathodes is mechanical, using overhead cranes for loading anodes to, and unloading cathodes from the cells to mechanical tilting racks. Cathodes are baled into 4,000-lb. bundles and loaded to box cars by lift trucks.

Anode Slimes are carefully collected and delivered to the anode Slime Plant, where they are filtered, dried and packaged for delivery to the Raritan Copper Works for final treatment.

It is expected that as the ore changes in character and less copper is leached and more is concentrated and smelted, the refining capacity of the Tank House will be increased, as the need for commercial cell operation falls off. Ultimately it is expected that the entire output of copper from Inspiration will be shipped as refined electrolytic copper.

TYPICAL REFINING SECTION DATA
(October, 1957)

Rate of flow of electrolyte	10-15	GPM/Cell
Copper content of electrolyte	45.0	G/L
Acid content of electrolyte	160.0	G/L
Average weight of cathode	225	Lbs.
Average voltage between anode and cathode . .	0.35	Volts
Average current density	19.5	
Average ampere efficiency	93.4	%
KWH-AC per Pound Copper	0.158	
Number of cells in service	30	

CONCENTRATOR OPERATIONS

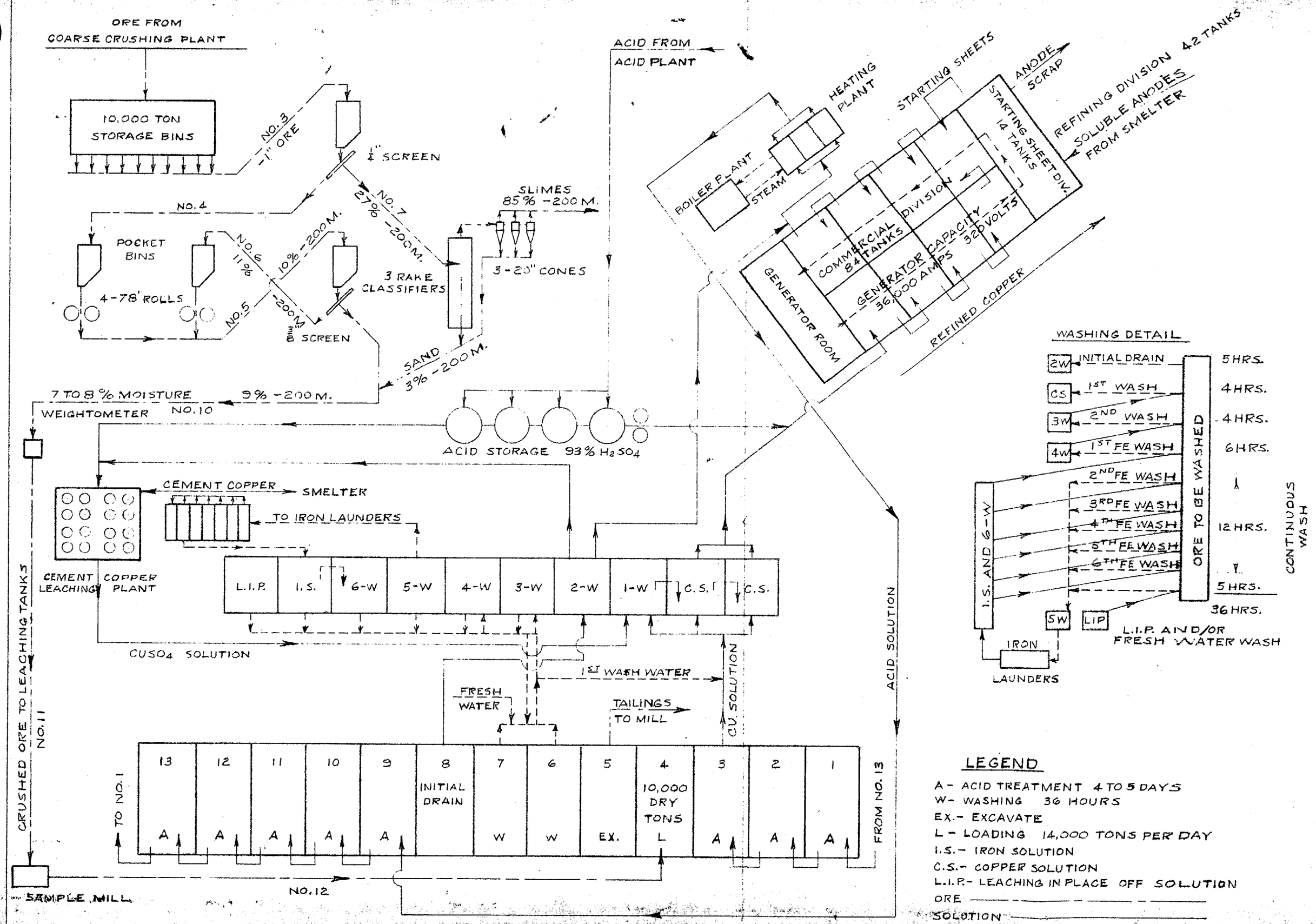
The Dual Process involves treating the ore twice, in two separate plants, to obtain maximum recovery of both oxide and sulphide copper values. The oxide copper is recovered in the Leaching Plant by acid leaching and electro-deposition from the solutions. The leached ore is then sent to the Concentrator, where the sulphide copper is liberated by grinding and recovered by flotation as a concentrate, which is shipped to the Smelter.

Leached ore, previously crushed to 3/8", is excavated from the leaching tanks and hauled by railroad to the Concentrator, about one mile distant. There, the 60-ton gable-bottom cars are dumped into 15,300-ton capacity caternary ore bins. Although the leached material is 9 to 11% moisture, and contains some residual acid, as well as copper and iron salts, there has been little evidence of attack on the steel of the bins.

The presence of the high moisture, however, dictated one of the several novel design features of the Concentrator. In order to draw the wet, sticky material from the bin, twenty-one inverted pyramidal hoppers were attached to the bottom; three for each ball mill. The slope of the sides is 60° and the mouth of each hopper is 4-1/2 feet square inside the wear plates. There has been no tendency for the wet ore to bridge over the openings. Under each hopper is a 60-inch belt feeder, which draws the ore from the outlet. With some modifications of the original design, this type of feeder has been very satisfactory. A vertical gate regulates the depth of the ore ribbon on the belt feeders to give virtually a metered rate of feed to the ball mills for each gate setting.

FLOW SHEET OF THE INSPIRATION CONSOLIDATED COPPER CO. LEACHING PLANT

DRG. NO. X-1292



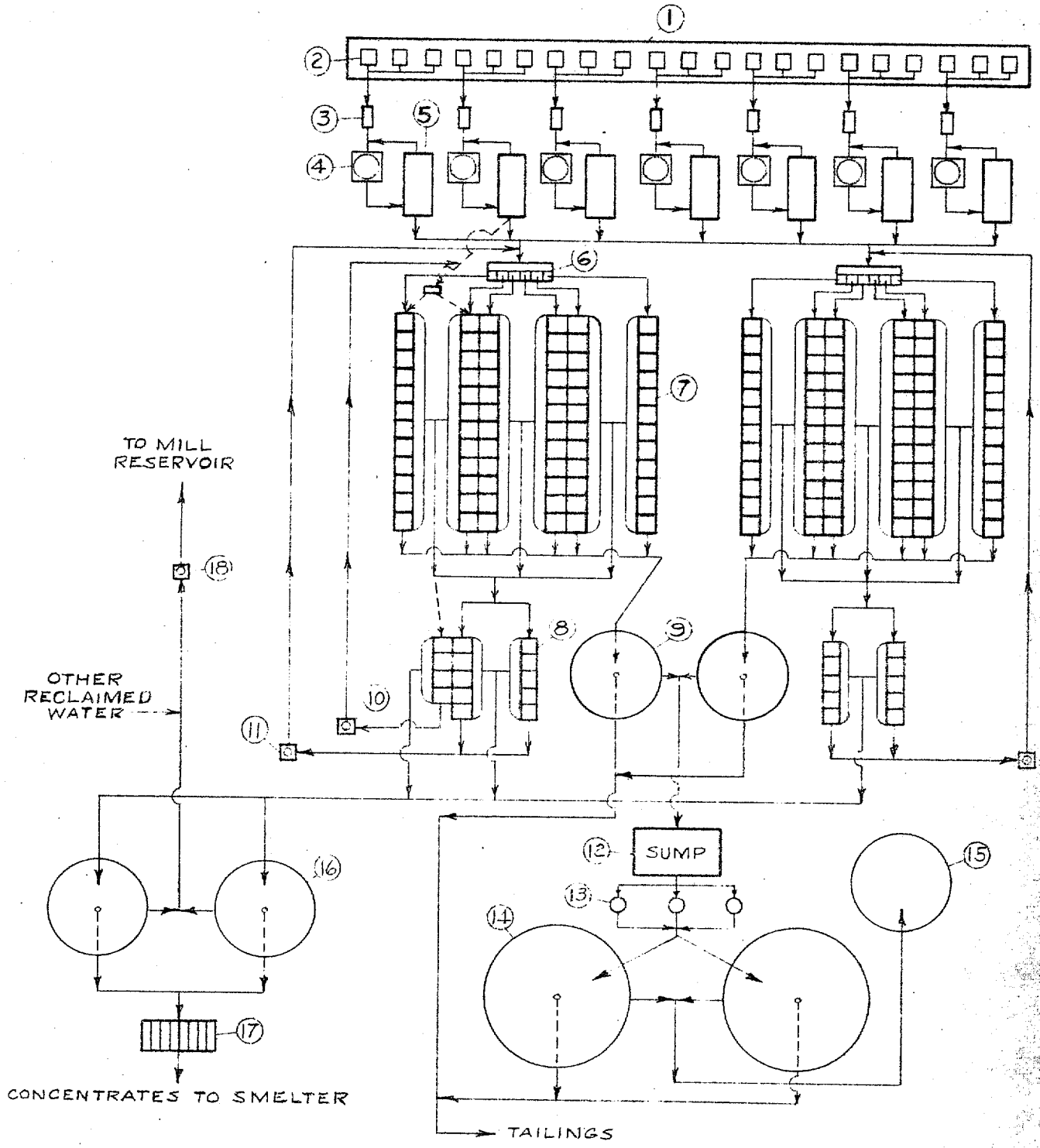
LEGEND

A - ACID TREATMENT 4 TO 5 DAYS
W - WASHING 36 HOURS
EX. - EXCAVATE
L - LOADING 14,000 TONS PER DAY
I.S. - IRON SOLUTION
C.S. - COPPER SOLUTION
L.I.P. - LEACHING IN PLACE OFF SOLUTION
ORE
SOLUTION
WASH SOLUTIONS

1-1-58 R.A.T.

DRG. NO. X-1293

FLOW SHEET OF THE INSPIRATION CONSOLIDATED COPPER CO. CONCENTRATOR



1. 15,300 ton capacity, steel catenary fine ore bin.
2. 21 Belt feeders, 60 inches wide.
3. Transweigh belt scales.
4. 7 Allis Chalmers 10-1/2 x 14 foot ball mills; 6 in service, 1 standby.
5. 7 Dorr HX, 16 x 38 foot classifiers.
6. Galigher sampler and pulp distributors; 3 in the circuit, 1 for test section.
7. 12 banks of 12-cell, Fagergren 66 inch flotation machines, roughers.
8. 4 banks of 5-cell and 1 bank of 4-cell Fagergren 66" flotation machines, cleaners
9. 2 Dorr, 32 foot diameter hydroseparators.
10. 1 Hazleton, 3 inch, twin-volute pump; test cleaner tails.
11. 6 Hazleton, 5 inch, twin-volute pumps; cleaner tails; 2 in service, 4 standby.
12. Concrete sump, 36 x 25 x 16 feet deep.
13. 3 Hazleton, 12 inch, centrifugal pumps; two in service, one standby.
14. 2 Dorr, 275 foot traction thickeners.
15. 1,500,000 gallon capacity mill reservoir; mill head tank.
16. 2 Dorr, 60 foot diameter, concentrate thickeners.
17. Eimco, 8 disc, 6 foot filter.
18. 1 Hazleton, 8 inch, twin-volute pump.

Three of the above 60-inch feeders discharge onto one 18-inch collector belt, which, in turn, transfers to a 24-inch belt conveying the ore to the ball mill scoop box.

BALL MILLS

Grinding units consist of seven 10-1/2 x 14 ft. diaphragm ball mills driven by 1,000 horsepower synchronous motors. Each mill is in closed circuit, with a 16 by 38 foot rake classifier, which is equipped with a 42 inch by 16 foot spiral conveyor for returning the classifier sands to the scoop box.

A unitized control panel is provided for each grinding unit. Located on it, in the direction of ore flow, are push-buttons for the control of feeders, conveyor belts, classifier, spiral conveyors, ball mill and exciters. These controls are tied together with electrical interlocking to prevent spillage of ore, should any unit fail. On the same panel is a weightometer recorder and along-side is the panel of the pH recorder.

The Concentrator was designed to handle 15,000 tons per day, using six grinding units at 2,500 tons each, leaving one unit to serve as a spare. This spare unit was necessary since all mill repairs must be done in place. The steel structure of the building was not strong enough to support a crane large enough to lift a loaded ball mill and carry it to a repair bay.

In actual practice, the mills are capable of grinding more than the 2,500 tons called for in the original design plans. As shown in the table of grinding data, the tonnage rate has been increased to better than 2,600 tons per day. Therefore, in order to maintain the 15,000 tons per day total, it is only necessary to run five units continuously and start and stop a sixth section as needed to keep up with the train haulage from the Leaching Plant.

In operation, each ball mill operator has charge of three mills. The operator is responsible for putting as much tonnage as possible through the mills, consistent with maintaining the specified mesh of grind and percent solids in the classifier overflows. To aid in controlling the mesh of grind, a wet screen analysis of each classifier overflow is run every two hours. Generally, enough water is added to the classifier to make a pulp of 33% solids in the overflows, which will contain about 3% plus 48 mesh.

Adjacent to the grinding bay is the lime grinding section, which consists of two 6 foot by 48 inch ball mills in closed circuit, with two 6-inch cyclone classifiers. The lime slurry produced in the cyclone overflows is stored in three agitated storage tanks. From these tanks the slurry is circulated through the Concentrator in a looped piping system along the line of columns near the ball mill scoop boxes. The ball mill operators add a sufficient quantity of lime into each scoop box to maintain a pH of 9.5 in the classifier pools. pH is measured and recorded automatically.

Despite the acidic character of the ore, only about 5 to 6 pounds of lime per ton of ore are required to attain the pH value.

Both the collector and frother reagents for the subsequent flotation are added in the classifier overflow boxes. The frother reagent feeder is located at the overflow end and the collector feeder is located at the sand end of the classifier, with piping provided to allow the collector to be added either to

the classifier overflow or to the ball mill scoop box. This gives the necessary flexibility to handle those reagents which might require additional conditioning of being passed through the ball mill.

BALL HANDLING

Grinding balls are directly unloaded and stored in four concrete bins of 300 tons capacity each. These bins are located on a hillside below a railroad spur, about a quarter of a mile from the Concentrator. They can be loaded with balls either from railroad cars or from trucks. From the dispensing gates on each bin, the balls roll down a launder to the hopper of a recording scale, where they are weighed in specified charges and the weight is printed on a scale ticket. Each charge is then dropped into one of six separate compartments on the bed of a 12-ton truck, which is parked directly below. The truck hauls the balls to small concrete bunkers, which are located outside the mill behind the ore bins and at an elevation about ten feet above the center line of the ball mills. Each weighed charge is unloaded from the truck compartments into one of 21 of these bunkers, which allows for three days of ball storage for each mill. When the gates of the bunkers are lifted, the balls roll by gravity into the drum feeder on the front of each ball mill scoop.

Present consumption of 2-inch balls is between 1.25 and 1.35 pounds per ton of ore. The weighed charges, added each day, are, therefore, about 3,400 pounds. Ball load level in each mill is inspected about every five or six weeks and is maintained as nearly as possible to 45% of the mill volume.

TYPICAL GRINDING DATA

	March through July, 1957	October 1957
Days operated	149	27
Wet Tons of Feed	2,018,195	421,523
% Moisture	10.16	10.57
Dry Tons of Feed	1,812,232	376,957
Dry Tons per Day	12,169	13,961
Avg. Number Sections Running	4.836	5.296
Avg. Tons per Section/Day	2,516	2,636
% +48 Mesh	3.1	2.8
Ball Consumption Lbs. per Ton	1.27	1.34

ROUGHER FLOTATION

The rougher flotation cells are divided into two sections, each consisting of six rows of twelve Fagergren flotation machines. Ahead of each section of roughers is a rectangular pulp distributor, which receives the overflows of three classifiers, combines the pulp into a single stream for feed sampling, and then splits it into six portions for the individual rows of roughers.

As mentioned previously, always one and possibly two of the seven grinding units may be shut down at any particular time. Therefore, the overflow of classifier number 4 is provided with a junction box from which the pulp (when that grinding unit is in operation) may be diverted to either, or split to both, of the distributors, and thereby equalize the load on the two flotation sections.

For ease and safety in starting and stopping the rougher machines, individual push-buttons for all roughers are centralized in one control console at the end of the rougher bay. On the same platform is a smaller console with push-buttons

for cleaner cells and hydroseparators, which are located on the floor below. From this one control platform, excellent visibility is afforded to all the machines of both the rougher and cleaner floors.

CLEANER FLOTATION

All rougher concentrates from each rougher section are combined and then split to the cleaner cells. Cleaners consist of two rows of five 66" Fagergren cells for each rougher section.

Cleaners are likewise arranged in a single-stage circuit, making a final concentrate, which flows by gravity to the concentrate thickeners and a cleaner tail, which is pumped back to the distributors ahead of rougher flotation.

The final concentrate is thickened to 50 to 60% solids in a 60-foot thickener and filtered by a 6-foot diameter 8-disc filter. Filtered concentrate is conveyed to a loading station, where it falls into a railroad car for transport to the International Smelter, about one mile away.

TEST SECTION

Since the flowsheet is simple and uncomplicated, it was an easy matter, in design, to provide for a test section which can be isolated from the rest of the mill circuit. This test section consists of one grinding unit, a two-way distributor, two rows of rougher cells, a four-cell row of cleaners, and a cleaner tailings pump for re-cycling to the test section distributor.

The advantages of this test section in evaluating reagents, alkalinities, pulp densities, feed rates, etc., are obvious. It is particularly useful in reagent testing, giving a four-step procedure. In the first step, the reagent is tested on small batches in the Testing Laboratory. If it shows promise, it can then be tested on 2,600 tons per day in the mill test section. Then, if results justify, the reagent can be tried on 7,500 tons per day in an entire rougher section. The final step would be adoption for the entire mill.

ELECTRICAL CONTROL ROOM

Push-button consoles in the grinding bay and flotation bay are actually only remote control stations for the switches which control the flow of power to the various motors. Switches, circuit breakers, transformers and electrical recorders and meters are all located in a separate sub-station building within the Concentrator. This is constructed of hollow concrete blocks and is cooled in the summer by an air washer and a 90,000 cubic foot per minute fan.

From the control room, the wiring is carried in overhead cable trays to the motors and control panels in the grinding bay. Wiring to the flotation machines, hydroseparators and push-button consoles is led on cable trays through a tunnel, which runs the length of the building under the flotation bay.

The use of cable trays reduced the need for buried conduit, with its attendant difficulties, to a minimum. Further, cable trays have the advantages of neatness, flexibility, and easy circuit tracing in replacement of faulty wiring.

TYPICAL FLOTATION DATA

	March through July, 1957	October 1957
Assay - Concentrator Feed		
% Oxide Cu.	0.054	0.033
% Sulphide Cu.	.239	.193
% Total Cu.	.293	.226
Assay - Tails		
% Oxide Cu.	0.041	0.020
% Sulphide Cu.	.055	.046
% Total Cu.	.096	.066
Assay - Concentrates		
% Copper	29.723	22.043
% Moisture	14.75	14.04
% Insoluble	12.70	9.14
% Iron	17.44	32.27
Concentrator Recoveries		
Tons Concentrates	12,067	2,757
Lbs. Total Copper	7,173,431	1,215,331
Ratio of Concentration	150.264	136.727
% Cu. Recovered	67.433	71.244
Lbs. Cu. per Ton Feed	3.956	3.224

RECOVERY OF SILVER AND MOLYBDENUM

Another salient feature of the Concentrator operations is the recovery of two valuable metals, which were formerly left untouched in the Leaching Plant tailings, namely, silver and molybdenum. While silver is not present in the ore in large quantities, its recovery in the copper concentrates in the amount of one to three ounces adds at least a small extra margin of profit for the Concentrator operations. Gold values are negligible.

While it was known that molybdenum occurred to some extent in the ore body, the extent of its occurrence was indefinite. However, it was quickly noted that the copper concentrate produced had a substantial and, more or less, consistent molybdenite content. Early in the year a program of test work for the recovery of this valuable mineral was inaugurated. As a result of this work a molybdenite recovery plant has been designed and is now under construction.

TAILINGS DISPOSAL

At Inspiration, the maximum recovery of mill water is of vital importance. To further this purpose, it was necessary to install two 275-ft. diameter tailing thickeners to provide for adequate settling. However, because of lack of space, below the mill site, it was necessary to locate these thickeners on a site,

which is 105 feet above the mill. This involved pumping to the thickeners. To decrease the burden of pumping, two 32-ft. Hydroseparators were installed in the tailings circuit.

Rougher flotation tails flow through the Hydroseparators. Hydroseparator overflow running 25% solids, with about 3 to 5% plus 100 mesh material, flows by gravity to a concrete sump. From the sump the thin pulp is pumped to the thickeners on the hillside above. Pump installation consists of three 12" centrifugal pumps, two of which are constant speed, the third being driven through a variable speed coupling, which is automatically controlled. In normal operation, one fixed speed pump and the variable speed pump adequately handle the pumping load.

Clarified overflow water from the two 275-foot traction thickeners flows by gravity to a 1,000,000-gallon reservoir, which serves as the head tank for the Concentrator. Make-up water is added in this reservoir and blended with all reclaimed waters before flowing to the grinding bay through a 20-inch transite line.

Thickeners are constructed partly on solid ground and partly on tailings, but the center piers of both are on solid ground. To prevent leakage and water-logging of the old tailings, the bottoms are of reinforced gunite.

The thickened underflow pulp at 48 to 50% solids is carried back down the hill to the Hydroseparator floor through an 18-inch transite pipe. Inside the mill building, the Hydroseparator sands at 58 to 60% solids are recombined with the thickened slimes to give a final tailings pulp of 52 to 54% solids, which flows to the tailings dams through an 18-inch transite line.

The present main tailings dam has an area of about 105 acres and had been developed by previous operations. It was made ready for use by building up an initial level berm about ten feet high on three sides. The fourth side lies against an older and higher dam. The pulp from the Concentrator crosses this higher dam and can be diverted to fall through concrete drop-boxes into either leg of a horseshoe-shaped loop of 18-inch transite pipe located on top of the initial berm.

This tailings disposal system is a modification of the Morenci system. The pulp is discharged into the pond through 3-inch plug valves, which are spaced along the header pipe every 26 feet. A short length of rubber hose and a 20-foot length of 3-inch light gauge steel pipe carry the pulp into the pond.

When the tailings have filled to the top of the initial berm, a new berm 5 feet high will be erected along the inside of the header pipe and extra lengths of steel pipe will carry the tailings up and over the new berm. It is expected that by extending these riser pipes and building new berms, a total lift of 20 to 25 feet may be made before it will become necessary to raise the main 18-inch header to a new elevation.

In addition to the main dam, a new dam with a potential area of about 150 acres is being developed adjacent to it on the north. Here the system is the same, except that the loop of header pipe is 12-inch transite instead of 18-inch. The tailings stream is split to feed both ends of the loop simultaneously.

Water is reclaimed from the tailings dams by means of decant chimneys and

buried decant lines, which carry the water to a settling basin below the dams. From there the water flows by gravity to one of the main pumping stations, where it is combined with other industrial waters and returned to the main plant reservoir.

As shown in the table of water data, the recovery at the tailings dam had averaged better than 25% of the water in the tailings pulp until October, when considerable tonnage was diverted into the new tailings dam. Most of the water in this pulp was lost to seepage.

TYPICAL WATER DATA

	March through July, 1957		October 1957	
	Gals./Ton Ore	% of Total	Gals./Ton Ore	% of Total
Total Make-up Water	165	70.7	179	86.4
Tailings Dam Return Water	68	29.3	28	13.6
Total Water Used	233	100.0	207	100.0

DISCUSSION OF METALLURGY

The design of the Concentrator has several novel and unique features, but certainly the most unusual thing about the operation is the metallurgy itself. As described in the discussion of Leaching Plant metallurgy, up to the present time the extraction of sulphide copper in the Leaching Plant has been higher than had been anticipated. At times this higher extraction does not leave much copper in the leached ore for the Concentrator to work on. Such a situation may vary within wide limits, depending on the amount of oxide and of chalcopyrite contained in the ore.

The mill ratio of concentration is fairly consistent and has remained between 130 and 145 to 1. Obviously, then, the grade of final concentrate made varies from day to day with the assay of sulphide in the feed.

The reasons why the ratio of concentration seems to have an upper limit are not fully known, but it is thought that they would include:

(1) The ready floatability of both the chalcocite and pyrite at the present pH of 9.5 in the flotation pulps. Efforts to depress the pyrite at higher pH values have not as yet been successful, as in so doing the tailings loss is increased. Rougher cells make nearly the final grade of concentrate and leave little work for the cleaner cells to do. The concentration ratio of the cleaners is only about 1.2 to 1.5 to 1.

(2) The absence of a regrind mill in the cleaner circuit. In the design discussions, the subject of a regrind mill was given full consideration, but was ruled out on the basis of test work, which had shown that a marketable concentrate could be made with only a single-stage roughing and single-stage cleaning circuit. It is still not felt that a regrind mill would pay for itself in further upgrading the concentrates. Microscopic examination of

various mill products has revealed little interlocking of the copper with iron or gangue minerals.

(3) The lack of selectivity of the present collector reagent. During the design and construction period, exhaustive laboratory flotation tests were conducted on most of the collectors and frothers now in commercial use. All reagents were judged solely on their ability to minimize the loss of copper in the tailings. On this basis, Xanthate for the collector and Pine Oil for the frother were chosen as the start-up reagents. The problem of final upgrading of concentrates was left for actual operations. In fact, until the low heads of recent months, there was little or no problem in maintaining a satisfactory grade of concentrates.

Now that operational difficulties of starting up the new mill are fairly well straightened out, the test section of the circuit will be used more continuously in an effort to learn more about the present flow sheet and just what modifications and changes are necessary to improve it. Early in this test work, will be a reappraisal of other collector reagents to try to find one of equal promoting power, but greater selectivity than the Xanthate being used.

J. H. Courtright

1974

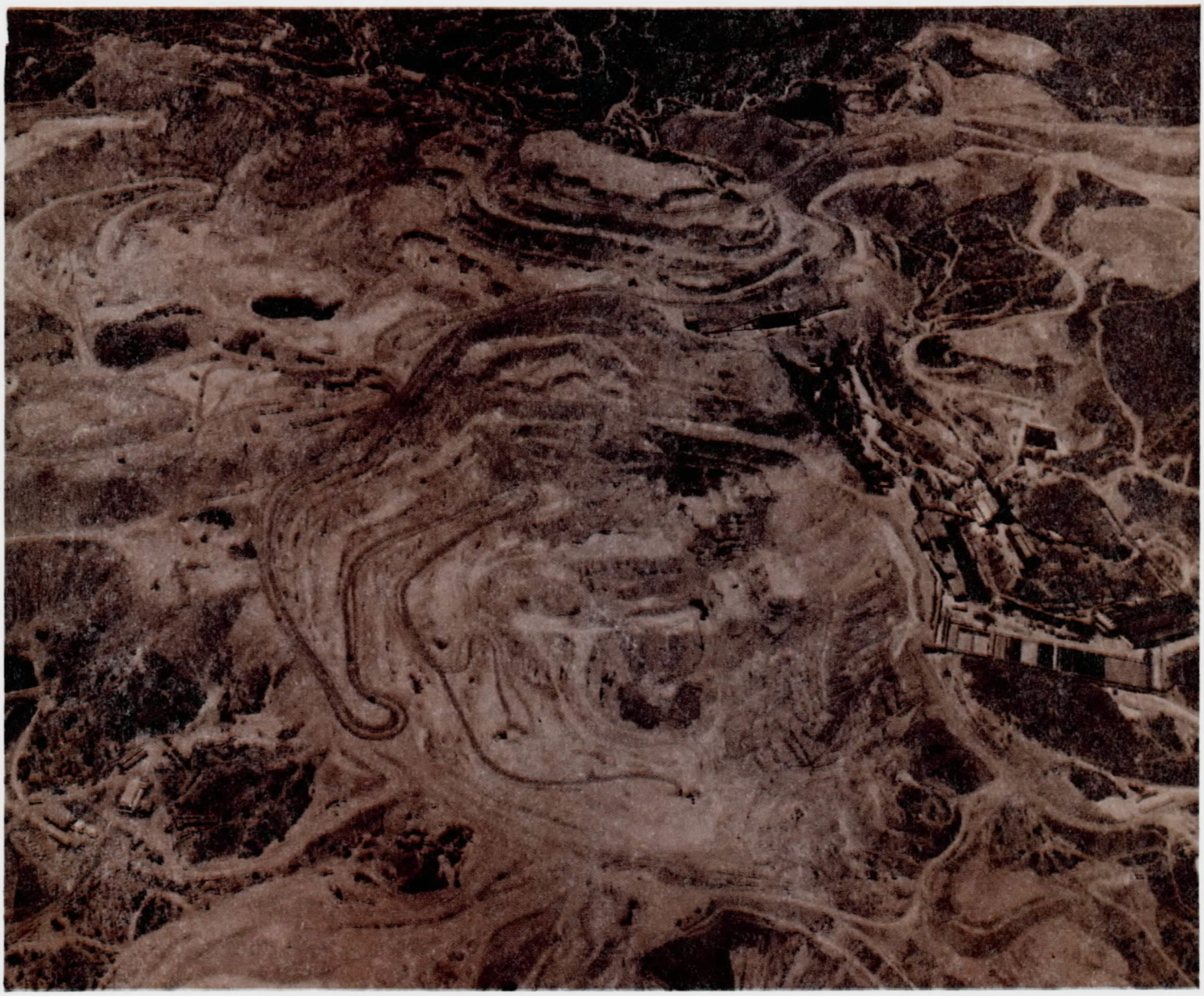
A.I.M.E.

OPEN PIT DIVISION

SPRING MEETING

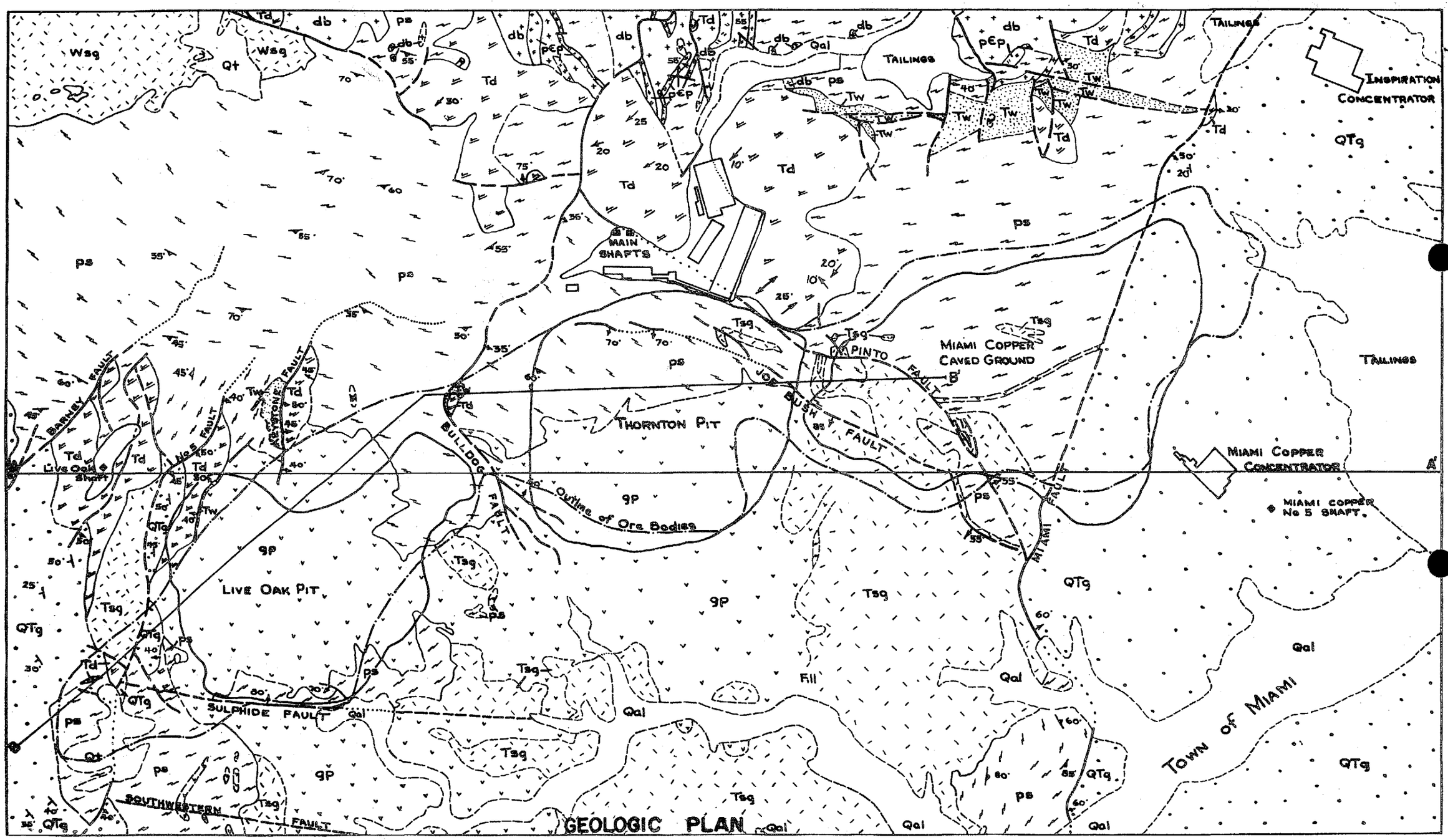
file

Inspiration Consolidated Copper Company

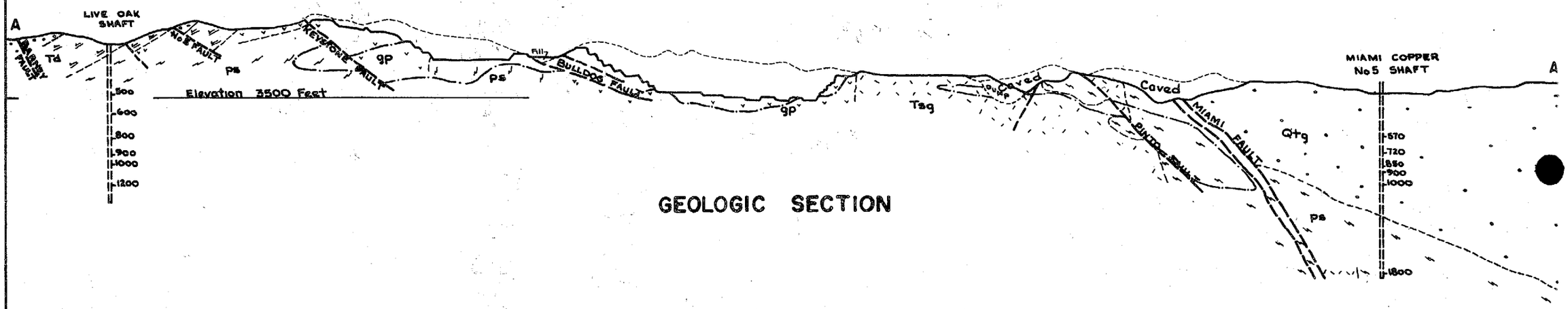


AERIAL VIEW
OF
INSPIRATION CONSOLIDATED COPPER COMPANY

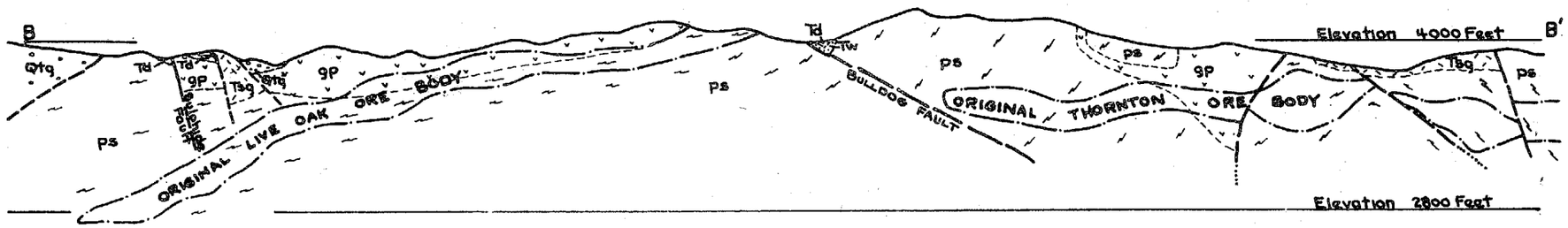
THORNTON, LIVE OAK & RED HILL
OPEN PIT OPERATIONS



GEOLOGIC PLAN



GEOLOGIC SECTION



GEOLOGIC SECTION ORIGINAL ORE BODIES

EXPLANATION

Quaternary	Qal Qt	Talus and Alluvium
	QTg	Gila Conglomerate
Tertiary	Td	Dacite
	Tw	Whitetail Conglomerate
	gp	Granite Porphyry
	Tsg	Schultze Granite
	Wsg	Willow Spring Granite
P.	db	Diabase
Precambrian	p€p	Pioneer Formation
	Ps	Pinal Schist

35 Strike and dip of beds

60 Strike and dip of foliation

20 Bearing and plunge of lineation

Horizontal lineation

Shaft

Outline of ore bodies

Outline of pits and caved ground

Faults

1000' 0' 1000' 2000' 3000' 4000' FEET

SCALE

GEOLOGIC PLAN and SECTIONS

INSPIRATION MINE

GILA COUNTY, ARIZONA

N

A I M E

ARIZONA SECTION

OPEN PIT DIVISION

SPRING MEETING
at
INSPIRATION CONSOLIDATED COPPER COMPANY

May 10, 1974

PROGRAM

8:00 - 9:00 A.M.	Registration - Miami High School
9:00 - 10:30 A.M.	Welcoming & Orientation - Miami High School Auditorium - Tom Anderson, Chairman Open Pit Division, H. D. Harper, General Superintendent.
	Inspiration History - Del Wisner, Assistant Leaching & Refining Superintendent.
	Inspiration Geology - Jack Eastlick, Chief Resident Geologist
	Inspiration's Ox Hide Mining and Leaching Operation - Bill Sorsen, General Foreman
	Inspiration's Open Pit Operations - Jim Lundy, Open Pit Superintendent
	Inspiration's Dump Leach Operations - Fred Rice, Chief Planning Engineer
10:30 - 12:00 Noon	Tour Ox Hide Operation
12:30 - 2:00 P.M.	Lunch - Elks Club, Miami
2:00 - 4:30 P.M.	Tour Inspiration Open Pit Operations
5:30 - 7:00 P.M.	Cocktails - Cobre Valley Country Club
7:00 P.M.	Dinner - Cobre Valley Country Club

INSPIRATION CONSOLIDATED COPPER COMPANY

INSPIRATION, ARIZONA

HISTORY

What is in a name?

Webster's dictionary defines Inspiration as:

1. An inspiring or animating action or influence
2. Something inspired, as a thought
3. Theology: A divine influence directly or immediately exerted in the mind or soul of a man

In Inspiration Consolidated Copper Company, the name itself is a mouthful of words, and to its employees and the other primary copper producers, an "Inspiration."

Inspiration, like many western mines, owes its discovery to the early prospector and his burro. Mineralization was recognized in the local area prior to 1900. The earliest exploratory working was the 1000 foot Woodson Tunnel driven into a local hill side in 1908. By then the local owners had consolidated claims and groups of claims into a single holding and induced outside capital to form the Inspiration Mining Company. Later mergers and aquisitions led to the formation of Inspiration Consolidated Copper Company in 1911.

From this time on the history of the company has been a series of "Inspirations." Dr. Louis Ricketts, William Boyce Thompson, Charles E. Mills paved the way to what is now one of the most complete metallurgical plants in the country.

The original concept was to build a plant to treat 7500 TPD by tabling. As work progressed it became apparent that a10,000 TPD plant would yield more profits. Meanwhile flotation work directed by J. M. Callow showed promising results.

The management at Inspiration Consolidated Copper Company had an "Inspiration." They cancelled the steel ordered for the tabling plant and re-designed the mill for Mr. Callow's flotation process, resulting in a 14,400 TPD plant on stream in June of 1915. It had been decided to use the unique mining method known as under cut or block caving. Ore would be crushed, and delivered to the Concentrator by train, where it was ground in 36 -(8' X 6') Marcy Mills. The ground slurry was floated with coal tar in a modified callow cell known as the Inspiration Air Cell. The sands, separated from the tail, were tabled for oxide recovery. Metallurgy for the first six months was as follows:

	% Total Copper	% Oxide Copper
Heads	1.702	0.226
Tails	0.373	0.180
Float Concentrate	37.63	
Table Concentrate	13.12	

At the end of the six months it was decided to again increase capacity to 16,000 TPD by adding four more ball mills.

At the end of the first operating year Inspiration estimated it had spent 20.5 million dollars to build the new metallurgical plant and mine. The net profit for this period was 20.6 million dollars. Within one and one-half years Inspiration had lived up to its name to become one of the largest copper producers and a major profit maker.

Because of Inspiration's unique ore body, containing approximately 50% oxide minerals, it was decided in 1918 to try to develop a metallurgy more amenable to this unusual ore body.

The first experiments on leaching the oxide minerals and then floating the sulfide minerals called "Dual Process Ore" were to continue until 1922. At this time

Dr. Ricketts engaged Mr. G. D. Van Arsdale for some small scale lab work on the ferric sulfate leach, using sulfuric acid leach for the oxide minerals and the ferric sulfate leach for the sulfide minerals. Recovery of the copper was accomplished through electrowinning and iron launder cementation.

From these results it was determined that an ore of - 1.19% total copper
0.77% oxide
0.42% sulfide

would yield a 0.182% total copper tail

In September, 1925 construction started on a Leaching Plant to treat 7500+ TPD. This was quickly scaled up to 9000 TPD and production started in October 1926.

The mill continued to treat a portion of the high sulfide ores. In 1929 it was found that desliming the ore prior to leaching improved overall recovery, lowering the tail from 0.198 to 0.127% copper. The slimes treatment plant was designed and put into service in 1930. The slime slurry was pumped to the mill, floated for sulfide recovery and then leached and washed in four counter current decantation thickeners, with copper being recovered from the leach solutions in iron launder cementation.

When the price of copper fell to 5.5¢ per pound Inspiration shut down in May of 1932 and it did not resume operations until September, 1935.

Upon resumption of operations the ore was primarily oxides and the plant was converted to the faster acid leach sulfide float, Dual-Process method. When the market sagged again in 1937 the process reverted to the cheaper and slower ferric sulfate leach, where it was to remain with one exception until 1957.

In 1942 Inspiration was asked to produce more copper for the war effort. A unique metallurgical process resulted. Mixed ores were ground while acid was added, the copper leaching solution passed over scrap iron, and then floated to recover cement

copper and the sulfide minerals.

In 1944 the Tank House burned to the ground. Again the plant switched to the Dual-Process recovery while the Tank House was rebuilt with copper being recovered in the iron launders.

The rapid development of modern earth moving equipment, and rising costs made Inspiration look at its "hole card." In 1947 the decision was made to begin open pit mining. Stripping for the new open pit began in 1947, the first ore was mined in 1948. The last of the underground ore was mined in 1954.

The phasing out of the underground operation and the building of large waste dumps led Inspiration into the dump and underground leaching. Iron launders were expanded at the Leaching Plant and still more copper was produced.

Meanwhile, the oxide content of Inspiration ore was decreasing, affecting the electrowinning, and it became necessary to re-dissolve cement copper produced at the Leaching Plant and add this copper sulfate to the leaching-electrowon circuit. This decrease in oxides was accompanied by an increase in chalcopyrite. Chalcopyrite is not soluble in ferric sulfate. It was now time for another change in the metallurgical process.

The old mill was revamped and the latest mill equipment added. When the plants switched from ferric sulfate leach to Dual-Process in 1957 the plant capacity was increased to 15,000 TPD and again to 17,500 TPD within the year. The Concentrator was now offered the opportunity to recover molybdenum as a by product. The moly circuit was added in 1958.

The change from ferric sulfate to Dual-Process reduced the number of electrowinning cells required in the Tank House and these were converted to electrorefining cells.

The development of the Christmas mine resulted in a 4000 TPD mill for treating ores from the underground mine. Poor ground conditions kept tonnages below design. Abundant oxide minerals on the surface were not amenable to leaching because of their high lime content. Inspiration metallurgists devised a circuit to recover the oxide minerals and a small open pit operation was started to supply additional tonnages. In 1966 the underground operations ceased and the operation was converted to surface mining.

The International Smelter was purchased in 1960 and the Tank House was expanded by 40%. Inspiration now had the capacity to treat, smelt and refine all of its production. Refinery capacity was further increased by adding a rectifier to the New Tank House.

A unique method of upgrading cement copper through floation was developed in 1962 to produce what we call "Cu Pels", a cement copper of 97+% copper with annual sales of 3.5 million pounds per year.

In 1964 it was decided an increase in plant capacity to 20,000 TPD was needed to offset rising costs and dropping grade. A 6300 foot conveyor was constructed to haul leached ore from the vats to the Concentrator. To speed up the excavation of the vats a new bucket-wheel excavator was built to feed the overland conveyor system.

Early in 1966 the new excavator collapsed and a modified ore treatment program was called for. The old clam-shell excavator was again used to load trains to convey the leached ore to the Concentrator at 15,500 TPD. A bypass belt system was added from the crusher to the new overland belt, thereby bypassing the leaching vats allowing a high sulfide ore to be fed directly to the mill. This change completely altered all previous mining plans and the mine was now producing a Dual-Process ore and Direct Mill Feed ore.

To treat the additional concentrates from the 20,000 TPD mill, the Smelter added a suspended arch reverbatory furnace, and started phasing the old Great Falls converters out in favor of new Pierce-Smith converters.

About the same time someone thought the mine had it entirely too easy and decided, "let's go to three ore products."

1.	Dual Process Ore - now approximately	0.7% total copper 0.35% oxide 0.35% sulfide
2.	Direct Mill Feed -	0.6% total copper 0.15% oxide
3.	Discard Ore -	0.6% total copper 0.2% sulfide

By leaching the discard ore and using the old excavator to load 40 ton haulage trucks to carry tailings to the dump, the mill could then receive an equal amount of Direct Mill Feed ore. This, along with changes in the primary ore storage for three ore types, belt conveyors in place of train haulage from the primary to secondary crushers, allowed Inspiration to aim at 25,000 TPD.

In 1968 Inspiration decided to include a "CMCR" continuous melting casting rolling plant in its metallurgical line. Now cathodes from the refinery could be melted, cast, and rolled into 5/16" rod for direct sale, instead of costly shipment of cathodes back East for melting and refining into ingots. This was the first plant West of the Mississippi to do this and made Inspiration the most completely integrated plant of its kind, truly an "Inspiration."

During the same year the Smelter converted from log poling to natural gas refining of blister copper, one of the first major users of this new method. The mine started several new projects; stripping Red Hill for future ore development, Black Copper, a million + ton ore body of low grade oxide copper which was developed

adjacent to the Leaching Plant. The Ox-Hide mine, a 12,000 TPD low grade oxide, pad leach, cementation operation was started just west of Inspiration. It was now time for the mine to upgrade its trucks, shovels and garage facilities for the larger haulage trucks now coming into service.

Due to the many diversifications of the company each plant generated and collected its own data; this was becoming an overwhelming avalanche of paper work, so a computer was added to receive and correlate much of the clerical activity.

The years 1966 to 1968 were busy ones for Inspiration, gearing the old facilities for increased production and expanding its workings to include many outlaying projects like Black Copper, Live Oak and Ox-Hide.

The years 1969 to 1972 were busy with the realization of 24,700 TPD in 1970. Taking place during this time was the development of the Sanchez property located near Safford, a large low grade oxide deposit, a preliminary look at the possible reworking of old mill tailings using LPF, and the use of liquid Ion Exchange (LIX) for leaching. Smelter capacity was increased by adding an air preheater and the realization that air pollution was becoming a serious problem led Inspiration to investigate many new pollution control plans. During this period all of the scattered assay labs and sampling rooms were consolidated into a new modern centralized Analytical Lab.

The old tertiary crusher built in 1926 for 750 TPH and now running at 1200 TPH is in advanced stages of old age. This coupled with subsidence in the old primary crusher area led to the design and construction of a new primary and tertiary crusher.

1973 saw the start of Willow Springs Leach Area and treatment facilities, and an expansion of the Ox-Hide mine. But most importantly, a commitment of 54 million dollars by Inspiration to meet the new State Air Pollution Laws was made.

Again Inspiration would commit itself to an "Inspirational" undertaking. A large new electric furnace, five Hoboken siphon converters and gas-collecting apparatus are teamed with a Lurgi double adsorption acid plant to treat 1500 TPD through the Smelter while reducing incoming sulfur content of the ores by 90%, thereby enabling Inspiration to meet the State and Federal Ambient Air Standards. Scheduled start up for the new reduction plant is midyear 1974.

1974 will bring the new reduction plant into operation, a new lime plant is under design and Willow Springs will start. Mining will begin on the Joe Bush Area, a joint venture with Cities Service to produce 49 million tons of ore for Inspiration.

All in all it looks like a busy future for Inspiration. The past 60 years have seen many changes in process and attitudes. Each change took courage and foresight. To remain competitive the Inspiration Consolidated Copper Company must remain an "Inspiration in Copper."

Del Wisner

GEOLOGY OF THE INSPIRATION-MIAMI DEPOSIT

by John T. Eastlick
May 3, 1974

INTRODUCTION

The Inspiration-Miami ore deposit has had a long productive history, and in terms of production, it ranks among the large orebodies of the world. As presently developed (including the OxHide and Bluebird Mines), it has a strike length of over 27,000 feet with widths ranging up to 4,000 feet and thicknesses up to 900 feet. Although severed by faults and mined as separate orebodies, it is essentially one zone of mineralization. This mineralized zone occurs along the contact between schist and porphyritic intrusive rocks, which trend generally to the east and to the southwest.

Tonnages of ore mined from the various mines within the mineralized zone up to January 1, 1974 totals in excess of 530,500,000 tons. To date, total copper produced from these operations amounts to more than 7,200,000,000 pounds including both concentrates from milling and cement copper from in-place and dump leaching.

ROCK DESCRIPTIONS

Rocks represented in the general area within and surrounding the orebody include only those of Precambrian and Tertiary ages. Paleozoic sediments are absent, but are exposed elsewhere in the district in association with other mineral deposits indicating that a thick section of covering material existed during the time of the intrusion of the granitic rocks and the formation of the ore deposit.

Descriptions of the rocks presently exposed in the general are are as follows:

Precambrian Rocks

Pinal Schist: The Pinal Schist of early Precambrian age consists of a metamorphosed sequence of sedimentary and volcanic rocks which are represented in the general area by coarse grained quartz muscovite schist, fine grained quartz serecite schist, and chloritic schist. The general schistosity strikes about N50°E and dips steeply to the southeast with some local variations. In general, the outlines of the orebodies and the trend of the intrusive porphyries appear to closely parallel the local schistosity.

Pioneer Formation: This unit comprises the lower part of the younger Precambrian Apache Series. The upper portions of this formation are fine grained, thin-bedded sandstones, whereas the basal part is a pebbly arkose 15 to 20 feet thick. In places, the lower part grades into hard, fine grained, reddish-brown sandstones, and in some localities interfingers with fine grained gray sandstones and arenaceous shales. Pre-Whitetail erosion has removed much of the Pioneer Formation from the area containing the mineralized deposit, but it is mineralized where found within the limits of the orebody.

Diabase: The age of the diabase has been the subject of some controversy, but it is believed by the author to be of Late Precambrian age. It clearly intrudes the Pinal Schist and younger Precambrian Apache Series, but contacts with the later Precambrian Troy Quartzite and the Paleozoic sediments are generally fault contacts.

In the mine vicinity, this basic intrusive ranges from fine grained to coarse grained varities with the composition grading from a hornblende to an augite diabase. Larger bodies of diabase occur near and in the

Warrior and Geneva workings, but only narrow dikes are found within the Thornton and Live Oak Mine areas. Apparently this rock was very receptive, both from hypogene and supergene sources, as a precipitant for mineralizing solutions.

Tertiary Rocks

Schultze Granite: The Schultze granite occurs as a large irregular stock, forming a noteable part of the Pinal Mountains to the south and west of Inspiration. Generally the rock has a characteristic granitic texture, consisting of a coarse grained matrix of quartz, oligoclase, and biotite which often encloses phenocrysts of sodium-rich orthoclase from one to four inches long. Chemically the rock is a sodium-rich granite, but mineralogically it is a quartz monzonite.[4] Age determinations by Creasay and Kistler[1] indicate the Schultze granite to have an age of about 62,000,000 years which would date it as early Tertiary.

The relationship of the Schultze granite to the Inspiration-Miami orebody is obscure, but it generally believed to be part of the parent magma from which the porphyritic intrusives and associated hypogene mineralization were derived. Although both the Bluebird and OxHide deposits rest on Schultze granite, both are fault contacts and only very weak pyrite disseminations are noted in the immediate footwall at these mines.

Biotite Granite and Granite Porphyry: Intrusive rock types within the mine area are classified into two types. Along the north and northeast sides of Live Oak Gulch, the most prevailing rock is designated as granite prophyry in conformance with local rock nomenclature. Specifically this rock is a quartz monzonite porphyry composed of medium sized phenocrysts of orthoclase and plagioclase, together with varying amounts of quartz and biotite set in a phaneritic matrix.

Outcrops to the south and southwest of Live Oak Gulch, extending towards the Bluebird Mine, consist of both granite porphyry as above and a somewhat coarser grained igneous rock, which is designated as a biotite granite. Mineralogically this so-called biotite granite is also a quartz monzonite. In many places the boundaries between the porphyry and the biotite granite appear to be graduational and the exact contact is poorly defined; in other places, however, these are well-defined contacts which may indicate a later intrusive phase. Age determinations by Creasey and Kistler[1] date the granite porphyry to about 58,000,000 years which would favor the latter.

Actually, the two rock types are rather uniformly altered, being light gray in color, intensely silicified with numerous quartz veinlets, and moderately sericitized. Both contain abundant secondary biotite, and veinlets and replacements by K-feldspar. Near the contact zone in the Inspiration area, the granite porphyry is usually intensely altered, with quartz "eyes" and sericite forming the predominant minerals.

Generally these rocks are regarded as separate marginal or younger intrusive phases of the Schultze granite. These rocks are particularly significant in that roughly one-third of the orebody in the Inspiration area extends into the granite porphyry and ore occurs in the Bluebird and OxHide deposits in both granite porphyry and biotite granite.

Breccias: Several separate breccia structures are exposed within the granite porphyry mass along Live Oak Gulch to the south of the Open Pit office, being composed mainly of angular to subangular fragments of granite porphyry and quartz, together with a few fragments of schist. These have been recemented by later quartz which fills the spaces between

the individual breccia fragments. Generally these zones contain more quartz fragments near their centers with granite porphyry fragments becoming more abundant towards the outer edges. Sizing of the individual pieces of breccia commonly range from one to two inches in diameter, but some fragments range up to 12 inches in size. At their perpheries the breccias grade outward to fractured and shattered granite porphyry with a stockwork of intersecting quartz veinlets.

Intruded into the breccia structures and numerous small dikelets of later intrusive material, ranging from a few inches to two feet in width. These intrusives are very irregular, in places cutting steeply through the breccia and in other places intruding the breccia in flatly dipping sinuous patterns. In contrast to the granite porphyry this later intrusive material is finer grained and darker colored, containing more inherent biotite and small to medium sized phenocrysts of feldspar and quartz set in an aphanitic groundmass. Locally small breccia fragments appear within this finer grained igneous material, possibly indicating a later stage of microbrecciation.

The significance of these breccias and later intrusives to the mineralizing cycle is not clear, but the pipe-like structures evidently served in part as principal conduits for the large amounts of quartz, which flood the general area. Evidence of primary mineralization is meager, but scattered psuedomorphs of hematite after pyrite indicate some sulfides were present.

The age of the breccia and associated igneous material is clearly post-porphyry and pre-mineral.

Whitetail Conglomerate: Thin Conglomerate beds, regulated to this unit, are noted on the northwest corner of the property near the Warrior and Geneva workings, and are found overlying the 550, the South Barney,

and Montezuma ore zones. This conglomerate also covered parts of the Live Oak, and Red Hill pits before these areas were excavated. Generally this detrital material overlies the older rocks along an erosion surface that existed prior to the eruption of dacite, and it probably post dates the period during which the major part of supergene enrichment occured. However, some exotic copper enriched material due to those processes is present in certain areas.

Dacite: Eroded remnants of dacite in many places overlie the Whitetail conglomerate, but in others the basal portions of the dacite rest directly on the older rocks. The dacite is composed generally of plagioclase, sanidine, quartz, and biotite in a glassey aphanitic groundmass, together with occassional fragments of older rocks. Commonly, these dacitic flows are underlain by a bed of water-lain tuff from 10 to 100 feet in thickness which grades upward into a layer of black vitrophyre. The lower part, including the tuff and vitrophyre layer is regarded by Peterson[4] as an earlier volcanic eruption, and the dacite as part of later volcanic activity being dated at about 20,000,000 years in age.[1]

Quaternary Rocks: Thick sections of Gila Conglomerate cover the mineralized zone to the west of the Barney Fault and to the east and southeast along the hanging wall side of the Miami Fault. Within these areas, fragments of schist and the various intrusive rocks are the prevailing rock constituents with schist being predominant towards the bottom. Cementing material consists generally of clay minerals and calcareous material.

These conglomerates generally overlie a post-dacite erosion surface with the bedding trending northwest-southeast and dipping 30-55 degrees southwest.

STRUCTURE

Pre-porphyry and pre-mineral controls within the area are to some extent related to the schistosity of the Precambrian schist which trends to the southwest, and to other fault structures and fault veins which strike east-west to northeast-southwest, generally paralleling the schist-porphyry contact. Structures of this type include the Warrior, Sulphide, Southwestern, and other paralleling faults and fault veins located further to the south. All of these have had some post-mineral movement, but apparently these trends had some expression before the emplacement of the porphyry and the formation of the ore deposit.

The breccias, as previously mentioned, provided some control, for the extensive quartz that is exposed throughout the area. These breccias outcrop at the surface generally as oval, pipe-like bodies with their long exes oriented approximately N60°W. Although none are fully exposed, the largest is approximately 800 feet long on its major axis and 300 feet wide across the minor axis.

Several systems of pre-mineral and post-breccia fractures and shears are in evidence. One system, striking N25°-60°E and dipping 20-55 degrees southeast, is abundantly represented throughout the mine area. Another set trends with a north direction dipping steeply to the east, and another system strikes about N60°E, dipping to the northwest. These pre-mineral systems of fractures and shears, many of which are quart filled and sulfide enriched, cut both the schist and igneous rocks, including the breccias.

Some controversy exists as to the age of other fault systems exposed within the area. While there is some agreement that these faults may have followed along pre-mineral zones of weakness, evidence points

strongly to the fact that the major part of their movements postdate the formation of the chalcocite blanket and probably followed the deposition of the greater part of the Gila conglomerate.

Post-mineral faulting throughout the area is reflected by two main systems. One of these of which the Pinto, Williamson and Schulze faults are a part, trends to the northwest and dips from 35 to 55 degrees northeast. The other system appears as north-northeast trending faults which dip flatly to the east and southeast. Faults of this system include the Barney, Porphyry, Number Five, Keystone, Bulldog, and Miami Faults. Both of these fault systems cut and displace the enriched chalcocite zone, the Whitetail conglomerate, and the dacite; and several, including the Miami, Barney, and Williamson Faults, also offset thick sections of Gila Conglomerate. These faults generally have broad crush zones with strong development of clay along several fault strands. Movements are normal, with the various blocks being downdropped to the east.

Another strong fault, the Joe Bush, strikes approximately parallel to the Pinto Fault, but dips steeply to the southwest. As exposed, this fault apparently displaces the schist-porphyry contact and ore zone about 1,000 feet northwest.

Other paralleling smaller faults, known as the Colorado Fault system, occur as splits in the hanging wall of the Bulldog Fault. Some of these have displacements of as much as 100 feet offsetting the secondary-enriched zone with movements down to the northeast.

At least, one reverse fault or overthrust is represented and others may be present in the Barney-Number Five Fault block and in the fault block to the west of the Barney Fault. This overthrust apparently trends

about east-west and dips about 40 degrees to the south with the hanging wall side displaced upward between 500 to 600 feet, over-riding the enriched chalcocite zone, Whitetail conglomerate, dacite, and Gila Conglomerate on the footwall side.

Indications are that the general area remained relatively undisturbed during the erosional period preceding the deposition of the Whitetail Conglomerate and probably remained stable throughout the period of volcanic activity that followed. Regional tilting to the southwest possibly began during the erosion cycle following the volcanic activity and culminated during or after the deposition of the Gila Conglomerate with the development of the major fault trends that strike north to northwest. This is evidenced by the drilling to the west of the Barney Fault which shows, the enriched chalcocite blanket and overlying Whitetail and dacite to plunge sharply to the southwest, being covered on the west end by as much as 1,500 feet of Gila Conglomerate. It is also suggested to the east of the Miami Fault where the dacite, Whitetail, and underlying enriched zone is covered with thick sections of Gila Conglomerate up to 4,000 feet in thickness.

MINERALIZATION

In general, the better mineralization in the Inspiration orebody is confined to a belt on either side of the schist-porphyry contact, but further to the east in the Miami Copper workings and in the blocks to the east of the Miami Fault the orebody appears to extend into the schist some distance from the porphyry contact. Some porphyry in the form of dikes and irregular bodies, however, is present in these areas intruding

into the schist.

Hypogene alteration effects are shown by the intense silicification and moderate to strong sericitization throughout the general mine area. Secondary biotite is evident within the breccias and surrounding igneous rocks, occuring as thin bands parallel to the trend of quartz veining and locally as irregular masses and filling along fractures. Pervasive and veinlet replacement by K-feldspar occurs within the igneous mass.

Although the original modular expression of the Inspiration-Miami deposit is now obscured by rotation and erosion, and by later enrichment and faulting, it probably was roughly cylindrical in shape consisting of an inner zone of potassic alteration (quartz-biotite-sericite-K-feldspar) outward through a phyllic zone (quartz-sericite-pyrite), an argillic zone (quartz-kaolin-montmorillonite), and a porphylitic zone (chlorite-epidote-calcite-magnetite). Over this same interval sulfide assemblages apparently varied from a weak pyrite-chalcopyrite-molybdenite inner zone outward through various other sulfide mineral assemblages to sphalerite-galena with minor gold and silver on the outer margins. These zonal arrangements are well represented to the northwest of the Miami Fault, but are cut off to the south by the Miami-Williamson Fault complex.

Primary mineralization is not strong; although below some of the thicker portions of the chalcocite enriched zones, diamond drill cores contain veinlets of chalcopyrite up to one-quarter of an inch thick, sometimes accompanied by blebs of bornite. Within the schist, the stronger mineralization generally occurs in granular beds in which the schistose structure is either poorly developed or destroyed. Quartz, pyrite and chalcopyrite occur as veinlets and pyrite and chalcopyrite is disseminated

throughout the schist. Mineralization in the porphyry is similar, with quartz, pyrite and chalcopyrite filling fractures to form veinlets, and with pyrite and chalcopyrite occuring as disseminations, replacing biotite in many instances.

Quartz appears to be early, occuring in several successive stages. Pyrite is later followed by chalcopyrite and bornite, and molybdenite and glassy quartz were introduced at a still later stage.

Most of the ore mined today at Inspiration was formed by supergene enrichment. Supergene chalcocite was deposited, mainly on pyrite, forming a blanket-like enriched zone of varying thickness. Sometimes the chalcocite replaces the pyrite completely or occurs as thin films on the surface of the pyrite. Above the chalcocite enriched zone, chrysocolla occurs as the principal oxidized mineral with later malachite and azurite cementing and filling fractures. Brochantite, atacamite, lindgrenite, libethenite, and minor metatorbernite are found locally, and some native copper and cuprite occur within and near the faulted zones.

The rocks capping the ore zone are generally moderately iron-stained at the surface and locally contain some varying amounts of copper oxides. The thickness of this capping is quite variable due to the tilting of the ore zone and subsequent erosion, but it apparently averaged about 400 feet in thickness. The usual limonitic colors are well-developed, and there are abundant crusts of transported limonite. The silic boxworks, although not in abundance, are characteristic of that resulting from a mixture of sulfides that contained a high pyrite content.

ORE GENESIS-CONSLUSIONS

The ore deposits in the Inspiration-Miami area show the inter-relationship of intrusive activity, hydrothermal alteration, metallization, and supergene enrichment. Hydrothermal alteration and ore deposition are late in the intrusive sequence, occuring after the formation of the breccias and intrusion of the finer grained igneous material.

The characteristics of the ore deposit are suggestive of a typical porphyry copper system, consisting of disseminated and stockwork veinlet sulfide mineralization emplaced in schist and intrusive rocks that are hydrothermally altered into roughly concentric zonal patterns. The configuration of these alteration zones and subsequent mineralization is apparently not related in detail to the distribution of the quartz monzonite porphyry bodies, but these processes were influenced to some extent by the porphyry trends.

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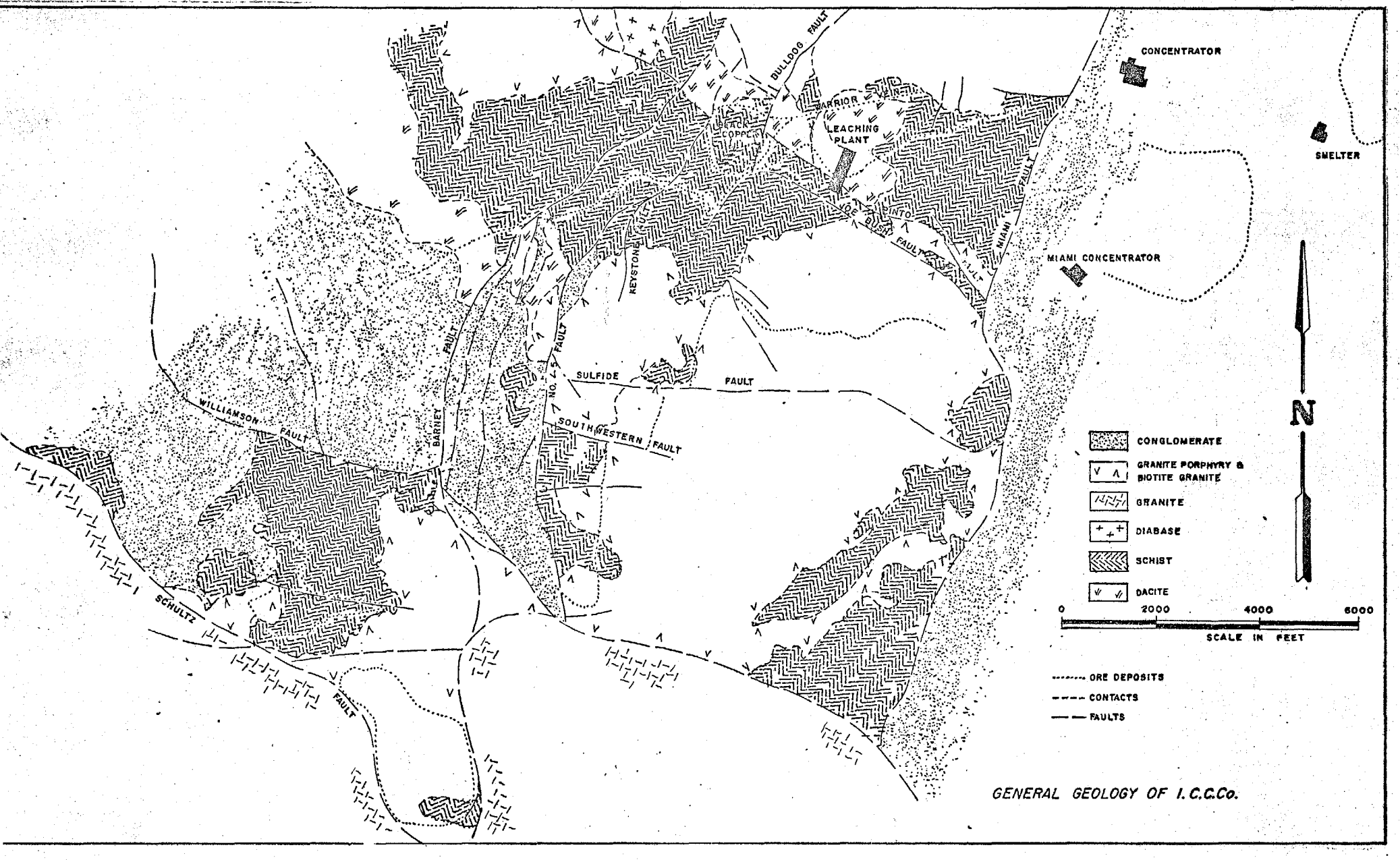

GENERAL GEOLOGY OF I.C.C.Co.

MINERALOGY OF THE INSPIRATION MINE AREA

by David W. Johnson

In discussing the mineral assemblage of the Inspiration Mine area the first major division would logically be that of rock type minerals and ore and ore associated minerals.

The rock type minerals and their alteration products comprise a very large and complex group of primary, hydrothermal and supergene minerals. In addition to the alteration products of argillization, serictization, and silicification, there exists alteration products of biotization, orthoclazation, epidotization and surface oxidation and leaching. It is sufficient to say that the list of minerals covered in this group is large and non-economic in nature.

The ore and ore associated minerals present in the Inspiration Mine area, that is those minerals with some economic conotation, should be divided into three main divisions: hypogene minerals, which are minerals formed by generally ascending solutions; supergene minerals, which are minerals that have been formed by generally decending solutions; and oxidation products. At Inspiration the original hypogene mineralization of the porphyry and schist was of low tenor and the ore zones as we know them today were dependent upon supergene enrichment.

In the hypogene mineral classification we have chalcopyrite (which is considered to be the most important hypogene copper mineral), minor bornite, minor chalcocite, molybdenite, magnetite, traces of gold and silver, very minor galena and sphalerite, and pyrite is here included because of its role in the enrichment cycle. The molybdenite, it should

be noted, is present in molybdenum-quartz veins and veinlets which cut all other mineralization and it is therefore considered to be the last phase of the hypogene metallogenetic phase.

The supergene minerals would include chalcocite (the most abundant copper sulfide mineral), covellite, chalcopyrite, minor amounts of bornite and native copper. Chalcopyrite has been included here because of its rather consistant occurrences of thin films on pyrite beneath the chalcocite blanket. Bornite was included because it is a product which will readily result from copper sulfate solutions in contact with chalcopyrite under the proper conditions.

The oxidation products consist of chrysocolla (the most important oxide copper mineral), malchite, azurite, and minor copper pitch, cuprite, chalcotrichite, brochantite, pseudomalachite, libethenite, lindgrenite, ferrimolybdite, hematite, goethite and limonite. Chalcanthite, a product of mine water seapage, is found in the underground mine workings and on many of the undercut faces which are exposed in the Thornton Pit and Live Oak Pit. It is interesting to note the more important recent discoveries in the area such as the OxHide, Blue Bird, and Montezuma ore zones seem to represent a former chalcocite blanket which has been oxidized in place.

By: William W. Sorsen

INSPIRATION'S OXHIDE MINING AND LEACHING OPERATION

INTRODUCTION

The OxHide Mine, formerly known as the Shulze Copper Deposit, is located on the lower slope of the Pinal Mountains at an elevation of 4,000 feet. The property adjoins U. S. Highway 70, about three miles Southeast of Miami, Arizona.

HISTORY

The first claims of record were located by the Shulze family in 1894 and ownership has been sustained by them to the present. No record of production is available but undoubtedly selected ore was shipped from some of the shallow workings scattered over the property. During the period from 1923 to 1962, several mining companies leased the Shulze property and drilled a total of 58 exploration holes. This drilling indicated the presence of two separate mineralized bodies.

In 1964, Inspiration Consolidated Copper Company obtained a lease and option from the Shulze family. Inspiration drilled 21 diamond drill holes and 55 churn drill holes to further delineate the mineralized zones.

GEOLOGY

The OxHide Mine is along the North contact of the main intrusive stock of Shulze granite and granite porphyry which extends Southwestward from the Miami-Inspiration area.

Copper mineralization is confined chiefly to the block of ground which lies to the East of a strongly faulted zone, known locally as the Shulze fault. This fault strikes almost due South where it crosses U. S. Highway 70 and trends roughly N85°W along the upper reaches of Needle Creek. The dip varies from 20° to 35° to the Northeast.

The Upper OxHide deposit is adjacent to the quartz porphyry contact in highly fractured and altered Pinal schist.

The Lower OxHide orebody is all in shattered quartz porphyry. These two rock types form the hanging wall of the Shulze fault, overlying barren Shulze granite in the footwall.

Surface exposures of both the schist and quartz porphyry show evidence of having been weakly mineralized by primary sulphides. In general, the schist orebody has a higher copper content than the porphyry orebody. The copper mineralization in the porphyry, however, is more widespread both in lateral and vertical extent.

MINERALOGY

In both of the mineralized bodies, the rocks are stained by copper and iron oxides and the numerous fractures are coated with copper silicates and copper carbonates. The principal supergene minerals are chrysocolla, malachite, and azurite. Melaconite, tenorite and cuprite are found as minor

incrustations along fractures, and occurrences of native copper are noticeable along the more intensely crushed zones.

Hypogene mineralization has been intersected in the deeper diamond drill holes. Pyrite is the most abundant sulphide followed by Chalcopyrite. Molybdenite is found in typical association with quartz occurring as minor blebs and disseminations. The protore of these deposits is very similar to that of other deposits in the district, assaying between 0.08 and 0.25 percent copper. Evidently there was very little migration of copper with subsequent weak supergene enrichment.

UPPER OX HIDE PIT

Early in 1968, Inspiration completed its evaluation of the OxHide orebodies and decided to go ahead with the construction of the launder precipitation plant and associated facilities. Three collection reservoir dams were constructed in the natural drainage areas to the West of the two orebodies. In July, mining was started in the Upper OxHide pit with tractors and scrapers. While construction was in progress, the number one heap area was cleared, graded, and sealed with two coats of asphalt.

Heap emplacement was started in the number one heap area. Initial heaps were emplaced in stair steps up the canyon so that the deepest part of the heap was 40 feet. Subsequently overlays were emplaced in 20-foot lifts. The objective was to create a large surface area as quickly as possible, to allow increases in the volume of solution applied.

Early heap construction was done by building a fill from the finished top elevation by casting. Scrapers spread their load near the crest of the dump and two graders windrowed the material over the edge. Two tractors with dozers and parallelogram rippers were used to loosen the rock and load scrapers. Four 24 cubic yard scrapers were adequate at the start of the job due to the short haul and favorable grades.

The schist ore in the Upper OxHide pit was well shattered but the fractures had been re-cemented in some areas. Rippability varied from easy to impossible. On two occasions, it was necessary to drill and blast hard ribs to maintain the continuity of the mining operation.

Excessive rainfall in 1972 caused flooding of the Upper OxHide pit to the extent that mining was temporarily suspended. The decision to move the mining operation to the Lower OxHide pit was hastened by the need for a large capacity storage reservoir to contain the excess discard water generated by runoff. By the time the water system was back in balance, the Upper OxHide pit contained 200 million gallons of water, standing at a depth of 190 feet.

During a 4 1/2 year period, 11.2 million tons of ore was deposited in the number one and number two leach areas. This ore contained 87.7 million pounds of copper, of which 40.4 million pounds, or 46%, has been recovered.

LOWER OX HIDE PIT

In January 1973, mining was started in the Lower OxHide pit. Drill holes indicated the quartz porphyry orebody contained 33 million tons assaying 0.296% copper with a 0.5/1 (waste to ore) stripping ratio. This orebody

is a fractured granite porphyry with two hard zones running the length of the pit. It was necessary to initiate a drilling and blasting program to avoid abusing the tractors that were attempting to rip this hard material. Most of the ore in the Lower OxHide pit digs readily and with the aid of blasting to ease the hard zones, we should be able to mine the 12,000 tons of ore per day required to meet the production goal of 34,000 pounds of copper per day.

The Research Department determined from bench leaching tests that 47% of the total copper in the ore is acid soluble. The refractory remaining portion is probably copper oxides locked up in clays formed by altered feldspar. Ultimate recovery in the heaps will probably exceed the level indicated in the bench tests.

LEACHING PRACTICE

Three experimental leaching pads containing 10,000 tons of ore each were built to ascertain the best method of emplacement for the granite porphyry ore. This was thought to be necessary since mining the Lower OxHide ore generated an excessive amount of fines. Even the coarse material was all minus 4 inches in size.

The first heap was built with a stacker conveyor and subsequently levelled with a bulldozer and ripped to a depth of 4 feet.

The second heap was built as a typical truck dump, levelled and ripped.

The third heap was laid in with scrapers in 30" lifts, ripped, and filled another 30" until a depth of 10' was reached. It was then levelled and ripped.

All of the ore in these heaps was mined in the same manner by ripping before loading. Water distribution lines were laid on the three pads and a one percent sulfuric acid solution was applied. After the first 30 days, copper recovery from the truck and scraper pads was about 10%, while the stacker dump had produced 17% of the copper contained in the ore. After 120 days, all three heaps had produced 85% of the acid soluble copper. On the basis of this test, the so called "laying in" method was chosen as the most economical method of the three and was adopted for the Number Three leach area.

After fifteen months of heap emplacement on number three leach area, the percolation rate was 14 GPM per 10,000 square feet of heap area. This low percolation rate continues to be a challenge for the development of improved emplacement methods.

In order to keep copper production at an acceptable level, it is necessary to bring new heap areas on stream about every 12 days. Each new area requires about 130,000 tons of ore. Currently, we have 40 acres of heap area under irrigation. A leach cycle for an individual heap is a minimum of 120 days, after which the area is allowed to dry for 60 days.

Sometimes the surface of an old heap requires considerable preparation to restore permeability before the addition of a new 15 foot lift.

The minimum preparation consists of ripping in two directions with a nine foot Kelly ripper shank that penetrates six and a half feet.

Frequently it is necessary to cut deep slots with scrapers to penetrate hard pans and replace the excavated material with coarse rock to re-establish percolation.

Currently, we are planning to acquire a dragline with a 50' boom. Deep narrow slots will be excavated to release entrapped solution from perched water tables. These slots will then be backfilled with coarse rock to insure future drainage.

Two old heaps have been drilled with blast hole drills on a 50' grid to within 10 feet of the original ground. Two inch perforated plastic pipe was then placed in the holes to assure drainage even if the holes cave. Water is sprayed on the surface and allowed to drain into the drill holes. Water running down the side of the hole is more apt to soak out into the surrounding ground than if it is introduced into the pipe.

Solution containing from 0.6% to 3.0% sulfuric acid is distributed on the heaps with 2" polyethylene plastic pipe laid on 18' centers and drilled on 4' centers with 3/16" holes. These pipes are fed by a 3" plastic pipe manifold which is connected to the 12" transite main line with a rubber pinch valve.

Another method of distributing solution is to divide a large area into several parts with low dikes separating the sections. Then 2" plastic lines with pinch valves are laid to conduct solution to each area by running the water down a header ditch which feeds each furrow created by ripping.

PRECIPITATION PLANT

The OxHide precipitation plant was designed to process 1500 GPM of pregnant solution in 10 conventional iron launders. Solution is piped from the three collection reservoirs by gravity through 12" epoxy lined 150 class transite lines to a receiving box which empties into the feed launder of the plant. Stainless steel gates are used to control the flow of solution to each of 10 launders or to isolate a launder while it is being washed or serviced. Each launder is made up of two halves 4' wide by 30' long. A center wall 26' long separates the two sides leaving a 4 foot gap so that solution can flow down one side and return on the other side. A stainless steel punched plate false bottom supports the scrap metal in the cell and below this is a sloping floor that drains to a pneumatic discharge 12" gate valve. The punched screen is 52" below the top of the cell. Recently 8" was added to the top of the walls to increase the capacity of the plant to 2200 GPM. The 10 cells operate in series giving an effective length of 600 feet when all the cells are in operation. There is a 6 inch drop between cells to insure an adequate flow velocity.

The tail water is conducted to a settling sump consisting of a head section with a splitter gate to adjust flow into two separate pump sumps. Acid is metered into each pump sump separately as needed. The number one sump is equipped with four vertical turbine pumps designed to pump against a 500' head. The number two sump is equipped with four vertical turbine pumps to

serve the lower heaps where the maximum head is 400 feet.

Fisher and Porter magnetic flowmeters are used to measure and record the flow of pregnant solution from the heaps and acidified solution pumped to the heaps.

Three 80-ton tanks for acid storage and a 260-ton iron storage area are located on the bench above the plant. A service road above this bench accomodates acid and iron truck deliveries.

Iron is fed by a front end loader through a hole in the floor to a pan feeder located in a short tunnel under the floor. This feeder discharges iron to a tripper conveyor which traverses the center line of the iron launders. A chute with splitter gate conducts iron to the left or right chute to distribute iron to any part of the plant as needed.

In practice, it is necessary to wash the cement copper out of 5 to 8 cells per day, depending on the grade of the pregnant solution. To wash a cell, gates are dropped into the feed launder to isolate the cell. The 12" pneumatic dump gate is opened to discharge the cement copper slurry into the decant sump. Two high pressure washing hoses are then clamped to a rail support at one end of the cell and the cell washer proceeds to wash the cement copper off the remaining scrap cans through the perforated floor into the decant sump. After both sides of the cell is washed, the discharge gate is closed and the cell is recharged with scrap cans. On day shift, the cells are washed and recharged with scrap cans, but only a few cells are filled with solution and put back on stream. Usually two cells are reserved for afternoon shift and two more for graveyard shift to be put into the circuit as needed to maintain a low tailing.

The cement copper is allowed to settle in the decant sump after which the water is pumped back to number one launder cell. The afternoon shift plant attendant removes the cement copper from the decant sump with a front end loader and piles it on a draining slab. The following day the cement copper is transferred to a drying slab and ultimately shipped via truck to the smelter. The cement contains about 20% moisture and assays from 80 to 90% copper when shipped.

PRODUCTION STATISTICS - Month of February 1974

Mine

Ore Mined (1.5 mile haul)	277,410 tons @ 0.30% copper
Waste Mined (0.5 mile haul)	94,185 tons
Average daily production	13,271 tons

Precipitation Plant

	To Launders	From Launders
Flow GPM	2061	2061
Copper GPL	1.40	0.06
H_2SO_4 GPL	5.4	5.1
Fe GPL	17.1	19.2
Fe^{+++} GPL	1.0	Trace

Copper Precipitated - 924,955 pounds or 33,036 pounds/day
Acid consumed per pound of copper - 11.34 pounds
Iron consumed per pound of copper - 1.95 pounds

FUTURE PLANS

The present mining rate will exhaust the ore reserves in about seven years. Leaching will continue as long as it is economically feasible, perhaps for an additional ten years period.

A small crew will be needed to operate the precipitation plant and to skim, trench, and drill the heaps and move solution lines as needed. The ultimate recovery will possibly be 60 - 70% of the total copper.

OX HIDE MINE LABOR FORCE - TOTAL PERSONNEL - 80

Administration
1 Superintendent
1 Mining and Leaching General Foreman
1 Maintenance General Foreman
1 Clerk

Shop Crew
1 Shop Foreman
3 Welders
9 Heavy Equipment Mechanics
1 Pump Mechanic
7 Mechanic Helpers and Servicemen

Mine Crew
4 Shift Foreman
4 Grader Operators
8 Tractor Operators
20 Scraper Operators
4 Water Truck Drivers
1 Equipment Operator Instructor
1 Driller

Precipitation Plant and Leach Pad Crew
1 Shift Foreman
1 Leach Pad Attendant
1 Plant Operator
4 Plant Attendants
1 Tripperman
2 Cell Washers
3 Laborers

OX HIDE MINE EQUIPMENT

Mine
4 Model D9G Caterpillar Tractors with dozers and rippers
6 Model 631C Caterpiller Scrapers (24 cu yd)
3 Model S24 Euclid Terex Scrapers (24 cu yd)
1 Model 16 Caterpillar Grader
1 Model 16G Caterpillar Articulated Grader

1 Model R22 Euclid Water Truck - 4,000 gallon capacity.
1 Model 769 Caterpillar Water Truck - 4,000 gallon capacity
1 Model 769 Caterpillar Lube Truck
1 Model 750 Chicago Pneumatic Rotary Drill
2 Model D1012 - 3" Pacific Centrifugal Pumps
Powered by 90 H.P. Deutz Diesel Engines - Trailer Mounted

Precipitation Plant

1 - 750 KW Transformer and Power Center
1 - Model 950 Caterpillar Loader - 2 1/2 cu. yd. bucket
1 - Model 7231 Euclid Terex Loader - 2 1/2 cu yd bucket
1 - WABCO 100 cu ft stationary compressor

Pumps - 316 Stainless Steel

No.	Make	Type	Stages	Head	GPM	H.P.	RPM	Application
3	Worthington	Turbine	2	400'	500	75	3550	On Solution
3	Worthington	Turbine	3	500'	500	100	3550	On Solution
1	Floway	Turbine	5	400'	500	100	3550	On Solution
1	Floway	Turbine	7	500'	500	125	3550	On Solution
1	Worthington	Centrifugal(S)		300'	300	40	1750	Decant
1	Layne Bowler	Turbine	2	300'	250	25	1750	Make Up

Transportation

2 - 3/4 Ton Pickups
4 - 1/2 Ton Pickups

INSPIRATION OPEN PIT OPERATIONS

SUMMARY

By

James H. Lundy, Jr.
Open Pit Superintendent

Open Pit Division - A.I.M.E.
Spring Meeting
May 10, 1974
Inspiration, Arizona

UNIQUE OPERATION

Although Inspiration's mine has much in common with its sister operations throughout the Southwest, in the employment of mining equipment and the techniques of their several operations, it is unique in the ore production area.

Here at Inspiration, we must produce three distinct types of ore, along with two types of heap leaching material, in addition to our stripping of barren waste. The three types of plant feed are (1) oxide - which term is applied to all acid soluable minerals; specifically Chrisocolla, Azurite and Malachite. (2) Sulfides - mostly chalcocite and some chalcopyrite - known as Direct Mill Feed; and (3) Dual process ore which contains such an intimate mixture of the first two that it must be treated first by vat leaching, and then flotation of the Leaching Plant tails.

In addition to the plant grade ores, we also mine heap leaching waste as a part of scheduled production. Then, of course, the barren waste which must be dumped in non-leachable areas. These, then, are the five products which we must separate continually on the basis of our ore control information.

To this we add one more operational activity - for every tank of "Discard" which is the oxide ore that is vat leached only in the leaching plant, we must, at the completion of the leaching cycle, send one fleet of haul trucks to dispose of the tailings.

One might conclude hastily that this should not be too great a problem, since the non-leachable overburden will be encountered at the surface - the leachable waste on the upper benches - the oxide ores on the intermediate levels - grading into a dual product zone, which the direct mill feed sulphides being found at depth.

Unfortunately this is not the case. Non only do we have sulphide ores high in the column in certain areas, but also previous mining by block caving has

moved large blocks of ground down - the result of which is that we find pipes of ore and waste throughout the mine - with all products being frequently encountered in close approximation, all on the same bench. Dealing with this complexity makes scheduling production from Inspiration's Mine a real challenge.

SCHEDULING PRODUCTION

Ore reserves are continually upgraded by the Mine Engineering Department in cooperation with the Geologists.

Exploration drilling, blast hole drilling, and analysis of recoveries of immediate past mining are used to supply management with a detailed analysis of all available reserves. This, in turn, is fitted to the overall plant capabilities, and modified by economic consideration to constitute annual goals.

Thus the Mine is tied to its own long-range commitments, 5-year schedules, annual production goals, and finally the quarterly and monthly mining plans - as approved by management.

The Mining Department is broken down into four areas of activity: Engineering, Mining Operations, Dump Leaching and Maintenance. Working closely with Engineering, Operations must ask Maintenance for a minimum level of availability of all necessary equipment to meet the production requirements of both plant and Dump Leaching.

Currently our schedule is this: 27,500 tons per day ore and 64,500 tons per day waste. The ore is broken down into an average of 6,800 tons per day Dual, 7,200 tons per day Discard and 13,500 tons per day sulfide. Since all the dual also goes to the concentrator, this amounts to a mill feed of 20,700 tons per day. The Leaching Plant generally keeps twelve tanks in circuit on a 9-day cycle.

It is obvious, then, that to maintain this rate of production; i.e., 6 discard tanks per 12-tank cycle, one fleet of haul trucks will be tied up at the Leaching Plant approximately fifty percent of the time.

To maintain this rate of production, the mine runs three shifts, seven days a week, with four equally staffed crews. Each crew works twenty-one shifts every four weeks. This provides equal strength of crews, even rotation, equal distribution of scheduled overtime, no short changes - one long change each four weeks and one crew off at all times for emergency call-outs.

PRODUCTION SEQUENCE

Inspiration, at this time, has three active mines combined into one operation. They are the Thornton Pit on the East, the Live Oak Pit on the West, and the Red Hill Mine just North of the Live Oak. Distribution of shovels indicates fairly well the rate of production from these three divisions.

We now have eight Electric Shovels, three 10-yard loaders and forty-two haul trucks, divided into four major fleets. In the Thornton Pit, we have one 5-yard and one 6-yard shovel. In the Live Oak, we have our 8-yard shovel and on the Red Hill we now employ one 5-yard, one 6-yard, and three 10-yard shovels distributed from the 4050 bench to the 3750 bench.

Each pit is developed initially by one main haul road breaking off to 50-foot benches. These 50-foot benches are in turn split into 25-foot benches wherever loaders are intended to be used or sorting is required. Each pit has some of both 50 and 25-foot benches.

Production crews are scheduled to run five loading units each shift. The crews consist of four 2-man shovel crews, one loader operator, three wheel dozer operators, two pitmen (laborers who may flow to any operating classification), three water truck drivers and twenty-six haul truck drivers. Although we would prefer to operate four shovels and one loader each shift, at times we find it necessary to schedule a second, and even a third

loader in place of shovels - to provide the required plant feed, or maintain some particular development schedule. When we are discarding from the Leaching Plant, one unit is of course shut down in the pit.

The crushing schedule for ore is 2 1/2 shifts for six days a week, two shifts on the seventh day. This allows for four hours six days for daily maintenance and eight hours on one day for major maintenance work each week to the crushing and conveying system. Of this scheduled crusher operating time of 20 hours/day, we actually average approximately 18 hours. This means that the crushing system must average 1,530 tons per hour. The maximum rate sustained for any length of time is 2,000 tons per hour. Generally we try for 1,800 tons per hour. To produce this amount of ore, we may use one 10-yard shovel; the 8-yard shovel with any other unit; two 6-yard units; one 6-yard unit plus one loader; two loaders or two 5-yard shovels with about 1/2 the production of one other small unit. As the larger units are seldom in ore, we usually end up operating two and three units in ore in some staggered sequence - thus sacrificing shovel efficiency to gain quality control in ore blending.

Loading capacities of the various units in well fragmented bench blasts have recently averaged the following tonnages:

TONS PER SHIFT	10-yard shovels	12,000
	8-yard shovels	9,000
	6-yard shovels	6,500
	5-yard shovels	5,000
	10-yard loaders	5,600

Frequently we choose to dig the undercut material - or other ground badly broken by movement - without blasting. While this reduces unit capacities by 10 to 15 percent, it greatly improves sorting and is a useful technique.

The Haulage fleet, as of April 1, 1974, consisted of:

1 - 105 ton - 2772 Dart
8 - 85 ton - 85-C WABCOs
3 - 85 ton - M-85 - Lectrahauls
11 - 75 ton - 2662 Darts
19 - 40 ton - 37SL Darts

This gives a weighted mean-haul capacity of 65 tons per unit. Considering the average lengths of hauls and the mean capacity of our loading units, we may calculate a production potential of 120,000 tons per day. So far, however, our high has been a little over 116,000 tons. Theoretical maximums are difficult to achieve.

Running ore units too lightly covered, to insure proper grade proportioning, cleaning up final toes, and frequent shovel moves all reduce shovel efficiency. Although we have scheduled crews to operate 105 loading units per week, scheduled discard and predictable absenteeism, along with unscheduled downtime reduces this to an average of 94 shifts per week.

IMPORTANT ANCILLARY OPERATIONS

Rock breaking here is rather conventional. Drilling is done by three C.P. 750 truck mounted rotary drills. These drill an average of 550 feet per shift of 9-inch blast holes. Drilling is scheduled on two shifts, seven days per week. Blasting agent is ANFO for dry holes, and bagged slurry for wet holes. Fifteen percent of our holes require slurry. All powder is bagged in 50-pound units. (We will be converting to ANFO bulk loading later this year.) We use 3,500,000 pounds per year, and break 4.5 tons per pound. Only 70% of our muck requires blasting, as much of the undercut material digs readily with the larger shovels.

Secondary blasting is a very minor part of the operation, and when it is necessary, it is done by drilling 2 1/2" percussion holes with a truck-mounted jumbo drill. These are shot with 2 x 12 stick powder at the end of day shift when danger of fly rock is minimized.

Building haulage roads, drill roads, and maintaining benches is accomplished with three D-9 bulldozers and one D-8. Three CAT, Number 16, motor graders operate 15 shifts per week. Haul road berms, bench and road repairs are maintained by use of a CAT 988 6-yard loader.

Flood waters which cannot be diverted around the perimeter of the pit, are allowed to drain to the bottom levels where whey gradually seep into old underground workings and eventually are pumped to the surface for use in leaching, from either our Live Oak or Inspiration shafts. Surface sumps are occasionally necessary, and they are pumped by four 200-foot head by 500 GPM trailer-mounted portable pumps.

It is not possible to mention here all the services rendered to Operations by Maintenance and Engineering, but this report would not be complete without mentioning the close cooperation required for an efficient operation from these supporting units. Operations ask Maintenance to provide a minimum of 28 haul trucks per shift; five of nine shovels; two of three loaders; four of six wheel dozers; four of six water trucks; and eight out of ten service vehicles each shift. In addition to this, we expect two of three drills on a two-shift basis. Two out of three bulldozers around the clock; and two of three motor graders each day shift.

Indeed, the success of Operations is dependent as much on mechanical excellence as on mining expertise.

By Fred D. Rice, Chief Planning Engineer

LEACHING OPERATIONS AT INSPIRATION

Leaching of waste dumps, old cave areas and heap leaching have provided Inspiration with a good source of supplemental copper. Through January 1, 1974, 230 million pounds have been recovered from this leaching. As far back as 1926, reference is found to the possible heap leaching of a waste dump at the sulphide tunnel, located on the South side of our Live Oak ore body. Natural leaching in the Block Caved areas of our ore body began to show up in 1939 when an iron pipe column from the underground mine, that had lasted 24 years, was eaten up by corrosive mine waters. The next column lasted about a year and others even a shorter length of time until lead lined iron pipe was installed in August 1941. No doubt this natural leaching was greatly accelerated by the 28.92 inches of rain which fell from October 1940 through March 1941, about two and one-half times the average for that period. Just before this final column was installed, a 150' launder filled with scrap iron was constructed to strip the mine water of some of its damaging copper before it entered the pump and pipeline.

By 1944, over 200,000 pounds of copper from these underground launders had been shipped to the smelter. Sprays of fresh water were installed over a limited area in the old underground caves and water was applied. Within four days there was a response of solutions running 15 GPL, more than the limited launders could accommodate. After adding additional launder capacity, water was again intermittently applied during the next 17 months. From the normal mine flow of 50 GPM, plus an estimated 6 1/4 million gallons of leach water, a quarter of a million pounds of copper were recovered during these 17 months. This led management to begin formalizing plans for the active leaching of mined out areas.

All earlier solution collection had been done on the Inspiration Division 600 level (3342 elevation). In an effort to get under more of the ore body, the new facilities were constructed on the Inspiration 850 level (3101) elevation). Here it was necessary to open up some old drifts, build dams and ditch boxes for water collection and control, and a pump station and sump. Both the pump station and sump were excavated below the normal haulage level, as an additional safety factor. From the pumps, solutions went up the shaft almost 700' vertically and then across to the leaching plant, 300' to the South for treatment. New launders and supporting facilities were constructed at the Leaching Plant for this additional flow. By April 1950, everything was ready and acid solutions were applied to the old cave areas. During this same period Open Pit mining was started at Inspiration, removing some cave areas from possible leach but adding waste dumps over others that were available for leaching. This method was continued until 1965 when down the hole leaching or L.I.D. (Leaching in Depth) was started. Churn Drill holes were drilled to within 40' of the old undercut elevation, perforated 4" P.V.C. was then installed to release the leach solutions at the desired depth. Excellent results were obtained for a number of years by continually adding new holes to the project. Open Pit mining has now limited the area available for this type of leaching and it will be phased out later this year with the new Joe Bush Pit. The final

phase of the underground collection of solutions was the automation of the 850 pump station during 1973, eliminating the need for a pumpman around the clock. A Motorola Control Panel was installed in the hoist house that allows automatic or manual control of both underground pumps and also provides selenoid activated-air operated valves on the two main sources of leach solutions. Power for these operations is 124 volt D.C. from constantly charged wet cells. Monitoring points covered by this system are: High and Low solution levels in the main sump, an alarm on the pump cooling water sump, and monitors that check vibration, cooling water and bearing heat on both pumps. Also located underground is an air receiver with sufficient supply to operate both valves several times in the event of a power failure. Problems are indicated at the control panel by a horn and flashing lights.

As open pit mining progressed, efforts were made to segregate waste into potential leach and non-leach dumps. During 1962 facilities were constructed in Davis Canyon, at the base of #5 dump, to collect solutions and pump them back to the leaching plant for treatment. This more than doubled our capacity to leach and led to the construction of still another plant to leach the Live Oak Dumps. The Live Oak Plant had 10 iron launders in series, iron charging facilities, drying pads for cement copper and pumps for returning solutions to the dump or discarding them to Webster Lake. It was soon discovered that 10 cells were not needed in series, so the plant was split in half into two 5 cell units - effectively doubling capacity. Later a tower, which is really a vertical launder, was also added. Here solutions are injected through nozzles in the bottom and flow upwards through tin cans and out around the top. While this does increase capacity, it is not nearly as efficient as launders, having a tail of 0.25 to 0.30 GPL. To improve this, tail water goes through a scalping launder to remove residual copper. High concentrations of ferric iron in the OFF solutions cause excessive consumption of tin cans. To help overcome this, all solutions from the Live Oak Dumps now go through a "heavy iron" launder before the regular launders to reduce ferric iron. Scrap iron for this launder is obtained from around the plant or bought at a considerably lower price than shredded iron. With both of these facilities on the South side of mining operations it seems only natural that something would be needed on the North side to keep haulage distances to a minimum. A dam and pipeline were built above the small Black Copper Pit and leach waste from Black Copper and the Red Hill Pit have been leached there for about five years. Solutions from here are conveyed by gravity to the leaching plant for treatment. ON solutions are pumped back up to the dump from the leaching plant. The Black Copper or #24 Dump is rather limited in capacity so another larger operation is being constructed in Willow Springs - North and West of Black Copper. This new plant, featuring three towers of modified design, is scheduled to go into production later this year. It will serve a vastly larger dump area that can accomodate a major portion of the leach waste from Red Hill. Presently being applied to the various projects are the following gallonages: L.I.D. 525 GPM with all OFF solutions pumped to the leaching plant; Davis Canyon 1600 GPM with 1/3 of the OFF solutions pumped to the leaching plant and 2/3 to Live Oak launders; Live Oak Dumps 2100 GPM with all OFF solutions treated at Live Oak and Black Copper 425 GPM and all OFF solutions treated at the leaching plant. Normally we expect to average about 95% return on our leach solutions to the treatment plants.

Present leaching operations call for dumping waste in overlays, over previously leached areas and whenever possible new lifts are limited to 50' in height. These areas are then leveled, bermed and ripped by dozers. Two inch plastic hose is laid on 18-foot centers and 3/16" holes are drilled in these plastic hoses every four feet. Since most of these dumps take solutions very well, average application rate is 30 GPM/10,000 sq. ft.

Acid is generally applied at a rate of 6 - 10 GPL with off solution acid desired in the 1/2 to 1 GPL range to maintain good iron launder efficiency. Some slope leaching has recently been done by spraying with generally excellent results. Number 5 Dump alone had 5 million tons in the slope that had not previously been leached. Some drilling has also been done to introduce solutions into the slopes and to drain parts of the Live Oak Dumps where old haul roads have made apparent water courses. Wetting agents have been tried but results at best are inconclusive.

With Inspiration shortly to have a good supply of low cost acid from its new acid plant at the smelter, increasing the potential for new and additional leaching projects in the future.

ASPECTS OF
PIT SLOPE STABILITY
at
INSPIRATION, ARIZONA

By
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INTRODUCTION

Geologic structure plays a very important role in the orientation of pit slopes and their susceptibility to failure. Faults, joints, and bedding provide planes of weakness, which when daylighted, are prone to failure. To exemplify this statement, two areas of the Inspiration Consolidated Copper Company Mine at Inspiration, Arizona, are to be examined.

BACKGROUND

Physiographically, the Globe-Miami district, of which Inspiration is a part, is situated in the middle of the Mountain Region of the Basin and Range Province. Geographically, Inspiration is situated 65 airline miles East of Phoenix and 85 miles North of Tucson (Figure 1).

The Inspiration property began production in 1915 as a block caving operation. Open pit mining began in 1948 and continues through the present. Underground mining ceased in 1954. The present operation is designed for 25,000 TPD and requires three distinct ore products. These are 1) a sulfide ore, subjected to flotation, 2) an oxide ore, treated by vat leaching, and 3) a mixed oxide-sulfide ore treated by both flotation and vat leaching. The resultant products of the two processes are in turn treated by smelting or electrowinning. The three ore products are mined from three mines - the Thornton, Live Oak and Red Hill (Figure 2).

GEOLOGY

There are four principal rock types at Inspiration, these are shown in Table A and Figure 3.

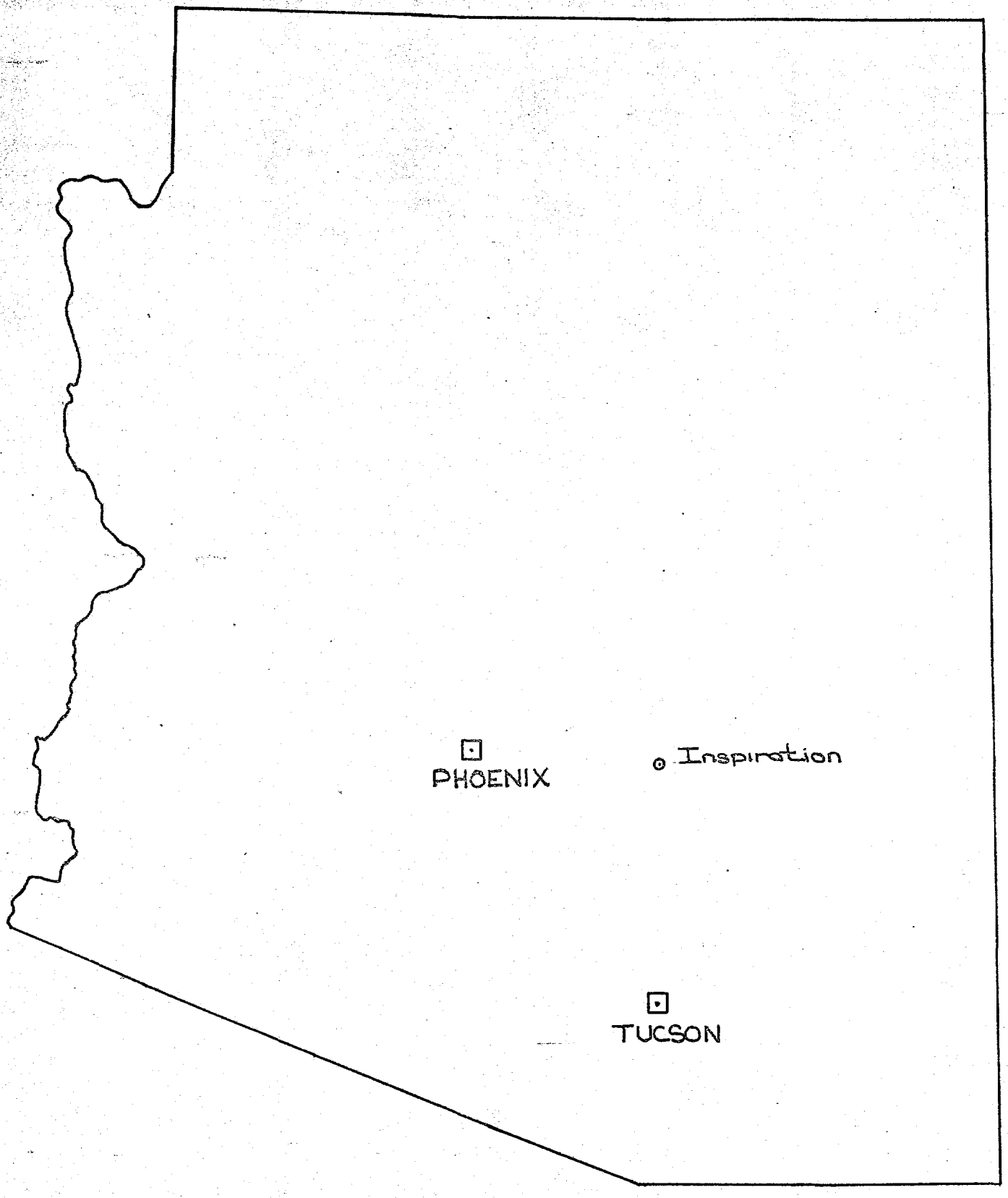
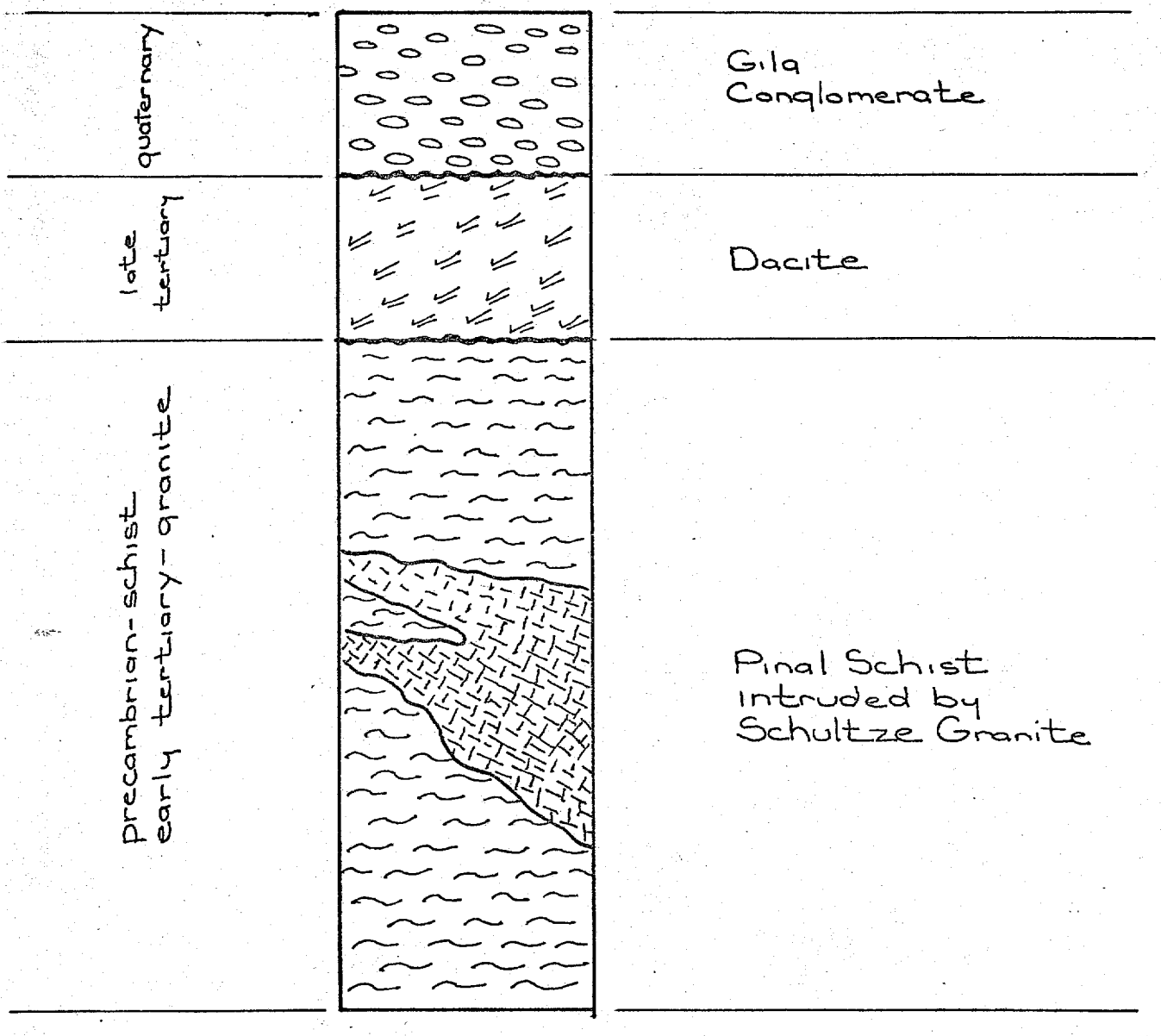
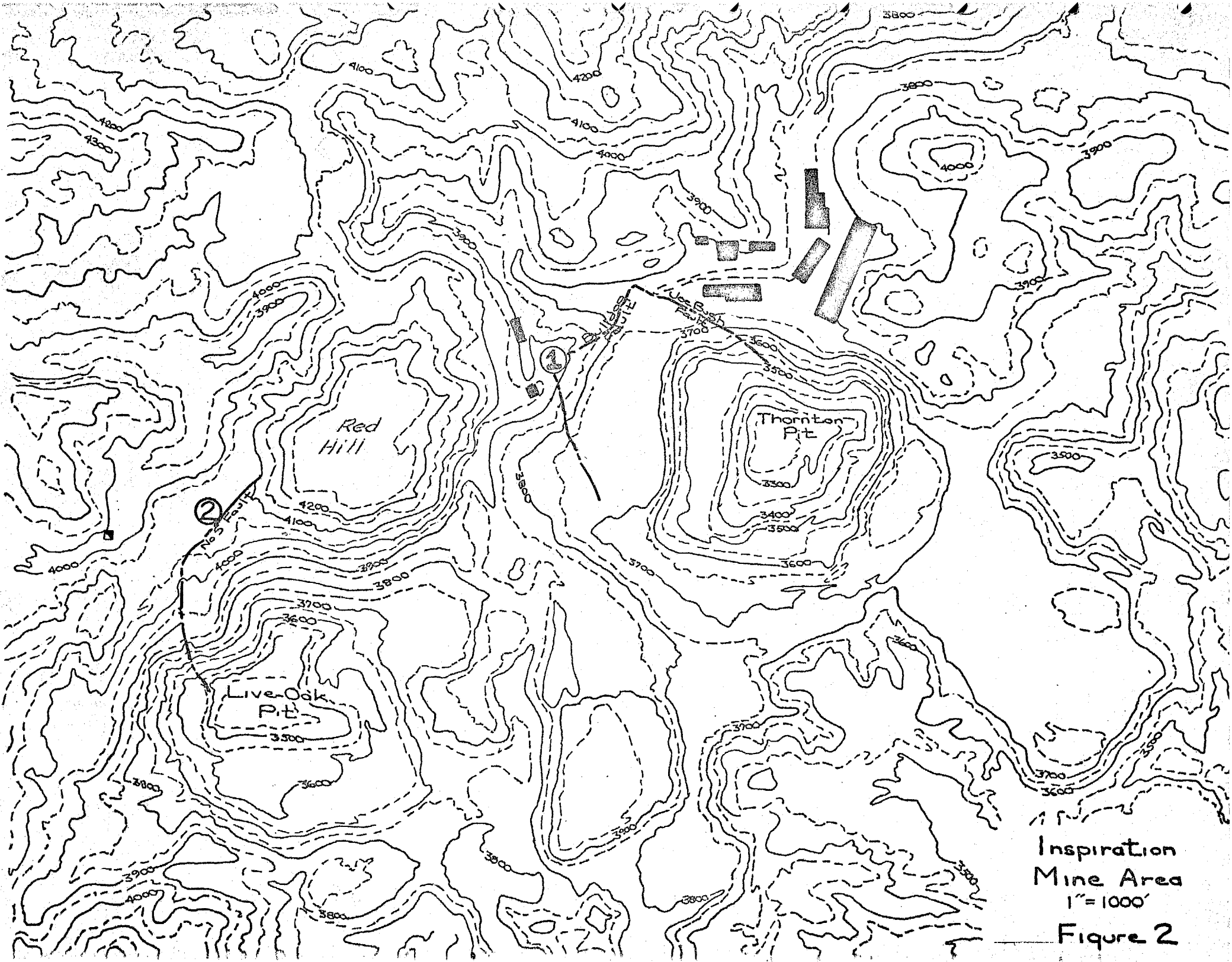


Figure 1
Location Map
Inspiration, Arizona

General Stratigraphic Column at Inspiration, Arizona

Figure 3





Inspiration Mine Area
1" = 1000'

Figure 2

TABLE A

TYPE	AGE	REMARK
1) Pinal Schist	Precambrian	Host Rock for approximately 1/2 of the ore mineralization
2) Schultze Granite	Early Tertiary	Intrudes the schist-host rock for approximately 1/2 of the ore mineralization
3) Dacite	Late Tertiary	Small remnants of dacite cap present
4) Gila Conglomerate	Quaternary	Moderately, cemented by caliche

There are two principal directions of faulting in the areas to be examined. Pre-ore faulting trends NW and dips steeply SW to 45° NE. Post-ore faulting trends NE and dips 25° to 60° SE. The Bulldog and Number Five Faults are of this type (See Figure 2). The Bulldog offsets the ore-body between the Thornton and Live Oak Pits, and ranges between 200 and 300 feet thick. It is apparent that all displacements in the areas of interest are due to normal faulting.

FAILURE BLOCK GEOMETRY

The pit wall failure in the northwest corner of the Thornton Pit (Figure 4) is governed by the Bulldog and Joe Bush Faults. The Bulldog strikes N 24° E and dips 28° SE while the Joe Bush strikes N 30° W to N 50° W and dips 83° SW. In places the hanging wall of the Bulldog has been penetrated into the fault zone by mining. Where the pit slope has been flattened, a basal failure plane exists. This failure plane strikes N 20° E and dips 13° SE. This plane is a joint set established by running a detail line along the north side of the pit in the hanging wall of the faults. The existence of such a mode of failure has been established within a probability of 98%.

A second failure block exists on the northwest corner of the Live Oak Pit (Figure 5). This zone of subsiding ground is bounded on the north and west sides by the Number Five Fault. This fault trends N 40° E, 35° SE on the north side of the failure and turns onto a N 18 W, 85° NE trend on the west side. A set of headwall cracks forms the east margin of the subsiding ground. These cracks are comprised of two intersecting predominant joint sets. The first trends N 70° W, 68° NE and the second, N 69° W, 45° SW.

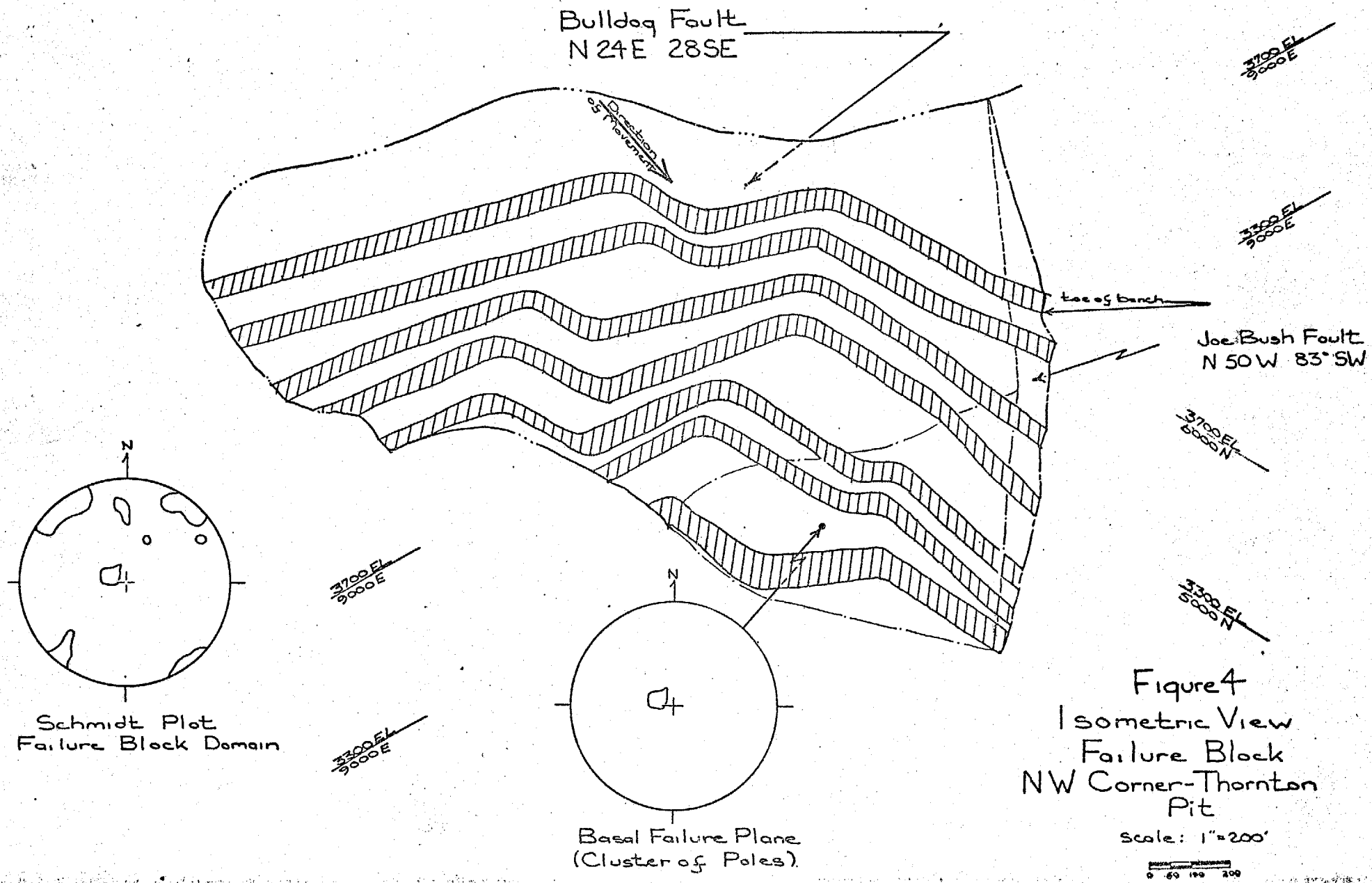


Figure 4
Isometric View
Failure Block
NW Corner-Thornton
Pit
Scale: 1" = 200'

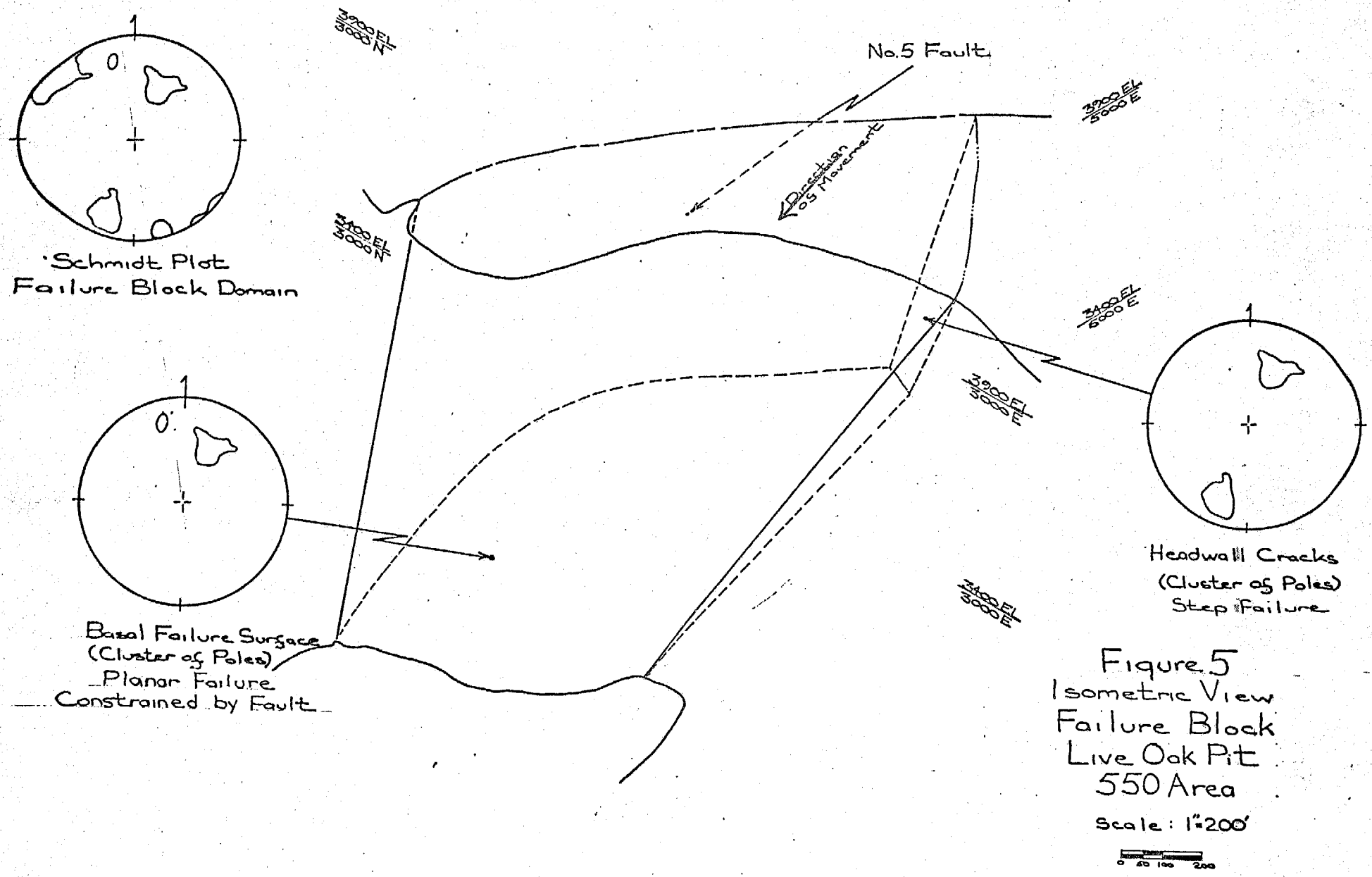


Figure 5
Isometric View
Failure Block
Live Oak Pit
550 Area

Scale: 1"=200'

Since the fault is not daylighted, the subsiding ground must be moving along a basal plane of failure. From a detail line along the north wall of the Live Oak Pit, it appears, within a probability of 99.9%, that a joint set exists which strikes N 69° W and dips 45° SW. It is along the intersection of this joint set and the #5 Fault plane that the movement takes place. The direction of movement is S 18° E which coincides with the plunge of the intersection.

It has been suggested that the mode of failure is a joint set gently dipping to the northwest due to heaved ground found at the toe of the slide. From the evidence gathered from detail line work, this failure mode does not seem likely. There is no statistically valid evidence of the existence of such a joint set. The heaved ground is probably the result of broken, subdrilled rock being displaced by the sliding mass above.

HISTORY OF SLOPE STABILITY PROBLEMS

The moving ground in the northwest corner of the Thornton Pit is of critical importance due to the proximity of its margins to the primary and tertiary crushers. The formation of headwall cracks east of the primary crusher was first noted as early as 1956. Steel pins were placed on opposite sides of the cracks and displacement between which, readings were taken. Movement continued during the early and middle 1960's as the hill between the pit and the primary crusher was mined. Headwall cracks continued to develop, "slumping" occured along the Bulldog Fault and cracking with subsequent ground movement developed between the pit and Joe Bush Fault.

Until Fall, 1969, movement in this area was of little consequence. However, at this time the tension cracks had proceeded to within 10 feet of the primary crusher base and these cracks exist for approximately 1000 feet in strike along the Bulldog Fault zone. A consultant from Anaconda's Mining Research Department was solicited and among his proposals are the following:

1) Move the primary crusher as soon as possible.
2) Drill the Bulldog fault zone for assay valves.
3) Mine out the fault zone and establish a new pit slope 300 to 400 feet to the west.
4) Establish a monitoring system. (Figure 6).

A monitoring system of 26 triangulation pins was established.

In the meantime, an increasing number of tension cracks developed adjacent to the road across from the tertiary crusher. It was evident at that time this crushing plant, also, should be moved.

Movement accelerated during the winter of 1970. Tension cracks had progressed within 30 ft. of the tertiary crusher. The accelerated break-up of this area is partially attributed to a decrease in rock mass strength due to increased pore water pressure. This was caused by seepage from Webster Wash, to the north of the crushing plant and runoff from precipitation flowing into the newly formed tension cracks. To prevent seepage from precipitation, the cracks were filled and the area affected was graded. Intermittent stream flow in the wash was restricted by a small holding dam upstream.

To decelerate the movement of the failed block, it was decided to leave a buffer zone in the hanging wall. This was attained by maintaining

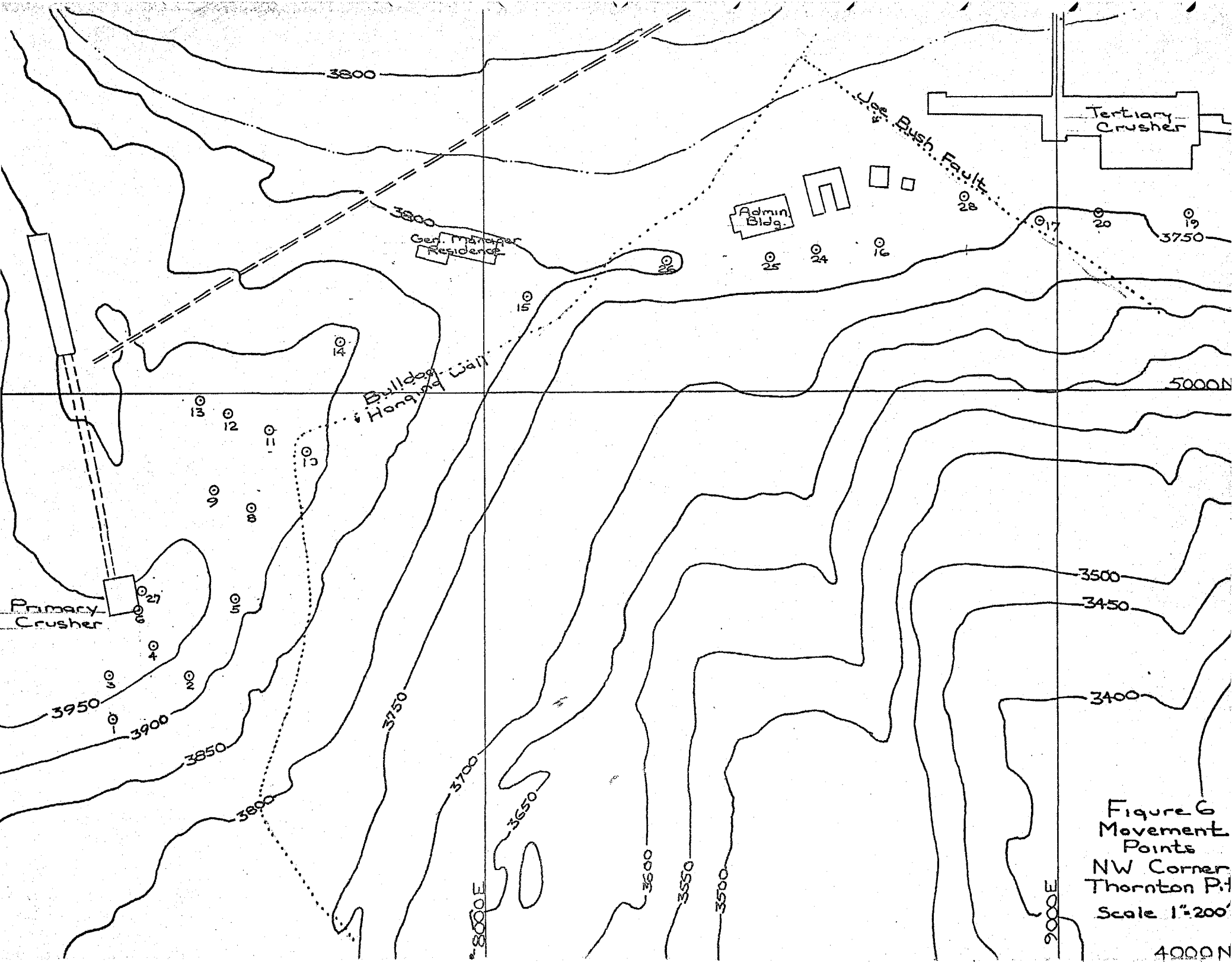


Figure 6
Movement Points
NW Corner
Thornton Pit
Scale 1"=200'

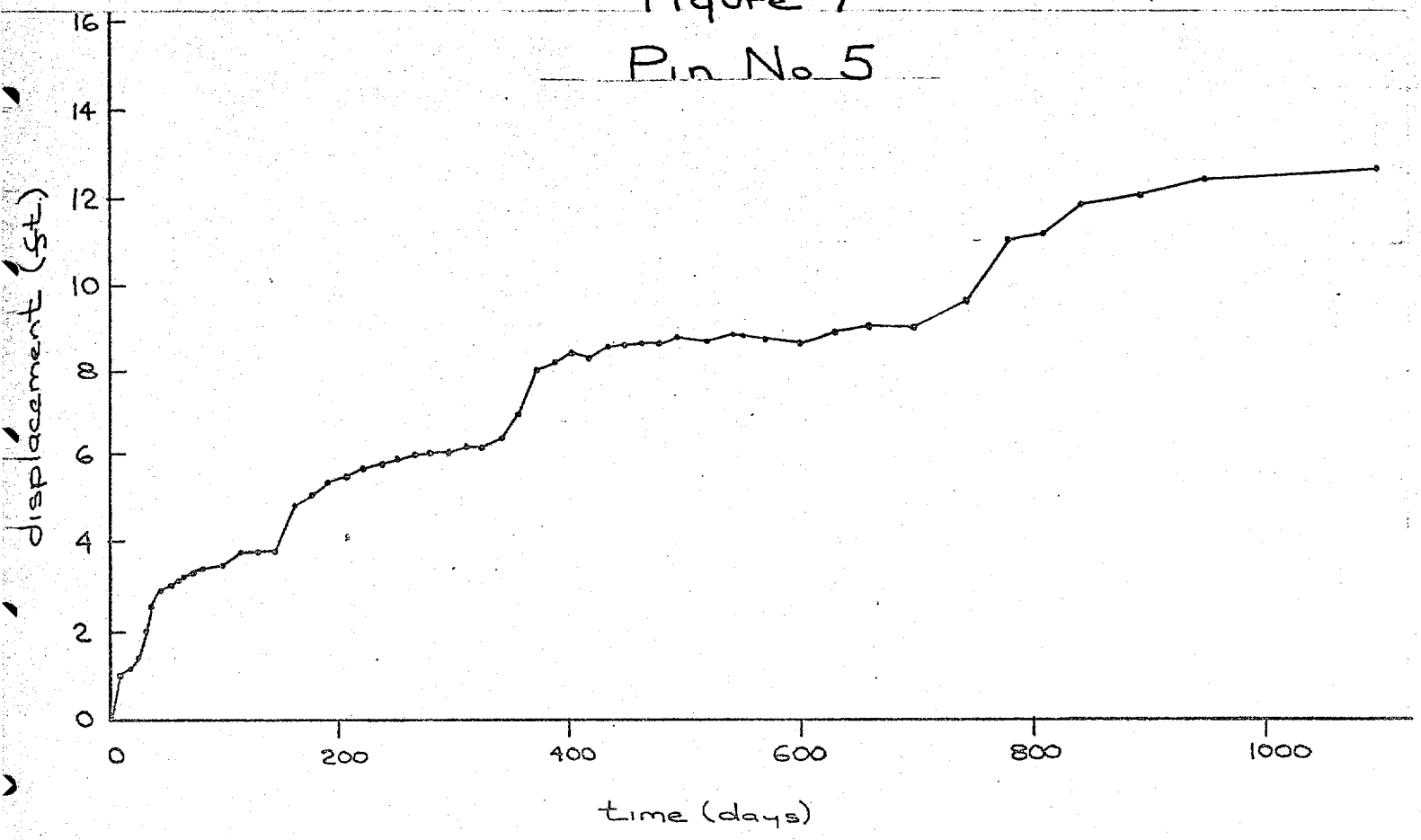
a flatter slope angle near the fault. This action was successful in ceasing movement along the north side of the pit. The cost was high in that a sizeable tonnage of previously stripped ore is tied up. This deceleration can be noted in the cumulative displacement plot of Pin No. 17 (Figure 7) which lies directly south of the tertiary crusher. Mining along the west side of Thornton Pit has, in places, progressed within a short distance of the Bulldog Fault resulting in movement. Pin No. 5 indicates the magnitude of displacement in this area (Figure 7).

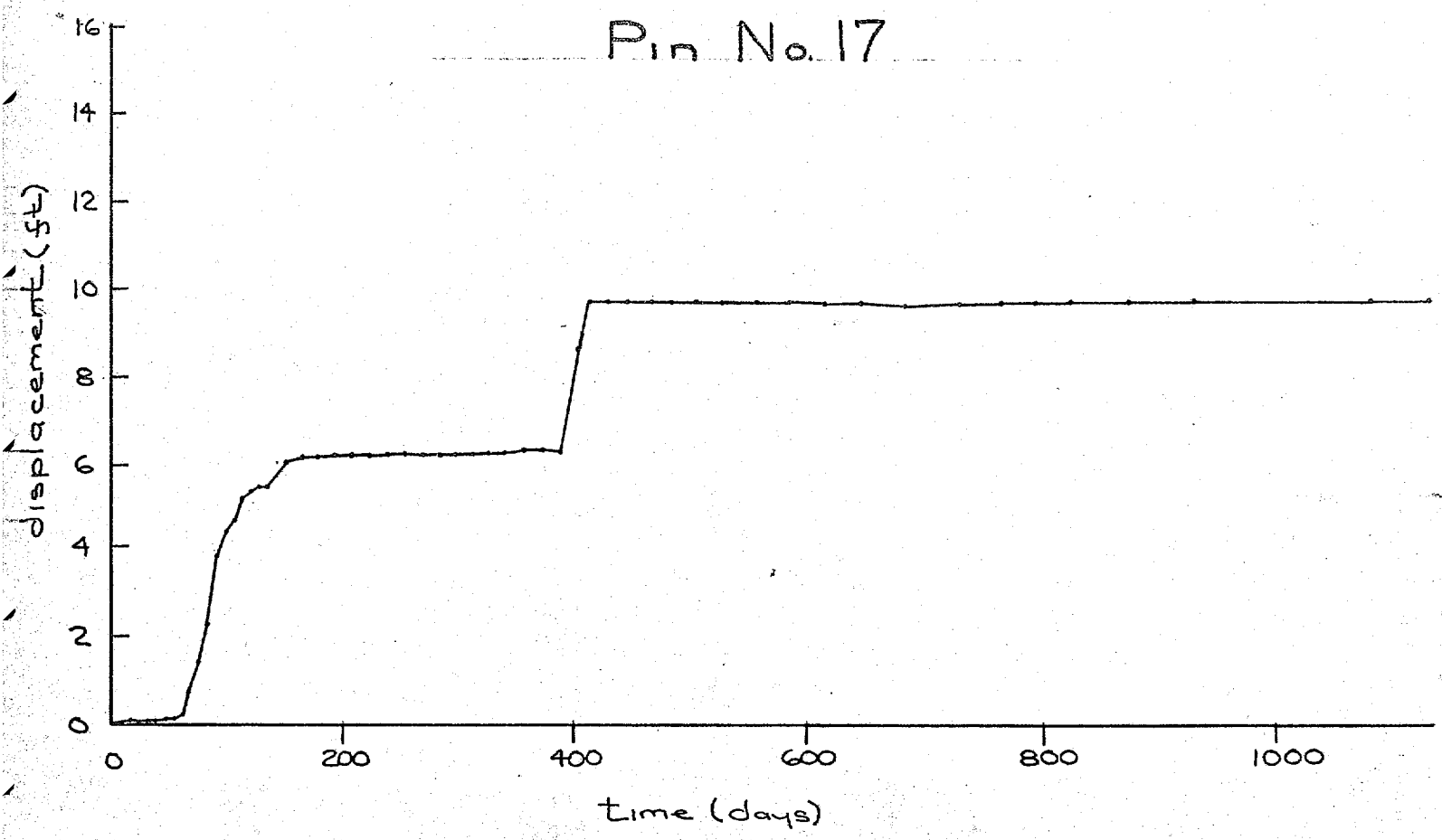
The failure in the Live Oak Pit (the 550 Area) has been of less importance from an operation's standpoint. A displacement of approximately two feet along the No. 5 Fault resulted from prior subsidence caused by the block caving operation. As early as the early 1960's, ground movement in the 550 Area has been a nuisance. Although the ground movement has caused some operational problems, it has not meant tieing up a large tonnage of ore. Displacement has been immense. Survey Station P-215 has moved over 100 feet in elevation since its establishment in 1954 (Figure 8). Pins now straddle the "plane of failure" and displacement readings are taken on a weekly basis. Figure 9 illustrates the vertical displacement of the failure block over a short time span.

CONCLUSION

The geometry of potential planes of failure is critical to pit slope design. A slope failure may only provide an annoyance or it could cripple an operation. Therefore, the evaluation of geologic factors is paramount in determination of a pit slope.

Thornton Pit
Horizontal Displacement
Slope Stability Points
Figure 7





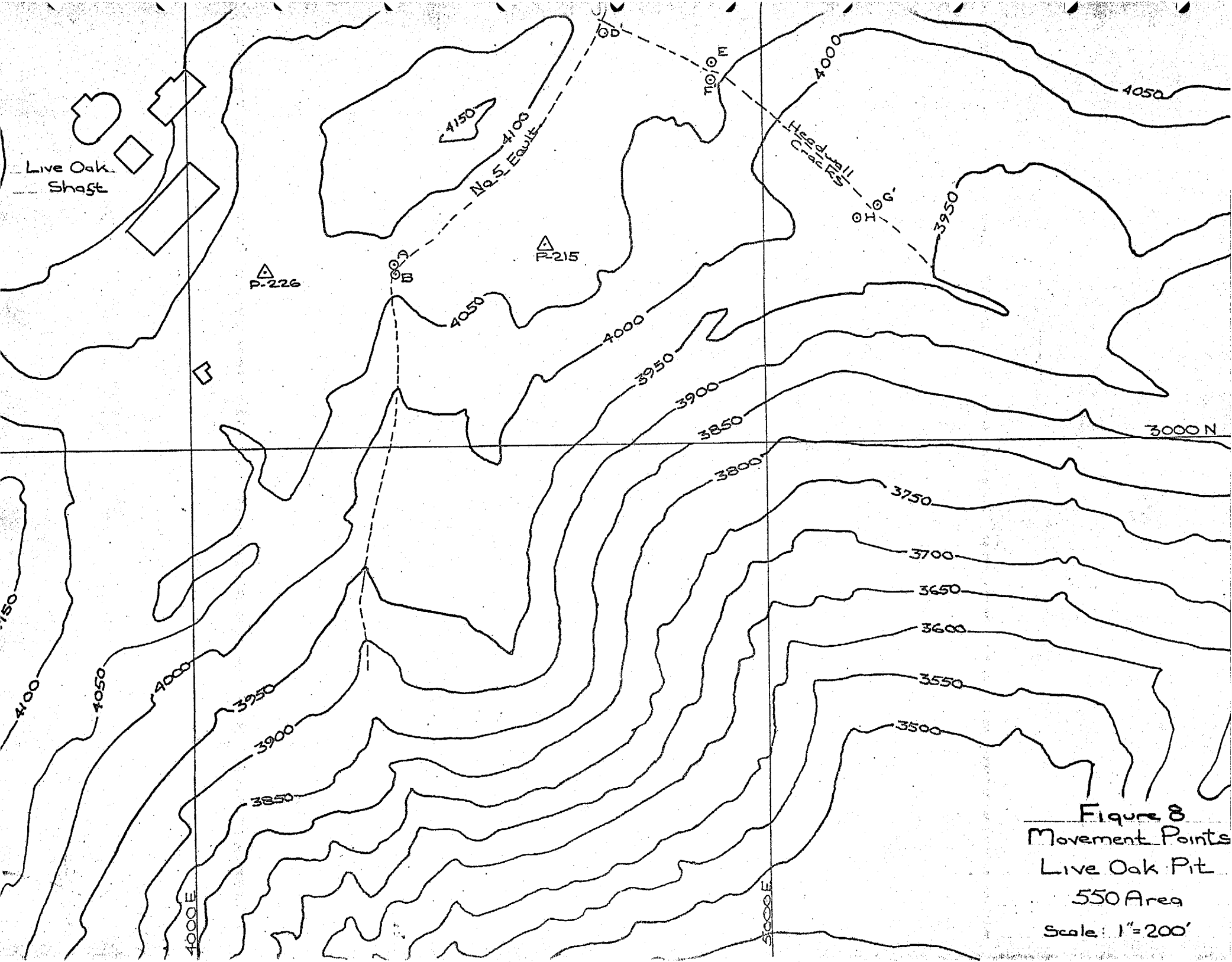
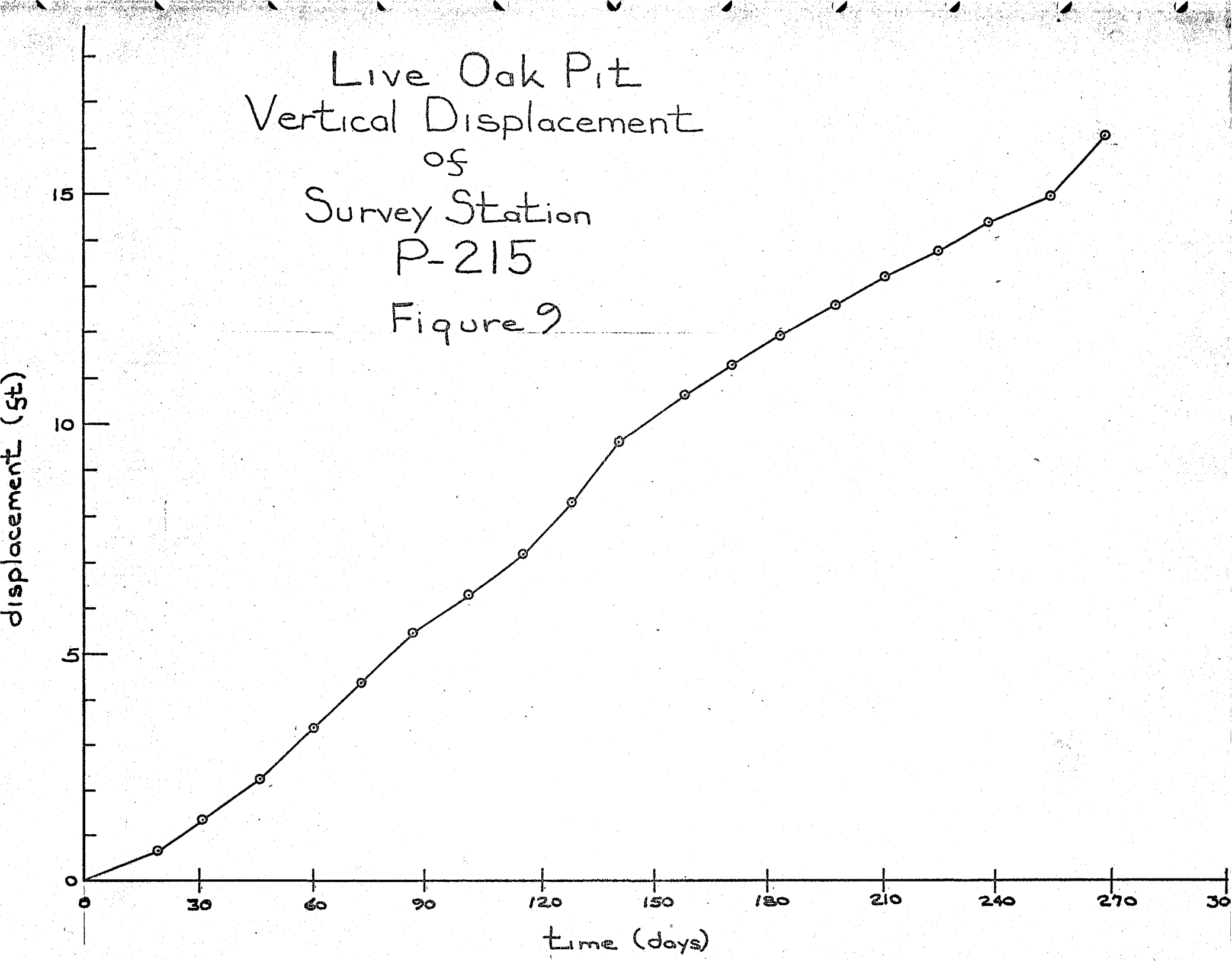


Figure 8
Movement Points
Live Oak Pit
550 Area
Scale: 1" = 200'


Live Oak Pit
Vertical Displacement
of
Survey Station
P-215
Figure 9
displacement (ft)
0
5
10
15
0
30
60
90
120
150
180
210
240
270
30
time (days)

INSPIRATION CONSOLIDATED COPPER COMPANY
Inspiration, Arizona

November 28, 1973

OPEN PIT EQUIPMENT

Shovels:

2 - P & H Electric, Model 1400	5 yd.
2 - P & H Electric, Model 1500	6 yd.
1 - P & H Electric, Model 1600	8 yd.
3 - P & H Electric, Model 1900	10 yd.

Haulage Trucks:

19 - 37 SL Dart Trucks	37 tons
3 - M85 Electra Haul	85 tons
1 - D2772 Dart	100 tons
1 - M70 Mack	75 tons
11 - D2562 Dart	75 tons
8 - Wabco 85C	85 tons
9 - Wabco 85C (March 1974)	85 tons

Front End Loaders:

3 - Caterpillar 992 (Rental)	10 yd.
1 - Caterpillar 988	6 yd.

Drills:

4 - T750 Rotary Drills CP	9" hole
1 - Joy-Beaver Mobile Drill	

Motor Graders:

3 - Caterpillar #16

Wheel Dozers:

5 - Caterpillar 824B
2 - Caterpillar 814

TIRE SHOP

1 - Tire Service Truck
1 - Hyster Forklift

HEAVY DUTY REPAIR

1 - Ford 2-ton Truck

WELDING SHOP

6 - Portable Welders
1 - 2-Ton Truck
1 - 1-Ton Truck

OPEN PIT SURFACE

1 - 3/4 Ton Stake
1 - 2-Ton Flatbed

SUPPORT EQUIPMENT

27 - Pickups
1 - Sedan
1 - Bronco
2 - Man Trucks
1 - Powder Truck
1 - Shovel Cable Truck

DUMP LEACHING

1 - Euclid 2.95 yard Front End Loader
1 - Hough 2.5 yard Front End Loader
1 - Terex 4.0 yard Front End Loader
1 - 25-ton B. E. Crane
1 - Marion Dragline

Crawler Tractors:

1 - Caterpillar 46A D-8
3 - Caterpillar D-9

Water Trucks:

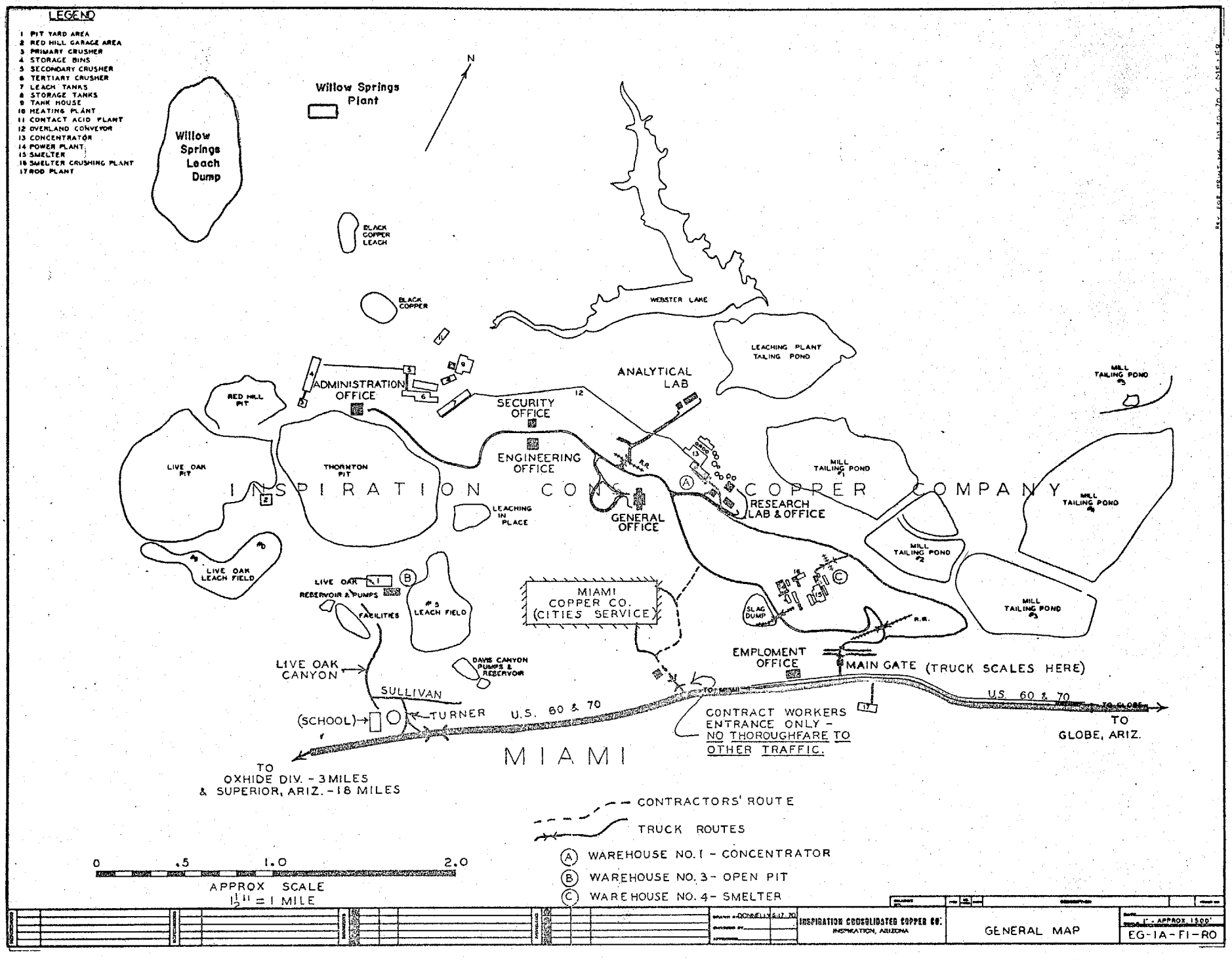
2 - 8,000 Gal
4 - 6,000 Gal
1 - 2,000 Gal

Service and Fuel Trucks:

1969 International 2-ton Service Truck
355L Fuel Truck
1972 Ford 2-Ton Lube Service Truck
1963 2-Ton Ford Heavy Duty & Fuel Truck

GARAGE

5 - Portable Pumps
1 - P & H 15-ton Crane (Rental)
1 - P & H 30-Ton Crane
1 - Austin Western 5-Ton Crane
1 - Galion 12 1/2 Ton Crane
4 - Hyster Forklifts
2 - Hyster Karry Cranes
3 - Portable Compressors


LEGEND
1 PIT YARD AREA
2 RED HILL GARAGE AREA
3 PRIMARY CRUSHER
4 STORAGE BINS
5 SECONDARY CRUSHER
6 TERTIARY CRUSHER
7 LEACH TANKS
8 STORAGE TANKS
9 TANK HOUSE
10 HEATING PLANT
11 CONTACT ACID PLANT
12 OVERLAND CONVEYOR
13 CONCENTRATOR
14 POWER PLANT
15 SMELTER
16 SMELTER CRUSHING PLANT
17 ROD PLANT
N
Willow Springs Plant
Willow Springs Leach Dump
BLACK COPPER LEACH
BLACK COPPER
WEBSTER LAKE
LEACHING PLANT TAILING POND
ANALYTICAL LAB
ADMINISTRATION OFFICE
SECURITY OFFICE
RED HILL PIT
LIVE OAK PIT
THORNTON PIT
ENGINEERING OFFICE
INSPIRATION CONSOLIDATED COPPER COMPANY
LEACHING IN PLACE
GENERAL OFFICE
RESEARCH LAB & OFFICE
MILL TAILING POND #1
MILL TAILING POND #2
MILL TAILING POND #3
MILL TAILING POND #4
MILL TAILING POND #5
LIVE OAK LEACH FIELD
LIVE OAK RESERVOIR & PUMPS
FACILITIES
#3 LEACH FIELD
MIAMI COPPER CO. (CITIES SERVICE)
SLAG DUMP
R.R.
EMPLOYMENT OFFICE
MAIN GATE (TRUCK SCALES HERE)
LIVE OAK CANYON
DAVIS CANYON PUMPS & RESERVOIR
SULLIVAN
(SCHOOL)
TURNER
U.S. 60 & 70
CONTRACT WORKERS ENTRANCE ONLY - NO THOROUGHFARE TO OTHER TRAFFIC.
TO GLOBE, ARIZ.
MIAMI
TO OXHIDE DIV. - 3 MILES & SUPERIOR, ARIZ. - 18 MILES
CONTRACTORS' ROUTE
TRUCK ROUTES
(A) WAREHOUSE NO. 1 - CONCENTRATOR
(B) WAREHOUSE NO. 3 - OPEN PIT
(C) WAREHOUSE NO. 4 - SMELTER
0 .5 1.0 2.0
APPROX SCALE
1½" = 1 MILE
INSPIRATION CONSOLIDATED COPPER CO.
INSPIRATION, ARIZONA
GENERAL MAP
EG-1A-F1-RO

Jack Eastlick – occid. in HW miami fault
50–100M @ .5– Bluebird has cc

H. Bishop – Ray – 50% diabase reserve – #40? in schist | all open pit
some primary ore in schist
no ore gr primary in porph

Bob West – Tyrone – 48000 tpd – was 4% –
min (not ore) extends NE to Mangus
st magnetite on E side of Mangus
Carson IP anom –

Jack E – Insp pit area – .17% Cu primary under smelter –
deepest holes 1700' – (not clear whether
they have potassic zone – but in answer to
question – grade of primary? – .7)

Chuilings – 375 m at .4 – considering
30 000 tpd operation – or possibly e.g. mining.

cc. TCO
JHC

cy ulk file Globe-Miami District

AMERICAN SMELTING AND REFINING COMPANY
TUCSON ARIZONA

J. H. G.
OCT 21 1974

September 24, 1974

file

FILE MEMORANDUM

Inspiration East
Smelter Area
Miami District
Gila County, Arizona

At the AIME meeting of September 17 at Kearny, Mr. Jack Eastlick, Chief Resident Geologist, ICC, offered the following comments:

They believe they have the larger part of the presently-being-developed Miami East orebody. (Miami East announced at between 52 and 75 million tons.)

The last hole drilled in the smelter area drilled through 4000 feet of Gila Conglomerate, then 600 feet of Dacite before entering pre-mineral rock. The premineral intercept contained 300 feet of plus one percent copper and 300 feet of plus 0.7 percent copper with the last assay over 0.7 percent copper.

Bottom hole temperatures are between 150°-160° F.

James D. Sell
J. D. Sell

JDS:lb

J. E. K.

JUN 7 1966

INSPIRATION CONSOLIDATED COPPER COMPANY
Inspiration, Arizona

HISTORY

Inspiration, like most mines in Arizona, owes its discovery to the old-time prospector and his burro. The beginning of mining operations on the Inspiration property dates back to the turn of the century.

The earliest exploratory working was known as the Woodson Tunnel. This tunnel, driven by hand, went into the hillside for 1000 feet. By 1908, local owners had consolidated claims and groups of claims into a single holding and had induced outside capital to form the Inspiration Mining Company. This name was later changed to that of Inspiration Copper Company. Following this, through a long series of events and negotiations, which saw a merger of the Inspiration Copper Company with the Live Oak Development Company, the Inspiration Consolidated Copper Company came into being in the year 1911. Later the Warrior Copper Company and the New Keystone Copper Company, as well as other properties, were acquired by Inspiration.

Plans were soon formulated to engage in a large-scale copper mining operation. The mine was developed and made ready for operations. A complete surface plant, railroad and concentrator were constructed. This concentrator was the first large-scale plant of its kind to make use of the Flotation Process to recover the copper minerals from the ground-up rock. In all, even at that time, it was necessary to spend close to $20,000,000 before one pound of copper was produced. Construction was completed and Inspiration went into production in 1915.

USES OF COPPER

Copper is one of the oldest known metals. The word "copper" originated many thousand years ago when half-savage tribes living on the Island of Cyprus called it "Cyprian Metal". It has kept the name through all the ages. Our tongues have changed it to "copper". Copper plays an important part in the industry of the United States. In fact, it is the backbone of the electrical industry. Because of this, 60% of the annual output of metallic copper in the United States goes into electrical machinery, power transmission lines and telegraph, telephone, radio and television communication lines and equipment.

Other typical uses of copper include sheet for roofing, tubing for gas, steam, water and oil lines, extruded shapes for industrial equipment, drawn shapes for molding, and all types of brass and bronze. It is also used in the coins of many nations; for jewelry; household articles and architectural designs and shapes.

A recently formed organization, The International Copper Research Association, is doing a large amount of research to find new uses for copper. Ninety-five percent of the copper producers in the Free World, including Inspiration, are members of this Association.

LOCATION

The Inspiration Consolidated Copper Company's operations are entirely in Gila County, Arizona. Inspiration is one of the large copper producers in the State, producing approximately 8.9% of the State's output. In comparison with the nation's copper production, Inspiration produces approximately 4.9% of all copper produced in the United States. The State of Arizona, with its many copper producing districts, accounts for more than 50% of all domestic production. The mine, the town of Inspiration with its U. S. Post Office, and the Company's plant and offices are just north of the town of Miami and are reached by turning off U. S. Highways 60-70, about three-fourths of a mile east of Miami and following the paved road for a distance of about three miles. It is approximately eleven miles around the property.

THE ORE BODY

Inspiration is designated as one of the "Porphyry Coppers". Such an ore body is one in which the copper minerals are widely distributed throughout a large rock mass. At Inspiration the distribution is such that one ton of ore contains less than seventeen pounds of copper. Peculiar to Inspiration is the fact that about half of the copper minerals are present in the oxidized form, the other half being sulphide minerals, mainly chalcocite (Cu_2S). It is the presence of the oxide minerals which gives the green coloration to much of Inspiration's ore.

MINING UNDERGROUND

From the start of operations in 1915, up until 1948, all of Inspiration's production came from underground mining, in which a mining method, known as "block caving", was utilized for the extraction of the ore.

"Block caving" is a method particularly adapted to the mining of large, low-grade ore bodies. The rate of production is high and the cost of breaking and handling ore from the "block" or "stope" can be kept relatively low. Largely, the force of gravity is used, both to break the ore and to deliver it to the ore trains operating on the haulage level under the "block".

Ore trains made up of twelve to twenty-four five-ton cars hauled the ore from the "stope" areas to the shaft, where it was hoisted to the surface in twelve-ton skips.

Inspiration's Main Shafts go to a depth of 850 feet and the Live Oak Main Shaft goes to a depth of 1200 feet, with stations at various levels. From the Live Oak Main Shaft bins, ore was hauled in train loads of sixty-ton railroad cars to the Coarse Crushing Plant at the Main Shaft.

Since 1954, all ore mined has been produced by Open Pit mining.

OPEN PIT MINING

The rapid development of modern methods and equipment for moving earth, coupled with the steady increase in underground mining costs, made it necessary to investigate the possibility of mining much of Inspiration's remaining ore tonnage by Open Pit methods. The decision to go to Open Pit mining followed,

and stripping of overburden was started in 1947. The first Open Pit ore was mined in March, 1948. The adoption of Open Pit methods required the expenditure of several million dollars to meet the cost of construction and equipment of new plant facilities and stripping of waste rock.

Ore and waste are mined by large electric shovels and transported by 40-ton diesel-powered haulage trucks. Considerable equipment, in the way of bulldozers and carryalls, is also required.

Open Pit ore is delivered to a large 42-inch gyratory crusher, where it is crushed down to five-inch size for delivery by train to the main Coarse Crushing Plant.

ORE TREATMENT

Early in Inspiration's operations it was recognized that large reserves of copper were available in the "oxide forms", which could not be recovered by treatment in a concentrator. Years of experimental and test plant work evolved a leaching process which would successfully treat the major portion of Inspiration's ore. A Leaching Plant was erected at a cost of six million dollars. This plant was put into operation in 1926. From 1926 to 1956, inclusive, this process accounted for all but a minor amount of Inspiration's production.

The Inspiration Leaching Plant during the 1926 through 1956 period was the only one of its kind in the world. In this treatment, copper in both the oxide and sulphide form was recovered by Leaching (dissolving). The solvent used was a solution containing both sulphuric acid and ferric (iron) sulphate, with the copper going into solution as copper sulphate. This leaching operation was carried on in large leaching vats, each of which holds 10,000 tons of ore. Nine days of contact time with the solvent solution was necessary to dissolve the copper in the sulphide portion of the ore.

After leaching, the copper dissolved from the ore is recovered from the solution in the electrolytic Tank House. In this process an electric current is passed through the solution, breaking down the copper sulphate and precipitating the copper on thin copper starting sheets suspended in the electrolytic cells. In the course of seven days these starting sheets, made at the plant and weighing fifteen pounds, are built up to a weight of one hundred and forty pounds, then the sheets are withdrawn and shipped as electrolytic copper. Such copper is over 99.9% pure. However, the copper sheets, or cathodes, as they are known, still must be melted and cast into commercial shapes as required by the market. In the electrolytic plant the electric power utilized would supply that needed by a good-sized city.

A vital cog in the Leaching Plant operation is the iron launder system. In these iron launders the last trace of dissolved copper picked up in wash solutions, used to wash ore after leaching, is precipitated out on precipitating material. This iron precipitation material is made up of processed tin cans. The so-called tin can is in reality an iron can coated with a very thin film of tin. Tin cans are cleaned, burned, and shredded and in this process form make an excellent material on which to precipitate dissolved copper from solutions. Most of the tin cans used by Inspiration come from the Houston area in Texas. Total consumption of processed cans amounts to about 1,650 tons per month.

To provide sulphuric acid for leaching, Inspiration operates two sulphuric acid plants which can produce up to 200 tons per day. Sulphur, in molten form, shipped in from east Texas mines, is used in this process.

PRESENT PROCESS

By 1954, increasing copper values in sulphide minerals, not soluble in the ferric iron solution, were noted. The grade of the remaining ore was dropping and the capacity to produce copper was limited by the nine-day leaching time. These factors brought about a study which resulted in a radical change in the metallurgical treatment of the ore. By 1957 the old concentrator had been completely rehabilitated and new, modern machinery installed. In the leaching process only that copper soluble in sulphuric acid, plus the sulphide dissolved in a low ferric iron solution, continued to be sent to the Tank House for electrolytic precipitation, as previously described. The contact time for leaching was cut to four and one-half days. Sulphide copper remaining in the ore is then sent to the concentrator for recovery by the flotation process.

The concentrate so recovered is sent to the Smelter. The concentrate is smelted, fire refined, and cast in the form of copper anodes. These anodes are returned to the Tank House.

Due to the change in process, with less dissolved copper being sent to the Tank House, excess capacity was available. This excess was converted to a Refining Section. Here, copper anodes returned from the Smelter are further refined to electrolytic cathodes. In this process the copper is dissolved from the anode and plated on a starting sheet as electrolytic copper. These cathodes are heavier than the Commercial Section cathodes and weigh as much as two hundred and fifty pounds.

To enable Inspiration to refine all of its own copper, including production from its Christmas Mine, a new eighty-tank refinery section was constructed and is now in operation. The electrolytic refinery at Inspiration is the only one in Arizona.

MOLYBDENUM RECOVERY

With copper concentrate being made in volume under the revised process, it was found that such concentrate contained a small amount of Molybdenum Sulphide (Moly). A section was added to the concentrator, to recover the Moly. This is a difficult and involved process.

SMELTING DEPARTMENT

The Smelter was built in 1915 to handle concentrates from the District's mines and to treat custom ores and concentrates. It was owned by the International Smelting and Refining Company. This plant was purchased by Inspiration in April of 1960. It continues to handle District concentrates and custom business.

At the Smelter, properly mixed concentrates and flux are melted in a reverberatory furnace at a temperature of approximately 2700 degrees. The copper collects in the bottom in the form of "Matte", which is an artificial copper-iron sulphide. Some of the sulphur is burned off. Impurities and waste

material float on top and are skimmed off and discarded as slag.

The matte is tapped off at a point below the slag level and is poured in molten form into a converter. Here, air is blown through the molten material and flux is again added. The air oxidizes (burns) the iron in the charge and the sulphur is burned off. Slag is formed and is poured off and returned molten to the reverberatory furnace. The reaction in the converter provides its own heat. Final product from the converter is known as "Blister Copper".

Blister copper may be cast into cakes for shipment to Eastern refineries or poured molten into the anode furnace.

In the anode furnace it is further refined to fire refined copper by blowing with air and "poling" with oak poles. The copper is then poured into anode molds and the anodes are returned to the Inspiration Tank House for further refining.

It is interesting to note the many steps in the processing of copper and the work necessary to produce a final product.

Inspiration Ore	0.80% Copper
Concentrates	30% to 45% Copper
Matte	40% to 50% Copper
Blister Copper	99.4% Copper
Fire Refined Copper	99.6% Copper
Electrolytic Copper	99.95%-99.97% Copper

POWER PLANT

Requirements for electric power at Inspiration are quite large. To meet the original need, a 25,500-KW Power Plant was constructed. In this plant, natural gas piped from New Mexico is burned under boilers to provide the steam to operate the turbo generators. Waste heat steam from the Smelter boilers is also utilized.

The Inspiration power system is tied into that of the Salt River Project, and most of the power needed is supplied by them. On many occasions the entire Inspiration load has been carried by the Salt River Project system, which derives much of its power from hydro-electric generating stations located below the dams along the Salt River.

SHOPS AND SERVICE

The Company has its own shops, warehouse, and service departments. The shops are fully equipped and are capable of handling almost everything in the way of repairs and maintenance which may be required.

RAILROAD

The Inspiration Company operates seventeen miles of standard gauge railroad. This railroad delivers ore to the treatment plants and concentrates to the Smelter, as well as handling inbound freight and outbound shipments of copper. The railroad connects with the Southern Pacific at the foot of the hill.

TOWNSITE AND STORE

The Warrior Cooperative Mercantile Company operates a general store at Inspiration. This store is operated to serve the needs of Inspiration employees.

Operations are on a non-profit basis and profits earned are returned twice yearly to employee customers in proportion to their purchases throughout the period.

HOSPITAL AND CLINIC

The Miami-Inspiration Hospital is maintained jointly with other companies in the District. Also jointly maintained is the Miami-Inspiration Clinic located on the Globe-Miami Highway. These facilities not only serve industrial cases, but the employee and his family is provided medical care at exceptionally low rates. In all, the families of some 2,600 mining employees in the District are served by these facilities.

STATISTICS

Tons of Ore Mined and Treated - 1915 to 1965		198,050,241
Pounds of Copper Produced - 1965		124,153,059
Pounds of Copper Produced - 1915 to 1965		3,725,271,647
Tons of Ore and Waste Moved from Open Pit in normal 24-hour work day		25-35,000
Wages and Salaries paid in 1965		$13,723,277
Supplies and Equipment Purchased in 1965 . .		15,779,002
State, County, and District Property Taxes. . 1965		2,707,885
Approximate Number of Employees:		
Inspiration	1,465	
Christmas	470	1,935
Number of Stockholders		8,679

In 1965, Inspiration produced approximately 4.9% of the copper produced in the United States.

CHRISTMAS MINE

Inspiration is also producing ore from the old Christmas Mine. This is an underground operation. The mine is located some forty miles from Inspiration. It is one mile west of State Highway 77 between Globe and Winkelman, Arizona.

Ore from the Christmas Mine is processed in a new crushing plant and concentrator at Christmas. The concentrates are being sent to Inspiration's Smelter.

ISSUED JANUARY, 1954
ICCCo.
Revised May, 1954
Revised July, 1960
Revised May, 1962
Revised July, 1962
Revised March, 1963
Revised April, 1965

SAMPLING DRILL CUTTINGS AT INSPIRATION —
A FIRST STEP IN QUALITY CONTROL

By James H. Lundy Jr.

"Drill holes make it possible to investigate blocks of ground that by any other means would be accessible only at much greater expense, if at all. In some investigations drill holes are intended merely to secure geologic information — the position of a contact, the attitude of a formation, or the sequence in a stratigraphic column. In others they are designed to determine the presence of veins or other guides to ore. In still others, drill holes are used to provide all the information that is required for an estimate of tonnage and grade."

Hugh E. McKinstry [1]

The last sentence of Professor McKinstry's opening paragraph on Drilling contains the scope of the role drilling has played at INSPIRATION. Indeed, no single phase of scientific endeavor has born a greater responsibility in supporting the industrial complex at Inspiration than has the compilation and utilization of the data derived from drilling.

In order to properly evaluate this basic part of the mining program today, it is necessary to quickly review the History of Inspiration.

Located in East-Central Arizona in Gila County, Inspiration is one of the oldest of the typical Porphyry Copper deposits of the Southwest. Systematic prospecting by churn drilling was started in 1910, with much of this early drilling data still being profitably studied today. The mine was a pioneer in block caving, with this system being used exclusively until 1948. At this time the open pit was started, and the two systems were continued until 1954 when the underground mining was phased-out in favor of the open pit exclusively.

[1] McKinstry, Hugh E., Mining Geology - New York - Prentice-Hall, Inc. 1948

From its inception until the fall of 1926, all the ores from Inspiration's several divisions were treated by flotation. In fact, this mill was the first of its kind ever to treat high tonnages of low grade ore by flotation. However, the high percentage of silicates and carbonates encountered in our orebody were not recoverable by this process, thus bringing about the introduction of the leaching system now used. The present Leaching Plant was then put into operation using a ferric sulfate leach. This process uses ferric sulfate-sulphuric acid leaching to recover Chalcocite, Chrysocolla, Malachite and Azurite, and perhaps some other lessor know copper minerals in minute quantities. All soluble copper ore at Inspiration is referred to as "oxide copper".

Inspiration is considered to be a typical Porphry-Copper deposit. These are generally agreed to have a leached Gossan Cap, a low grade "oxide" zone, a highly productive sulphide secondary-enrichment zone, and finally they grade out to a low grade primary-sulfide zone at depth. Since the primary sulfides at the bottom of the ore body could not be taken into solution with the same techniques employed in the existing leaching plant, and since a considerable amount of "oxide" ore remained, Inspiration rebuilt its flotation mill in 1957 and has employed a dual metallurgical process since that time.

Current production is supplied nearly equally from the west Live Oak Pit, and the east Thornton Pit by six electric shovels — 4, 5 and 6 yard sizes equipped with 5, 6 and 8 yard buckets — and a fleet of some 34, 40-ton trucks. Both these divisions are mined in 25 or 50 ft. benches, which are drilled for blasting by truck mounted 9 in. rotary drills. The cuttings from these blast holes furnish the immediate data for quality control of daily run-of-the-mine ore.

Necessity for close control of daily production is based on the fol-

lowing described operation. All of the ore bedded in the leaching tanks is subject to dual-process beneficiation and must be a mixed "oxide-sulfide" product of such grade and proportions to insure that both plants may treat it economically. Approximately 7% of all production is removed from the main flow of ore as slimes. This is done by running the ore through a classifier plant. The slimes are piped to a flotation unit at the mill; following this they are leached in acid bath thickeners. The copper from this last process is recovered in an iron launder as cement copper. The ore which is "practically-all-sulfides" may by-pass the leaching plant and go directly to the flotation plant in the mill. This tonnage is a variable dictated by local plant conditions and has not been a standard part of the flow sheet until recently.

Without going farther into the field of beneficiation it is apparent that the mine must evaluate all blocks of ground into the following classifications: 1. Sulfide ore which may be treated directly by flotation. 2. A mixture of oxide and sulfide ore which may be treated by dual-process. 3. Readily soluble ore known as oxide which is used as a leacher feed sweetner. 4. Sub-marginal ore which may be economically heap-leached on selected leaching dumps, and 5. Barren rock which must be removed to waste dumps in a stripping operation (if it is inside the pit slopes).

Classification into the five mentioned catagories is not dependent on copper assay alone. In some few areas the Molybdenite content must be evaluated in terms of equivalent units of copper and credited to the mill-feed product. Also the solubility of sulfides must be determined as percent "oxide" so that dual-process feed may be blended to meet the needs of both Leaching Plant and Mill.

It is now apparent that multiple process metallurgy does not allow the mine to send "anything above cut-off" to the crusher at any time, but rather demands a dynamic quality and grade control from shovel to final beneficiation plant feed.

This responsibility has been met by a program of careful sampling and evaluation of drill hole cuttings as a first step in mine planning and production quality control. The program as carried on now varies from past practice only in that blast holes were formerly drilled by churn drill, but now, as has been mentioned, are drilled by rotary drills using compressed air as the drilling fluid to cool and lubricate the bits and bring all cuttings to the surface.

Even though some Diamond Drilling was done in the mine in earlier days, and a little surface drilling may be carried on by this method today, the burden of exploration and development has been carried by Churn Drills, and only Churn Drilling is considered in this paper.

EARLY PRACTICE

The first systematic drilling was started in 1910, with holes being put down on the corners of 200 ft. squares. This pattern with fill-in holes in anomalous areas has been continued till the present. The results of this drilling revealed a very homogenous ore body. Logical contouring of ore grade lines between holes plotted on either plans or sections has given remarkably reliable pre-mining evaluation. This fact has been an important factor in establishing drilling proceedure. Many other mines use the equilateral hole spacing system, and then evaluate benches by single value polygons. Inspiration has been very successful in using the method first outlined, perhaps due to the fact that the ore body is very consistent. We do not quarrel with the polygonal

approach; we simply prefer the square pattern.

The original drill holes were logged on individual section sheets which included rock types, formation penetrated, and classification as capping, oxide, mixed and sulfide. Unfortunately for us today, only sulfide samples were assayed, that being the only ore which could be treated in those days. However, great care was exercised in taking samples on 5 ft. runs. Splitting was carefully supervised, equipment kept clean to avoid salting and composite samples taken to be checked by several assayers. After evaluation by drilling was nearly completed in the initial phase, underground samples were taken to check the validity of projected values. Subsequent mining verified the reliability of this evaluation.

RECENT PRACTICE

With the coming of the open pit mining in the late '40's, an additional drilling program was initiated. The original pattern was expanded, with fill in holes located to expose indicated anomalies. Much care was exercised in drilling previously caved ground, as it became necessary to re-mine large areas which were previously undercut or had subsided due to nearby mining.

This program of continued drilling to develop additional low grade tonnage has been carried on somewhat intermittently since the pit began, with the previously described sampling program followed. However, now assays are run for both oxide and sulfide values, with composites run for Mo also. Pulps are retained so that sections might be re-run for additional data, should this be necessary. This sampling and logging are done by the Geologic Department with the assaying being performed in the Mine Assay Office. Check assays are run periodically to assure accuracy. The Geology Department keeps a log of washed cuttings glued to log charts so that even though the five foot samples are thoroughly ground and mixed by the

drilling, a reasonable stratigraphic section of the hole is obtained. We might say this technique produces "Homogenized Stratigraphy".

With the completion of each hole, the Mine Superintendent is supplied with a geologic report. An excerpt from such a report follows:

CHURN DRILL HOLE NO. 281

Location: North side Thornton Pit near General Office
Inspiration coordinates - 5242.11 N. and 8566.71 E.
Collar elevation - 3789.36

Depth: 275 feet. Started 2-21-66. Completed 3-8-66.
Drilling Time - 37 hours at 7.43 ft./hr.

		Total	Oxide
Assays:	0 - 40'	Nil Cu	Nil Cu
	40 - 50'	0.27%	0.08%
	50 - 65'	0.45%	Trace
	65 -180'	0.31%	0.00%
	180 -275'	0.23%	0.00%

Mineralization: Chalcocite, Pyrite, Chalcopyrite, Hematite, and Magnetite with minor Chrysocolla, 40' - 65'.

Rock: Sericitized schist.

Faults: Bulldog fault - 155' - 190'.

Casing: Run - 28' of 15". 170' of 10".
Recovered - 18' of 15" left in hole.

Water: No water reported.

This information is also furnished to the Mine Engineering Department and is plotted on the Bench Assay plans with all other grade data.

BLAST HOLE DRILLING

As has been indicated, early data did not give an "oxide-sulfide" classification and not even an indication of grade in the upper oxide zones. Consequently when mining for dual process is being carried on in an area previously evaluated by per cent Cu only, the data for grading into the five catagories previously indicated can come only from the Blast Hole cuttings. The importance of accurate daily blast assays cannot be over emphasized. The Mine operators must rely on these, plus the results

of "yesterday's run" to dispatch ore and waste to meet the daily plant requirements.

This is how the system works. The Pit Superintendent lays out drilling for two truck-mounted 9" rotary drills on the blast plans of active benches. The Mine Engineers survey and stake these locations in the pit. Drillers collect a one gallon representative sample of cuttings from the bench column only -- although holes generally are put down 3' to 5' below grade to assure good toe fragmentation. These cuttings are collected automatically by the use of a wedge shaped sheet metal pan which sets underneath the drill platform normal to the hole collar. Thus a percentage of the drill cuttings are automatically taken from the cone of cuttings which build up around the collar of the hole. As is customary, the remaining cuttings are left at the collar for a convenient source of stemming, so any hole may be re-sampled prior to blasting, should this prove necessary. Drilling is generally carried out only on day shift, although it has been done on all three shifts. Blasting, however, is reserved for day shift. At the end of "C" shift each day, all the drill hole samples from the previous day are taken to the Mine Assay Office. These samples have priority and are run for both oxide and total copper, immediately. This assaying is by "Slop Cyanide", and although not extremely accurate, it has afforded reasonable mine control in the 0.08% to 2.00% range.

While an attempt is made to schedule all production well in advance, plant requirements can vary from day to day, so the Pit Superintendent must revise any production scheduling on the basis of the most recent blast hole assays. Ideally all drilling would be completed two full days in advance of blasting so that one full day of production might be evaluated before its immediate mining. Practically this may not always be,

so that sometimes the assays of a blast arrive while that particular blast is being dug. Seldom is there a grave error made as to whether material is ore, leaching-waste, or value-less waste, but frequently it is necessary to make adjustments due to high or low oxide or sulfide content. It is this factor which makes evaluation of drill cuttings from the blast holes so important.

One might wonder if it is practical to move shovels or send fleets of trucks from one producing bench to another each time new assays arrive. Certainly it is not. However, Inspiration has enough flexibility due to the amount of live storage in the primary and secondary crusher bins to conveniently sustain changes in grade being made on half shift intervals. This is frequently done.

It should be noted that visual assays, a knowledge of yesterday's run and the facility to obtain "quicky" assays of bench material within 90 minutes, greatly expedites the quality control program. However, in the final analysis, how well the mine is able to supply the consistent mill heads so necessary to efficient metallurgy is dependent almost entirely on how well the drill hole cuttings have been interpreted.

MINING GEOLOGY DIVISION - ARIZONA SECTION A.I.M.E.

SPRING MEETING: APRIL 30, 1966

Inspiration, Arizona

"STRUCTURE AND ORE CONTROL AT INSPIRATION"

by Hugh Olmstead, Inspiration Consolidated Copper Company

STRUCTURE

The Globe-Miami District lies near a postulated intersection of four continental lineaments. Locally these are: 1. The Texas lineament (N 74° W); 2. The Arizona Rockies lineament (N 33° W); 3. The Utah-Arizona lineament (N-S); and 4, the Raton-Globe lineament (N 57° E).[9] Regionally, they are traceable as Mesozoic and Tertiary intrusive and extrusive rocks found in mountain ranges and fault structures, which respectively include: 1. The Buckskin, Wickenburg, Pinal, and Pinaleno Mountains; 2. The Swisshelm, Galiuro, Dripping Springs, and McDowell Mountains; 3. The Tumacacorie, Picacho Mountains, and the Pleasant Valley fault; 4. The Datil volcanic field and Pinacate volcanic field in Mexico. Inspiration is situated in the lower elevations on the northeast side of the Pinal Mountains. At the mine, the Texas and Raton-Globe lineaments are reflected strongly in the Pinal schist and various faults. The Utah-Arizona and Raton-Globe lineaments are partially delineated by Tertiary Schultze granite and Granite porphyry intrusives.

Physiographically, the district lies in the middle part of the mountain region province.[16] It is in an area of flexing on the southern end of the Mazatzal Mountains. Although a broad deep valley separates the Mazatzals

on the north from the Pinal Mountains on the south they probably were one undisrupted rock mass during the Paleozoic Era. Before and during the Rocky Mountain orogenesis the level ocean beds were uplifted, warped, and distorted to their present position. This and accompanying Tertiary intrusion undoubtedly created a high, sharp-edged mountain range which has been eroded to the present rugged and irregular topography. In the Quaternary, masses of erosion products filled all of the lower elevations. Resulting equilibrium adjustment depressed conglomerate areas which are now the major valley and drainage paths.

Structures in the mine area are somewhat related to the general structure of the Pre-Cambrian schist which trends northeasterly and dips to the southeast.[17] Local granitic intrusives have distorted and obliterated the schist structure, but the schistosity prevails as the major lineation followed by mineralizing solutions.[17] A lack of schistosity in the mineralized zones may be due to more massive beds of schist in which fractures remained open for mineralization and enrichment.

The structural control of the intrusion of the Granite porphyry phase of the Schultze granite, with which the ore bodies are associated, is shown by some of the existing faults. The Miami fault, one of these structures, strikes north 25 degrees east and dips about fifty degrees to the east and drops the Gila conglomerate east of the ore bodies between two and three thousand feet. This fault or the ancient break it followed may have had some pre-porphyry movement which is indicated by the porphyry-schist contact extending along the general fault strike beyond where the present fault zone swings to follow the trend of the Pinto fault for some distance. East of the mining areas the ore bodies appear to be cut off by the Miami fault, although some sections show that the ore did not reach the fault zone or that there is a leached zone near the fault. Some ore in diabase has been

found at considerable depth east of the Miami fault and secondarily enriched sulphides were found below low-grade primary material near a branch of the Miami fault in one drill hole.

The Sulphide fault is also a pre-porphyry structure. It has an east-west trend and dips steeply and irregularly to the north. The contact between the schist and porphyry is in the fault plane for some distance, but the ore bodies do not appear to be displaced by it. The Sulphide fault may have been a conduit for ore-bearing solutions, but cannot be traced in the porphyry for any distance and passes under conglomerate to the west. This fault is paralleled by the Southwestern fault about one thousand feet farther south, which may be a later structure, as it can be traced in the porphyry and appears to displace the ore zone.

The Pinto fault strikes northwest and dips at about forty-five degrees to the northeast. It cuts through the mining areas in both the Miami Copper and Inspiration workings. At the junction of the Pinto with the Miami fault, the Miami swings to follow the trend of the Pinto for more than one thousand feet. The Pinto fault is irregular with a broad crush zone and some fault clay. There are some drag fragments of mineralized material in the zone and also indications of post fault mineralization. Although the ore bodies appear to be down dropped east of the fault, there also appears to be a zone of higher grade primary material in the footwall of the fault in the Inspiration workings, which was probably introduced in a pre-mineral structure. Also, the Miami Copper ore bodies are best developed in the acute angle near the junction of the Miami and Pinto faults, which would indicate pre-mineral structures, but there is no clear indication that the faults were channels for mineralizing solutions.

Other faults in the Miami Copper workings appear to have minor displacement, but where a crosscut from the No. 5 Shaft, which started in

conglomerate, encountered the conglomerate-schist contact east of the Miami fault, there was considerable clay developed at the contact, which might indicate some movement.

The Joe Bush fault is more or less parallel to the Pinto fault, but dips steeply to the southwest or is almost vertical. It passes south of the Miami Copper workings, but is well exposed by the workings on the Inspiration 600 Level. On this level there is an apparant horizontal displacement of the schist-porphyry contact of more than one thousand feet by the fault. This large movement is not shown on the surface and may be, in part, due to a pre-porphyry structure being a guide for the emplacement of the porphyry.

The Bulldog, Keystone, Number Five, and Barney faults are north to northeast trending faults dipping flatly to steeply to the east. The movement on these faults is normal with the east blocks down dropped. The amount of movement on some faults is shown by the displacement of the post-mineral dacite beds.

The Bulldog fault is roughly parallel to the Miami fault and dips at between thirty and forty degrees to the east. This may also be a pre-porphyry structure with some post-porphyry movement, since a dike of porphyry intrudes the fault zone and the fault cannot be traced far in the porphyry or granite. The fault branches to the south and there are a number of steeper hanging wall splits which have about one hundred feet displacement. These are locally called the Colorado faults. Some post secondary enrichment movement on these faults is shown by the displacement of the ore bodies down to the east.

The Keystone fault trends northeasterly and dips at about forty degrees to the southeast, as shown on the Geologic Plan. It probably swings to join the Bulldog fault north of where it is exposed on the surface. There is a displacement of about three hundred feet on this fault, but its location has

never been well established in underground workings.

West of the Keystone there are the Number Five fault, the Barney fault, with about six hundred feet displacement, and other smaller structures, besides the Sulphide and Southwestern faults which have been described. These have a general northeasterly trend and dip to the southeast. Some rotational movement on these structures is shown by tilting of the bedding of the conglomerate and by the slope of the dacite beds and the pre-conglomerate land surface. The Sulphide fault, which guided the emplacement of the porphyry, would have had a southerly dip before this tilting and continued movement on that fault may have caused some anomalous structural conditions, such as schist overlying dacite found in drill holes in the area south of the projection of the Sulphide fault.

From the underground mapping and drill hole information, the ore bodies appear to be cut off on the west by the Porphyry fault, which trends about north fifteen degrees east and dips twenty degrees to the east, although there is considerable decrease in size in the ore zone before it reaches the fault. This fault is either cut off by the Barney fault or there is a considerable steepening of dip towards the surface and the Barney fault is the same structure. Exploration west of these structures has encountered mineralized schist, but no ore bodies.[17]

OREBODY GEOLOGY

does this include Miami also — or is limits made by property lines? JCR

The Inspiration orebody is 8,300 feet in length and has a maximum width of about 2,500 feet. The ore attained a total thickness of as much as 700 feet but will average about 300 feet. It has an arcuate, elongated shape which thins in the middle and on its southwest end. (REFER TO MAP). Elongation follows a general trend of N 72° E with a gentle southwest pitch. The original high-grade chalcocite ore was regarded as an irregular sulfide

What is he talking about? JEK

replacement blanket deposit. Some of the low-grade ore now being mined is regarded as a disseminated or "porphyry-type" deposit. In the early 1900's high-grade chrysocolla was mined from veins in the porphyry; therefore, at least three types of deposits occur, in which each is an integral part of the other.

Primary hydrothermal mineralization is believed to be intimately associated with the porphyritic intrusions of Schultze granite, where faults and relevant crushing created a favorable environment for solution emplacement. There is some evidence suggesting simultaneous action of faulting, crushing, stretching and magma flowage. The primary ore solutions appear to have succeeded the introduction of the Granite porphyry differentiate. These solutions carried pyrite, chalcopyrite, and possibly bornite and chalcocite. They likely were injected into small fractures opened by stretching[1 and 6] which was a resultant of the active diastrophism and volatile pressure. Later differentiates apparently of moderate magnitude carried pyrite, quartz, and molybdenite. Microscopic study indicates a period of orthoclasization and silicification which preceded the flow of ore mineral solutions. It is believed that super-saturated solutions carrying a high potassium content derived from a parent magma and from circulating juvenile solutions altered permeable Schultze granite, and strongly impregnated the granitic wall rock. Data available from a study of the compositional variations of the alkali feldspars states that "the variation in many cases is not independent of the protore distribution."[7] Kuellmer has presented two hypotheses concerning the origin and significance of compositional variations of the alkali feldspars. His second hypothesis postulates the possibility of alkali feldspar compositional difference resulting from a "secondary compositional adjustment of primary crystals during a hydrothermal stage in addition to crystallization during the hydrothermal stage."[7] Microscopic study[10] supports this

hypothesis and strongly suggests secondary orthoclase.

In addition to the alteration products of silicification and orthoclasization, there exists alteration products from kaolinization, sericitization, biotization, hydration, argillization, surface leaching, and enrichment. These are profound within and near the orebodies while epidotization is more prevalent around the north and east fringes.

Over the major portions of the ores, surface oxidation and leaching has imparted a moderate to weak brown coloration. Early prospectors were attracted to this area by conspicuous stains of copper silicate and carbonates, but now strong coloration from ore minerals exists at only four small areas other than the pits. Although not plentiful, residual limonitic boxworks occur near the surface within small quartz veinlets in schist, Granite porphyry and Schultze granite.

In the ore areas dark minerals comprise a small percentage of the whole, partly due to alteration and partly due to an original small supply. Biotite is the main ferromafic and it seldom exceeds five percent of the rock composition. Although restricted, some biotite flakes reveal marginal chlorite alteration. The disintegration of biotite varies but has generally produced kaolin, chlorite, or hydrous micas.

Often in the zones of strong alteration, it is difficult to megascopically distinguish the outlines of the feldspars. Sericite prevails as a hydrothermal alteration product and kaolin is intricately an associate of the alteration cycle. In the crushed zones, near the more prominent faults, there is a weak indication of brecciation[14] but it appears that supergene alteration has etched the fragments (quartz) beyond recognition as breccia. Sericite and kaolin penetrate at least as deep as the enriched zone.

At Inspiration, secondary supergene mineralization is as important as primary mineralization. One without the other would have not produced a mine.

Hypogene metallization penetrated schist, granite porphyry, and Schultze granite. Economic minerals included pyrite, chalcopyrite, bornite, and chalcocite, with later impregnations of molybdenite and pyrite. Pyrite is included as an economic mineral because of its necessity in the supergene cycle. During the metallization a distinct zonal pattern was established, which somewhat controlled the mineral distribution and the mineralization intensity.[15] The ore minerals filled small fractures in the host rock, creating a low-grade protore. Subsequent alteration and erosion by various chemical reactions decomposed the primary minerals and produced copper chlorides, copper silicates, copper carbonates, copper phosphates, and copper rich sulphosalts, which, because of their relative solubility, resulted in chalcocite replacing pyrite, chalcopyrite, and bornite. During this process a protore from hypogene mineralization was oxidized and leached and transformed to an ore of supergene enrichment. Between the leached zone above and the supergene sulfide enriched zone below, there is an intermediate zone of oxidation and hydration containing malachite, azurite, chrysocolla, and ferric hydroxide minerals. Within the zone of oxidation there occurs primary quartz veinlets with chalcopyrite and pyrite which have not completed the enrichment cycle, nor undergone significant oxidation. Since the supergene chalcocite replaced mainly pyrite, ore bodies are localized in zones of primary mineralization regardless of the amount of primary copper present.[17] Colloidal solution activity has not been ascertained, but there is evidence supporting hypogene origin for part of the chalcocite.

I don't believe AER

The higher grade primary mineralization occurs as bands along the Miami and Pinto faults, and between the Joe Bush and Bulldog faults. (SEE MAP). Very little primary copper mineralization was encountered in the Live Oak mine. However, some primary chalcopyrite was associated with the crushed zone near the Sulphide fault.[17] The deepest zone of ore (supergene) on this property

occurred on the 1200 Level in the southwest corner of the Live Oak Mine. Below the enriched zones there exists a zone of protore of primary mineralization assaying from 0.15% to 0.40% copper. The thickness of this material is not known.

The leached zone of capping varies in thickness from Nil to 1,000 feet or more and will average about 400 feet. Near the upper extremities of the leached zone the copper content is often not more than a few parts per million. The oxidized zone, fairly consistent in thickness, averages about 200 feet, although this varies somewhat in fault zones or other permeable zones. The oxide ore zones resulted from the oxidation of the supergene enriched zones. They are much higher grade than mineralization in protore. At least half of the present ore is produced from this zone. The supergene enriched zone varies but will average about 200 feet and attains as much as 450 feet in thickness. OK

The age of the ore extends from early Tertiary through the present. It is generally believed that primary metallization first occurred in late Cretaceous or Early Tertiary. However, secondary enrichment processes were active throughout the Tertiary Period and have been moderately active until today. ? Not la pre-alt fail (Mid-T) ? also see "A" pg 10

The minerals present in the Inspiration orebody should be divided into hypogene minerals and supergene minerals. As the original hypogene mineralization in the porphyry and schist was of low tenor, the orebody, as we know it today, is dependent upon the supergene enrichment. The ore now averages less than one percent total copper, about 0.02 percent molybdenite, with traces of gold and silver.

The hypogene minerals consist of pyrite, chalcopyrite, molybdenite, minor bornite, minor chalcocite,[4] traces of gold and silver, and a few very minor occurrences of galena and sphalerite. In a few places

chalcopyrite-quartz veins cut earlier pyrite-quartz veins, indicating at least part of the chalcopyrite mineralization is later. The molybdenum-quartz veins cut all mineralization and are considered the last phase of the hypogene metallogenetic phase.

The supergene copper sulphide minerals consist of chalcocite, bornite, covellite, and chalcopyrite.[8 and 18] The chalcocite blanket varies from the complete replacement of the original pyrite and chalcopyrite by chalcocite to films of chalcocite on pyrite crystal surfaces. Although argumentative, the consistent occurrences of a chalcopyrite enriched zone beneath the chalcocite blanket is attributed to supergene solutions. Thus, chalcopyrite in the enriched zone[18] occurs as extremely thin films on pyrite crystal surfaces. interesting. Secondary Cpy suggested by short et al (Ariz Bur Mines Bull) at Magma - limited

refer "A" pg 9

Chrysocolla, malachite, azurite with minor copper pitch, brochantite, atacamite, lindgrenite, libethenite, and extremely minor metatorbernite,[20] occur as products of a supergene enriched area in the oxidized zone. An association has been established between the occurrence of granite porphyry, radioactive minerals and ore minerals.[20] interesting - a hot relationship?

also identified at Castle Dome (Peterson)

Some notations must be made of the very minor occurrence of native copper and cuprite in a few fault zones. Also, chalchanthite, a product of present mine water seepage, is found in the underground workings.

See PhD thesis by Art Sell.

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Notes on Open Pit Meeting April 16, '62

Block cave mine showing extensive funneling exposed in both pits — extraction ± 50 % —

mining rate ± 15000 tpd of ore
w/o — .66/1.00

4 day leach — then flotation for mixed ore

good cc ore in schist in ~~Thornton~~ Live Oak pit
some por with cu sil — por shows sul cov. indicating oxidation in place.

In Thornton pit unaltered schist contains exotic cu in the form of silicate.

± 15 mil tons .6 sulphide ore under Red hill north extension of Live Oak ore body — capping and outcrops show strong sericitized schist + mod alt por — limonite often c'c generally sparse except in occasional vein —

INSPIRATION'S APPROACH

TO THE

GRADE HAUL PROBLEM

By

T. M. Anderson

Inspiration Consolidated Copper Company

Inspiration, Arizona

ARIZONA SECTION AIME
OPEN PIT DIVISION

April 16, 1962

INSPIRATION'S APPROACH TO THE GRADE HAUL PROBLEM

At Inspiration, to date less than fifty percent of the total material moved has required drilling and blasting, due to pit mining being carried on to a considerable extent in areas which were broken by earlier underground block caving operations. In the solid or virgin area, the material ranges only from soft to medium hardness, thus simplifying drilling and blasting operations. Thus the bulk of mining costs and problems have centered around truck haulage, maintenance, and the ever increasing length of the grade haul. Currently, most of our ore is being lifted +400 feet vertically by haulage trucks over adverse grades, or what will be referred to here as grade hauls, up to 7% maximum with some haul lengths now approaching 10,000 feet.

Originally our open pit operations, which started in 1947, were to only supplement the underground ore production with approximately 4,000 tons per day by mining areas with which had a low stripping ratio. Subsequently, a relatively small 42" gyratory crusher was installed; small shovels, 4 yd. capacity, and 24 ton capacity trucks were purchased. However, as the advantages of open pit mining and its lower costs were realized, the underground operations gradually gave way to the open pit. Since 1954 all ore production has come from surface operations.

For the most part, the history of truck haulage at Inspiration is similar to that of other pits using trucks as their means of haulage. Our first trucks had relatively low horsepower engines with dry clutches, standard transmissions, tandem rear axles and would creep up a 7% grade with a 24 ton load at approximately 6 miles per hour. In 1951, additional trucks were purchased. These, taking advantage of the technological improvements which had been made since our first trucks were purchased, were equipped with engines of higher horsepower ratings, torque converters and would haul a 30 ton load up grade at approximately 6 miles per hour. Some of the original trucks were then disposed of and the remaining trucks were modified to haul 28 tons and larger engines and torque converters

were installed. In 195[illegible] as surface mining operations w[illegible] expanding, additional trucks were purchased. These had a 40 ton capacity, twin engines, converters, power-shift transmission and retarders. The retarders, at that time, being one of the latest technological improvements to be added to off-highway trucks. These trucks would also move up grade at approximately 6 miles per hour.

During the first five years of open pit operations, our hauls were relatively short, generally favorable with some down hill. But as both pits deepened, waste dumps lengthened and ore production increased, the hauls became longer and more adverse. Figure #1 shows how the length of haul has increased through the past 13 years. Even this does not tell the whole story since the percentage of the total grade haul has also increased. The natural result was an increase in trucks shifts required per shovel shift and increased haulage costs.

Faced with this problem of ever increasing haul lengths we began to investigate the possibility of increasing the truck speed on the grade. At this point, one might ask why other alternatives were not feasible. Three such alternatives are listed below with the reasons why we felt they were not the answer to our particular problems.

(1) Larger haulage units (60 ton range)

(a) We felt the advantage of larger haulage units could not be fully realized when used with our 4 and 5 yd. shovels.

(b) Our crusher was not designed to accommodate trucks of this capacity and would require modification to do so.

(c) We were not convinced that the larger haulage units on long grade hauls, due to their higher depreciation, operating and repair costs, could haul as economically on a cost per ton mile basis as our 40 ton units under our operating conditions.

(2) Skip Haulage.

(a) In the Thornton Pit we have neither a final nor a stable slope on which to put a skip way.

(b) The location and shape of the Live Oak Pit would not lend itself to skip haulage.

(c) In both pits, due to the mining in old undercut areas, waste and ore sorting operations will be necessary throughout the ore body. Due to the location of the plant on one side of the pit and waste dumps on the opposite side, it would not be practical to hoist both ore and waste from one skip location. See Figure 4.

(3) Continued and expanded use of present underground haulage facilities.

(a) From 1951 to 1961, we passed approximately 14 million tons through our ore pass to underground operations, to be hauled to the shaft and hoisted to the surface at the coarse crusher. At the beginning this operation was an economical one, but as larger haulage trucks were purchased, speeds up grade increased and underground handling, haulage and hoisting costs increased, the total per ton costs of this operation exceeded our overland haulage and handling costs and was therefore abandonded. In the future, when lower depths are reached in the Thornton Pit, this operation may again become economical.

Our first step toward attaining higher speeds up grade began in 1956 when we turbocharged our first engine. By turbocharging, we were able to up our speeds up grade from 6 - 6.5 miles per hour to 8.5 - 9.00 miles per hour. During 1956 and 1957 we turbocharged all haulage units. R. V. Bamerio, who heads our Industrial Engineering Department, presented a paper to the Open Pit Section meeting in Tucson in 1959 in which he discussed this subject.

In 1957 we purchased our first single rear axle haulage unit - a Model 35SL Dart with Cummins NVH-12 engine - which would haul approximately 38 tons up grade at 8.5 - 9.0 miles per hour. We were pleased with the cost performance of this truck and have subsequently purchased 9 more similiar units. Since these trucks had naturally aspirated engines, we again began to envision higher speeds in these

units by turbocharging. However, in 1960 we were approached by Twin Disc Clutch Company on the possibility of installing a new converter and transmission as a test unit in one of our single rear axle trucks, with the assurance that this change would increase our speed up grade by approximately 15% and decrease our fuel consumption. We agreed to do this and were very anxious to see if we could obtain additional speed without any increase in horsepower.

It would be well to describe this converter - transmission package to understand how the increase in speed and decrease in fuel consumption over the present units is achieved. The Twin Disc Converter - Transmission assembly is not an intergal package but consist of an 1800 series, single stage, rotating housing, hydraulic torque converter independently mounted with a drive line and universal joints connecting to the TA51-2000 series power-shift transmission. The converter contains a lock-up clutch for direct drive and a hydraulic retarder. When truck speed reaches a point at which within the converter the impellor speed approaches that of the turbine, the lock-up clutch is engaged and the impellor, turbine and stator rotate as a unit. The transmission is of straight-through converter-shaft type construction with constant-mesh gears and multiple-disc, oil-cooled, hydraulic-actuated clutches. These transmissions have five forward speeds and one reverse.

The additional speed and fuel saving comes in selecting the proper gear ratio by up-shifting or down-shifting so that the truck operates with the converter in lockup when engine is at peak rpms. This eliminates the loss of efficiency that is inherent in a converter and makes that much more horsepower available for additional speed. It has also alleviated a cooling problem which we have during the summer months. Figure 2 shows effect in speed up a 7% grade and the difference in fuel consumption the new installation had on truck performance.

In our haulage operations we do not allow passing on the grade. Consequently, the slowest truck on a particular haul governs the cycle time of that

haul so the advantage of only one fast truck cannot be realized. In order to assemble a fleet of these, we ordered three new trucks equipped with this new converter - transmission and three assemblies to install in our older units giving us a total of seven units so equipped. We now have three more trucks on order which will be equipped with the same power train.

Our NVH-12 engines are rated at 450 horsepower. Discounting for altitude and accessory loss we figure we are getting approximately 380 horsepower at the flywheel. Since these new converters have a maximum 450 horsepower rating we decided to turbocharge the truck which had the test unit installed, and set the engine to deliver 450 at the flywheel. This resulted in an additional fuel saving and speed upgrade. Figure 3 illustrates this increase in performance. We are now considering turbocharging other units to further increase the speed of this fleet.

What adverse effect has the increased speed had on truck components? As far as we can determine, little or none. It must be kept in mind that we have not increased speed on the level or down hill return haul - only on the up grade, and are speaking of speed variances from 8 to 13 miles per hour. We are convinced that we are on the right track in coming up with a solution to holding our haulage costs in line as our grade haul lengths increase. As we have visited other operations and compared haulage costs, we know that with our 40 ton trucks moving up long grades at 11.2 miles per hour, we can compete on a cost per ton basis with the larger, slower haulage units.

How far do we intend to go with this speed approach to our grade haul problem? Our present position is this: we have trucks which by turbocharging we could increase horsepower at the flywheel by approximately 25% and, in all probability, this we will do, when the manufacturer has either larger converter-transmission units available or allows increased horsepower ratings in the present units.

Several years ago, we set what we thought at that time would be an ultimate goal of 15 miles per hour up 7% grade. But, as we are now approaching this speed, we feel we set our goal too low.

Figure #1

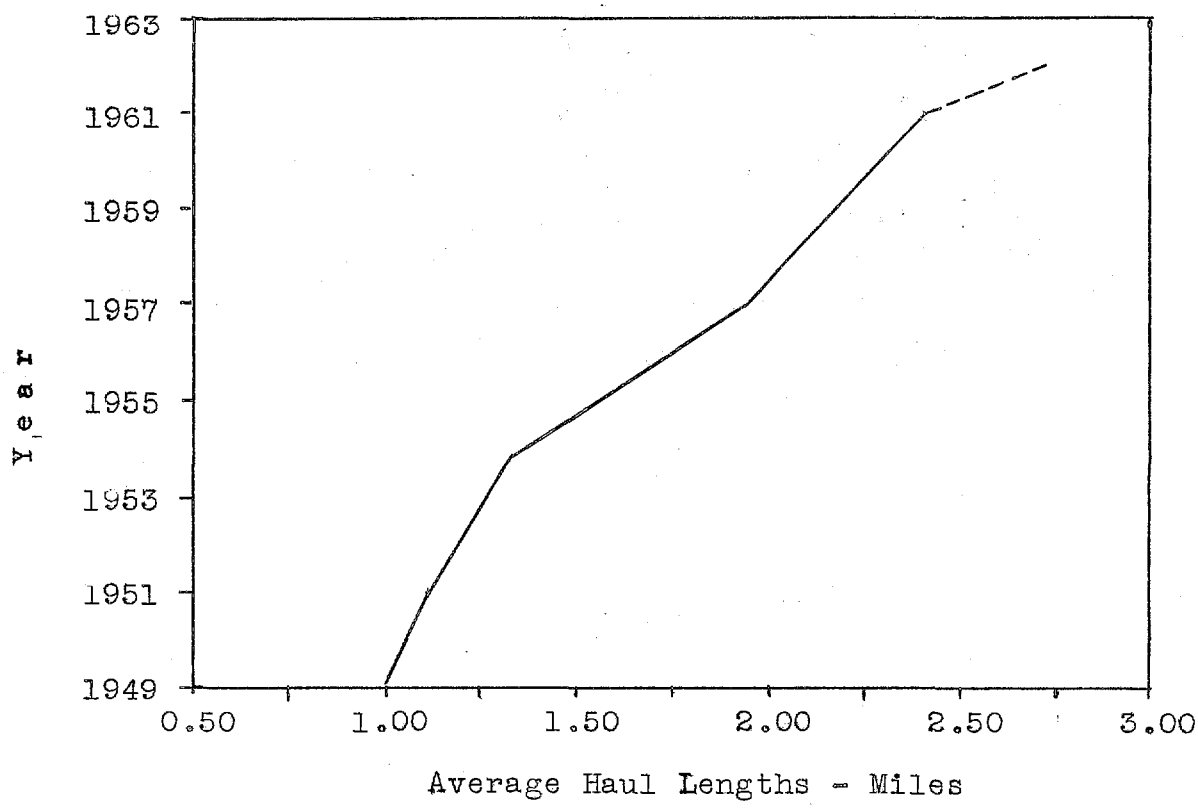


Figure #2

37SL Darts before and after installation of TA51-200 power-shift transmission

	Average Miles Per Hr. 7% Grade	Fuel Consumption Gal./ Hr.
Before	8.8	10.63
After	11.2	9.45

Figure #3

37SL Dart with TA51-200 power shift transmission before and after turbocharging.

	Average Miles Per Hr. 7% Grade	Fuel Consumption Gal./ Hr.
Before	11.2	9.45
After	13.2	8.78


MAIN SHAFTS
ORE BINS
CRUSHER
GENERAL OFFICE
LEACHING TANKS
N
THORNTON PIT
LEACHING AREA
LIVE OAK PIT
WASTE AREA
OFFICE and SHOP AREA
DWN. BY

INSPIRATION CONSOLIDATED COPPER COMPANY

OPEN PIT

GENERAL INFORMATION

EQUIPMENT

ROTARY DRILLS:

1 - Joy Model 225 - Truck Mounted
$6\frac{1}{4}$" Hole
335 feet per shift
1975 feet per bit

1 - Reich Model 750T - Truck Mounted
9" Hole
348 feet per shift
2984 feet per bit

ELECTRIC SHOVELS:

3 - P & H 1400 - 4 yd. - 5200 tons/shift
2 - P & H 1500 - 5 yd. - 5400 tons/shift

HAULAGE TRUCKS:

2 - Dart 30T - 30 tons
9 - Euclid 4FFD - 40 tons
2 - Euclid 10FFD - 40 tons
10 - KW-Dart 37SL - 38 tons

OTHER:

8 - D-8 Tractors
2 - Motor Graders
2 - Rubber Tired Dozers
4 - Sprinkling Trucks
3 - Portable Compressors
2 - Wagon Drills
6 - Misc. Service Trucks
4 - Carryalls

MISCELLANEOUS

DAILY PRODUCTION:

Ore - 15,200 tons
Waste - 10,000 tons - Stripping ratio - 0.66/1.00

OPEN PIT PRODUCTION - TO DATE:

Ore - 49,000,000 tons
Waste - 92,700,000 tons - Stripping ratio - 1.90/1.00

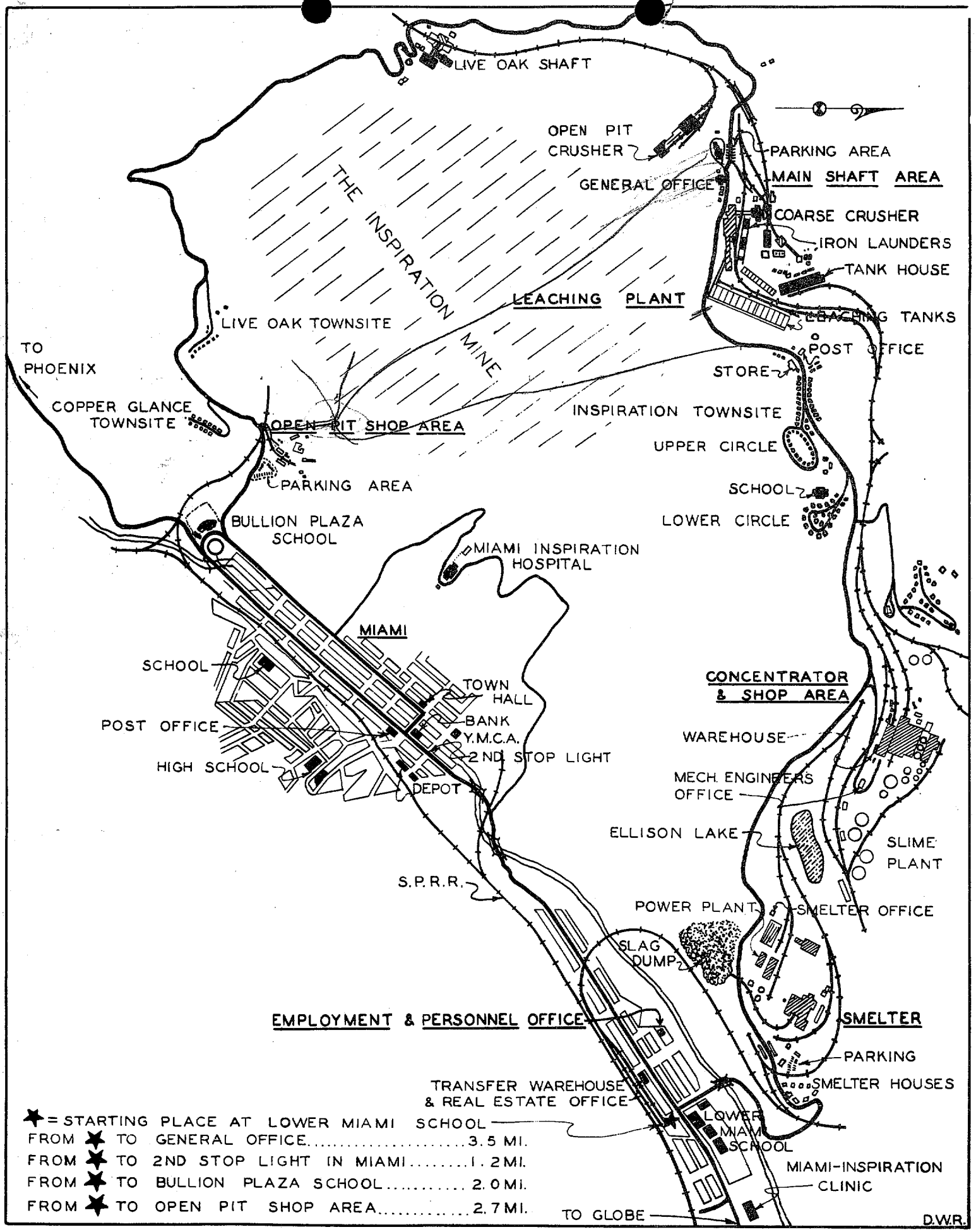
PIT ROLL:

158 Men
220 Tons per Man Shift

BLASTING:

Carbamite
Pro-Core Boosters
6.5 tons per lb. of powder

INSPIRATION CONSOLIDATED COPPER CO. INSPIRATION, ARIZONA



★ = STARTING PLACE AT LOWER MIAMI SCHOOL

FROM ★ TO GENERAL OFFICE.....3.5 MI.

FROM ★ TO 2ND STOP LIGHT IN MIAMI.....1.2 MI.

FROM ★ TO BULLION PLAZA SCHOOL.....2.0 MI.

FROM ★ TO OPEN PIT SHOP AREA.....2.7 MI.