CYRUS FOSS WEEKS
CONSULTING ENGINEER
KINGMAN, ARIZONA

March 13, 1953

Bureau of Mineral Resources, State of Arizona,

Attention; Mr. Roger I. C. Manning, Chief Engineer,

Phoenix, Arizona

Dear Mr. Manning;

Mr. George Reed advised me this morning that you were inquiring about the present status of the Emerald Isle mining property. The property is now open for a deal.

Thanking you for your interest,

I am,

Very truly yours,

C. F. Weeks

CFW/ww

C. F. Weeks

C-8 1½ lt 14 x 9.5 x 5 cm MINERAL SPECI[MEN] FOR DEPARTMENT OF LIBRARY AN[D A]RCHIVES K079

This specimen now in the ADMR Museum

(Do not write in this space)

Ore ________

Cabinet ________

No. ________

(Wrap each specimen separately, or place it in a substantial bag, by itself, with a number attached, identical with the number on this card.)

Specimen No. 27, collected by Elgin B. Holt
Field Engineer

Name of ore Crysocolla

Minerals contained 15% copper

Gangue Breccia

Depth at which taken 100

Approximate mineral content (in terms of average per ton) 15% copper

Name of mine or claim Emerald Isle

Group ________

District Wallapai Dist., Mohave Co., Ariz.

Location (distance and direction by highway from what town ________

Owner of property Emerald Isle Copper Co.

Operator Emerald Isle Copper Co

Mine active or inactive Active

If inactive, when operated ________

Specimen presented by Ogden C. Chase

Date Dec. 1, 1939

Notes (Any general information regarding the history of the property.) ________

If more space is desired for notes, use other side.

1-19-40

11/4/42 Emerald Isle

Coupal:-

Weeks says Emerald Isle now being examined by an Engineer, representing the Western Knapp Co., mill builders, San Francisco. American Cyanamid Co. will run the leaching tests, and if the same are satisfactory, Ayers will recommend the R.F.C. loan - $450,000 - for this operation.

About time!

Elgin

B. S. McCUTCHEN
PRESIDENT

AUGUSTUS GUMPERT
MANAGER

HOTEL HARRINGTON

ELEVENTH, TWELFTH & E STREETS, N.W.

WASHINGTON, D.C.

Oct. 30, 1942

H. C

DEPT. MINERAL RESOURCES
RECEIVED
NOV 2 1942
PHOENIX, ARIZONA

Emeral Isle

Dear Earl,

Re your letter to Ogden Chase.

I think this deal is principally in the WPB and he should authorize me to handle it with them for him.

My memo said WPB but your letter said RFC.

Bill

It is the WPB that is mostly reluctant to discuss deals without authority. I haven't had much trouble with RFC, though might if going in detail into a very large deal such as this and two letters to me with authorization to deal would prevent delays.

COPPER	EMERALD ISLE COPPER MINE
	Mohave County
Ogden C. Chase, Pres.(lessee)	Mineral Park Dist.
16-16 Boggs Bldg.	
Las Vegas, Nevada	
Report by E.B. Holt	10-3-39
" "	9-26-42

TYPE NO. 1

DEPARTMENT OF MINERAL RESOURCES
STATE OF ARIZONA
FIELD ENGINEERS REPORT

Mine EMERALD ISLE COPPER MINE Date October 8, 1942.

District Mineral Park, Mohave Co., Ariz. Engineer Elgin B. Holt

Subject:

PRODUCTION POSSIBILITY SURVEY

(Supplementing my report, dated Sept. 26, 1942, on the Emerald Isle Mine.)

-------o-------

Attention: Mr. J. S. Coupal, Director.

I am herewith enclosing a copy of a letter, dated Oct. 2, 1942, I have just received from Mr. Ogden C. Chase, President, Emerald Isle Copper Company, Boggs Building, Las Vegas, Nevada, to which he attached a detailed report from the General Engineering Company, of Salt Lake City, Utah, regarding laboratory tests made by that company on a sample of oxidized ore from the Emerald Isle property, near Chloride, Arizona.

As you know, Mr. Chase some time ago applied for a Federal loan in the sum of $450,000, to be used in completing a 300-ton heap leaching plant at the property. You will note by his letter he is very much perturbed and disappointed concerning the many delays he has experienced in getting this report, as well as getting action on his loan application.

Yesterday, Mr. C. F. Weeks, Consultant for Mr. Chase, calleed on me, and we went over the said laboratory tests carefully. I asked him for his opinion concerning the results of the tests mentioned, and he dictated the following statement:

"Broadly speaking", he said, "the report referred to shows that a 95 per cent recovery of copper values is practicable. That the acid consumption per pound of copper recovered will show an average slightly in excess of 2 pounds of H_2SO_4; and that the consumtion of scrap iron for copper precipitation will be between one and 1.25 pounds of iron per pound of copper produced.

"Furthermore", he stated, "to anyone familiar with copper leaching practice, it is quite evident from the details of the tests referred to, that Emerald Isle ores are unusually favorable for acid leaching.

"In conclusion", Weeks stated, "that while the said tests were not run parallel to Mr. R. C. Jacobson's work, in the operation of the 50-ton leaching plant in 1917-18, these tests very plainly support the conclusions as set forth in Jacobson's report."

Personally, it seems to me that the strength of acid in Test No. 13, was away lower than generally employed in other acid leaching plants. Also it will be noted in said Test No. 13, on minus 3" material the acid strength was carried extremely low. The total time of leach in Test No. 13 was 21 days. During 10 days of that time the strength of acid was kept under 10 pounds of H_2SO_4 per ton of solution, while at the same time the chemist who conducted these tests made the following statement, in reference to Test No. 12, in the said report: "The acid strength was maintained within a 10 to 20 pound range, which we believe to be desirable."

As a matter of fact, I have gathered from Jacobson's report that in treating 13,000 tons of Emerald Isle ore, by acid leaching in 1917-18, he used an initial acid strength of 5 per cent H_2SO_4, or 100 pounds, per ton of solution, and made a recovery of copper values in the ore in excess of 95 per cent. In proof of this statement, I have personally visited the Jacobson tailings pile at the property and I failed to find any stain of copper in the said tailings pile, or adjacent thereto. Again, Jacobson crushed to minus 3/4-inch and his period of leach was never in excess of five days.

Furthermore, I have tabulated Mr. Heginbotham's own results, referring to the chemist who conducted the tests, as set forth in Test No. 13; and I find that his average recovery for minus 1" plus ½", and minus ½" plus ¼", and minus ¼" plus 20-mesh, and minus 20-mesh, equals 88.3 per cent of the copper values in the ore. While on the other hand, his average recovery of copper values for minus ½" plus ¼", and minus ¼" plus 20-mesh, and minus 20-mesh material treated, equals 92.3 per cent. And these most excellent results were obtained notwithstanding the extremely low acid solution used.

Again, it will be noted by a close study of the tests, that the time element of leach is materially reduced by increasing the strength of the acid solution employed.

Another important item to consider is the following: The cost of crushing Emerald Isle ore, conceding that finer crushing is desirable, from minus 3-inch to minus ½" would amount to very little. So it appears to me, with reference to the Emerald Isle heap leaching set up, it would be immaterial, as far as costs are concerned, as to whether the ore is crushed to minus 3-inch or minus ½". Also, it seems immaterial whether heap leaching, or leaching in tanks, is employed, excepting, of course, that the installation of leaching vats would increase the cost of the final plant.

In conclusion, here is a most excellent example as to why more copper is not coming out of the ground. Every engineer who knows Emerald Isle, also knows that the copper in the ores of this property can be easily recovered by acid leaching methods. So it is plain that something should be done in the way of unwinding a lot of red tape, with the end in view of having the Federal authorities get busy and grant the loan mentioned, so that Emerald Isle can be listed as one of our active producers of copper. Elgin B. Holt.

THE GENERAL ENGINEERING COMPANY

CONSULTING ENGINEERS

Salt Lake City, Utah.

September 11, 1942.

Emerald Isle Copper Co.,
Boggs Bldg.,
Las Vegas, Nevada.

Attention Mr. Ogden C. Chase

Gentlemen:

TEST RESULTS - OUR LOT No. 1978
OPEN CUT OXIDIZED COPPER ORE

Laboratory tests have been completed on the sample of oxidized copper ore obtained from your property near Chloride, Arizona, to represent the material from open pit operations and we are submitting herewith our detailed report of procedures and results covering work on this sample.

A separate report is being prepared to cover similar work on the sample of underground ore, our lot No. 1977.

SCOPE OF WORK COMPLETED

At the time the original sample of Emerald Isle ore was sent to us, your instructions for this work provided for tests to determine:

1. Acid consumption
2. Iron consumption for precipitation
3. Leaching time.

We were further informed for our guidance in this work that it was proposed to heap leach this material at a minus 3", which plan has been given careful attention in the test work.

Before proceeding with testson material of this size it was considered advisable to carry out leaching tests on material crushed to a smaller size as a means of obtaining information relative to the leaching behavior of this material at various acid strengths and to establish relative acid consumptions and leaching periods. The tests made for this purpose were carried out on material crushed to minus 20 mesh.

Results of 13 individual tests are tabulatedon the attached sheets to show copper recoveries, acid consumption, acid strength used and leaching periods under various conditions. These results provide a basis for the following general conclusions:

1. Acid consumption for leaching operationsonly will not exceed 100 lbs. per ton of ore, or 2.5 lbs. per pound of copper dissolved as in tests showing satisfactory copper extractions.

2. Acid strengths of 10 to 35 lbs. per ton of solution should be maintained for best results.

TYPE NO. 1

DE. ARTMENT OF MINERAL RESOUR ES
STATE OF ARIZONA
FIELD ENGINEERS REPORT

PRODUCTION POSSIBILITY SURVEY

Mine EMERALD ISLE COPPER MINE   Date September 26, 1942

District Mineral Park, Mohave Co.   Engineer Elgin B. Holt

Subject:

PRODUCTION POSSIBILITY

LESSEE: Emerald Isle Copper Co., Ogden C. Chase, Pres., 15-16 Boggs Bldg., Las Vegas, Nevada.

METALS: Copper. No other metals.

LOCATION

This property is located 15 miles north of Kingman, Arizona, and 5 miles east of U. S. Highway 93, with which it is connected by a county maintained dirt road, in fair condition.

MINERALIZED CONGLOMERATE

A bed of conglomerate, with slight overburden at the main deposit, is from 60 to 100 feet thick; said conglomerate being impregnated with copper silicate over a considerable area. Under the conglomerate the basal rocks are pre-Cambrian granit complex rocks. A copper-bearing vein from 5 to 7 feet wide, cuts both the conglomerate and the granite underneath. This vein has been opened by a shaft 100 feet deep, as well as by 1,600 feet of drifts and crosscuts. The vein ore runs considerably higher than the conglomerate ore beds above the granite mentioned.

HISTORICAL

In 1917-18, R. C. Jacobson, Assayer and Chemist, Kingman, Arizona, and those associated with him at that time, installed a 50-ton test leaching & electrolytic plant on property, and thereby made a recovery of copper in excess of 95%. It required 100 pounds of concentrated sulphuric acid to leach a ton of ore containing 50 pounds of copper.

The Jacobson plant was being operated successfully, notwithstanding high electric power costs at that time, until the close of World War I, and the consequent slump in copper caused operations to cease. In carrying out this operation, Jacobson milled 13,000 tons of ore from a surface quarry that produced net about 50 pounds of copper per ton of ore treated. Ore was crushed to 3/4-inch mesh before being placed in acid leaching vats; hence fine grinding was found to be unnecessary.

The conglomerate ore bed, which, as stated is from 60 to 100 feet thick, has been proved by test pits to have an area on the surface of 161,850 square feet. Assuming an average depth of the conglomerate to be 60 feet, each square feet of surface would contain 4 tons of ore underneath. This would give 647,400 tons of ore, averaging 2.38% copper, per Jacobson. An average of 120 samples underground, gave 3.36% copper. Jacobson says that further development of the surface conglomerate area should yield an additional 500,000 tons of ore, making over 1,000,000 tons of indicated ore, in this immediate area. Should the conglomerate copper-bearing bed continue under the flat for some distance to the west in unexplored ground, it is possible that several million tons of additional ore may be found later on in this area. The 1,000,000 tons of indicated ore mentioned can all be broken by surface quarry mining methods and removed by means of power shovels.

ASSAYS

Per Jacobson, in 1917 a composite sample of 900 tons of ore from a surface quarry gave the following analysis:

Cu	Fe	Mn	Al	CaO	Insol
2.43%	2.6%	2.56%	3.26%	0.06%	84.6%

EMERALD ISLE COPPER COMPANY

During the year 1940, Ogden C. Chase organized the Emerald Isle Copper Company and succeeded in raising sufficient money to start the erection at the property of a heap acid leaching plant with a capacity of 300 tons of ore per day. This plant was partly constructed; but the company ran out of funds and had to discontinue operations.

APPLICATION FOR RFC LOAN

During June of 1942, Chase applied for a loan of $450,000, from RFC, to be used for the purpose of completing the said 300-ton leaching plant mentioned. Also a part of the money applied for, if granted, will be used for the purpose of drilling the property, with a view to hunting for new ore reserves. It is believed such drilling operations will result in the discovery of large areas of conglomerate, unknown at the present time, carrying about the same copper content as is now proven in known ore reserves.

This property warrants careful consideration, as undoubtedly it is an important source for new copper production.

Elgin B. Holt

DEPARTMENT OF MINERAL RESOURCES
STATE OF ARIZONA
FIELD ENGINEERS REPORT

Mine	Emerald Isle	Date	April 16, 1957
District	Mineral Park, Mohave Co.	Engineer	MARK GEMMILL
Subject:			

In late 1955 C. G. Paterson, Box 174, Chloride, Arizona, acquired the property under lease and option to buy and mined and shipped ore from one rich streak. After working several months, apparently worked out the richer ore, and closed down.

Mr. Paterson recently stated that he is dealing with some people with the idea of getting a leaching plant in operation on the property.

Frank P. Knight Notes

Emerald Isle Copper Co.
1942 MR
Ogden C. Chase, Pres. & Gen.
C. I. Chase, Sec-Treas.
C. F. Weeks. Gen. Supt. 825 W
Kingn
Office Bogg Bldg., Las Vegas
1946 MR
Office VNB Bldg., Tucson Chase, Pres.
1949
Office Kingman Chase, Pres.
1952
Office Kingman No officers listed
1956
Inactive List

C. G. Paterson, Box 174, Chloride
Lewin-Mathes Co. 1947-48 St. Louis

Ogden Chase deceased 3-5-47 report
His brother manager
Property owned by MD. Laujon & C.L. Weeks

E

5 of 5

DEPARTMENT OF MINERAL RESOURCES
STATE OF ARIZONA
FIELD ENGINEERS REPORT

Mine EMERALD ISLE MINE Date March 14, 1953

District MINERAL PARK, MOHAVE COUNTY Engineer George F. Reed

Subject: Report

Re-Present Status of Emerald Isle Copper Property, it has been inactive for some time. The crushing plant and leaching plant are largely intact. I do not have accurate data on grade of ore, etc., and there are old reports on this in the office at Phoenix. I asked Cy Weeks about it and he said he would write you direct about the property, so I think you will hear from him by the time you get this letter.

As you probably know, Lewin-Mathes Company operated there under management of Earl Hastings about 1947 and 1948. They got up to about 700 tons per day production of ore running around 2.0% copper in form of Silicates and oxides. The bulk of the ore lays more or less flat and is cemented gravel laying on bedrock. The ore is 20 or 30 feet thick, more or less, and has some barren gravel over it, possibly 5 to 20 feet, or more. Richer ore running on the order of 5. to 8.% was mined from a steep dipping fissure which comes up through the bedrock and into the gravel. This ore was shipped to a smelter by earlier operators. It was five to 12 feet wide. Lewin-Mathes mined with power shovels, trucks, etc.

EMERALD ISLE

DEPARTMENT OF MINERAL RESOURCES

TO ALL PRODUCERS OF COPPER, LEAD and ZINC IN ARIZONA:

This department and others are making strenuous efforts to bring about legislation which will help ameliorate the restrictions and difficulties faced by the producers of copper, lead and zinc, and other strategic minerals.

To assist in these efforts it is advisable that we have an authentic survey of the results of the President's veto of the Allen Bill, and the results that would take place if a new bill, such as the Russell Bill, were passed by Congress. The Russell Bill includes all strategic minerals.

While we have all learned to love questionnaires just as we love stomach ulcers, will you please give the answers in your best judgment to the following questions:

1. What was your approximate production in pounds per month for the period preceding the President's veto of the Allen Bill?

   (Copper 70000 Lbs.) (Lead ________ Lbs.) (Zinc ________ Lbs.)

2. What has been your average production per month since that veto has affected your price?

   (Copper 100,000 Lbs.) (Lead ________ Lbs.) (Zinc ________ Lbs.)

3. What is your estimate of your production per month for the first few months of 1948 if prices remain as they are now and no premiums are in effect?

   (Copper None Lbs.) (Lead ________ Lbs.) (Zinc ________ Lbs.)

4. What is your estimate of production per month if some incentive plan such as the Russell Bill were in effect?

   (Copper 300,000 Lbs.) (Lead ________ Lbs.) (Zinc ________ Lbs.)

5. General remarks: ____

An addressed envelope is enclosed for your convenience, but you will have to help with the stamp.

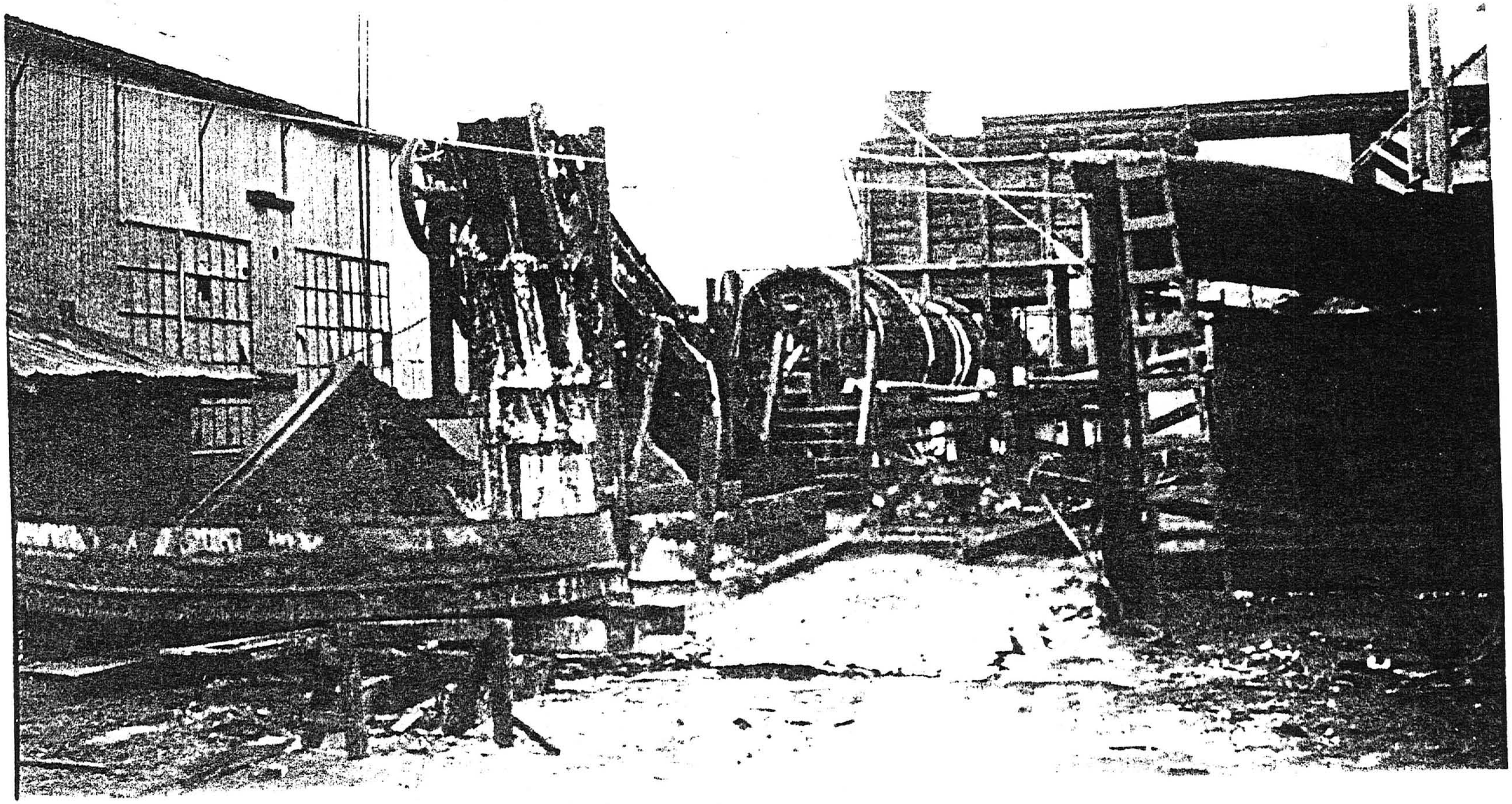
Yours very truly,

Chas H Dunning

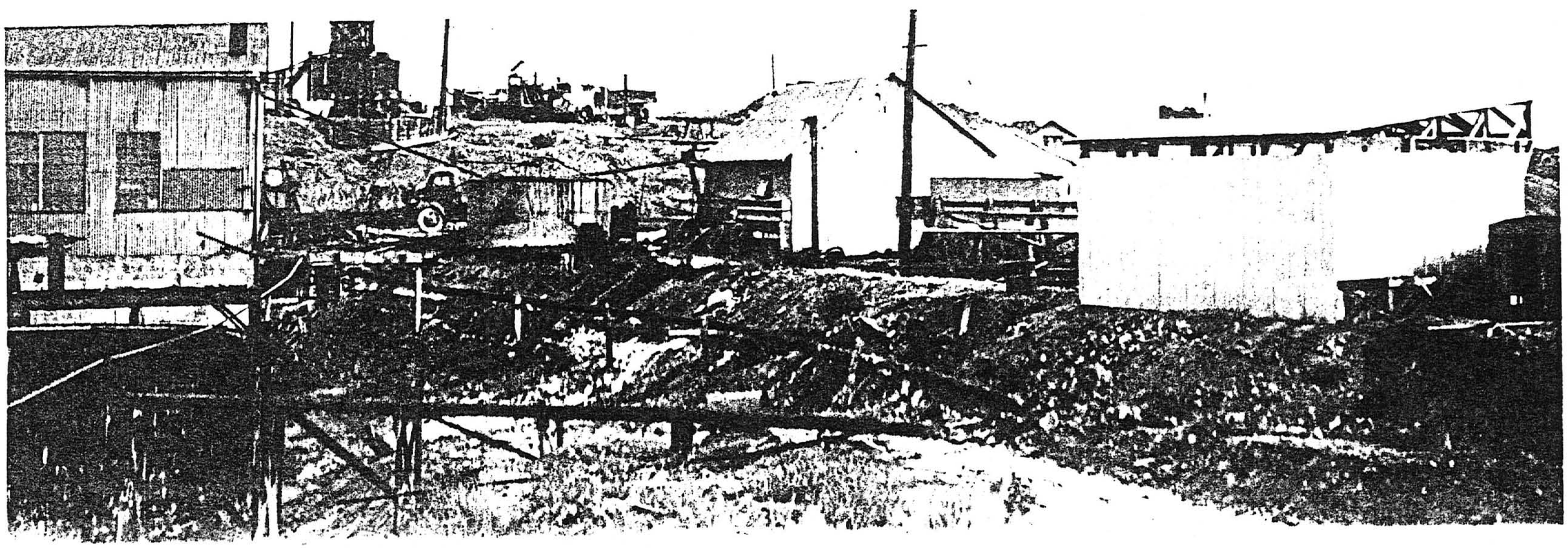
Chas. H. Dunning
Director

CHD:mh

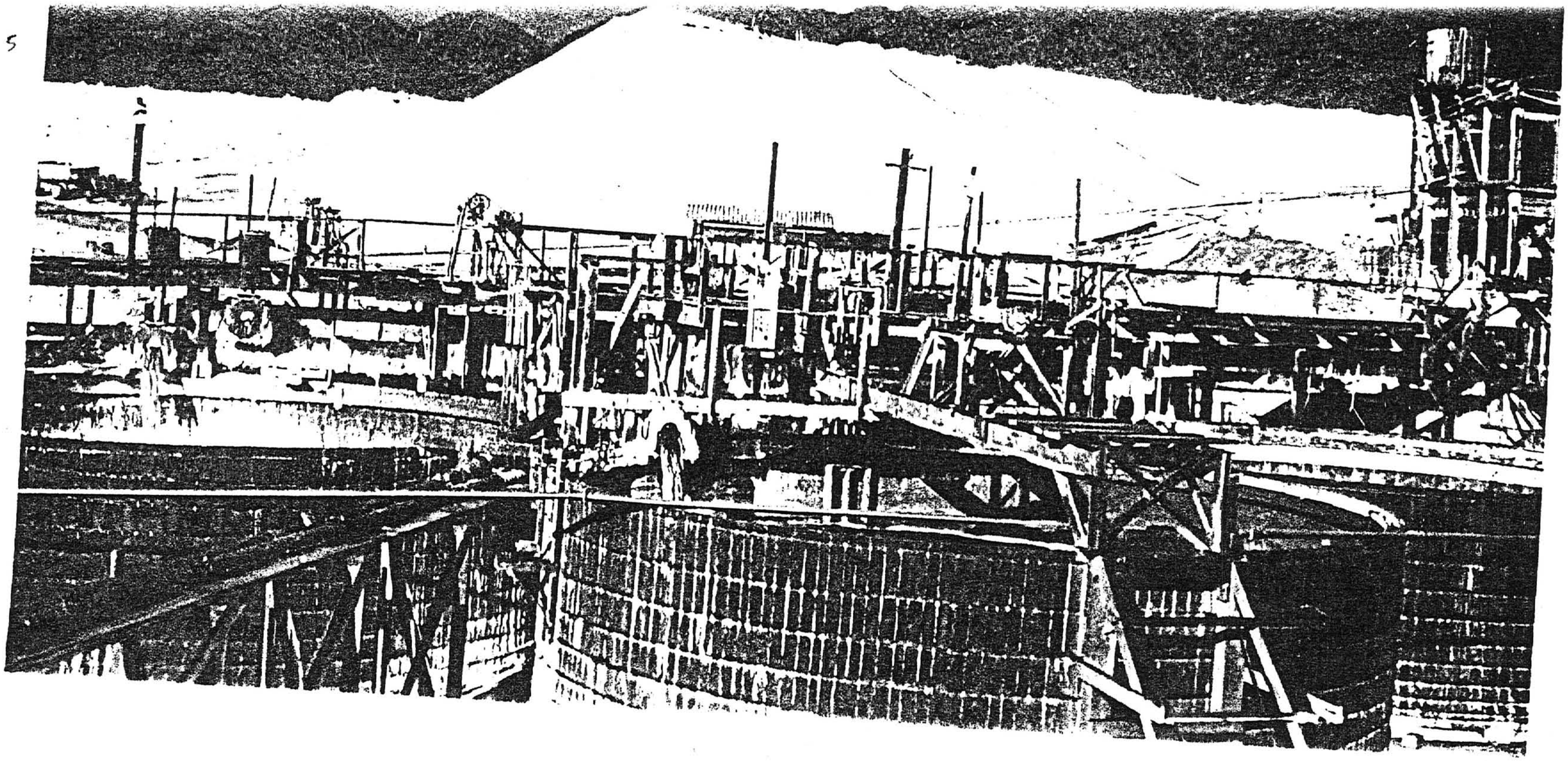
A-40-4
(1947)



A-40-3
(1947)



A-40-5



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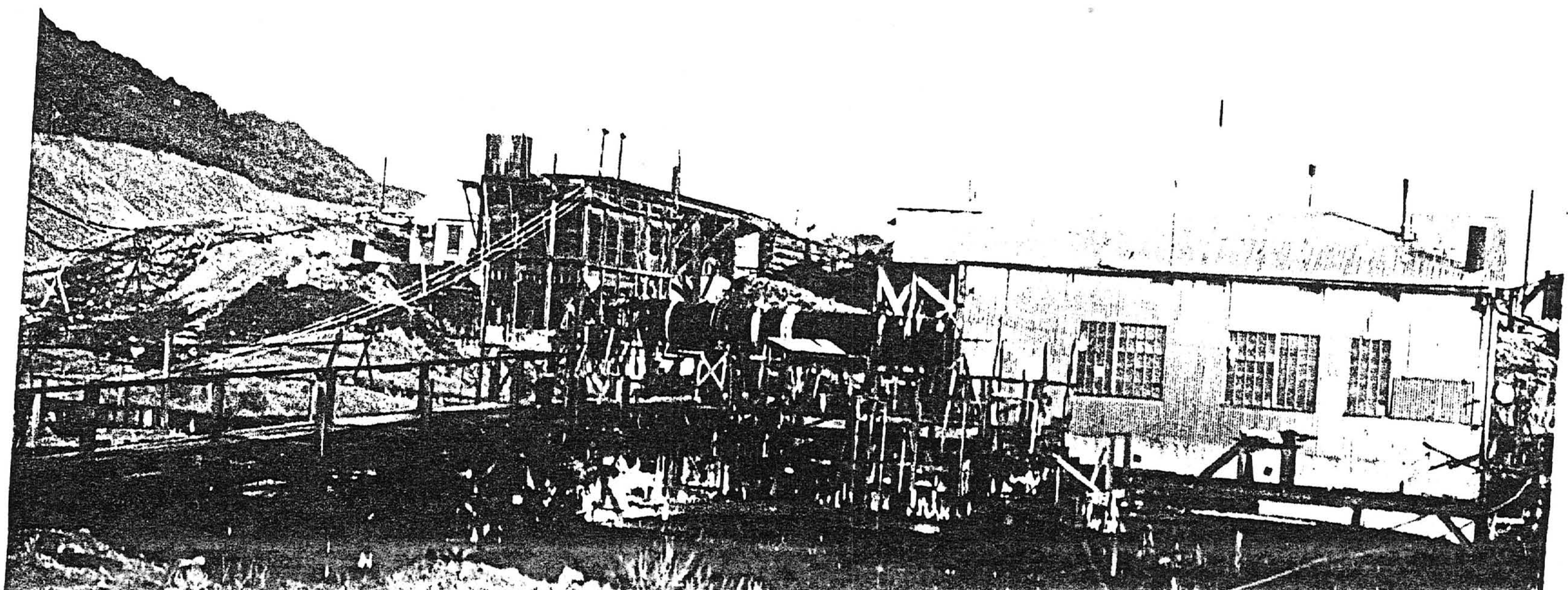

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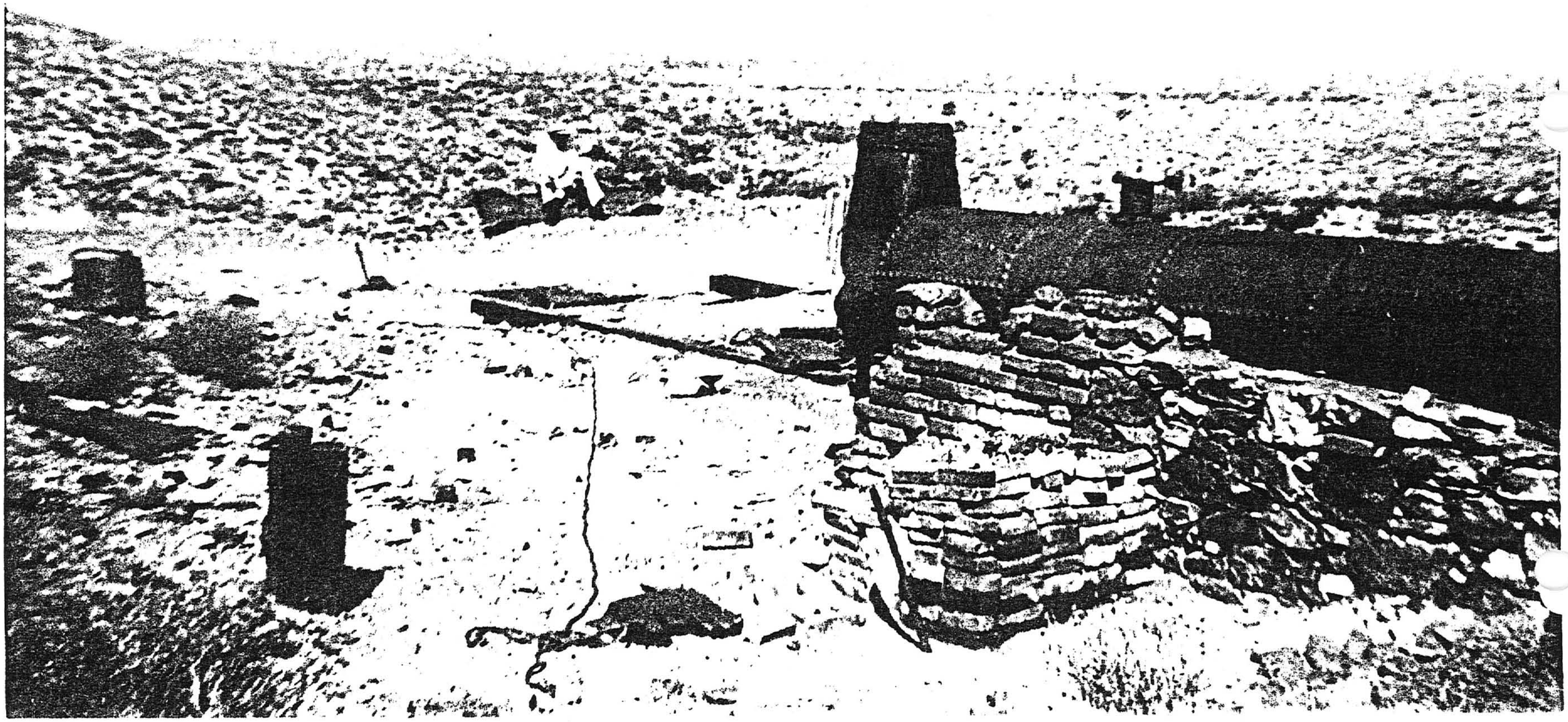
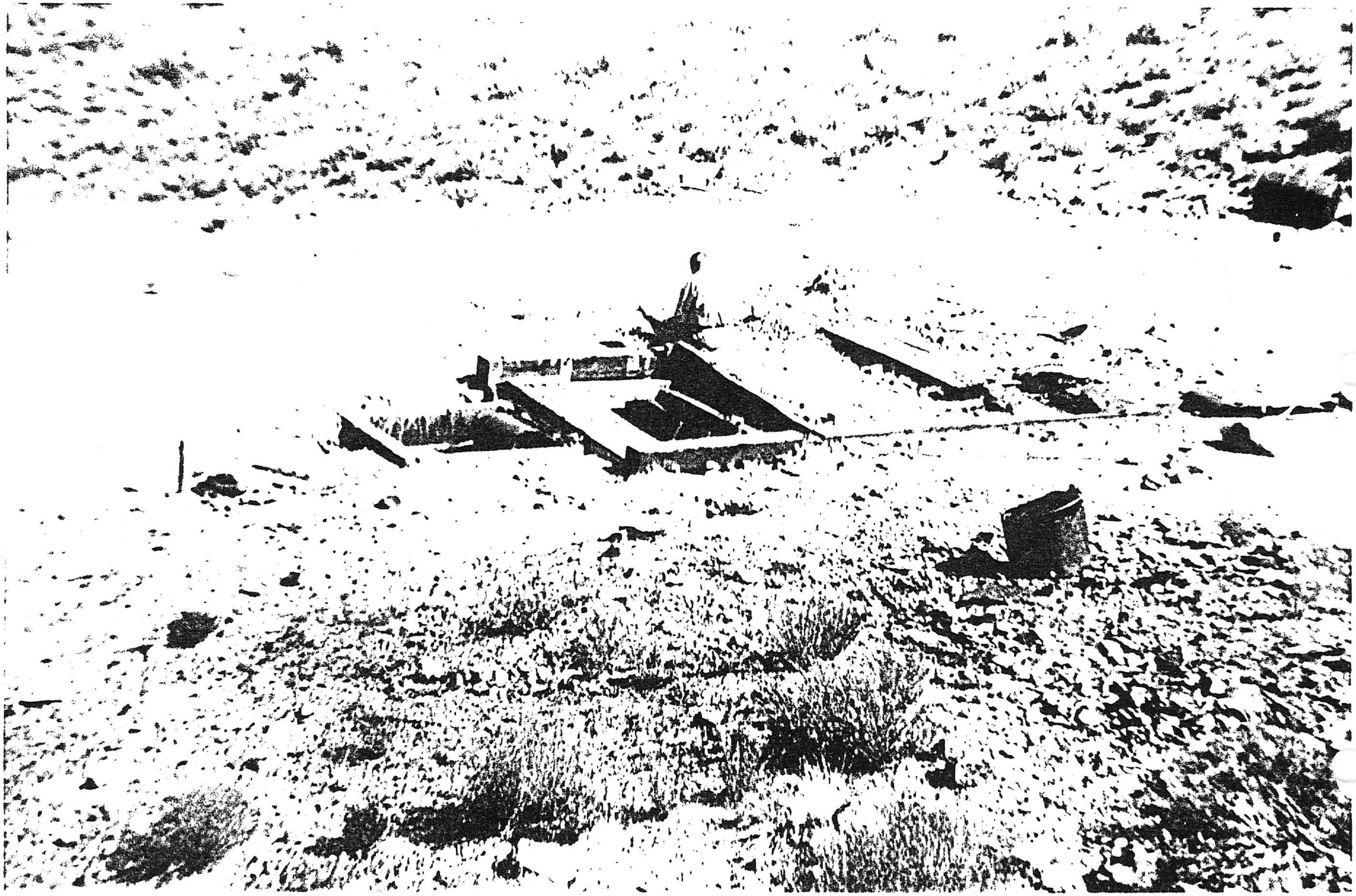
(1944)



A-40-2
(1944)

MAN
ELGIN HOLT



DEPARTMENT OF MINERAL RESOURCES
STATE OF ARIZONA
FIELD ENGINEERS REPORT

Mine Emerald Isle ✓ Date July IO I947

District Mineral Park, Ariz. Engineer A. C. Nebeker

Subject: Operations

The Emerald Isle mine and Leaching plant is located I7 miles north west of Kingman, Arizona, and about 3 miles off the main oiled highway, on the slope of the west side of Cerbat Range.

The property has recently been taken over , under long time lease contract, by the Lewins-Mathes Co, of St Louis Mo. and Mr Bob Payne is superintendent of operations.

The ore consists of a coating on conglomerate with the values occuring in the copper silicate, chrysocolla. The values in the conglomerate average I½ % copper, and shipments underground from the bottom of a shaft have run 6 to 8 % in car load lots. It has been found recently that the conglomerate boulders are impregnated with enough copper values to justify crushing the boulders and sending it to the leaching vats.

The ore is mined by open pit methods. The surface overburden, which is from zero to twentyfive feet thick, is removed, and if the conglomerate is tight it is shattered or loosened by powder and then the power shovel and dragline scraper are used to load the ore on trucks which haul it to the crushing plant of 500 tons capacity per 8 hours. The ore is crushed to minus 3/8 inch and passes on to the classifier where the slimes are removed. These slimes are being saved waiting for the construction of agitation tanks where they will be leached at a later date.

The 3/8 inch and minus 3/8 inch is trucked too the head of the leaching vats where a push scraper is used to fill the vats and level them off.

There are four rectangular vats of 300 tons capacity each. Thevats are worked in rotation. One vat leached per day, which at present making a 300 ton per day capacity plant. After the vat is filled, the leaching solution is pumped through the bottom of the the vat. The solution is a sulphuric acid solution with strength of 3.5 lbs of acid per I lb of copper. The impregnated solution is drawn from the vat and feed to a 6 ft. X 60ft. revolving drum where shredded iron is added to precipitate the copper. 3/4 lb to I lb of iron is used per pound of copper. The drum revolving one revelution per min. It is said that the cement copper assays 80% copper.

It is now estimated there is proven I,300,000 tons of commercial ore. This ore has been outlined by pits, one shaft with drifts and numerous drill holes. These test drill holes are put down with a Wagon Dill outfit using bits of 3 inch gauge for starters and finishing with 2¼ inch gauge at 25 foot depth.

The company is now stockpiling a large tonnage of prepared ore for heap leaching and expects to bring the production up to I,000 tons per day.

The premium or bonuses for copper is not worrying this company as they are producing copper for their own manufacturing plant.

A force of 30 men is now employed mining and leaching the ore, and doing some revamping of the plant to remove some of the 'Bugs' found while operating.

A good camp has been constructed consisting of several residences, Assay office, machine shop and other usual mining equipment.

DEPARTMENT OF MINERAL RESOURCES
STATE OF ARIZONA
FIELD ENGINEERS REPORT

Mine Emerald Isle

Date March, 5th, 1947

District Mineral Park, Mohave Co, Ariz.

Engineer A.C.Nebeker

Subject: Check on operations.

On Mar.5th. I went out to the Emerald Isle property to check up on reports I had received, that the property was getting underway and was under the direction of a Mr Chase, a brother of Odgen Chase, deaeased, the former president.

I found that the property was inactive as far as, leaching and mining were concerned but efferts are being made to get the property producing again.
The property is owned by Mr. Maurice D. Laujon and C.L. Weeks, and they are the ones who are putting the property in shape for operations, and expect to be running within the next month. They expect to leach the 2% ore from surface ores and ship the higher grade ore which will come from the shaft.

ACN

REPORT ON THE PRELIMINARY EXAMINATION

OF THE

EMERALD ISLE MINE

MOHAVE COUNTY, ARIZONA

By

FRED GIBBS

Prescott, Arizona

November, 1959

CONTENTS

REPORT ON PRELIMINARY EXAMINATION

of the

EMERALD ISLE MINE

MOHAVE COUNTY, ARIZONA

PURPOSE AND FORESTATEMENT:

The purpose of this preliminary examination was to determine whether or not the indicated economic potential was great enough to warrant a more thorough and detailed study.

The property had been presented to the Company by Mr. Wm. Hampton of 639 S. Donna Beth Ave., West Covina, Calif., who has a short term sub-lease and option contract on it.

The field work was carried out on November 3rd, 4th and 5th. Mr. J. T. Jordan, a registered engineer and geologist of Kingman, was assigned by Mr. Hampton to escort me to and over the property on the first day. Mr. C. G. Patterson of Chloride, who is thoroughly familiar with the property, helped me with the underground examination on the second day.

I had access to a short report written by Mr. Jordan in April, 1959, to which was appended an estimate of probable leaching plant cost and operating costs compiled by Charles T. Wyatt of Chloride. Mr. Jordan's report was based chiefly on data collected and compiled by the Lewin-Mathes Mining Company when it operated the property in 1947-48. Most of this data consisted of maps showing vertical sections thru lines of drill holes and on which depths of overburden and thickness of ore, with assays, was shown. Copies of some of these maps were made available to me and are included in the

Appendix. Unfortunately, other and additional Lewin-Mathes maps are missing and Mr. Patterson said that they had been loaned to somebody and had never been returned. Since practically all of the deposit is covered with recent alluvium, and since most of the underground workings are now inaccessible, the missing maps which contain the balance of the drilling data would be very helpful in trying to arrive at an overall picture of the ore reserves, both developed and potential.

In addition to the above data, Mr. Jordan loaned to me a copy of "Ore Deposits of The Wallapai District, Arizona" by Blakemore F. Thomas, and reprinted from Economic Geology, Vol. 44, No. 8, published in December 1949. This article was the subject matter of a thesis written by Thomas in connection with his application for a Doctor's Degree in Geology from Cal Tech. The article carries a description of the Emerald Isle Mine.

CONCLUSIONS:

The known tonnage in the deposit is too small, and the grade too low at present copper price, to permit of an economically profitable operation by any method of treatment. Additional exploratory drilling might disclose additional tonnages of ore, but there is no reason to believe that this potential additional tonnage would be of better grade than the present known tonnage, which, as above stated, is too low to provide a worthwhile profit.

PROPERTY:

The property consists of six unpatented claims located in the Wallapai Mining District of Mohave County about three miles south of the old mining town of Chloride and about 16 miles north of Kingman. It is owned by a Mrs. Downey, address not obtained. C. G. Patterson of Chloride holds a lease and option contract from Mrs. Downey which runs until 1969. Under this contract Patterson does the annual assessment work and pays a 10% royalty, based on the net smelter returns, on any ore or concentrates which he may market. Mr. Hampton has a short term (exact length not disclosed) sub-lease and option from Mr. Patterson.

HISTORY:

Initial exploration and development dates back to pre-World War I days. At that time an 80' vertical shaft was sunk on the mineralized cropping of the so-called fissure vein and drifting and crosscutting over a length of approximately 1,000' was carried out at that level. A little stoping was done on direct shipping ore above this level. Records of the amount and grade of ore shipped are not available. Subsequent thereto, and up until late 1946 or early 1947, several attempts were made by different leasers to leach the oxidized copper minerals, with indifferent results. In late 1946 the Lewin-Mathes Mining Company obtained a lease and option on the property and made a serious attempt to exploit the oxidized copper deposit. They carried out an exploratory drilling campaign using both wagon and churn drills on the mineralized alluvial blanket. Following this they built a percolation leach plant which they operated for a time on crushed ore mined in an open pit. They also attempted heap leaching on a portion of the ore. The project was apparently unsuccessful and terminated in late 1947 or early 1948. One reason given for failure was the inability to procure acid at a reasonable price and in the quantities needed.

In the early 1950's, Mr. Cy Weeks, an engineer of Chloride and Kingman, conducted a small direct-shipping operation and another attempt at leaching on a small scale under lease from the owner. His operation was not successful.

In 1953 Mr. Patterson obtained a lease and option contract and proceeded to mine and ship ore from the so-called fissure vein and he also shipped screened fines left by the Lewin-Mathes operation. When the price of copper dropped in 1957 Mr. Patterson ceased operations, but still holds the property under a long-time lease and option contract as above mentioned. Only assessment work has been carried out in the last two years and Mr. Patterson is concentrating his efforts now on an attempt to locate capital to be used in additional exploration and in establishment of a leaching plant if same seems justified.

GEOLOGY:

The deposit consists of a mineralized alluvial blanket of probably Gila age. Its shape, size and general attitude is shown on the Index Map in the Appendix. This blanket is overlain in most parts by recent unmineralized alluvials of varying depths, and is underlain by granite porphyry of pre-Cambrian age. The chief copper mineral is chrysocolla and in my opinion it was deposited in the Gila conglomerate by meteoric waters which had picked up their copper content thru leaching of porphyry type copper deposits located in the Mineral Park district on the west flank of the Cerbat Range a mile or so to the east. From observation, and from data compiled from the Lewin-Mathes operation, it seems likely that the mineralized conglomerate bed occupies an old river channel incised in the underlying granite porphyry. It is probable that during the period of copper deposition in the conglomerate that the bed was many times thicker than now, so that it not only filled the old river channel but covered to some depth the areas outside of the channel lips. Subsequent erosion probably accounts for the fact that the remaining conglomerate is that now occupying the old river channel. Structure-wise, a parallel to the situation can be found in some parts of the Colorado Plateau where old river channels incised in the Moenkopi formation are filled with Shinarump Conglomerate, particularly so where the Shinarump has been eroded from the areas outside the river channels exposing the Moenkopi on both lips and leaving only the conglomerate filled channels.

In one part of the mineralized blanket there is what appears at first glance to be a vertical fissure vein.

This is a fracture zone 3 to 12 feet wide in which enough chrysocolla has been deposited to raise the copper grade to several times that in the adjacent unfractured alluvial, high enough, in fact, to permit direct shipping of some of it. It was on this fracture zone that the original 80' shaft was sunk and on which the drifting therefrom was carried out.

It has been generally considered by most observers that this fracture zone constitutes a true fissure vein and that it carries thru into the underlying granite porphyry, a concept which has given rise to much controversy as to the origin of the copper mineralization. Thomas, in his thesis article mentioned above, subscribes to this same fissure vein idea and ascribes the chrysocolla mineralization to hydrothermal action. His theory is that the ascending solutions spread out into the pores and fractures of the conglomerate and deposited the copper as chrysocolla, a concept which I cannot accept for the following reasons:

The western part of the open pit mined by Lewin-Mathes reached this fracture zone, and later Patterson extended the workings along the zone to the old shaft, and subsequently sank a 50' winze below the 80' level. From the bottom of the winze he drifted in mineralized conglomerate about 150' north and south and did some stoping. All of this work disclosed that the old vertical 80' shaft reached the underlying granite and that the top surface of the granite at this point pitched steeply off to the west. Patterson's winze followed down over the top of the granite at about a

60 degree pitch. This winze, and the work done on the fracture zone near the shaft, has exposed the underlying granite for a length of at least 100 feet. In this exposure there is absolutely no hint of a fissure vein or any other fracturing in the granite. From the top of the granite just east of the strike of the fracture zone in the conglomerate, the upper surface of the granite pitches steeply to the west and is slicken-sided. Furthermore, a pronounced fracture carries up thru the conglomerate from the top edge of the granite with the same pitch as the steeply dipping slicken-sided surface. (See Section G-G' in Appendix).

Accordingly, it seems certain to me that a profound fault dipping steeply west cuts both the underlying granite and the over-lying conglomerate bed and that the down-thrown block is the westerly one. Also, that the fault is post mineral, but still early enough that the overlying conglomerate bed was perhaps many times thicker than now. The so-called fissure vein in the bed is merely a vertical-trending hanging wall break off the main fault and its higher grade is due to further post-faulting leaching from both the original source and from the bed itself - the fracturing lending itself well to extra deposition. Actually, this fault could well be an extension to the south of the big regional Sacramento fault west of Chloride - its projected but alluvial-masked extension would pass thru or close to the Emerald Isle. On the Sacramento fault the downthrown block is also on the west.

I feel that my concept of the situation is about the only explanation for the relatively great depth (over 100') of mineralized conglomerate cut by churn drill hole No. 7. Mr. Thomas's concept seems irreconcilable with the evidence, because (1) there is no trace of a fissure in the underlying granite (2) there is no evidence anywhere of sulphide mineralization nor any relics of same and (3) grade in the bed does not diminish with distance from the vertical "fissure", as would be a logical expectation if mineralization was due to solutions emanating sidewise from the "fissure". Sheet G-G' in the Appendix is a sketch showing my idea of the structural situation as exposed in the present workings.

All of the above is perhaps academic so far as the purpose of this report is concerned. After all, initially at least, it is a matter of how much and how good that counts - and not how it got there. However, the matter of which concept is the correct one could have a bearing on further exploration of the deposit in the future, assuming that conditions in the industry presently unforeseen and unexpected should make such exploration justifiable. If Mr. Thomas is correct, the possibility of extending known reserves is practically nil. If my thought is correct, exploration both to the east and to the west (particularly the latter) could well add additional reserves, and in addition geophysical prospecting might well find other mineralized erosional segments of the old Gila bed which are now entirely masked by recent alluvials.

As might be expected, the thickness of the mineralized bed is variable as is the overlying barren recent gravel. East of the fault the thickness of the mineralized bed averages 25' as does also the overlying barren recent alluvials. The surface of the old granite basement rock to the east of the fault is uneven but fairly level so far as the drilling has progressed. The open pit discloses that the granite slopes gently up to the north from the vicinity of the old shaft and must level off somewhere to the north between pit edge and the excavation for the leach plant 200' to the north since said excavation exposes the granite within 4' of surface and covered by recent alluvials but no Gila conglomerate. This gentle slope up to the north suggests that this is the north side of the old river channel. The south side is completely alluvial covered and the drilling in that direction is quite limited so that the shape of the south side of the channel is unknown. Probable width of channel from lip to lip east of the fault would be in the order of 500' and depth at center possibly 80 feet.

Aside from the enrichment on fractures such as the "fissure vein", the mineralization appears to be rather uniform both vertically and laterally. In addition to the chrysocolla there are minor amounts of other copper oxide minerals, all of which occur as coatings on the sand grains and gravel boulders but do not penetrate same.

ORE RESERVES:

Jordan's estimate of 509,000 tons of 1.056% copper

EMERALD ISLE COPPER CO., Mohave County, Arizona. Ogden C. Chase, Pres., Las Vegas, Nevada.

This is a copper silicate deposit of magnitude which has not been profitably operated though frequently promoted.

The General Engineering Co. of Salt Lake City, Utah recently completed a series of metallurgical tests, which are reported to show that the ore is amenable to beneficiation on an economical basis. There is approximately 500,000 tons of copper bearing conglomerate assaying according to reports, 2.25%.

Application has been made for an RFC loan with which to complete the 300 ton leaching plant and to explore for an additional 500,000 tons of ore which is believed to exist.

On completion this plant would yield in excess of 10,000 pounds of copper per day or 300,000 pounds per month.

A rather elaborate plant would be required and, with the shortage of labor for construction of it as well as operating labor following construction, an early production could not be anticipated.

REPORT BY Earl F. Hastings, October 9, 1942, to Copper Branch, War Production Board.

Against this net smelter value must be charged the following:

Mining (includes stripping)		$1.00
Milling	(Labor --------0.40) (Acid ---------0.85) (Iron ---------0.62) (Water --------0.04) (Maintenance --0.05) (Power --------0.03)	1.99
Overhead at operation	(Management ---0.06) (Office -------0.03) (Assaying -----0.06) (Insurance ----0.03)	0.18
Personal property taxes		0.05
Royalty (10% of net smelter)		0.46
Amortization ($250,000 plant)		0.50
	Total	$4.18

$4.63 - $4.18 = $0.45 net profit per ton before taxes.

$0.45 - $0.23 (Fed. Income Tax) = $0.22 profit per ton.

This figure is considered insufficient to warrant exploitation of the deposit under present conditions especially since the above does not take into account legal expense, head office expense, and various unforeseen contingencies which probably would arise. Amortization cost is high because of the relatively costly leach plant necessary and the low available tonnage. The royalty cost is excessive on ore of this grade but I understand that this is the figure which would apply. The figure of $250,000 used as cost of a 500 ton plant may be too low, especially since very little of the old plant used by Lewin-Mathes and still on the ground is useable.

The water supply situation as of now is not clear. The water would have to come from the old Tennessee-Schuylkill mine at Chloride and be carried to the operation thru a 3-mile pipeline. Mr. Patterson says that the Tennessee is owned by the Internal Revenue Department and that he is the agent for it. The IRD took over the property for non-payment of certain taxes and it is conceivable that it might sell the property at any time for the amount involved. Thus, since it is the only source of water in the area, a firm commitment would have to be had from the IRD for whatever period of time might be necessary.

RECOMMENDATION:

My recommendation, under the conditions now obtaining, is that the Company give no further consideration to the leasing or acquisition of the property at this time.

At some future time, when the title situation may be less complicated and copper prices somewhat higher, it might be desirable, if some gambling funds are available, to explore via geophysical methods the adjacent and surrounding areas in the hope that other deposits of similar nature may be found in sufficient quantities to justify exploitation.

Fred Gibbs
Prescott, Arizona

November 1959

ARIZONA DEPARTMENT OF ENVIRONMENTAL QUALITY

Rose Mofford, Governor
Ronald Miller, Acting Director

NOTICE OF INTENT TO ISSUE A GROUNDWATER QUALITY PROTECTION PERMIT(S)

Pursuant to Arizona Administrative Code, Title 9, Chapter 20, Article 2, the Director of the Arizona Department of Environmental Quality intends to issue a Groundwater Quality Protection Permit(s) to the following applicant(s), subject to certain special and general conditions.

Public Notice No. 111-88AZGW
Emerald Isle Mine (file) Mohave Co.
TSC Enterprises, Inc.
4449 East Monte Vista
Tucson, Arizona 85712
Groundwater Quality Protection Permit No. G-0053-08

On or about November 16, 1988

The applicant will operate an in-site copper leaching operation located four (4) miles south of Chloride, Arizona in Mohave County on the Old Chloride Road over groundwater of Sacramento Valley Basin in Township 23 North; Range 18 West; Section 22; North of the Gila and Salt River Baseline and Meridian.

The facility will operate seven days a week by discharging dilute sulfuric acid solution through leach lines to the bottom surface of the open pit. Pregnant leach solution shall be recovered by pumping no less than three (3) of the six (6) downgradient recovery wells simultaneously. Pregnant leach solution shall then be pumped to the cone precipitation where the copper shall be removed from solution. The tail or raffinate solution shall then be discharged to a High Density Polyethylene (HDPE) lined raffinate pond.

The permit includes requirements for groundwater monitoring on a monthly and bimonthly basis. Groundwater samples shall be obtained through two (2) downgradient monitoring wells.

The permit and related material are available for public review Monday through Friday, 8:00 a.m. to 5:00 p.m. at Arizona Department of Environmental Quality, Water Permits Unit, 2005 North Central Avenue, Phoenix, Arizona 85004.

Persons may submit comments or request a public hearing on the proposed action, in writing, to ADEQ at the above address within thirty (30) days from the date of this notice. Public hearing request must include the reason for such request.

COMPLETE AND MAIL TO:

STATE MINE INSPECTOR
1616 WEST ADAMS, SUITE 411
PHOENIX, ARIZONA 85007-2627

MINE INSPECTOR
NOV - 8 1989

FOR OFFICE USE ONLY
START-UP NUMBER 94365958
STATE NUMBER 091200
MSHA NUMBER Dave

EMERALD ISLE (F) MOHAVE CO.

NOTICE TO ARIZONA STATE MINE INSPECTOR

In compliance with the Arizona Revised Statute Section 27-303, we are submitting this written notice to the Arizona State Mine Inspector of our intent to start X stop ____ move ____ (Please check one) a mining operation.

If this is a move, please show last location: ____

If you have not operated a mine previously in Arizona, please check here: X If you want the Education and Training Division to assist with your mine safety training, please check here: X

If this operation will use Cyanide for leaching, please check here: ____

COMPANY NAME: Arimetco Inc.

DIVISION: TSC Enterprises (a wholly owned subsidiary of Arimetco, Inc.)

MINE OR PLANT NAME: Emerald Isle Mine TELEPHONE: Tucson office-290-9200 Mine-565-4554

CHIEF OFFICER: H. R. Shipes

COMPANY ADDRESS: 8835 E. Speedway Blvd.

CITY: Tucson STATE: Arizona ZIP CODE: 85710

MINE OR PLANT LOCATION: (Include county and nearest town, as well as directions for locating property by vehicle: SE ¼ Section 22 T23N R18W GSRBM

Highway 93 North of Kingman to Mineral Park Road, East 1.5 mile, North 1.7 mile, East into mine.

TYPE OF OPERATION: in situ leach PRINCIPAL PRODUCT: copper

STARTING DATE: Construction in progress Mining11/30/89 CLOSING DATE: not determined DURATION: ____

PERSON COMPLETING NOTICE: Harrison Matson TITLE: Mining Geologist

DATE NOTICE MAILED TO STATE MINE INSPECTOR: November 3, 1989

FORM 101-106 REV. 01/88

EMERALD ISLE (f)

K 74353274.

[illegible]

#10156500

Office of State Mine Inspector

705 West Wing, Capitol Building
Phoenix, Arizona 85007
602-255-5971

NOV [illegible] 1987

NOTICE TO ARIZONA STATE MINE INSPECTOR

In compliance with Arizona Revised Statute Section 27-303, we are submitting this written notice to the Arizona State Mine Inspector (705 West Wing, Capitol Building, Phoenix, Arizona 85007) of our intent to start/stop (please circle one) a mining operation.

COMPANY NAME TSC ENTERPRISES, INC

CHIEF OFFICER G H Staggers

COMPANY ADDRESS 4449 E MONTE VISTA, TUCSON AZ 85712

COMPANY TELEPHONE NUMBER 602-795-1183

MINE OR PLANT NAME Emerald Isle Mine

MINE OR PLANT LOCATION (including county and nearest town, as well as directions for locating by vehicle)

Chloride, Mohave Co., 2 mi. ~~[illegible]~~ North of Cyprus Minerals' Mineral Park Mine. Cyprus will take messages for TSC at 565-2226

TYPE OF OPERATION Leaching PRINCIPAL PRODUCT Copper precipitate

STARTING DATE Jan 30, 1988 CLOSING DATE 1994

DURATION OF OPERATION 6 years

PERSON SENDING THIS NOTICE Robert L Clayton

TITLE OF PERSON SENDING THIS NOTICE VP & Treasurer

DATE NOTICE SENT TO STATE MINE INSPECTOR 11/10/87

PLEASE NOTE: Any operation found operating, without having sent this notice to the Arizona State Mine Inspector, will be charged with a petty offense.

May 31, 1942

ERATING MINES
Mohave County

To: J. S. Coupal

From: Elgin B. Holt

EMERALD ISLE COPPER COMPANY: Located 15 miles north of Kingman, Mohave County. Ogden C. Chase, Pres., 15-16 Boggs Bldg., Las Vegas, Nevada.

PRODUCTION: None.

APPLICATION FOR RFC LOAN: While this property is not operating at present, company has applied for a $450,000 RFC loan, to be used as follows:

Company now has a 300-ton heap acid leaching plant partly completed. If RFC loan obtained, this plant is to be completed. Also this money will be used to drill the property, which now has 650,000 tons of surface ore partly exposed, assaying 2.5% copper. It is believed extensive drilling should uncover several million tons of the same tenor of ore, as ore bed dips to the S. W. Also new housing, mining equipment, new power units, acid plant, etc., to be provided, in case the loan should be obtained.

NOTE: In my opinion, this property warrants careful consideration by RFC.

ARIZONA DEPARTMENT OF ENVIRONMENTAL QUALITY

K

FIFE SYMINGTON, GOVERNOR
EDWARD Z. FOX, DIRECTOR

NOTICE OF PRELIMINARY DECISION TO MAKE A MAJOR MODIFICATION TO AN INDIVIDUAL AQUIFER PROTECTION PERMIT

RECEIVED
JUL 28 1992

Pursuant to Arizona Administrative Code, Title 18, Chapter 9, Article 1, the Director of the Arizona Department of Environmental Quality intends to make a major modification to an individual Aquifer Protection Permit to the following applicant (s):

Public Notice Number. 7-92AZAP
John D. Bracole
Emerald Isle Mine
Arimetco International, Inc.
6245 E. Broadway, Suite 350
Tucson, AZ 85710
Aquifer Protection Permit No. P-101846

On or about
July 24, 1992

Emerald Isle Mine is located four (4) miles south of Chloride, Arizona in Mohave County on the Old Chloride Road over groundwater of the Sacramento Valley Basin in Township 16 North, Range 19 West, Section 5, S 1/2, Section 8, and Section 17 N 1/2-Gila and Salt River Base Line and Meridian. Latitude 35° 21' 42" North and Longitude 114° 11' 28" West.

Presently, the facility is in operation pursuant to the terms and conditions established in the APP issued on September 13, 1991.

The modified permit incorporates major modifications proposed for the facility. These are:

1. Change the application rate of leach solution from 250 gpm to 320 gpm at the pit bottom.

2. Addition of two lined heap leaching pads and associated lined PLS ponds in order to apply leach solution at a maximum rate of 320 gpm. The heap leaching pads will be constructed on the impermeable bedrock overlaying by 12" thick compacted clay subgrade and 40 mil VLDPE synthetic liners. The solution storage pond and PLS ponds will also be constructed with 40 mil VLDPE synthetic liners placed over 12" thick compacted clay liners.

3. A fully contained solvent extraction and electro-winning (SX/EW) plant and associated tank farm to extract copper from the pregnant leach solution (PLS) extracted from on-site leaching operation. This plant is capable of handling PLS at a rate not exceeding 500 gpm.

4. The modified permit will restrict the total leach solution application to open pit and the heap pads to a maximum 500 gallons per minute so that the SX/EW will be able to handle the total PLS at the time of optimum production capacity.

5. The permit will restrict a maximum amount of copper recovery to 1.6 gpl as specified in TABLE I.A. of the permit. The presence of free acid in PLS pond will be restricted to 4.0 gpl.

6. The permit also includes additional groundwater monitoring using three downgradient, one upgradient, and two downgradient barrier wells. The new monitoring program as listed in TABLE I.B. and TABLES II.A. through E. requires monthly monitoring for extended list of parameters.

7. Finally, the modified permit reflects a minor modification, as to the format of the permit, to the current permit, which was issued on September 13, 1991. This minor modification has been proposed to maintain consistency with the current format of the permits.

The modified draft permit and related materials are available for public review Monday through Friday 8 a.m. to 5 p.m. at the Arizona Department of Environmental Quality, Plan Review and Permits Section, 3003 N. Central Avenue, 5th Floor, Phoenix, Arizona 85012.

Persons may submit comments or request a public hearing on the proposed action, in writing, to Syed Amanatullah, ADEQ at P.O. Box 600, Phoenix, AZ 85001-0600 within thirty (30) days from the date of this notice. Public hearing request must include the reason for such request.

SA:me

STATUS OF DORMANT MINES

MINE NAME: EMERALD ISLE MINE

LOCATION: Chloride, Arizona

OWNER AND/OR LEASEE: Cyrus F. Weeks and Anna I. Lauzon

ADDRESS: Box 227, Kingman, Arizona

APPROXIMATE PRODUCTION (Year of 1945):

COPPER 500,000 Lbs. LEAD ________ Lbs.

ZINC ________ Lbs. (OTHER) ________

CHECK THE CHIEF CAUSE OF YOUR DISCONTINUED PRODUCTION:

(A) Easily available ore worked out.
X---- (B) Increased costs, but have quantity similar to past grade of ore.
(C) Too close a margin to develop more ore.
(D) ________

If you have ore ready to mine please give your estimate of the amount of metal (name each metal) that you could produce in one year (after allowing 60 days to get started) if there were premiums above present market prices. Name amount with a low premium, and amount at a high premium; such as:

Copper at $22\frac{1}{2}$¢ plus 5¢ premium.............. 1,000,000 Lbs.
X---- Copper at $22\frac{1}{2}$¢ plus 10¢ premium............. 1,500,000 Lbs.

Additions to existing plant necessary. Would require 4 to 5 months to complete.

If you do not have ore ready to mine please discuss the following:

(A) Do you think a reasonable development program would produce a justified tonnage of commercial ore at above mine?

1,000,000 tons of $1\frac{1}{2}$% ore partially developed now. Additional ore could be developed by more drilling.

(B) With a premium price (guaranteed for one year) could you carry out such a development program yourself? What premium?

Premium price for one year inadequate. Would require guarantee of 4 to 5 years to amortize additional investment in plant.

(C) If you could not do this yourself, would a quick drilling program by some government agency (at government expense) be sufficient?

Additional drilling at this time not required.

Sufficient ore available to operate plant four years.

(D) Or would you prefer a loan plan similar to the arrangements during World War II?

A loan of approximately $450,000 would be required

to complete plant, and provide operating capital.

How about a combination plan in two stages such as follows?

Stage 1: Government engineers review project and, if a little drilling appears to be justified and a preliminary key to the situation, such drilling program to be agreed upon by owner and government engineer, paid for by the government, but let by contract.

Stage 2: If results of drilling (or without drilling) justify underground development and/or production equipment, same to be obtainable via a mortgage loan on property.

Please discuss the above: Parts of the present plant are large enough and in order for the production of 700 tons per day. Additions to the plant and equipment for mine will require the above amount of capital.

If 10¢ premium available for time required to pay off loan, then a smaller premium of, say, 3¢ to 5¢ would be sufficient to carry on operation. During period of high premium (10¢), additional drilling should be carried on to block out available ore on the property.

SUGGESTIONS:

If income-tax-free period of, say, 5 years, would be allowed, this capital investment of $450,000, could readily be arranged from private capital. However, with the present tax outlook, private capital is slow to invest in new businesses of any kind.

DATE August 5, 1950

SIGNATURE Cyrus F. Weeks

EMERALD ISLE MINE FILE COPY Copies Requested Bud & Tucson

RI 8236

Bureau of Mines Report of Investigations/1977

In Situ Leaching Research in a Copper Deposit at the Emerald Isle Mine

UNITED STATES DEPARTMENT OF THE INTERIOR

Report of Investigations 8236

In Situ Leaching Research in a Copper Deposit at the Emerald Isle Mine

By Dennis V. D'Andrea, William C. Larson, Larry R. Fletcher, Peter G. Chamberlain, and William H. Engelmann

UNITED STATES DEPARTMENT OF THE INTERIOR
Cecil D. Andrus, Secretary

BUREAU OF MINES

This publication has been cataloged as follows:

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Bibliography: p. 42-43.

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ILLUSTRATIONS

ILLUSTRATIONS--Continued

TABLES

IN SITU LEACHING RESEARCH IN A COPPER DEPOSIT AT THE EMERALD ISLE MINE

by

Dennis V. D'Andrea,[1] William C. Larson,[2] Larry R. Fletcher,[3]
Peter G. Chamberlain,[1] and William H. Engelmann[4]

ABSTRACT

The Bureau of Mines and El Paso Mining and Milling Co. conducted a cooperative in situ leaching research program at the Emerald Isle mine near Kingman, Ariz. The objective of this research was to develop in situ leaching methods for 200,000 tons of ore exposed in the pit bottom and also 3,000,000 tons of ore under 200 feet of overburden adjacent to the pit. The research included core drilling for fragmentation analysis, inplace permeability testing, seismic surveys, blasting tests, blast vibration measurements, inplace leaching, and ground water monitoring.

A 15,000-ton test area in the pit bottom was blasted and leached inplace for 117 days. This successful test was followed by leaching 100,000 additional tons of unblasted pit bottom ore. Two test blasts, under 205 feet of overburden and extending to 290 feet, were detonated in an area near the open pit. The second blast was detonated because the first blast did not create adequate permeability for leaching. Pit bottom leaching was discontinued and the test area under 200 feet of overburden was not leached because the company terminated operations at the Emerald Isle mine. Based on the research completed, an expanded in situ leaching system was designed, but not implemented, to recover copper from a 1,500,000-ton area at the Emerald Isle mine.

INTRODUCTION

The Bureau of Mines is conducting research to develop technology for in situ copper leaching (3-6, 11-12).[5] Although in situ leaching is generally considered applicable to deposits that are too low in grade to be mined by conventional methods, it can also be applied to higher grade deposits as an

[1]Geophysicist.
[2]Geologist.
[3]Engineering technician.
[4]Research chemist.
All authors are with the Twin Cities Mining Research Center, Bureau of Mines, Twin Cities, Minn.

[5]Underlined numbers in parentheses refer to items in the list of references at the end of this report.

alternative to conventional mining. The advantages of in situ leaching are lower capital investment, shorter development time, reduced health and safety hazards, minimal detrimental environmental effects, and lower energy consumption during mining and beneficiation.

In situ copper leaching has been practiced at Miami, Ariz. (8); the Old Reliable mine near Mammoth, Ariz. (7); the Zonia mine near Kirkland Junction, Ariz.; the Big Mike mine near Winnemucca, Nev. (10); and the Mt. City mine in Mt. City, Nev. (2). Although initial results from these operations have been encouraging, the depressed price of copper has resulted in temporary or permanent closing of all of these mines. However, an improved copper market will encourage companies to resume some of these operations.

The major problems with in situ leaching are (1) inadequate reliable methods for evaluation of potential in situ leaching properties including estimating chemical reagent consumption costs and final copper recoveries, (2) insufficient fragmentation systems which can produce adequate permeability at reasonable costs, (3) inadequate leach solution injection and recovery systems that assure continued contact between leach solutions and broken ore, (4) final copper recoveries must be maximized, and (5) ground water contamination must be prevented. Although the basic technology exists to conduct in situ leaching, this technology must be improved and presently available equipment must be modified (9) to increase efficiency and economy.

This report presents the results of a Bureau of Mines-El Paso Mining and Milling Co. cooperative in situ leaching research program at the Emerald Isle mine near Kingman, Ariz. Both the Twin Cities (Minn.) Mining Research Center and the Salt Lake City (Utah) Metallurgy Research Center of the Bureau of Mines participated in this program. Research was conducted in two phases. In the phase I area ore was exposed on the surface of the pit bottom, and in the phase II area the ore was buried under 205 feet of overburden.

ACKNOWLEDGMENTS

The authors acknowledge the cooperation and assistance of El Paso Mining and Milling Co., El Paso, Tex., and particularly its employees--Sidney Runke, chief metallurgist; Harold Horst, mining engineer; and Lester Mead, mine superintendent. The cooperation and assistance of George M. Potter, metallurgist, and Rees D. Groves, metallurgist, Federal Bureau of Mines, Salt Lake City (Utah) Metallurgy Research Center, is also greatly appreciated.

EMERALD ISLE DEPOSIT

Geology

The cross section through the Emerald Isle deposit, figure 1, shows vertical variations in copper grade in the mineralized zone, the phase I test area in the pit bottom, and the phase II test area under 205 feet of overburden. The deepest portions of the open pit have been excavated to about 200 feet exposing the top of the ore (locally known as the Gila Conglomerate). During open pit operations El Paso mined 1,400,000 tons of ore that averaged

1.0 percent copper. Conventional open pit mining stopped because of increased stripping costs as the operation proceeded in the downdip direction.

Copper mineralization occurs in the Gila Conglomerate which averages about 70 feet thick and dips 10° to 15° to the southwest. A plan view of the mine, figure 2, shows contours of the grade in the Gila Conglomerate and again shows the phase I and phase II test areas. The copper mineralization

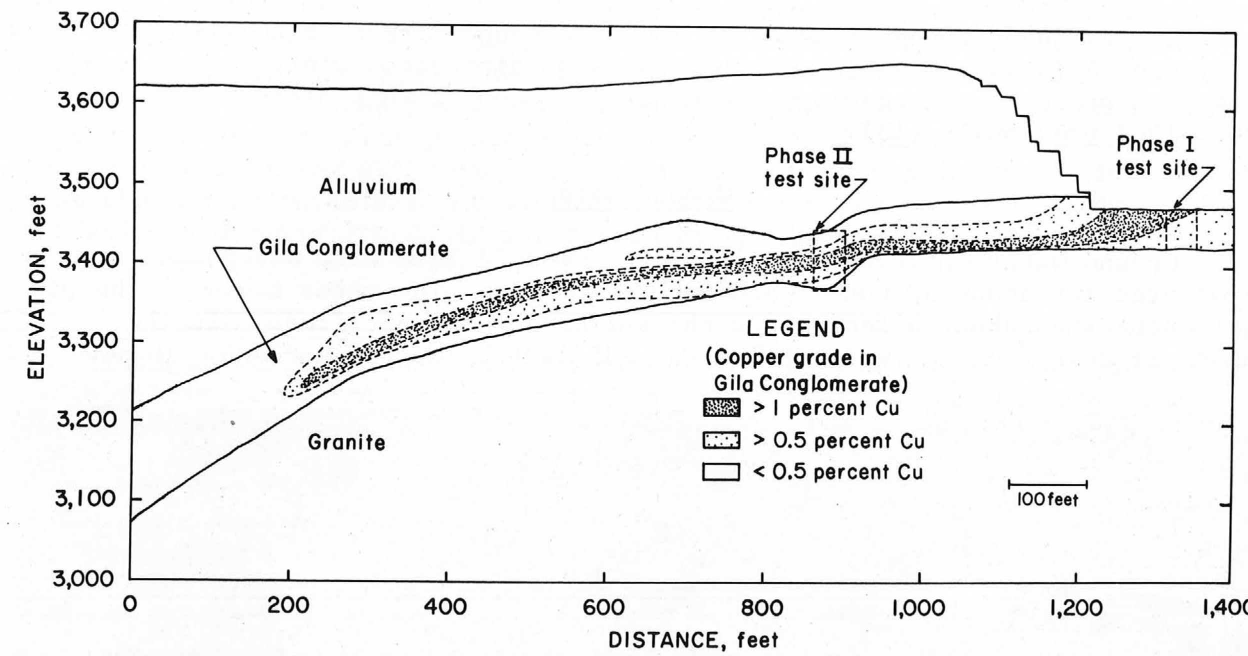

FIGURE 1. - Cross section through the Emerald Isle mine showing copper grade and phases I and II test sites.

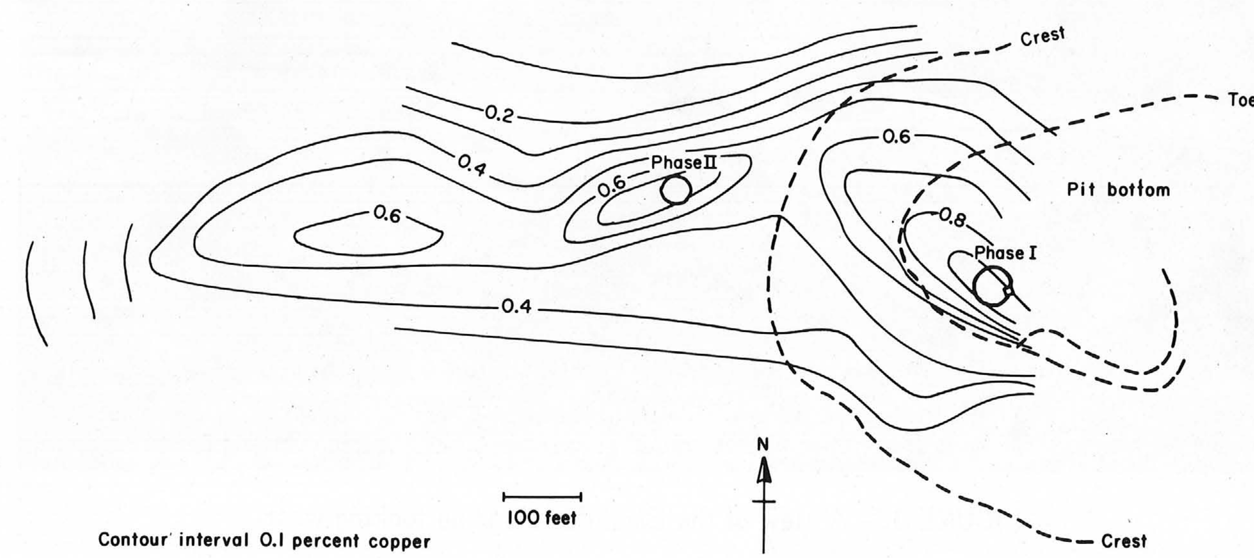

FIGURE 2. - Plan view of the Emerald Isle mine showing copper grade and phases I and II test sites.

continues beyond the pit in the downdip direction in a channel-type deposit that generally decreases in grade. At 1,000 feet west of the pit crest the grade of the ore is about 0.1 percent. Total copper resources are estimated to be about 3,000,000 tons of ore greater than 0.1 percent and 1,500,000 tons of ore at a grade above 0.4 percent. Figure 3 is a view of the pit and surface facilities at the Emerald Isle mine looking west.

Mineralization

The dominant copper mineral in the Gila Conglomerate at the mine is chrysocolla ($CuSiO_3$ 2 H_2O). Minor amounts of dioptase, tenorite, and cuprite are also present. The geology and mineralization of this deposit have been described previously (13).

Ground Water

Ground water entering the pit at the Emerald Isle mine was pumped from a sump area for about 30 hours each week at 90 gpm. The water table in the pit was maintained about 5 feet below the surface of the pit floor. In the phase II test area the water table was 235 feet below the surface. Ground

FIGURE 3. - A view of the Emerald Isle mine looking west.

water table elevations decreased in the downdip direction tending to follow the granite gneiss bedrock in the area around the open pit.

Figure 4 shows the water table elevations in the pit before leaching. Figure 5 shows these elevations during the phase I leach test when the water table was drawn down about 22 feet at the recovery well, which created a cone of depression.

An attempt was made to establish the pattern of ground water movement in the pit bottom using a red dye (Rhodamine B). The dye was injected into a drill hole in the middle of the phase I test area and water samples were taken from monitor holes around the test area. This test was unsuccessful because none of the dye was ever detected at any of the monitor holes. Another test conducted before and after blasting showed much more rapid dissipation of dye from the injection hole after blasting. The major problem with using Rhodamine B dye was its rapid absorption by the host rock.

Rock Properties

Table 1 lists the physical properties of the Gila Conglomerate at the Emerald Isle mine from phases I and II areas as measured in the laboratory using core samples. The porosity, permeability, and strength values probably reflect the characteristics of the cementing material in the conglomerate and not the actual in situ values.

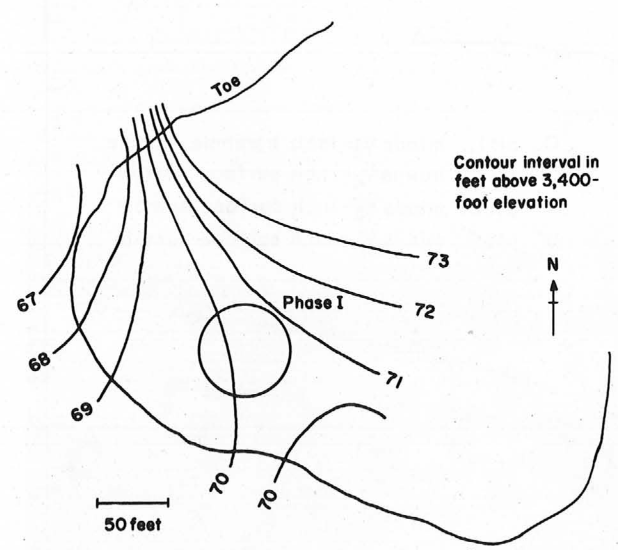

FIGURE 4. - Ground water elevations before the phase I leach test.

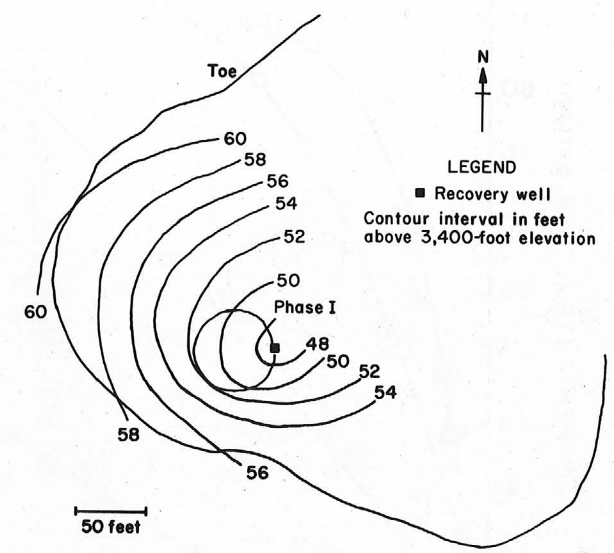

FIGURE 5. - Ground water elevations during the phase I leach test.

TABLE 1. - Physical properties of the Gila Conglomerate at the Emerald Isle Mine

Property	Phase I	Phase II
Porosity.................................percent..	20.6	16.3
Density..................................g/cu cm..	2.29	2.28
Permeability.............................darcies..	-	.65
Longitudinal velocity......................ft/sec..	8,100	9,500
Torsional velocity.........................ft/sec..	4,200	5,100
Compressive strength.....................lb/sq in..	2,600	-
Tensile strength.........................lb/sq in..	72	-
Young's modulus.................10^6 lb/sq in..	0.85	-

Laboratory Leaching Tests

Figure 6 shows the results of laboratory acid trickle leach tests on minus 0.5-inch ore from the phase I test area. These tests, conducted at the Salt Lake City (Utah) Metallurgy Research Center, show the effect of pH on the rate of copper recovery indicating that a pH of 1.0 would be desirable. During these tests acid consumption averaged about 6 pounds of H_2SO_4 per pound of copper, and iron consumption averaged 1.3 pounds of iron per pound of copper.

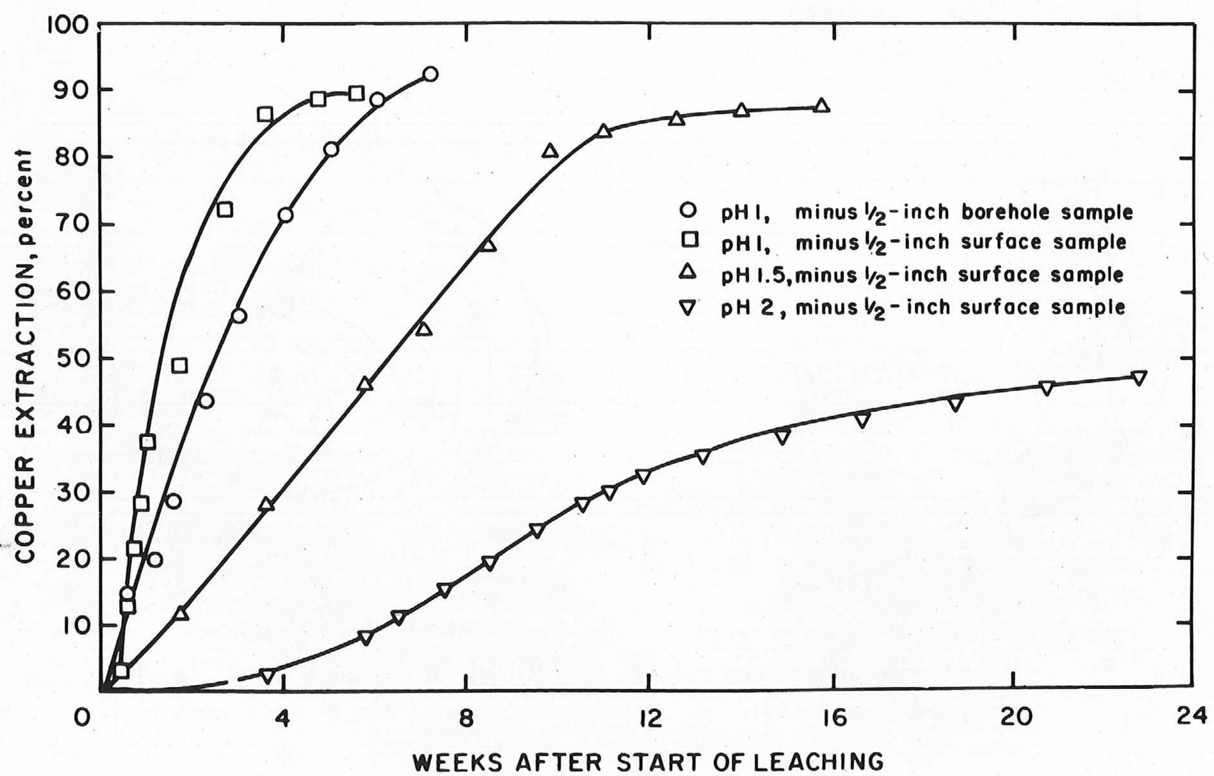

FIGURE 6. - Acid trickle leach test on ore from the Emerald Isle mine.

DRILLING PROCEDURES

Core Holes

Core holes were drilled before and after blasting in both test areas. These holes, drilled with a double-barrel wire line system using drill mud, produced core about 2 inches in diameter. In the phase II area the top 200 feet of overburden were drilled with a rotary system and cased before core drilling began in the ore zone below 200 feet (fig. 7). Drilling into ore that had been broken by blasting did not present any unusual problems and was done with return circulation of drill muds.

Blastholes

All blastholes were drilled with a rotary system using tricone bits and circulating drill mud. Blastholes in the phase I area were 8-3/8 inches wide and about 50 feet deep. In the phase II area blastholes were 9 inches wide and about 280 feet deep.

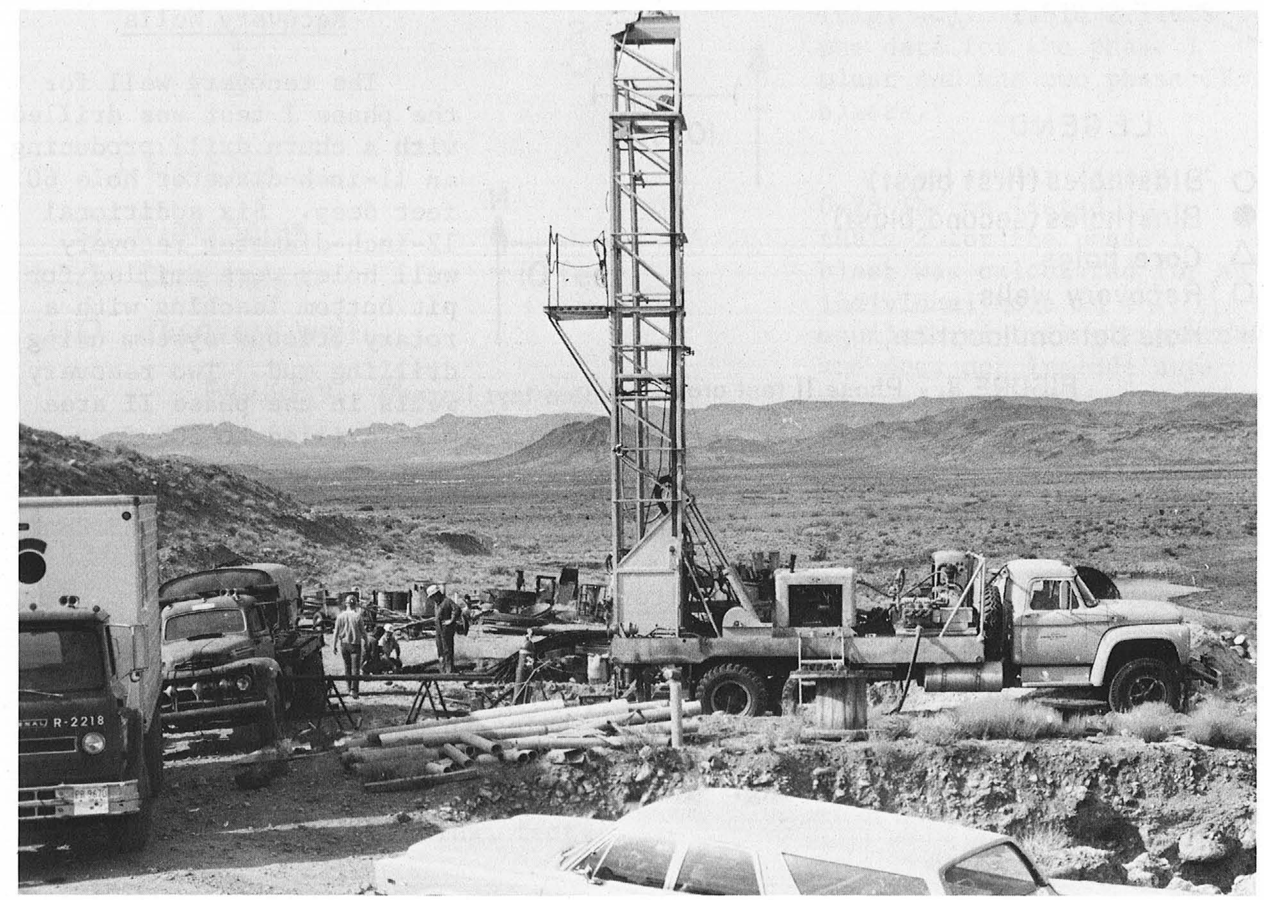

FIGURE 7. - Core drilling in the phase II area.

Borehole deviation surveys were run on the three 9-inch-diameter holes for the second phase II blast and one of the 12-inch-diameter recovery wells in the phase II area. The horizontal borehole deviations are shown in figure 8. The maximum borehole deviation was 14 feet for the 280-foot-deep blastholes and 12 feet for the 300-foot recovery well. These borehole surveys point out the need for improved drilling methods that will minimize borehole deviations.

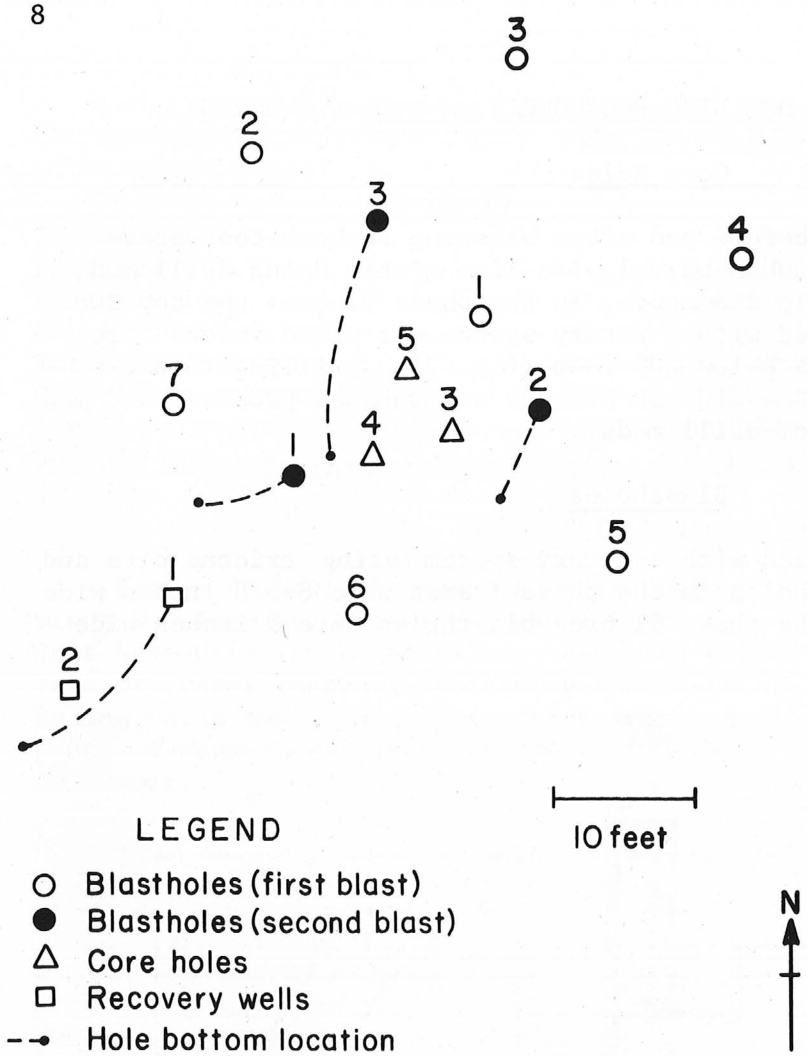

FIGURE 8. - Phase II test area.

Recovery Wells

The recovery well for the phase I test was drilled with a churn drill producing an 11-inch-diameter hole 60 feet deep. Six additional 12-inch-diameter recovery well holes were drilled for pit bottom leaching with a rotary tricone system using drilling mud. Two recovery wells in the phase II area were drilled to 300-foot depths with 12-inch-diameter tricone bits and circulating drill mud. All recovery holes were cased with 10-inch diameter polyvinyl chloride (PVC) casing with the bottom 20 feet perforated.

Monitor Holes

Four-inch-diameter monitor holes in the pit bottom were drilled about 50 feet deep with an airtrack drill. These holes were not cased and remained open after the phase I test blast. Monitor holes in the phase II area were drilled with a rotary tricone system to depths of 320 to 360 feet and cased with NX (3-inch-diameter) casing. The bottom 60 feet of this casing was perforated.

BLASTING PROCEDURES

Phase I Blast

Figure 9 shows the phase I test blast design. Seven 8-3/8-inch-diameter blastholes, about 50 feet deep, were spaced 25 feet apart in a seven-spot pattern with one central blasthole. These holes had an average 22-foot powder column with 25 feet of stemming. A total of 4,500 pounds of slurry was detonated without delays. The slurry was bulk loaded using a slurry pump truck (fig. 10). Table 2 lists the data for the phase I blast and the two phase II blasts.

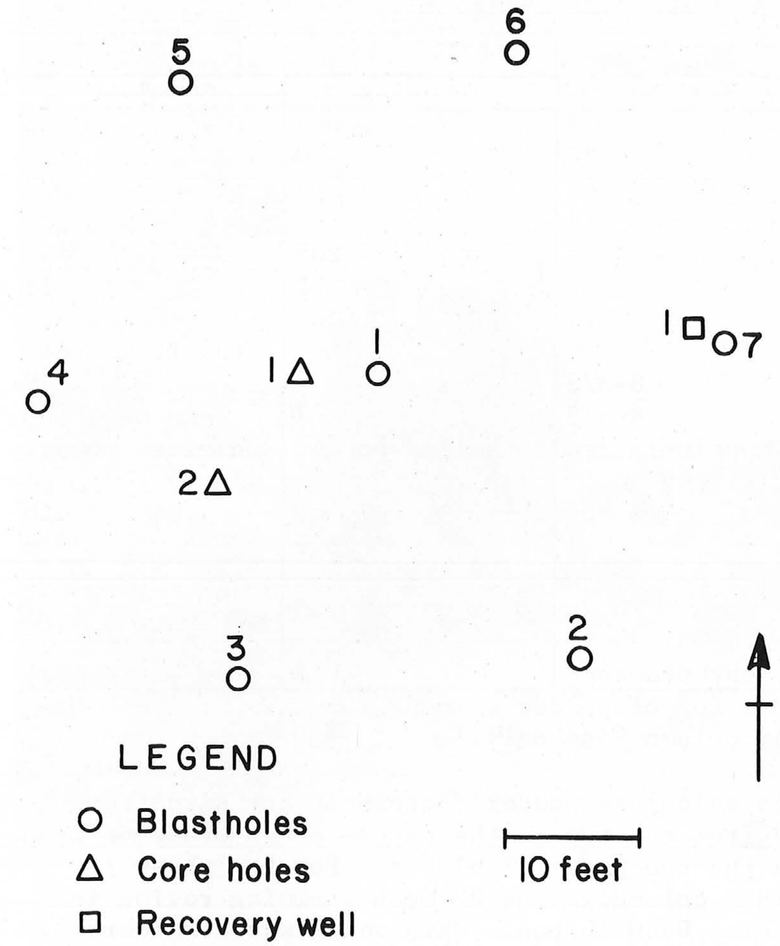

FIGURE 9. - Phase I test area.

The powder factor of 0.78 lb/ton listed in table 2 for the phase I blast was calculated for an individual hole in an equilateral triangle pattern and does not include any volume of ore above or below the powder column. For this situation the powder factor (PF) in pounds per ton is given by

$$PF = 1{,}814 \frac{Pe}{Pr}\left(\frac{D}{S}\right)^2,$$

where Pe = specific gravity of explosive,

Pr = specific gravity of ore,

D = blasthole diameter, feet,

and S = blasthole spacing, feet.

TABLE 2. - Test blast design data

	Phase I	Phase II	
		1st blast	2d blast
Number of blastholes.........	7	7	3
Blasthole spacing.......feet..	25	20	18
Average hole depth.......do...	47	277	277
Average top powder column..................do...	25	205	192
Average powder column....do...	22	72	85
Average stemming.........do...	25	205	192 (90 ft gravel)
Blasthole diameter....inches..	8-3/8	9	9
Explosive diameter......do....	8-3/8	7	7-in bags cut
Explosive....................	Nonaluminized slurry	Smokeless powder slurry	Smokeless powder slurry
Total explosives......pounds..	4,500	12,000	7,450
Loading density........lb/ft..	29.2	23.7	29.3
Powder factor[1]........lb/ton..	0.30	-	-
Powder factor[2]........lb/ton..	0.78	0.95	1.47
Delays between each hole...........milliseconds..	Instantaneous	17	25

[1] Powder factor includes ore above top of powder column.
[2] Powder factor for ore in powder column zone only.

This formula can be used to calculate powder factors in any situation where the powder column extends from the top to the bottom of an ore zone buried under overburden as with the two phase II blasts. For the phase I blast, if the ore above the powder column in the 22-foot stemming region is included, the powder factor becomes 0.30 lb/ton. This value was considerably lower than the powder factors of from 0.67 to 0.89 lb/ton used at the Old Reliable, Zonia, and Big Mike blasts (11).

Phase II Blasts

Two blasts were detonated in the phase II test area under 200 feet of overburden. The first blast had seven 9-inch-diameter blastholes drilled to an average depth of 277 feet. Blastholes were spaced 20 feet apart, again in a seven-spot pattern, with one central blasthole. Figure 8 shows the blasthole patterns for the phase II blasts. The second blast had three blastholes spaced 18 feet apart. Blast design data for the phase II blasts are listed in table 2. Powder factors were 0.95 lb/ton and 1.47 lb/ton for the first and second blasts, respectively.

FIGURE 10. - Phase I blasthole loading.

The first blast was loaded with 7-inch-diameter, 50-pound bags, of smokeless powder slurry in 9-inch-diameter blastholes. The second blast was loaded with the same explosive but the bags were cut before lowering them again into 9-inch holes as shown in figure 11. The cutting procedure, where several 3-inch slits were cut in each bag, increased the loading density from 23.7 lb/ft for the first blast to 29.3 lb/ft for the second blast. The 50-pound bags of slurry were lowered to the top of the water column in each blasthole on a lowering rope with a release hook. At the top of the water column the bags were released and allowed to freefall through the water to the bottom of the blastholes. This blasthole loading procedure was time-consuming but all blastholes were successfully loaded.

FIGURE 11. - Cutting 50-pound bags of slurry before lowering into blastholes.

Blast Swell

Detailed topographic surveys were run to determine elevation changes produced by the blasts. Figure 12 is a contour map of elevation increases produced by the phase I blast. The maximum surface rise for this blast was only about 1.4 feet. A swell factor, defined as the final volume divided by the original volume, is difficult to determine because the radius of blast damage needed for the volume calculation was not accurately known. With the assumption that the original volume was contained in a cylinder 47 feet deep and 75 feet wide, the swell factor for the phase I blast was 1.014 with a volume increase of 110 yd^3.

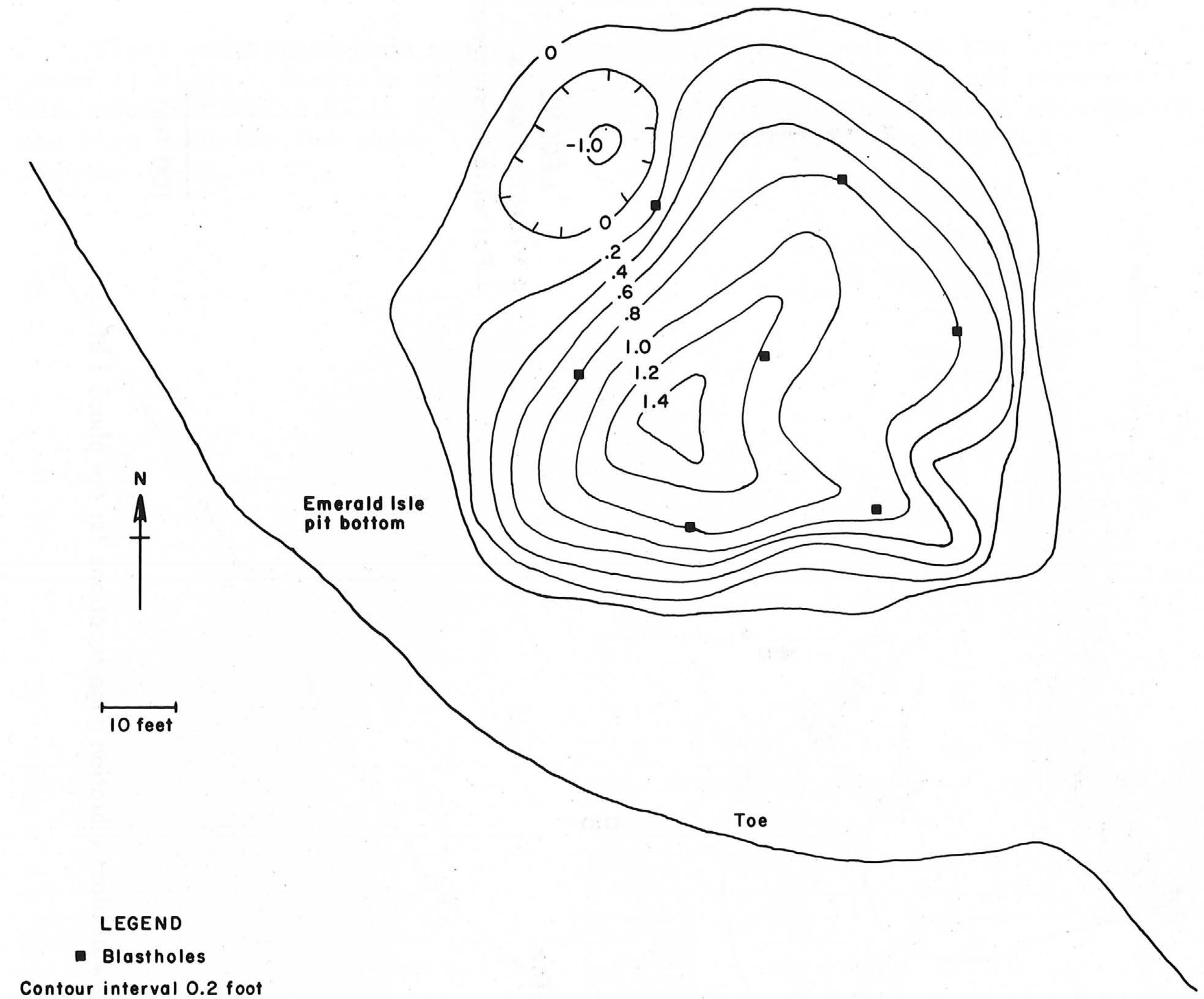

FIGURE 12. - Elevation changes produced by the phase I blast.

Elevation increases produced by the phase II blasts were generally less than 0.2 foot, which was about the estimated accuracy of the topographic surveys. Elevation increases of about 0.5 foot were observed near the collars of the blastholes that vented but these increases were considered to be a local effect.

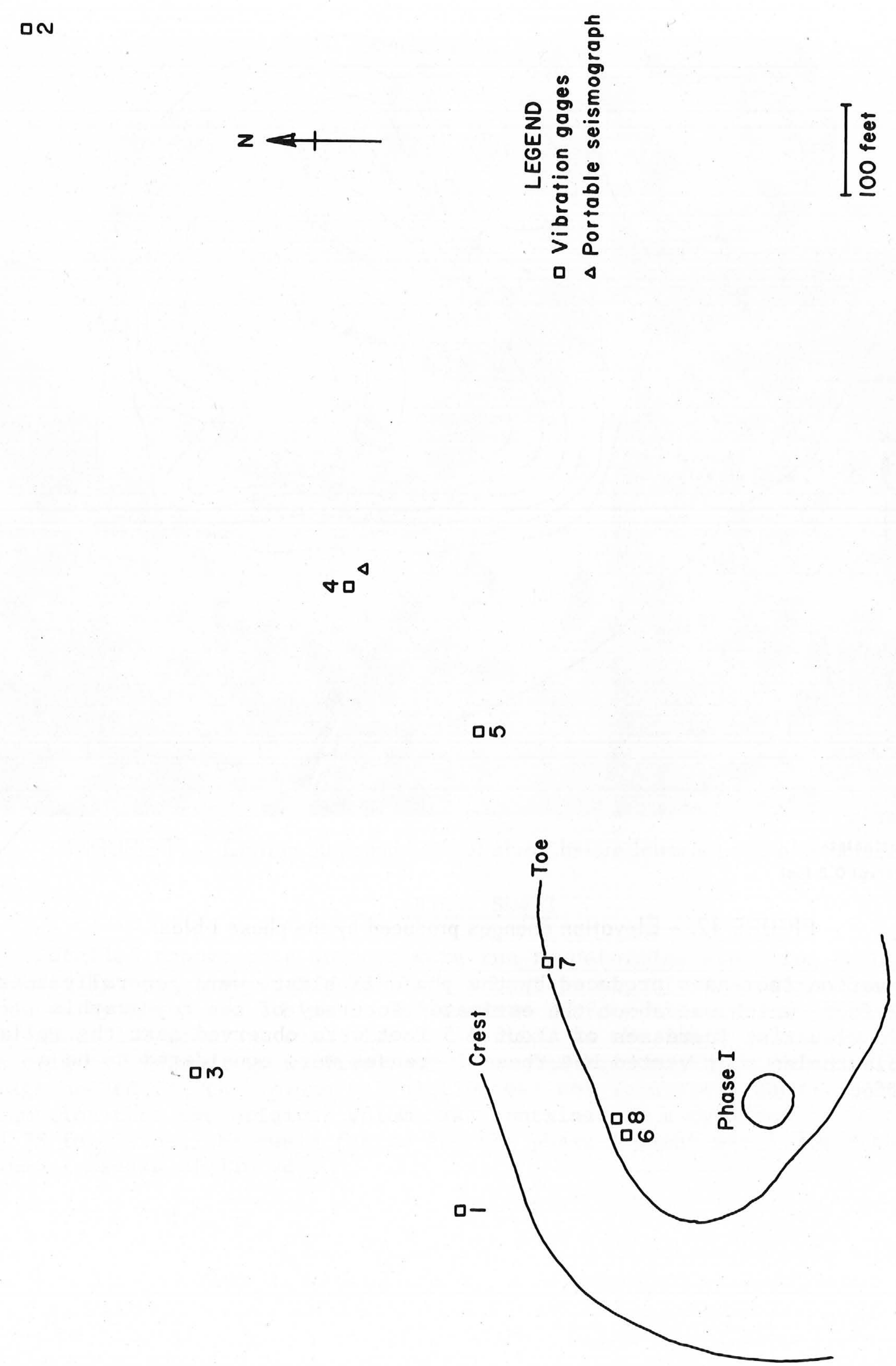

FIGURE 13. - Plan view of blast vibration gage locations for the phase I blast.

Blast Vibrations

Blast vibrations were measured for the phase I blast and the first phase II blast. Particle velocity gages and a 14-channel FM tape recorder were used to record blast vibrations. A portable three-component seismograph was also used for the phase I blast. The gage locations are shown in figures 13-15.

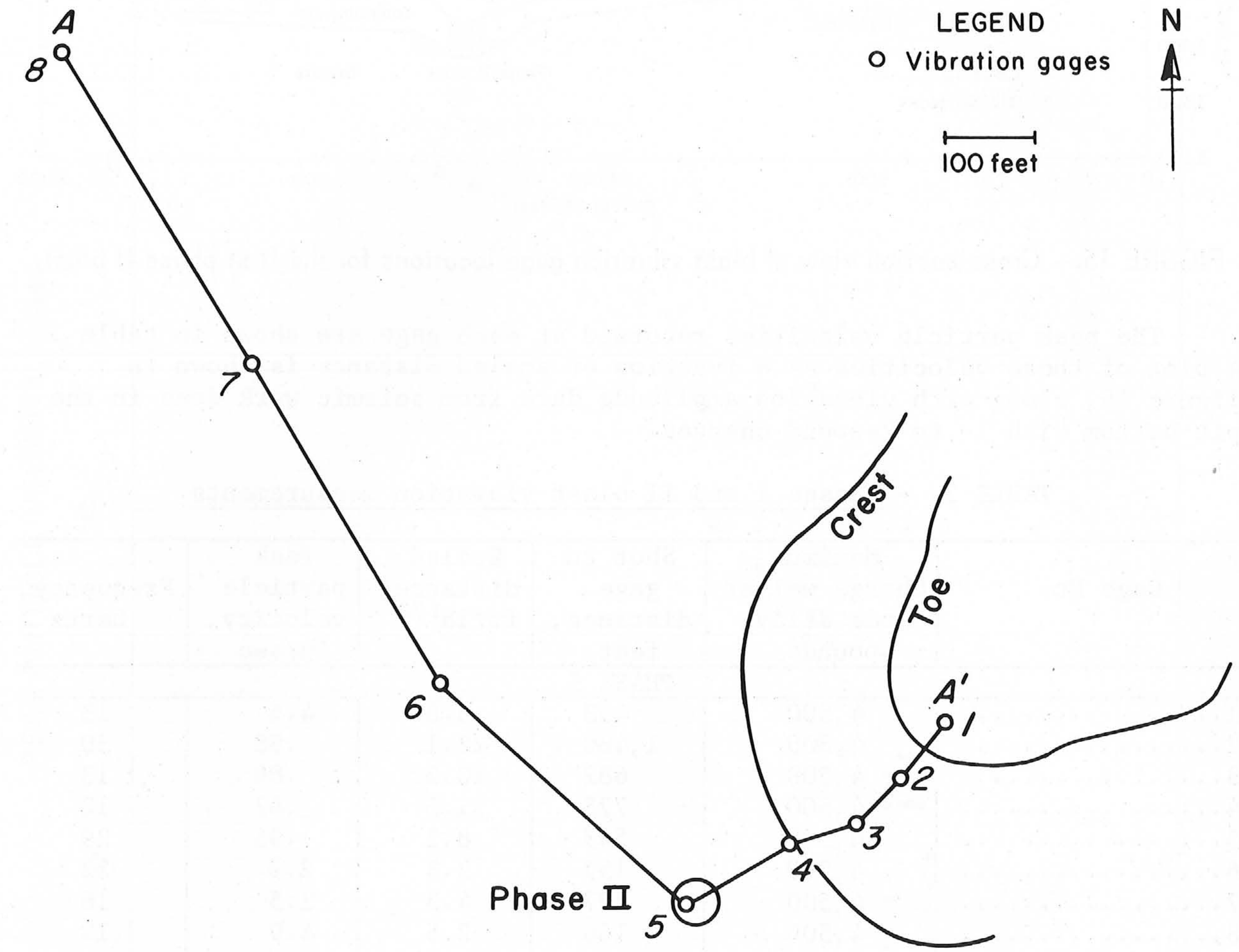

FIGURE 14. - Plan view of blast vibration gage locations for the first phase II blast.

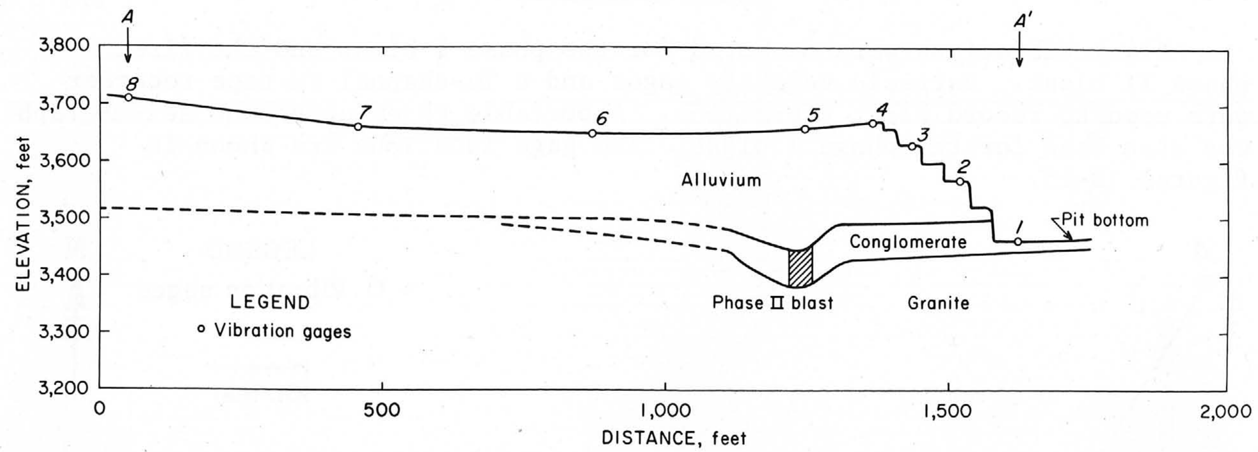

FIGURE 15. - Cross section view of blast vibration gage locations for the first phase II blast.

The peak particle velocities recorded at each gage are shown in table 3. A plot of these velocities as a function of scaled distance is shown in figure 16, along with vibration amplitude data from seismic work done in the pit bottom with 1- to 2-pound charges.

TABLE 3. - Phases I and II blast vibration measurements

Gage No.	Maximum charge weight per delay, pounds	Shot to gage distance, feet	Scaled distance, $ft/lb^{1/2}$	Peak particle velocity, in/sec	Frequency, hertz
		PHASE I			
1................	4,500	453	6.8	4.4	13
2................	4,500	1,480	22.1	.58	30
3................	4,500	682	10.2	.89	13
4................	4,500	773	11.5	.82	12
5................	4,500	547	8.2	.95	29
6................	4,500	157	2.3	2.9	22
7................	4,500	287	4.3	2.5	16
8................	4,500	166	2.5	4.9	19
Transverse........	4,500	733	10.9	1.65	9
Vertical..........	4,500	733	10.9	.60	11
Longitudinal......	4,500	733	10.9	.65	16
		PHASE II			
1................	1,858	376	8.7	1.7	67
2................	1,858	289	6.7	[1]2.2	-
3................	1,858	255	5.9	[1]3.4	28
4................	1,858	246	5.8	3.4	-
5................	1,858	200	4.6	[1]7.4	-
6................	1,858	421	9.8	[1]2.3	-
7................	1,858	809	18.8	.62	67
8................	1,858	1,223	28.8	.21	17

[1]Estimated from records that went off-scale.

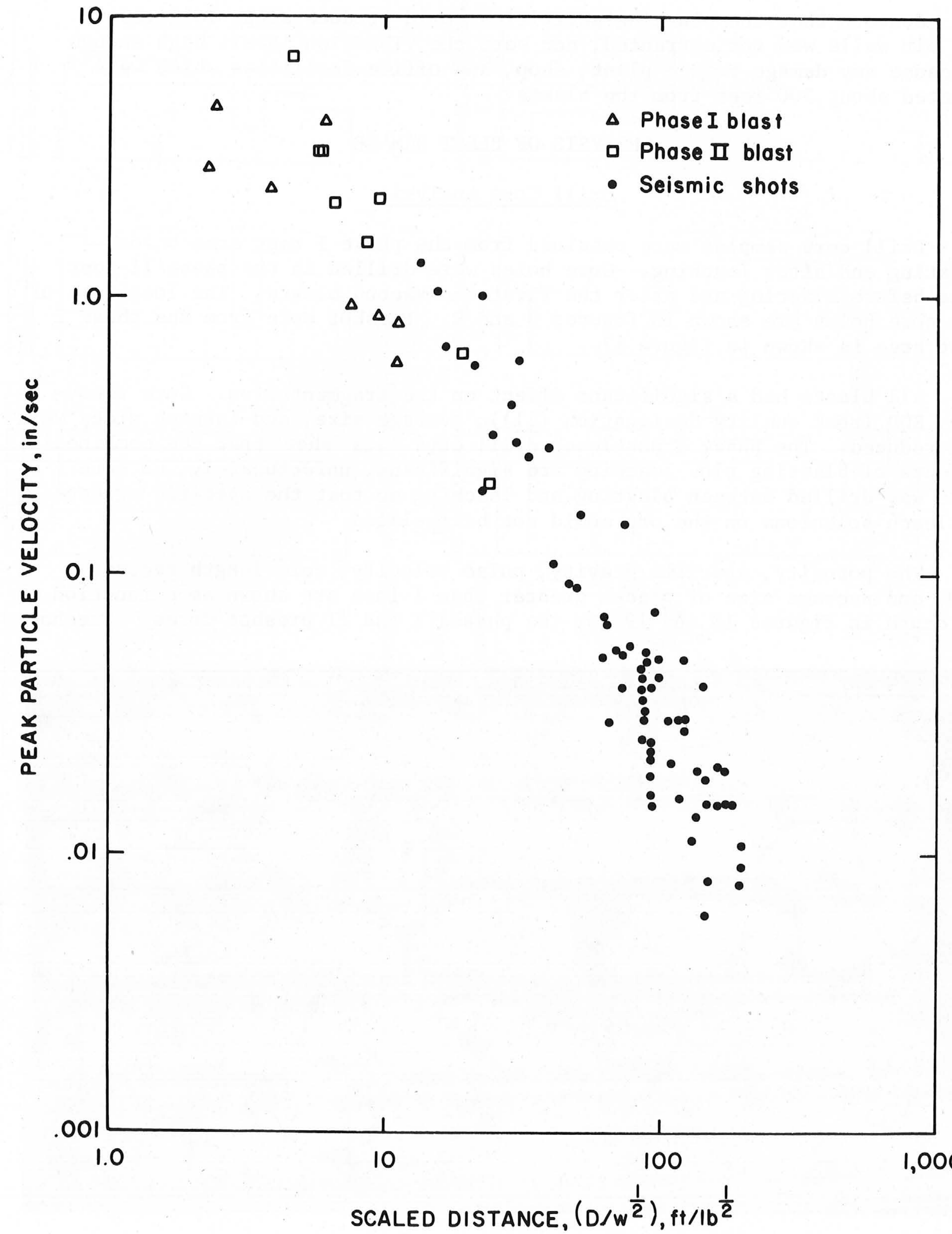

FIGURE 16. - Peak particle velocity versus scaled distance.

Concern that the blasting activities would produce slope failures in the pit walls was not warranted, nor were the vibration levels high enough to cause any damage to the plant, shop, and office facilities which were located about 300 feet from the blasts.

ANALYSIS OF BLAST DAMAGE

Drill Core Analysis

Drill core samples were obtained from the phase I test area before blasting and after leaching. Core holes were drilled in the phase II test area before blasting and after the first and second blasts. The locations of the core holes are shown in figures 8 and 9. Preshot core from the phase I test area is shown in figure 17.

All blasts had a significant effect on the fragmentation. Core recovery, RQD (rock quality designation (11)), average size, and largest piece were all reduced. The phase I postleach drill core data show that the combined effects of blasting plus leaching are significant, unfortunately, no core hole was drilled between blasting and leaching so that the specific effects of leach solutions on the ore could not be isolated.

The porosity, specific gravity, pulse velocity, core length recovery, RQD, and average size of pieces greater than 1 inch are shown as a function of depth in figures 18 and 19 for the phases I and II preshot cores. Preshot

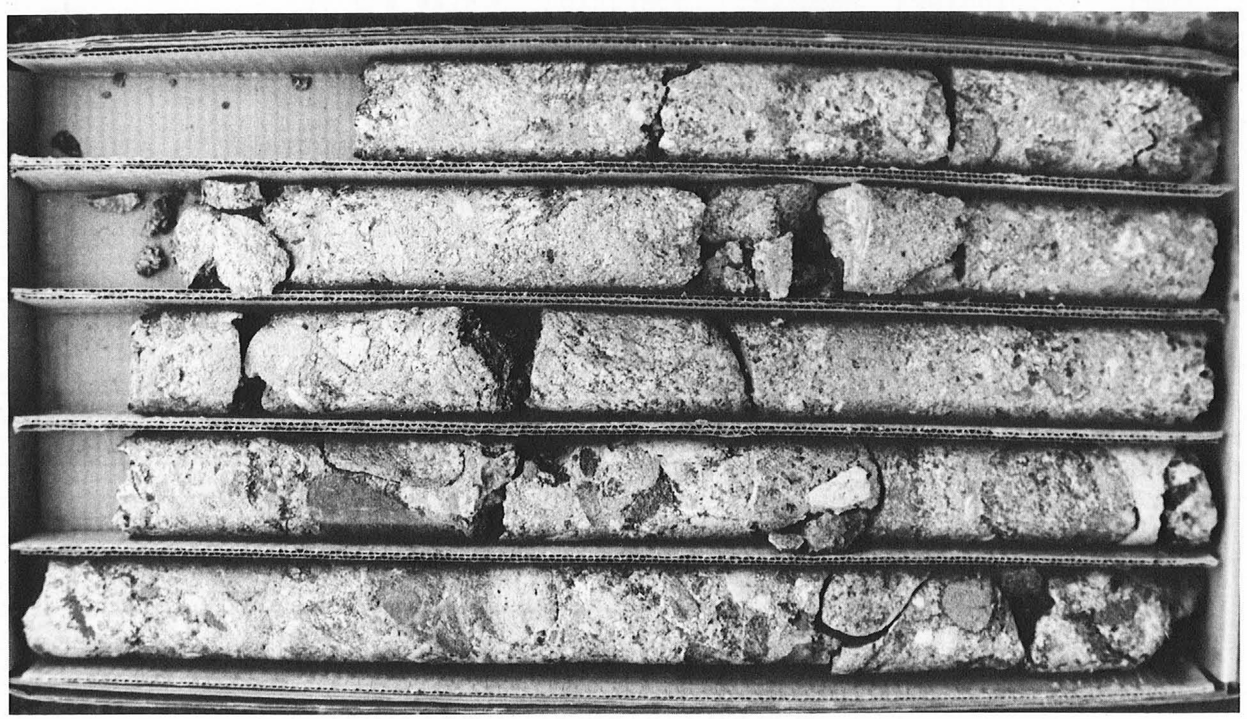

FIGURE 17. - Phase I preshot core.

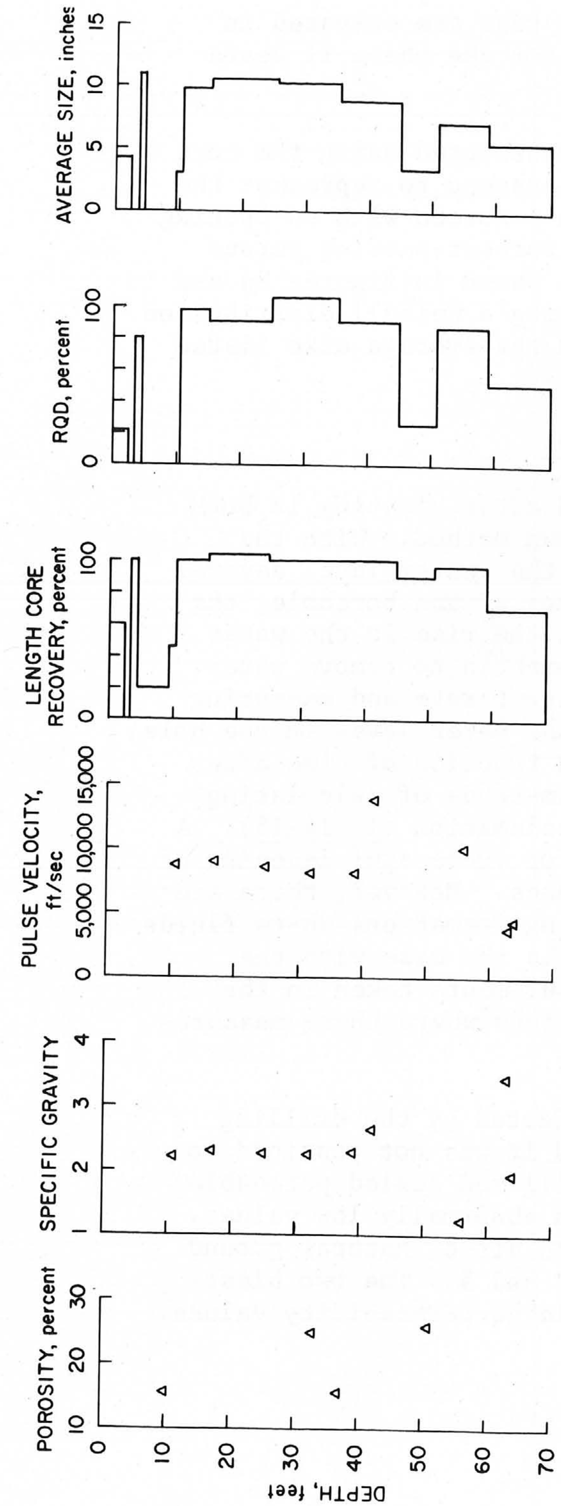

FIGURE 18. - Characteristics of phase I preshot core.

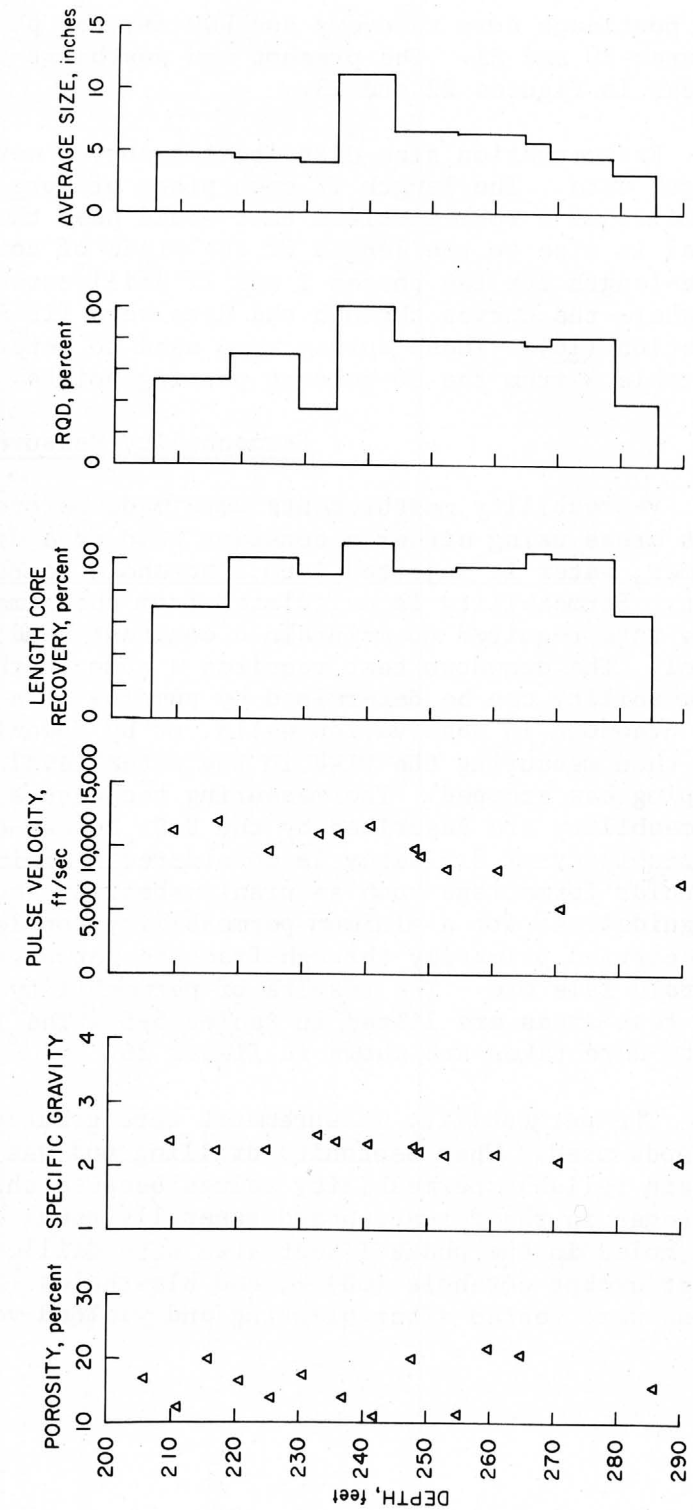

FIGURE 19. - Characteristics of phase II preshot core.

and postleach core recovery and RQD for the phase I test are compared in figures 20 and 21. The preshot and postblast data for the phase II tests appear in figures 22 and 23.

Fragmentation size distribution curves were constructed using the core length data. The length of each piece of core was assumed to represent the diameter of a rock particle that would pass through a screen with an opening equal in size to the length of the piece of core. Percent-passing versus core-length for the phases I and II drill cores are shown in figures 24 and 25 where the curves through the data were fit assuming a Weibull distribution function (16). These curves were used to determine the average size listed in table 4 from the 50-percent passing points.

Permeability Measurements

Permeability measurements were made before and after blasting in both test areas using either a constant head or a drawdown method. With the former, water is injected into a borehole to raise the water level several feet. Permeability is calculated from the dimensions of the borehole, the flow rate required to maintain a constant head, and the rise in the water level. The drawdown test requires a pump in the borehole to remove water. Permeability can be determined by pumping at a constant rate and measuring the drawdown in observation wells, or by lowering the water level in the hole, and then measuring the rise in the water level as a function of time after pumping has stopped. The measuring techniques and methods of calculating permeability are described by the U.S. Bureau of Reclamation (1, 14-15). A permeability of 0.5 darcy is considered a minimum for successful leaching of granular formations such as uranium-bearing sandstones. However, there are no guidelines for a minimum permeability for leaching formations where fluids are carried primarily through fracture networks as is the case with the Emerald Isle ore. The results of permeability measurements taken in the two test areas are listed in tables 5-6. The locations where these measurements were taken are shown in figure 26.

The permeability measurements were greatly effected by the drilling methods used. When bentonite drilling mud was used it was not possible to obtain reliable permeability values because the drill mud sealed permeable passages in the formation and generally resulted in abnormally low values. All holes in the phase I test area were drilled with air or natural ground water except corehole (CH) 1, and blastholes (BH) 2 and 5. The two blastholes were tested after blasting and yielded very large permeability values.

TABLE 4. - Drill core data

	Phase I		Phase II		
	Preshot	Postblast-leach	Preshot	Postshot	
				1st blast	2d blast
Core run..............feet..	0 to 68.6	18.0 to 72.0	206.0 to 278.5	213.0 to 256.7	200.0 to 277.5
Core recovery.......percent..	86	35	93	83	67
RQD....................do....	65	11	70	40	31
Average size (>1 inch) inches..	6.7	2.6	5.1	3.0	3.7
Average size (50 percent by weight)............inches..	6.4	<1	6.4	3.1	1.5
Largest piece..........do....	26	9	33	20	16

TABLE 5. - Permeability measurements for phase I tests

Location	Description	Preshot permeability, darcies	Postshot permeability, darcies
CH 1..................	Core hole.............	2	-
MW 1..................	Monitor well..........	-	5
MW 3..................	.....do...............	.09	27
MW 4..................	.....do...............	.02	.3
MW 5..................	.....do...............	.2	.3
MW 6..................	.....do...............	-	.2
MW 7..................	.....do...............	-	.1
BH 2..................	Blasthole.............	-	200
BH 5..................	.....do...............	-	800
RW 1..................	Recovery well.........	-	50

TABLE 6. - Permeability measurements for phase II tests

Location	Description	Preshot permeability, darcies	Postshot 1 permeability, darcies	Postshot 2 permeability, darcies
CH 3..........	Core hole.......	<0.02	-	-
CH 4..........	.....do........	-	0.0003	-
RW 1..........	Recovery well...	-	.002	-
RW 2..........	.....do........	-	.0002	-
MH 1..........	Monitor hole....	-	.0004	-
MH 2..........	.....do........	-	6.2	-
BH 1..........	Blasthole.......	-	-	0.006
CH 5..........	Core hole.......	-	-	.020

Holes in the phase II area were all drilled with mud and the permeability values were too low for leaching. It is difficult to say whether the low permeability values are due to insufficient blast induced fracturing, overburden stresses resealing any fractures created, or drill mud clogging permeability channels. There is little doubt that the drill mud has some effect in producing low permeability values.

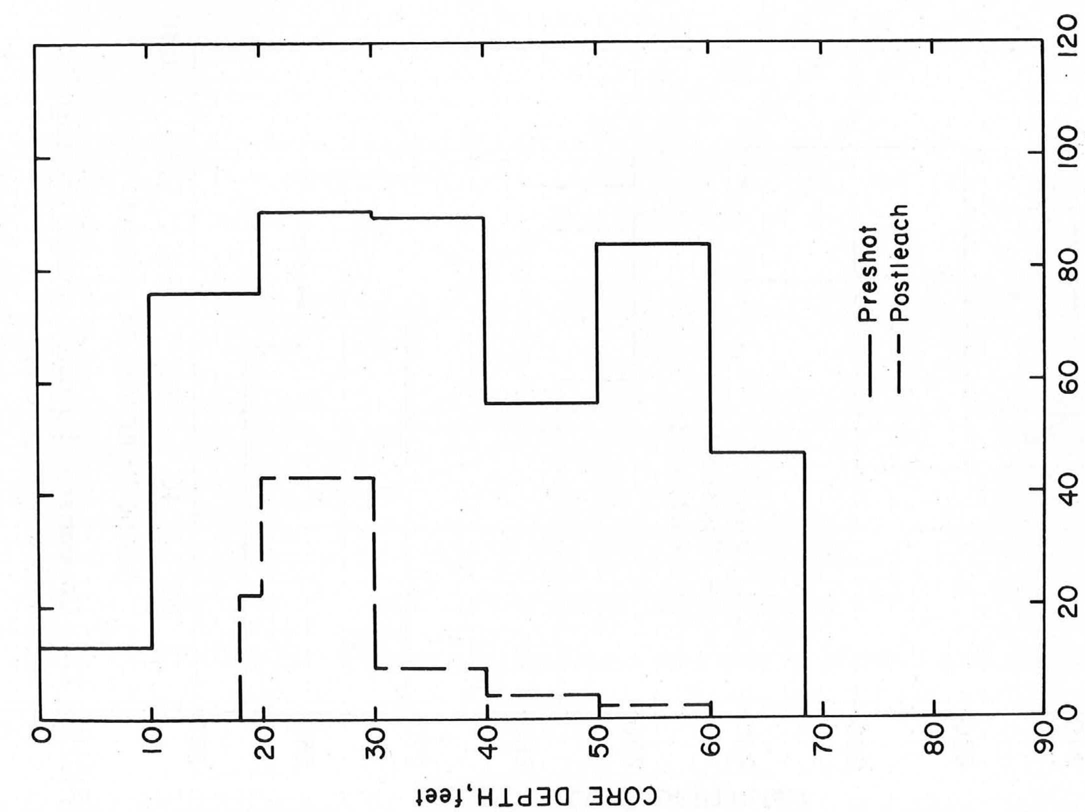

FIGURE 20. - Comparison of phase I preshot and postleach core length recovery.

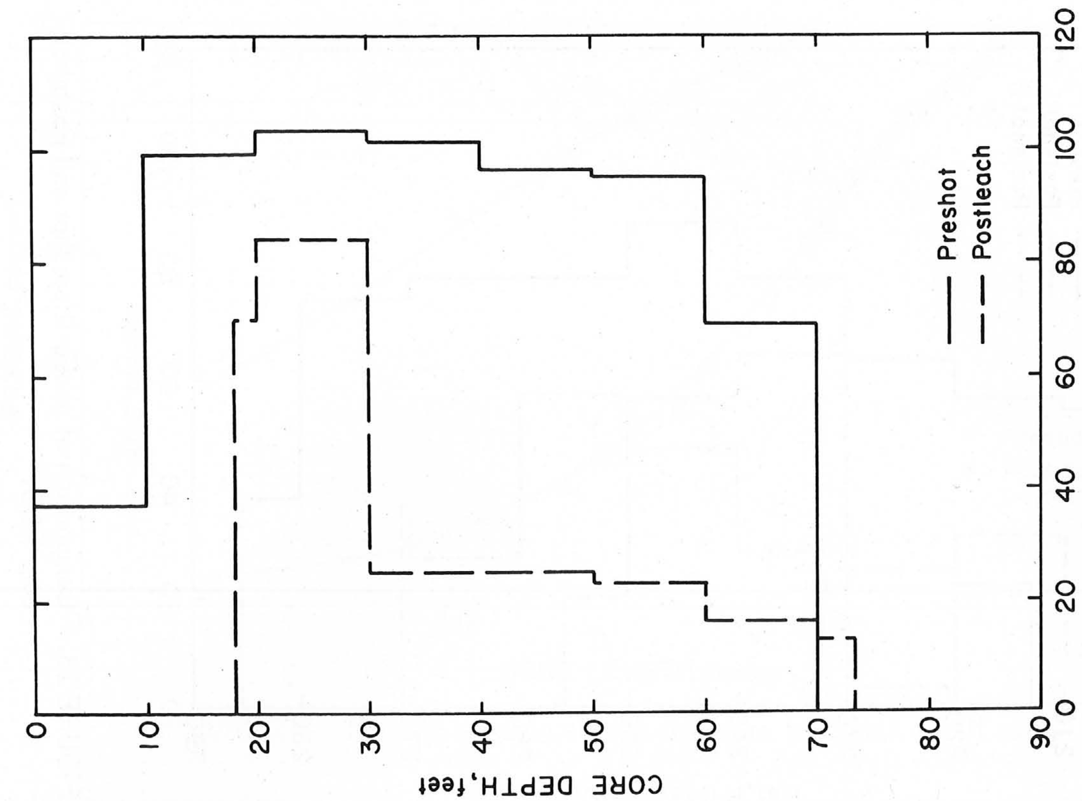

FIGURE 21. - Comparison of phase I preshot and postleach core RQD.

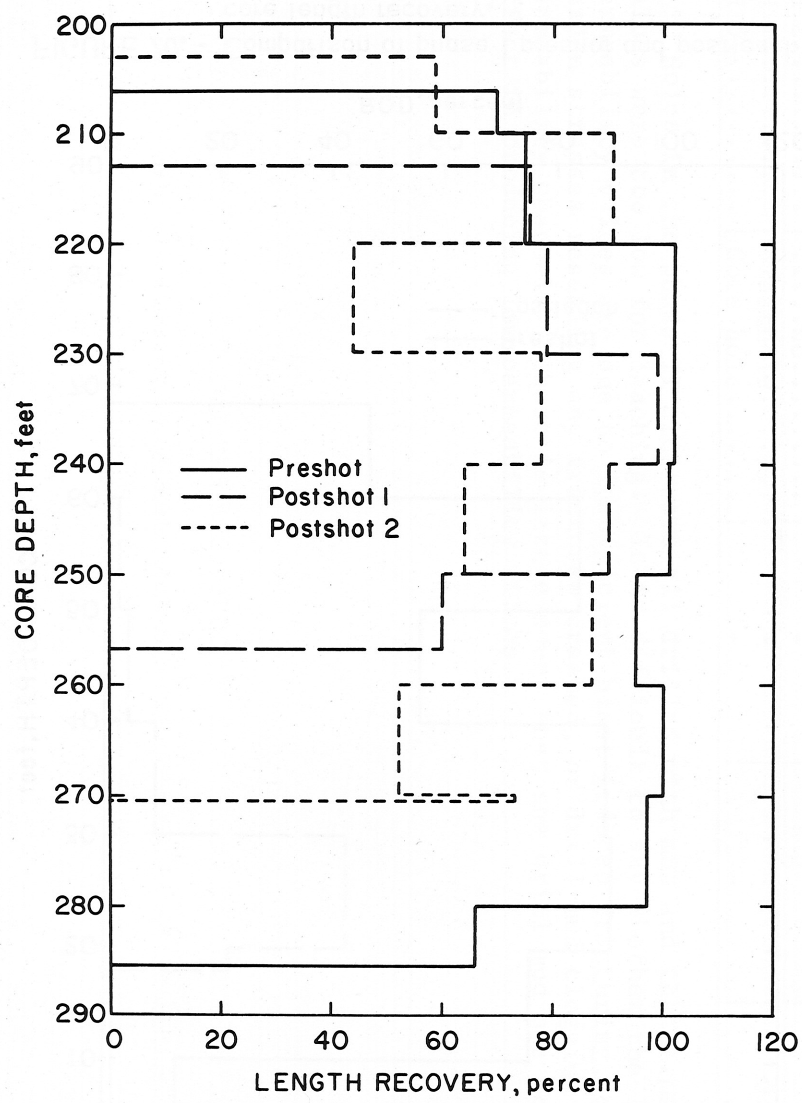

FIGURE 22. - Comparison of phase II preshot and postshot core length recovery.

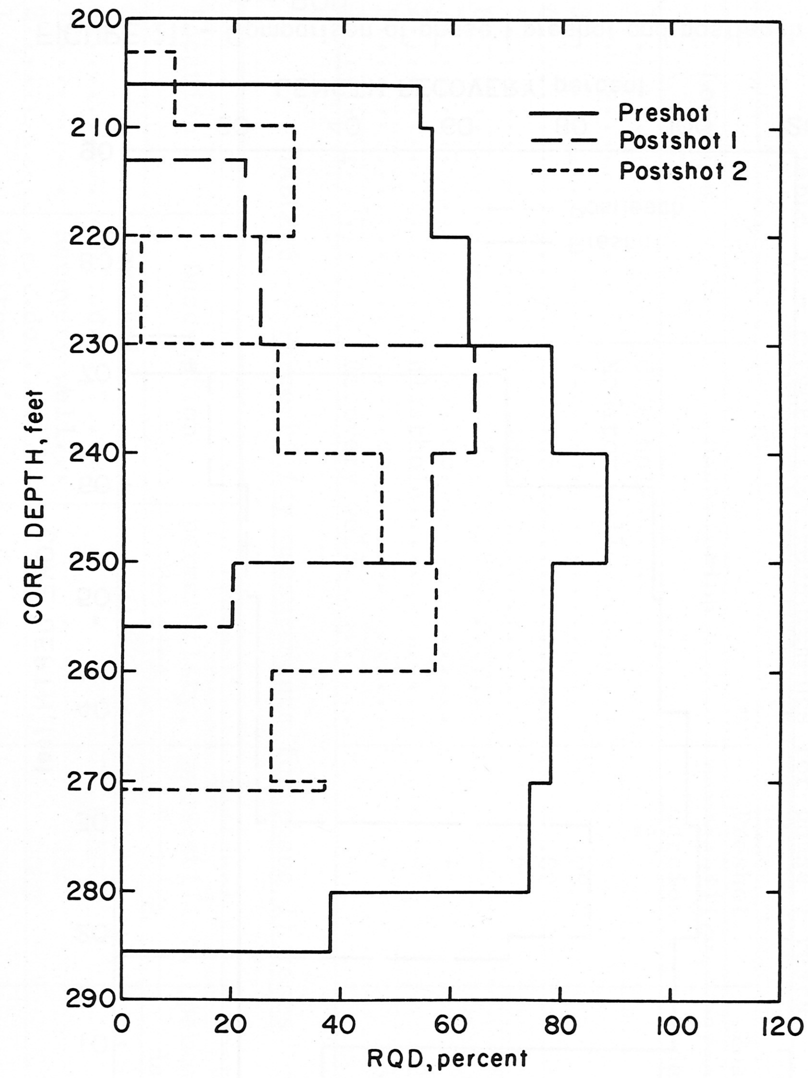

FIGURE 23. - Comparison of phase II preshot and postshot core RQD.

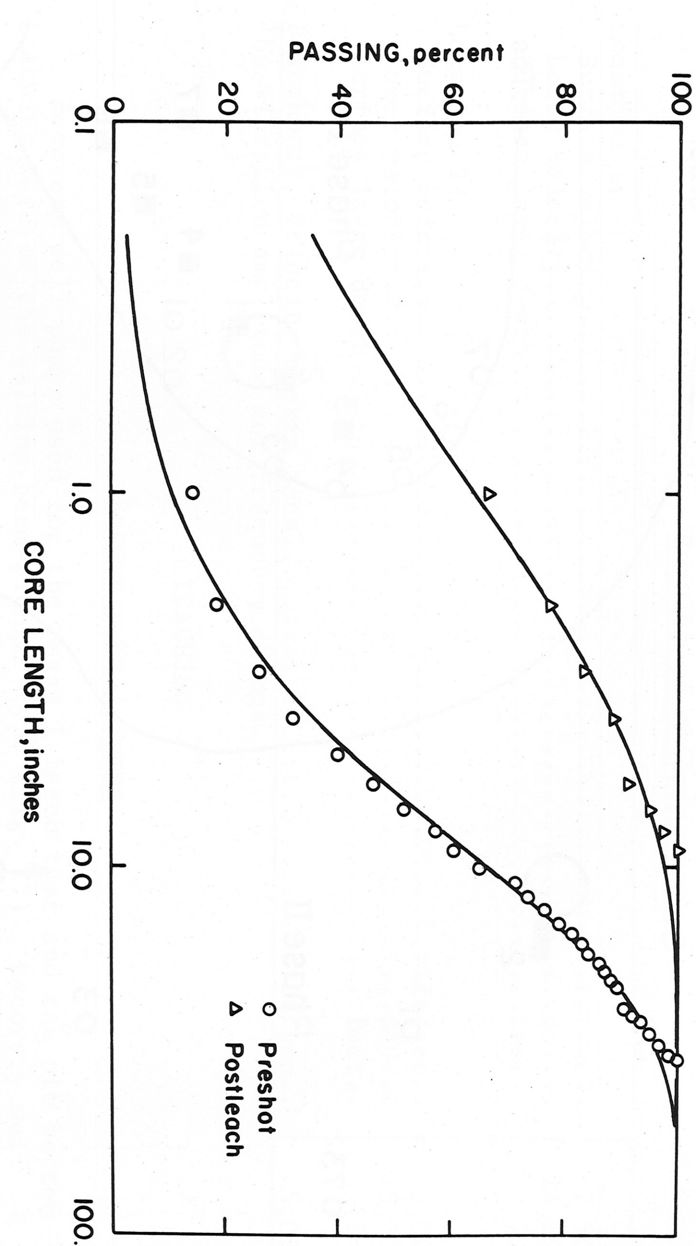

FIGURE 24. - Percent passing versus core size for phase I drill core.

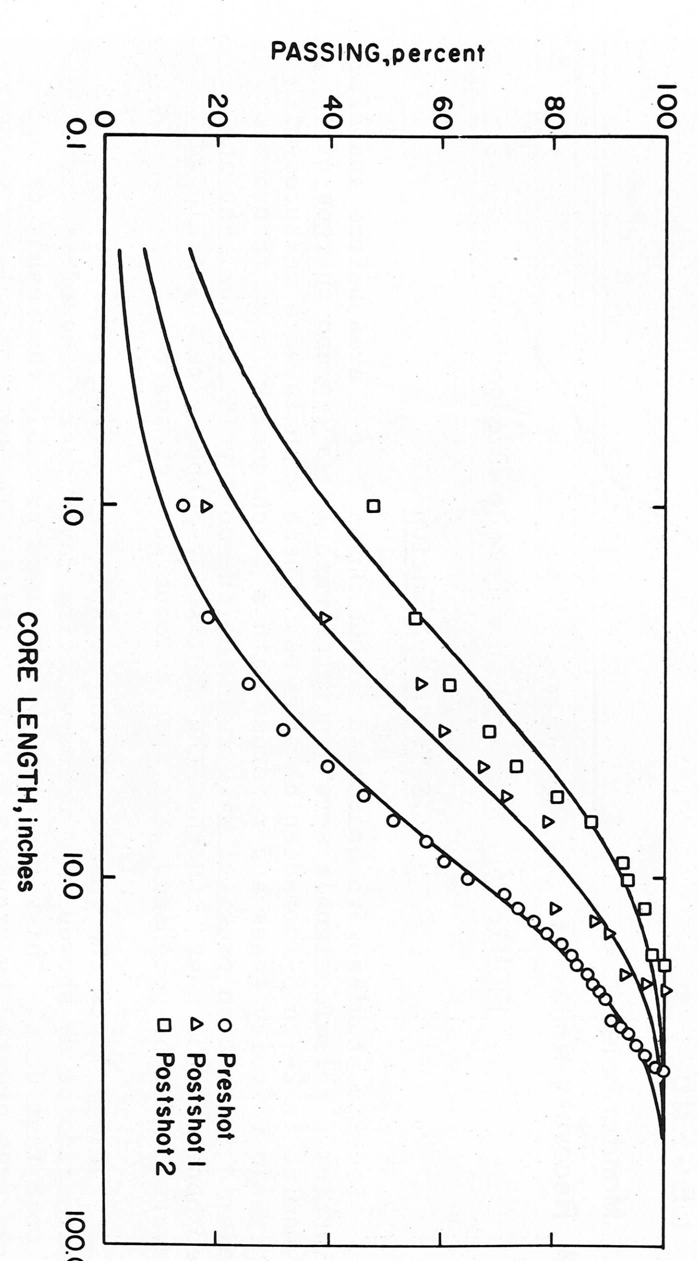

FIGURE 25. - Percent passing versus core size for phase II drill core.

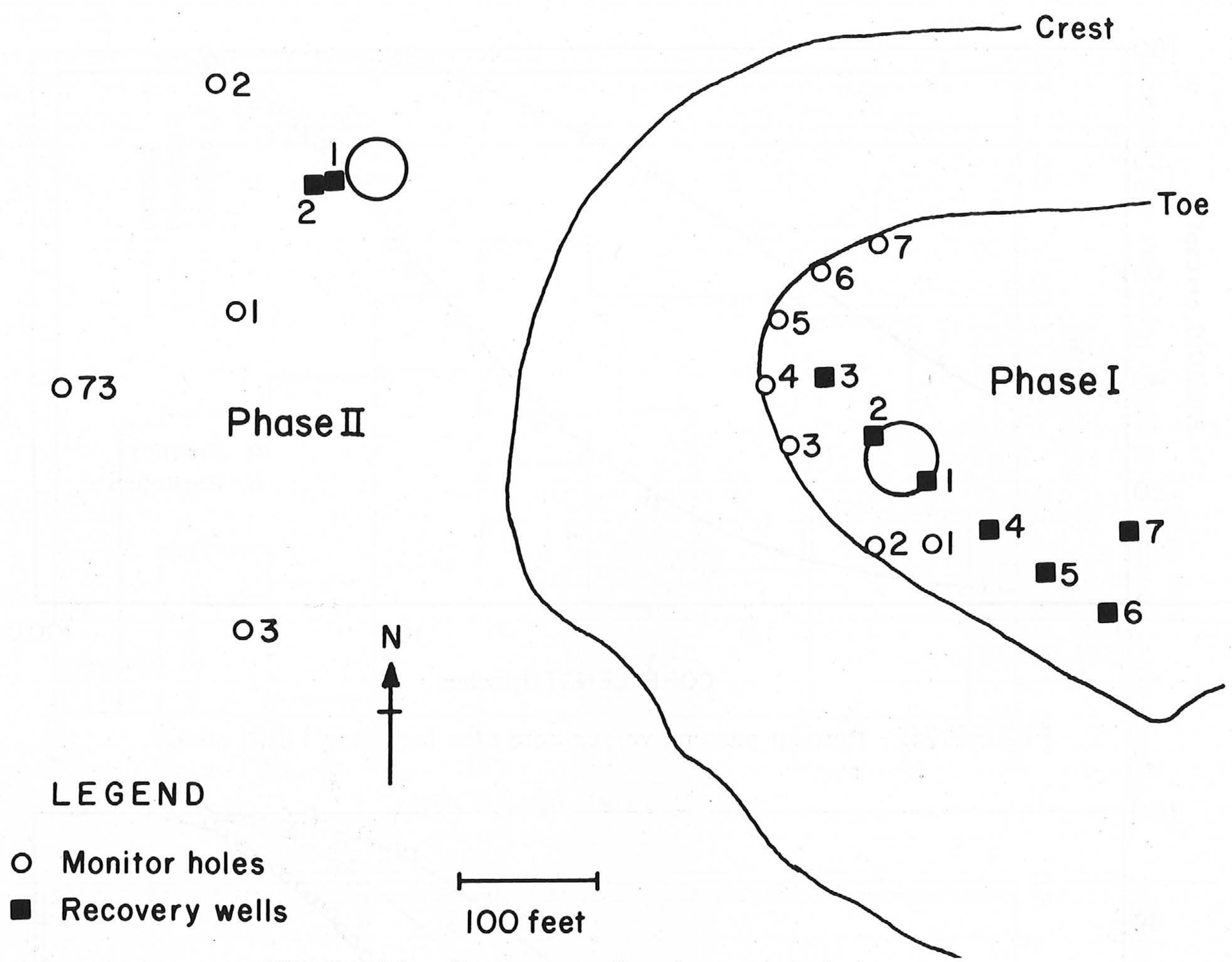

FIGURE 26. - Recovery well and monitor hole locations.

Seismic Evaluation

Seismic studies were conducted in the phase I test area before and after blasting. Seismic signals were generated with 1- to 2-pound charges of dynamite in 2- to 3-foot-deep blastholes. These signals were measured with particle velocity gages and recorded with a 14-channel FM tape recorder. Seismic refraction profiles were run with the shot point to the east and geophones to the west, and then the shot point-geophone relationship was reversed. Eight geophones were spaced about 20 feet apart.

Table 7 lists the results of the seismic refraction surveys. The refraction study showed that a low-velocity layer near the surface extended 3 to 8 feet deep. This low-velocity layer was probably the result of fractures created by previous mining activity. The preshot seismic velocity in the conglomerate ore was 15,000 ft/sec and dropped to 5,000 ft/sec after blasting. The seismic records revealed that the phase I blast did not produce a fractured zone that reduced the amplitude or changed the frequency of the transmitted seismic signals.

TABLE 7. - Results of seismic surveys

Location 1:	
Surface velocity..........ft/sec..	7,500
Lower velocity..........do....	15,200
Surface layer depth..........feet..	7.8
Location 2:	
Surface velocity..........ft/sec..	5,000
Lower velocity..........do....	15,200
Surface layer depth..........feet..	3.1
Postshot:[1] Velocity broken zone..........ft/sec..	5,000

[1]No vibration amplitude nor frequency change.

LEACHING

Recovery Well Pumps

Recovery well pumps used for the phase I leach test and the pit bottom leaching were of the walking beam-plunger type (fig. 27). Recovery well

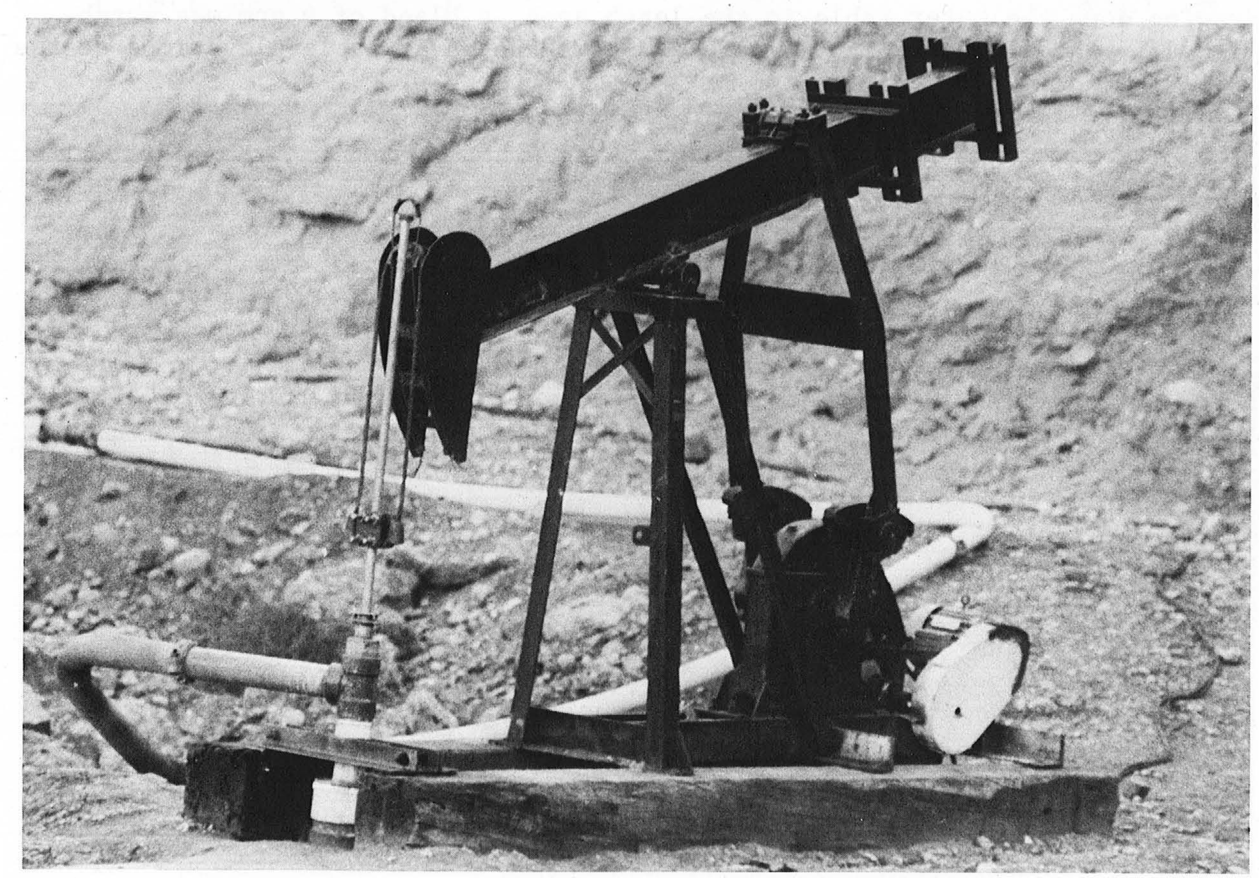

FIGURE 27. - Recovery well pump used during pit bottom leaching.

holes in the pit area were 12 inches wide and 50 to 60 feet deep, and were cased with 10-inch OD PVC casing, of which the lower 20 feet was perforated. The same hole and PVC casing diameters were used in the phase II area but the perforated casing was on the bottom 60 feet. A 20-hp submersible turbine pump was used in the phase II recovery well, which was positioned 290 feet down on a 4-inch ID stainless steel pipe inside the PVC casing.

Phase I Leach Test

The phase I leach test began in March 1974, and continued for 114 days. Leach solutions were distributed over the surface of the broken ore through perforated pipes and recovered in a well located on the east side of the blasted zone. The solutions were then pumped out of the pit to a recovery plant where cement copper was produced by precipitating the copper from the pregnant liquor with shredded iron in a cementation drum. Figure 28 shows the phase I test area during leaching. The 11-inch-diameter, 50-foot-deep recovery well hole was drilled with a churn drill. This hole was cased with a 10-inch-diameter PVC casing with the bottom 20 feet perforated.

During the leaching period flow rates averaged 63.5 gpm. The average flow rate was affected by significant shutdown time during the second and last weeks of operation. Because of ground water dilution and the desirability of drawing down the water table in the leach area, a bleed of 9 gpm was established in the flow circuit. Figure 29 shows the water table elevations at the

FIGURE 28. - Phase I leach test area.

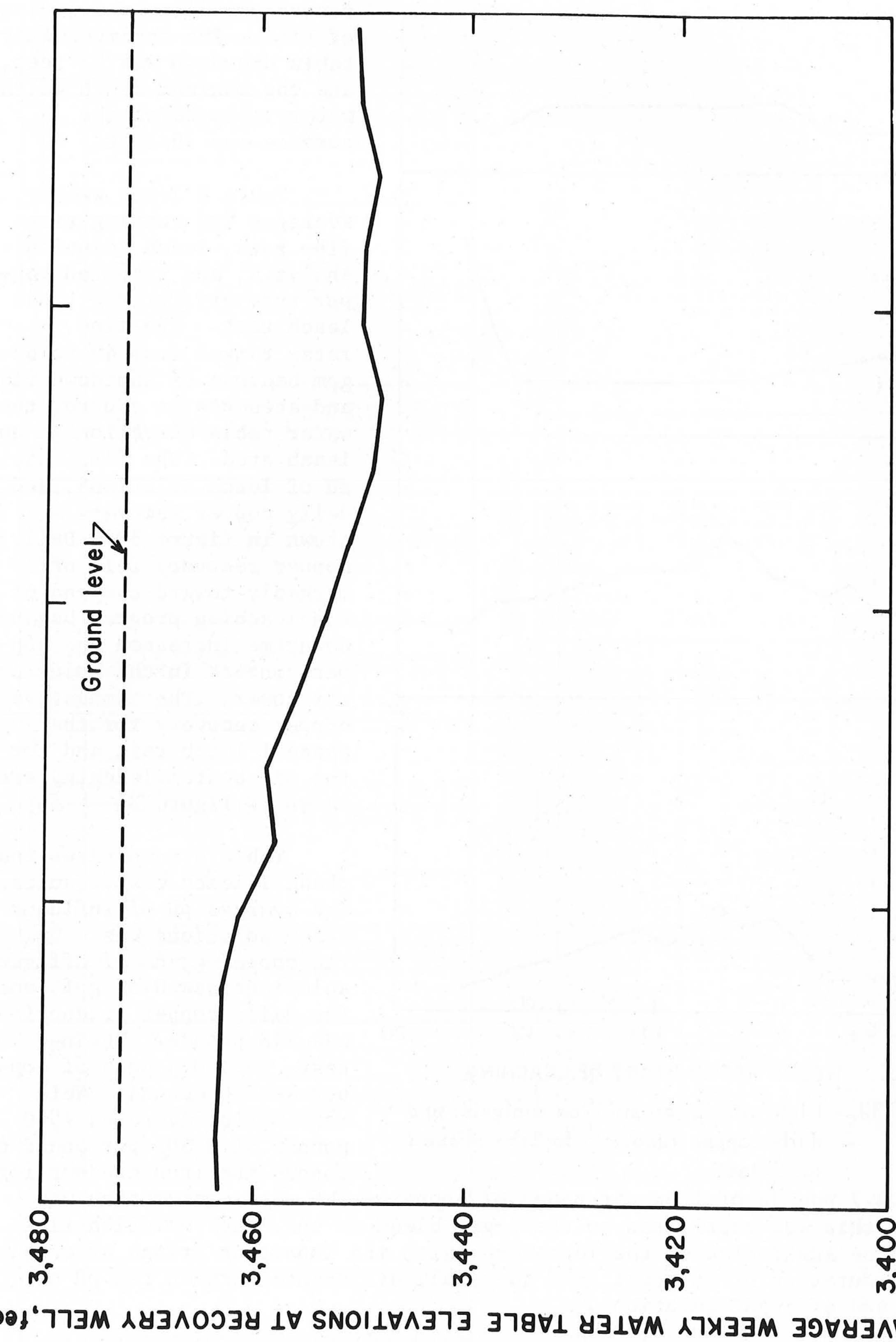

FIGURE 29. - Water table elevations versus time at the phase I recovery well.

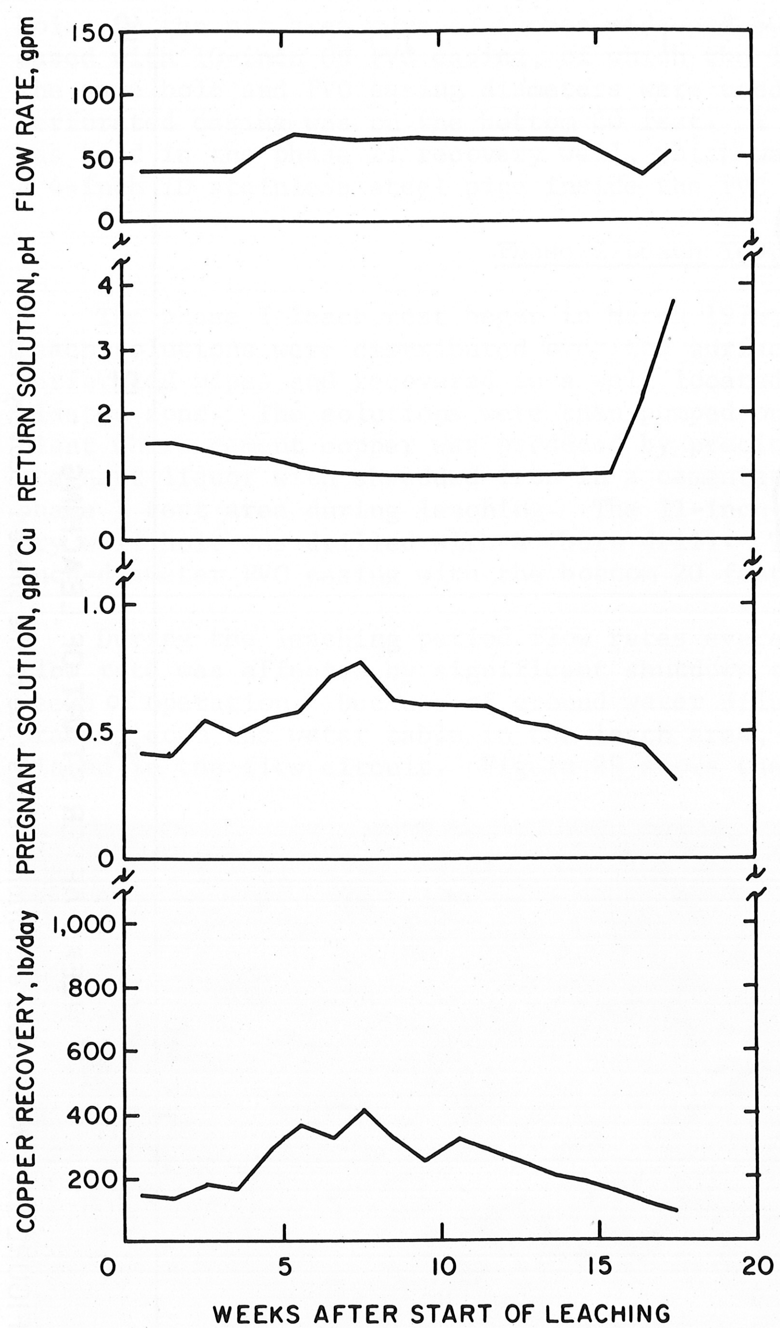

FIGURE 30. - Flow rate, leach solution analysis, and daily copper recovery for the phase I leach test.

recovery well as a function of time. The maximum water table drawdown was 23 feet, and the maximum depth of the water table below the surface was 25 feet.

Table 8 lists weekly averages for running time, flow rate, leach solution analysis, and computed copper recovery for the phase I leach test. The flow rates ranged from 40 to 68 gpm because of shutdown time and attempts to control the water table elevation in the leach area. The flow rate, pH of leach solutions, and daily copper recovery are shown in figure 30. Daily copper recovery fell off markedly toward the end of the leaching program because downtime increased and copper content in the solutions was lower. The cumulative copper recovery for the phase I leach test and for the pit bottom leaching are shown in figure 31.

Table 9 summarizes the phase I leach test results. The average pH of influent leach solutions was 1.04, the copper-grade of effluent solutions was 0.59 gpl, and the daily copper production was 248 pounds. During the test, 29,000 pounds of copper were produced. Acid consumption averaged 15.0 pounds of H_2SO_4 per pound of copper and iron consumption averaged 4.7 pounds of iron per pound of copper. The acid consumption was high; but this was attributed to the 9-gpm bleed in the leach solution flow circuit, the small size of the leach area with its important fringe effects, the short duration of the test, and the small difference between the pH of influent and effluent solutions.

TABLE 8. - Phase I leach test data

Week	Operating days/wk	Average operating time, hr/day	Total flow/wk, 1,000 gallons	Average flow, gpm	Copper content, gpl			Copper recovery, lb/day	pH levels	
					Pregnant solution	Precipitator	Return solution		Precipitator	Pit return
1.....	4	23.8	228.0	40.0	0.41	0.070	0.1	147.4	3.64	1.52
2.....	7	23.5	394.8	40.0	.40	.039	.1	141.2	4.02	1.53
3.....	7	20.4	343.2	40.0	.55	.033	.1	184.1	2.54	1.41
4.....	7	20.9	350.4	40.0	.49	.040	.1	162.9	1.85	1.28
5.....	7	21.9	529.7	57.5	.55	.059	.094	289.2	1.73	1.28
6.....	7	22.4	638.2	68.0	.58	.049	.099	365.9	1.65	1.16
7.....	7	17.4	488.5	66.7	.71	.058	.156	322.6	1.44	1.05
8.....	7	21.2	583.8	65.5	.77	.056	.171	416.8	1.41	1.03
9.....	7	23.7	655.2	65.8	.63	.064	.165	363.2	1.37	1.02
10.....	7	17.2	476.0	65.8	.61	.060	.157	257.0	1.36	1.02
11.....	7	21.7	588.0	64.5	.61	.072	.158	316.8	1.32	1.01
12.....	7	20.6	551.3	63.8	.60	.078	.171	281.9	1.30	1.00
13.....	7	21.7	581.3	63.7	.54	.098	.189	243.2	1.29	1.00
14.....	7	18.3	494.9	64.4	.52	.084	.177	202.3	1.33	1.00
15.....	7	20.3	528.2	62.0	.46	.070	.166	185.1	1.27	1.00
16.....	7	19.5	400.7	48.9	.47	.105	.160	148.1	1.26	1.00
17.....	7	16.1	373.5	35.3	.43	.093	.173	114.1	2.37	2.22
18.....	1	18.0	59.4	55.0	.29	.071	.110	89.2	3.15	3.75

TABLE 9. - Summary of leaching results

	Phase I leach test	Pit bottom leaching
Ore leached..............................tons..	15,000	100,000
Grade of ore..........................percent..	1.0	1.0
Duration of leaching...................days..	117	190
Average running time.................hr/day..	20.5	23.2
Leach influent:		
pH..	1.18	1.10
Copper..................................gpl..	0.147	0.091
Iron....................................gpl..	8.5	-
Leach effluent:		
Flow rate...............................gpm..	57.4	115.9
pH..	1.3	-
Copper..................................gpl..	0.562	0.646
Iron....................................gpl..	6.2	-
Precipitation effluent:		
pH..	1.71	1.62
Copper..................................gpl..	0.067	0.076
Acid consumption.............lb H_2SO_4/lb Cu..	15.0	10.0
Iron consumption................lb Fe/lb Cu..	4.7	2.75
Copper production....................lb/day..	245	748
Total copper production..............pounds..	29,000	142,000

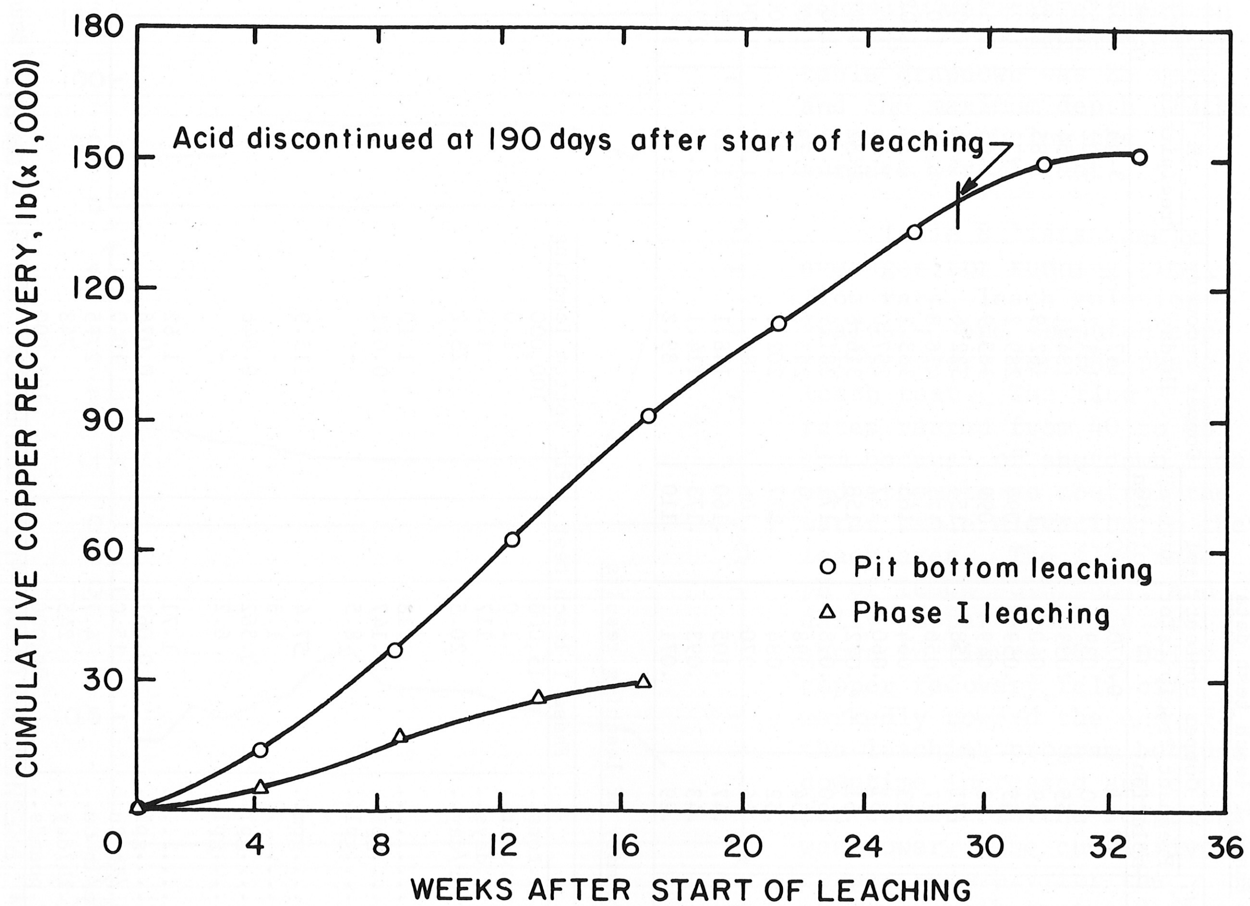

FIGURE 31. - Cumulative copper recovery for phase I leach test and pit bottom leaching.

Pit Bottom Leaching

Pit bottom leaching of about 100,000 tons of ore began in December 1974, and continued for 190 days. The system of seven 50-foot-deep recovery wells, spaced about 50 feet apart is shown in figures 26 and 32. The ore in the pit bottom was not blasted before this leaching in the hope that natural permeability would allow successful leaching.

The flow rate, pH of return leach solutions, copper content of pregnant leach solutions, and daily copper recovery are graphed in figure 33 and listed with other data in table 10. The copper content in the pregnant solution began to fall off only after acid was no longer added to the return leach solution after 190 days. Solutions were circulated for 6 weeks longer to bring up the pH of solutions remaining in the pit bottom, and an additional 9,000 pounds of copper were produced. The cumulative copper recovery for pit bottom leaching is shown in figure 31.

TABLE 10. - Pit bottom leaching data

Week	Operating days/wk	Average operating time, hr/day	Total flow/wk, 1,000 gallons	Average flow, gpm	Copper content, gpl			Copper recovery, lb/day	pH levels	
					Pregnant solution	Precip-itator	Return solution		Precip-itator	Pit return
1....	7	23.7	1,110.0	111.4	0.491	0.058	0.078	546.4	2.09	1.13
2....	7	22.7	920.4	96.5	.424	.029	.039	422.4	2.11	1.13
3....	7	21.1	838.1	94.4	.512	.033	.056	455.6	2.31	1.06
4....	7	21.8	1,005.4	109.9	.561	.066	.071	587.2	2.01	1.08
5....	7	24.0	1,390.7	138.0	.558	.055	.071	807.3	1.45	1.00
6....	7	23.1	1,465.5	150.8	.502	.062	.074	747.7	1.94	1.00
7....	7	24.0	1,599.6	158.7	.448	.048	.057	745.5	2.01	1.00
8....	7	24.0	1,321.9	131.1	.497	.055	.061	687.0	1.90	1.05
9....	7	23.4	1,254.3	127.9	.620	.064	.060	837.3	1.72	1.16
10....	7	19.9	916.2	109.9	.798	.091	.103	759.0	1.67	1.14
11....	7	24.0	1,178.3	116.9	.851	.089	.078	1,085.7	1.53	1.13
12....	7	24.0	923.9	91.7	.884	.888	.086	878.8	1.58	1.15
13....	7	24.0	1,221.6	121.2	.724	.107	.074	946.5	1.37	1.09
14....	7	22.4	1,219.4	129.4	.661	.091	.072	856.1	1.37	1.04
15....	7	23.3	1,392.5	142.4	.624	.059	.065	927.9	1.39	1.02
16....	7	21.3	1,202.6	134.5	.674	.080	.075	858.7	1.32	1.01
17....	7	24.0	1,155.7	114.7	.708	.053	.075	872.0	1.31	0.99
18....	7	24.0	1,027.5	101.9	.744	.089	.076	818.2	1.33	1.01
19....	7	24.0	1,150.1	114.1	.643	.085	.072	782.8	1.32	1.01
20....	7	23.7	797.8	80.1	.607	.083	.071	509.7	1.31	1.01
21....	7	23.9	1,022.9	101.8	.762	.154	.147	749.9	1.40	1.01
22....	7	23.0	881.1	91.2	.807	.134	.140	700.5	1.33	1.01
23....	7	23.6	861.0	86.7	.711	.101	.149	576.8	1.42	1.00
24....	7	23.9	1,172.8	117.0	.702	.074	.184	724.2	1.42	1.14
25....	7	23.4	1,240.2	126.2	.692	.097	.190	742.1	1.32	1.02
26....	7	24.0	1,146.1	113.7	.764	.073	.136	857.9	1.44	1.21
27....	7	22.6	1,067.7	112.6	.687	.056	.132	706.3	2.26	2.04
28....	1	24.0	187.6	130.3	.69	.089	.14	860.9	2.17	1.98
28....	7	17.9	793.9	105.4	.718	.077	.141	546.0	3.48	2.05
29....	7	22.0	1,002.7	108.5	.342	.039	.089	302.4	4.39	-
30....	7	23.9	1,043.2	104.1	.313	.054	.091	276.1	4.48	2.91
31....	7	24.0	1,094.2	108.6	.215	.060	.100	150.0	4.40	3.00
32....	7	21.4	821.7	91.3	.198	.059	.117	79.3	4.64	3.09
33....	7	19.6	830.8	100.7	.205	.068	.132	72.3	4.72	3.24

FIGURE 32. - Pit area during pit bottom leaching.

Table 9 summarizes the pit bottom leaching results. Average values are for the 190-day period only and not the 41-day period during which acid was not added. The average flow rate was 115.9 gpm, pH of return leach solution was 1.10, copper content of pregnant leach solution was 0.65 gpl, and copper recovery was 748 pounds per day. Acid consumption averaged 10 pounds of H_2SO_4 per pound of copper and iron consumption averaged 2.75 pounds of iron per pound of copper. The acid and iron consumption were lower than the phase I leach test but were still high. However, acid consumption was dropping as the test continued.

Pit bottom leaching was stopped because the flow rates of leach solutions were not as high as desired. With a total flow of 116 gpm and seven recovery wells, the average was only 17 gpm per well. Highest production was from wells near the blast-fractured phase I test area so the flows ranged from 45 to 5 gpm from individual wells. Pit bottom leaching was stopped and a plan to drill and blast the ore to improve permeabilities and flow rates was started. The drilling and blasting program began, but the Emerald Isle operation was closed before any further leaching of the pit bottom ore was done.

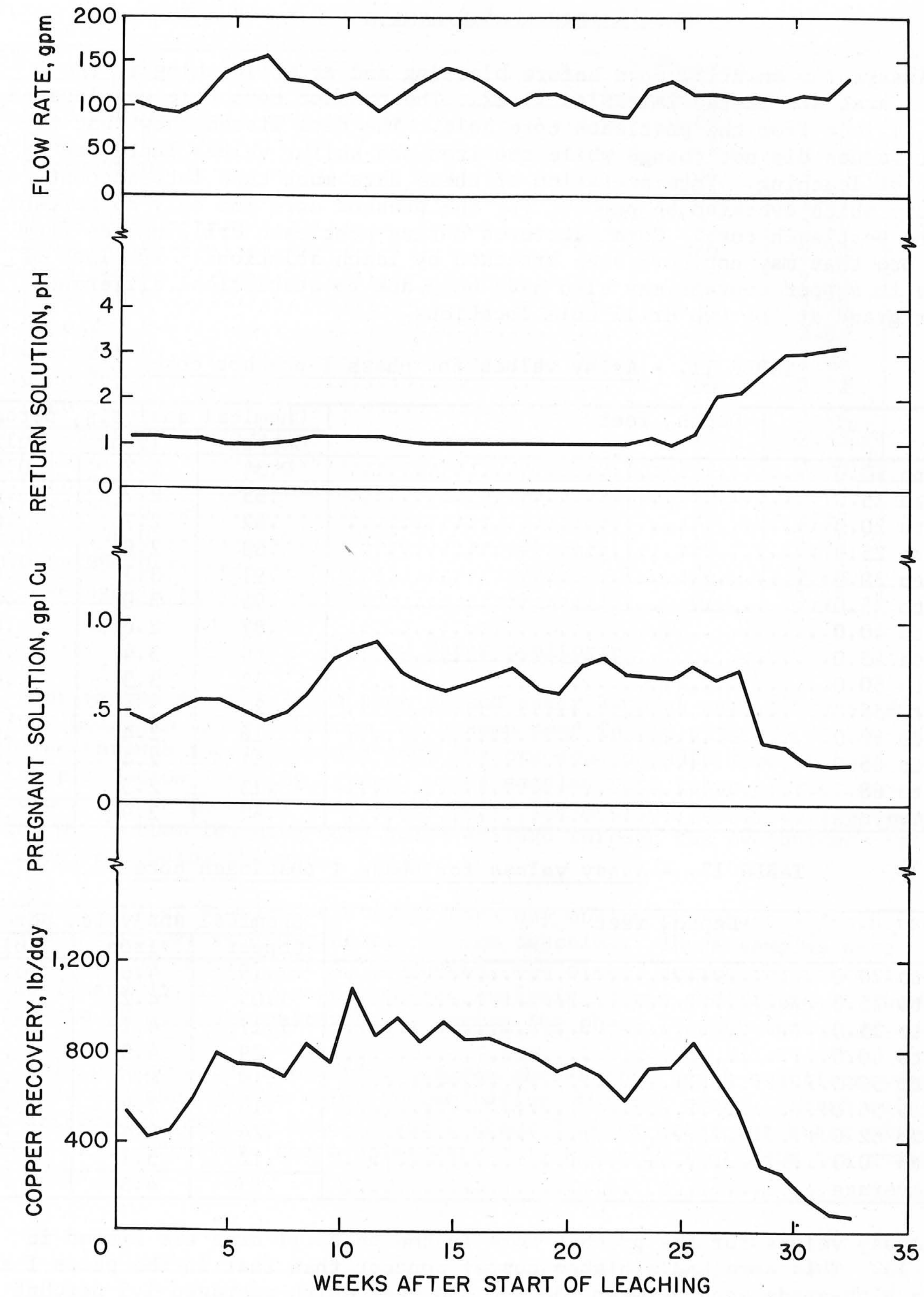

FIGURE 33. - Flow rate, leach bottom solution analysis, and daily copper recovery for pit bottom leaching.

DRILL CORE ASSAYS

Assays run on drill core before blasting and after leaching in the phase I area are listed in tables 11-12. The preshot core hole was located about 11 feet from the postleach core hole. The data listed show that the copper values did not change while the iron and sulfur values increased as a result of leaching. Interpretation of these data must take into account core recovery which averaged 86 percent for the preshot core and only 35 percent for the postleach core. Core recovered during postleach drilling was from solid ore that may not have been attacked by leach solutions. The lack of change in copper content may also have been due to statistical differences in copper grade at the two drill hole locations.

TABLE 11. - Assay values for phase I preshot core

Depth, feet	Chemical analysis, percent		
	Copper	Iron	Sulfur
0.0 to 10.0	0.34	2.7	0.023
10.0 to 15.0	.65	2.7	.041
15.0 to 20.0	.62	2.7	.056
20.0 to 25.0	.68	2.6	.110
25.0 to 30.0	.91	3.7	.039
30.0 to 35.0	.95	3.0	.090
35.0 to 40.0	.87	2.6	.092
40.0 to 45.0	.60	3.4	.052
45.0 to 50.0	.32	3.3	.049
50.0 to 55.0	.25	3.4	.120
55.0 to 60.0	.16	2.4	.017
60.0 to 65.0	.31	2.4	.026
65.0 to 68.7	.13	2.3	.017
Average	.52	2.9	.056

TABLE 12. - Assay values for phase I postleach core

Depth, feet	Chemical analysis, percent		
	Copper	Iron	Sulfur
15.1 to 20.0	1.19	4.6	0.140
20.0 to 25.0	1.03	4.2	.140
25.0 to 35.0	1.17	4.1	.110
35.0 to 40.0	.29	4.5	.100
40.0 to 50.0	.19	4.0	.029
50.0 to 55.0	.16	3.9	.037
55.0 to 62.0	.24	4.4	.210
62.0 to 70.3	.12	5.1	.056
Average	.53	4.3	.098

Assay values for the preshot core in the phase II area are listed in table 13. This core had a higher copper content than that in the phase I area with a high-grade zone between 230 and 250 feet which averaged 1.5 percent copper.

TABLE 13. - Assay values for phase II preshot core

Depth, feet	Chemical analysis, percent		
	Copper	Iron	Sulfur
206.0 to 210.0	0.69	3.08	0.160
210.0 to 216.0	.46	3.03	.080
216.0 to 220.0	.54	3.30	.088
220.0 to 225.0	.37	2.81	.055
225.0 to 230.0	.34	3.39	.062
230.0 to 235.0	1.69	2.90	.054
235.0 to 240.0	1.47	2.76	.084
240.0 to 245.0	1.70	2.90	.130
245.0 to 250.0	1.07	2.53	.150
250.0 to 255.0	.40	2.85	.064
255.0 to 260.0	.59	2.22	.022
260.0 to 265.0	.46	3.62	.110
265.0 to 270.0	.40	2.85	.024
270.0 to 275.0	.41	3.30	.064
275.0 to 280.0	.48	3.03	.058
280.0 to 285.0	.21	2.17	.045
285.0 to 290.0	.44	4.62	.026
290.0 to 294.0	.32	5.70	.028
Average	.67	3.17	.072

Ground Water Monitoring

During the phase I leach test ground water samples were taken at seven monitor holes in the pit bottom, at a sump pond in the pit, and at hole No. 73 (drilled through the overburden into the ore zone downdip from the pit area). During pit bottom leaching, ground water samples were taken at six monitor locations in the phase II area (fig. 26). The monitor locations sampled during pit bottom leaching were holes drilled through the overburden, into the ore zone, and below the water table.

Ground water samples were taken each day during the phase I leach test and about once per week during pit bottom leaching. These samples were analyzed for pH, copper, iron, manganese, and nickel during the phase I test. Tables 14 and 15 list average chemical analysis values and figure 34 shows changes in pH at four monitor holes during the phase I leach test.

This ground water monitoring program showed that leach solutions were contained and that ground water contamination did not occur. Leach solutions were detected in monitor holes 1 and 2 about 8 weeks after the start of the phase I test. However, these holes were only about 50 feet from the phase I test area.

TABLE 14. - Chemical analysis of ground water samples during the phase I leach test

Element	Days sampled	Location								
		MW 1	MW 2	MW 3	MW 4	MW 5	MW 6	MW 7	MW 73	Sump
Copper............ppm..	93	23.37	4.06	1.16	0.11	0.10	0.09	0.10	0.01	0.12
Iron..............ppm..	84	3.23	0.11	0.09	0.10	0.09	0.08	0.09	0.06	0.09
Manganese.........ppm..	93	33.92	6.68	9.20	0.12	0.08	0.04	0.07	0.02	0.21
Nickel............ppm..	84	11.31	0.29	0.46	0.14	0.12	0.12	0.12	0.05	0.24
Cadmium...........ppm..	18	0.36	0.08	0.05	0.04	0.04	0.04	0.04	0.03	0.04
Molybdenum........ppm..	18	0.07	0.04	0.05	0.05	0.05	0.06	0.06	0.04	0.04
Calcium...........ppm..	18	360	489	548	391	452	479	500	222	581
Magnesium.........ppm..	18	233	209	218	176	197	235	254	66	213
Aluminum..........ppm..	18	67.5	4.8	5.3	4.9	4.7	5.2	5.6	3.5	6.2
Zinc..............ppm..	18	27.64	3.05	1.06	0.10	0.09	0.09	0.08	0.10	1.91
Cobalt............ppm..	18	0.73	0.10	0.05	0.04	0.04	0.04	0.04	0.02	0.03
pH....................	36	6.3	6.6	6.8	7.2	7.2	7.0	7.0	7.6	7.6

TABLE 15. - Chemical analysis of ground water samples during pit bottom leaching

Element	Days sampled	Location in phase II area					
		MH 1	MH 2	MH 3	RW 1	RW 2	MW 73
Copper....................ppm..	37	0.04	0.04	0.06	0.05	0.05	0.02
Iron......................ppm..	37	0.06	0.08	0.04	0.05	0.06	0.04
Manganese.................ppm..	37	6.2	0.13	0.16	1.19	0.39	0.05
Nickel....................ppm..	37	0.08	0.09	0.04	0.09	0.10	0.06
Cadmium...................ppm..	37	0.03	0.04	0.03	0.03	0.04	0.03
Molybdenum................ppm..	37	0.08	0.08	0.16	0.13	0.13	0.08
Calcium...................ppm..	37	179	288	10.1	202	229	172
Magnesium.................ppm..	37	97.3	182	4.66	100	110	59.0
Aluminum..................ppm..	37	1.6	1.81	0.44	1.59	1.66	0.97
Zinc......................ppm..	37	0.73	0.07	0.14	0.18	0.09	0.11
Cobalt....................ppm..	37	0.05	0.07	0.04	0.06	0.06	0.06
pH...........................	37	7.14	7.45	8.05	7.64	7.73	7.58

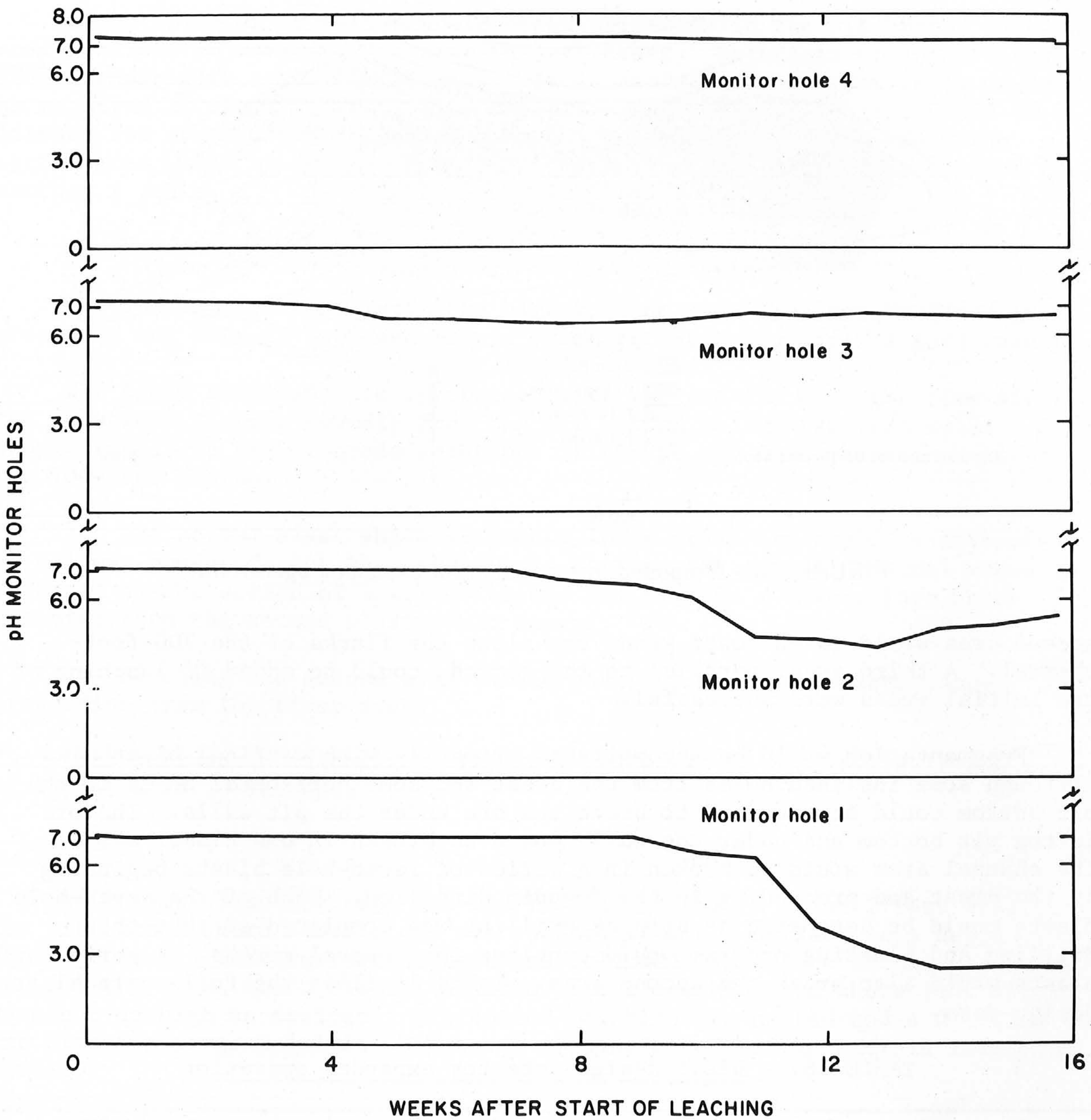

FIGURE 34. - Changes in pH at monitor holes 1-4 during the phase I leach test.

DESIGN OF EXPANDED IN SITU LEACHING SYSTEM

Based on the blasting and leaching test results, a full-scale in situ leaching system was designed to recover the majority of the remaining copper. This plan involved blasting and leaching a higher grade area first, then working on a lower grade area. The first area would include ore in the pit bottom, under the pit walls, and under 180 to 250 feet of overburden along a channel extending 700 feet from the crest of the pit. The channel would follow the high-grade copper mineralization as shown in figure 35. The

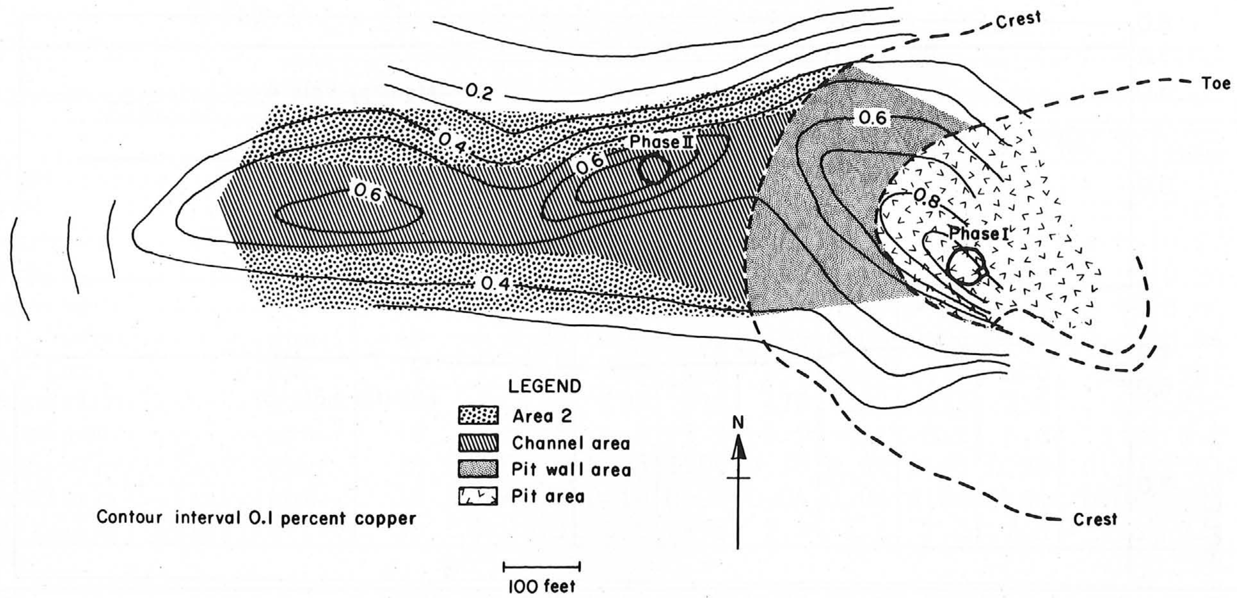

FIGURE 35. - Proposed blast design for expanded operation.

second area would leach lower grade ore along the flanks of the 700-foot-channel. A third area, parallel to the second, could be added if leaching of the initial areas were successful.

Fragmentation would be accomplished primarily with vertical blastholes although some inclined holes from the crest and some horizontal holes in the pit bottom would be required to break the ore under the pit walls. The ore in the pit bottom and under the walls would be broken in one blast, while the channel area would be broken in a series of seven-hole blasts beginning at the crest and proceeding in the downdip direction. Each of the seven-hole blasts would be detonated as soon as drilling was completed so that this drilling and blasting program would continue for several months. A series of blasts would also break the second area. Table 16 lists the full-scale blast design data.

TABLE 16. - Blast design data for expanded operation

	Area 1			Area 2
	Pit	Walls	Channel	
Area......................sq ft..	59,000	56,000	96,000	88,000
Thickness..................feet..	40	69	78	76
Volume....................cu yd..	87,000	143,000	276,000	246,000
Weight.....................tons..	167,000	274,000	530,000	472,000
Grade...................percent..	0.70	0.58	0.55	0.41
Total copper.............pounds..	2,300,000	3,200,000	5,800,000	3,900,000
Drilling depth.............feet..	40	125	287	297
Powder factor............lb/ton..	0.5	1.0	1.5	.75
Total drilling.............feet..	2,200	12,900	76,300	36,000
Total explosives.........pounds..	83,000	274,000	795,000	354,000

Leach solution injection would occur primarily in the pit bottom with some vertical injection holes near the pit crest. Solutions would be recovered downdip with a series of wells. It is estimated that four wells would be required at about 700 feet from the pit crest. The first area would be leached for about 2 years depending on the grade of solutions before the second area would be added. Both areas would then continue to be leached for another 2 years.

The major advantages of this in situ leaching system are:

1. Initial leaching is in the higher grade ore. The second lower grade area does not need to be added unless the first area had yielded good results.

2. Leach solutions are in contact with the ore for a longer time since the solutions migrate downdip from the pit bottom to the recovery wells. This should result in higher grade solutions if the pH could be maintained along the 700-foot channel.

3. The second blast which fragments lower grade ore could be detonated at a lower powder factor than the first, because the initial channel would be weakened by the action of leach solutions and provide a volume into which material from the second blast could expand.

4. Blasting the second area should shake up and restimulate copper production from the first area.

This full-scale leaching is feasible and would be profitable if the price of copper were to increase.

SUMMARY

An in situ leaching test was run for 117 days in a phase I test area in the pit bottom where blasted ore was exposed from the surface to a depth of about 50 feet. This successful test was followed by inplace leaching another 100,000-ton area, which was not blasted before leaching and where flow rates were not as high as desired. Pit bottom leaching was halted and a drilling and blasting program initiated but not completed before the Emerald Isle mine closed.

A phase II test area under 205 feet of overburden and extending to 290 feet was blasted. Water circulation revealed that the first blast did not create adequate breakage and permeability for leaching. A second blast did improve the fragmentation, but water circulation tests were not conducted because the mine closed.

An expanded in situ leaching system was designed to recover copper from 1,500,000 tons of ore in the pit bottom, under the pit walls, and under overburden extending 700 feet along a channel from the pit crest. This program was not implemented.

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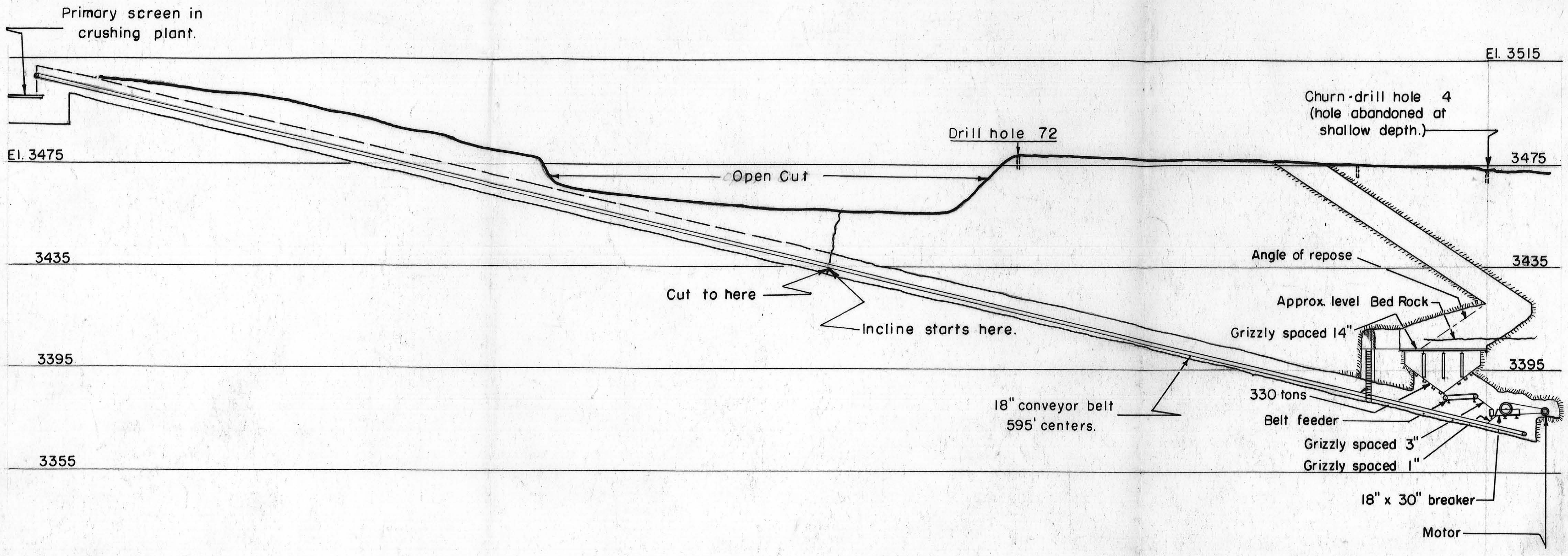


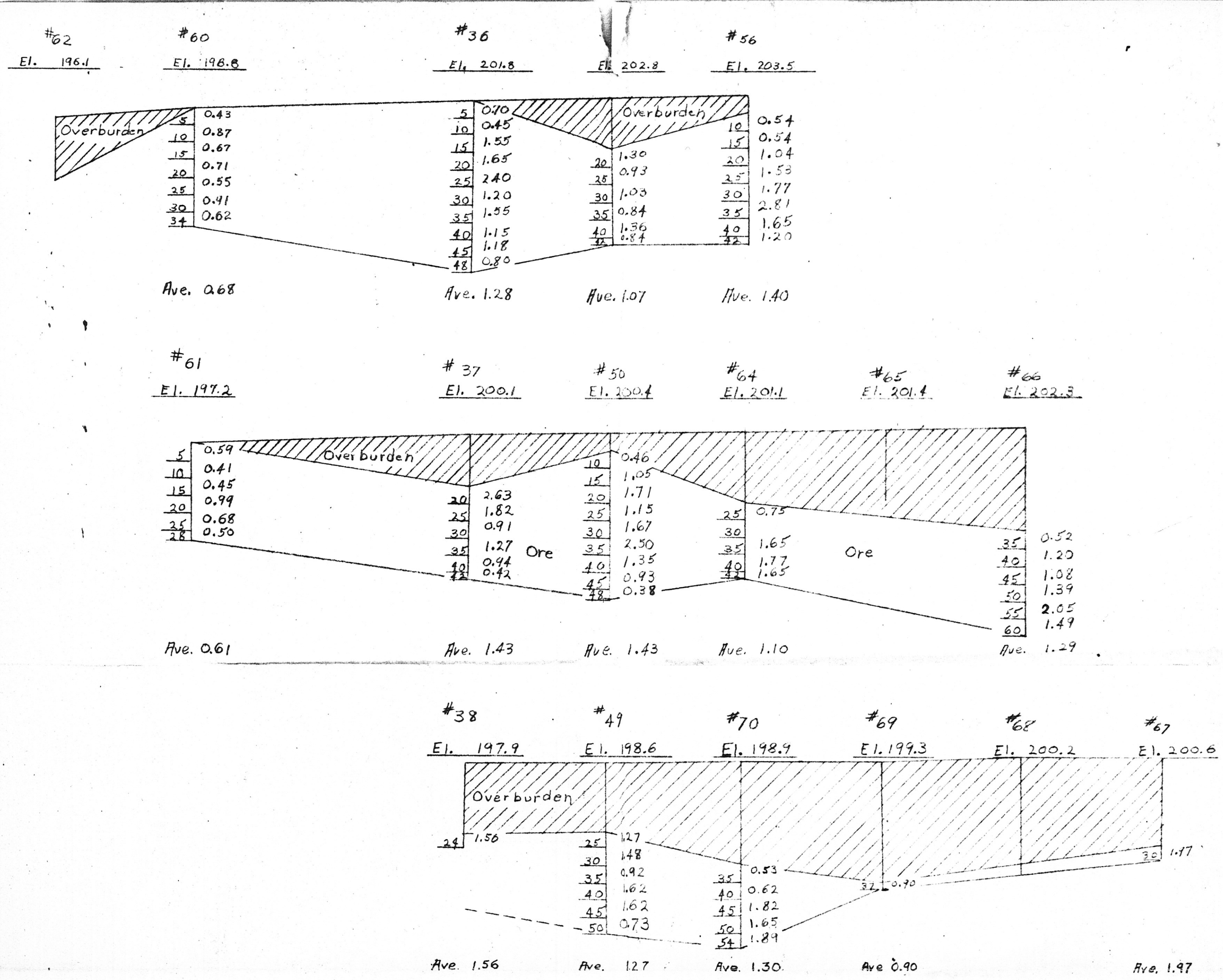
From sketch
By: C. F. Weeks

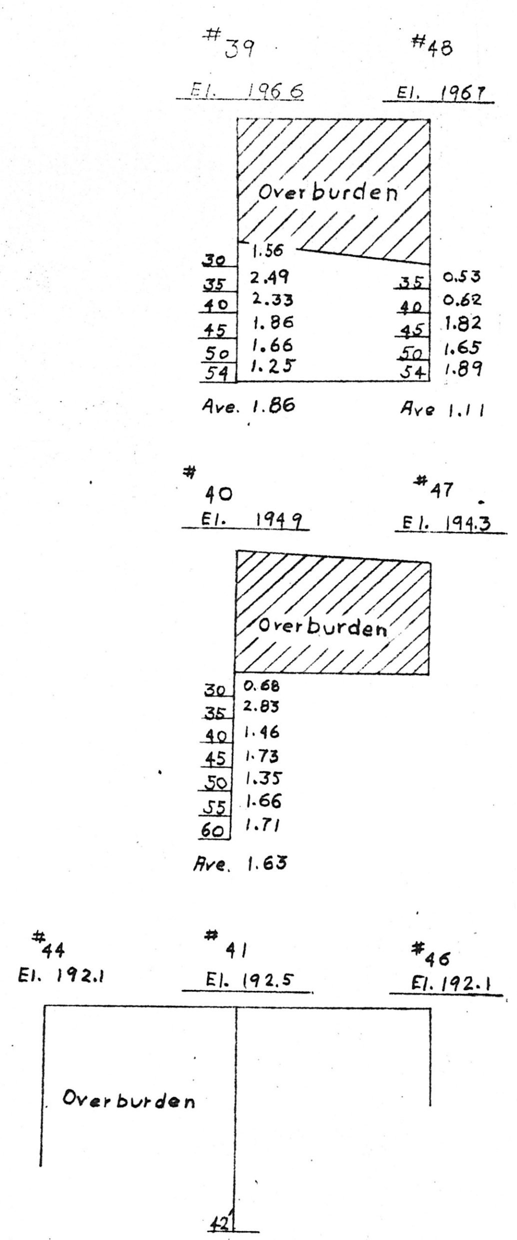
D.M.A. 374

Scale: 1" = 40'

Figure 4 .- Proposed pocket and conveyor, Emerald Isle mine.

copper

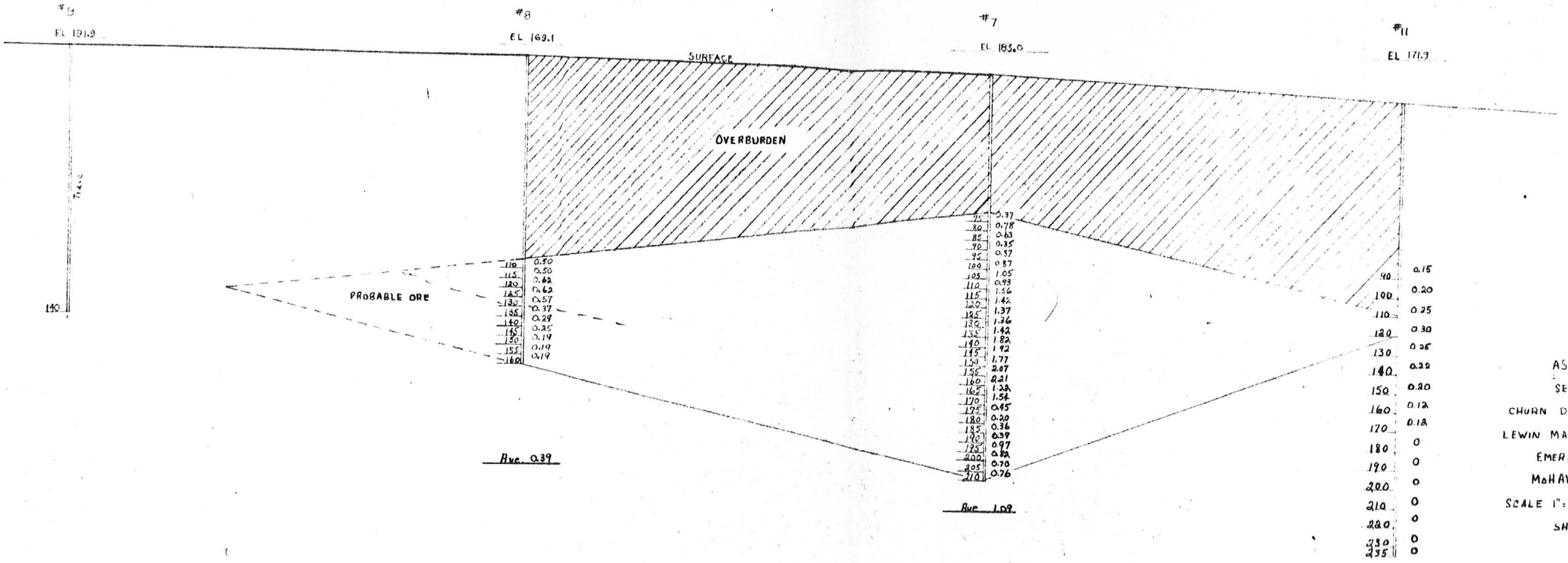




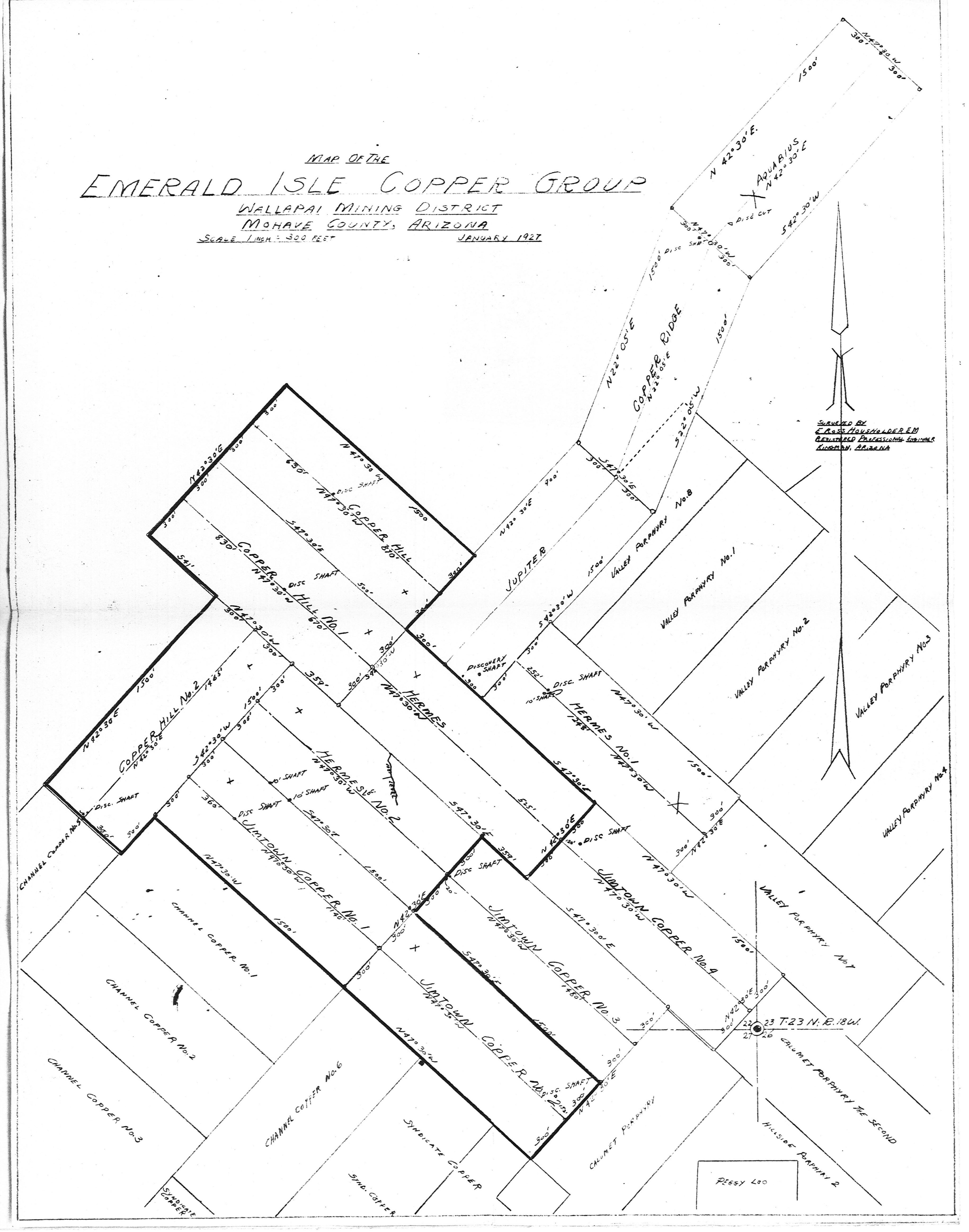
SHEET NO. "B"
Wagon Drill Holes

East-West Sections
1" = 20'

Emerald Isle



ASSAY MAP
SECTION ON
CHURN DRILL TEST HOLES
LEWIN MATHES MINING CO.
EMERALD ISLE DIV.
MOHAVE CO. ARIZONA
SCALE 1"=20' NOV. 1, 1947
SHEET No. 30


MAP OF THE
EMERALD ISLE COPPER GROUP
WALLAPAI MINING DISTRICT
MOHAVE COUNTY, ARIZONA
SCALE 1 INCH = 300 FEET
JANUARY 1927
SURVEYED BY
E. ROSS HOUSHOLDER E.M.
REGISTERED PROFESSIONAL ENGINEER
KINGMAN, ARIZONA
AQUARIUS
COPPER RIDGE
JUPITER
COPPER HILL
COPPER HILL No. 1
COPPER HILL No. 2
HERMES
HERMES No. 1
HERMES No. 2
JIMTOWN COPPER No. 1
JIMTOWN COPPER No. 2
JIMTOWN COPPER No. 3
JIMTOWN COPPER No. 4
VALLEY PORPHYRY No. 1
VALLEY PORPHYRY No. 2
VALLEY PORPHYRY No. 3
VALLEY PORPHYRY No. 4
VALLEY PORPHYRY No. 8
VALLEY PORPHYRY No. 7
CHANNEL COPPER No. 1
CHANNEL COPPER No. 2
CHANNEL COPPER No. 3
CHANNEL COPPER No. 5
CHANNEL COPPER No. 6
SYNDICATE COPPER
SYND. COPPER
CALUMET PORPHYRY
CALUMET PORPHYRY THE SECOND
HILLSIDE PORPHYRY 2
PEGGY LOO
T. 23 N. R. 18 W.
DISCOVERY SHAFT
DISC. SHAFT

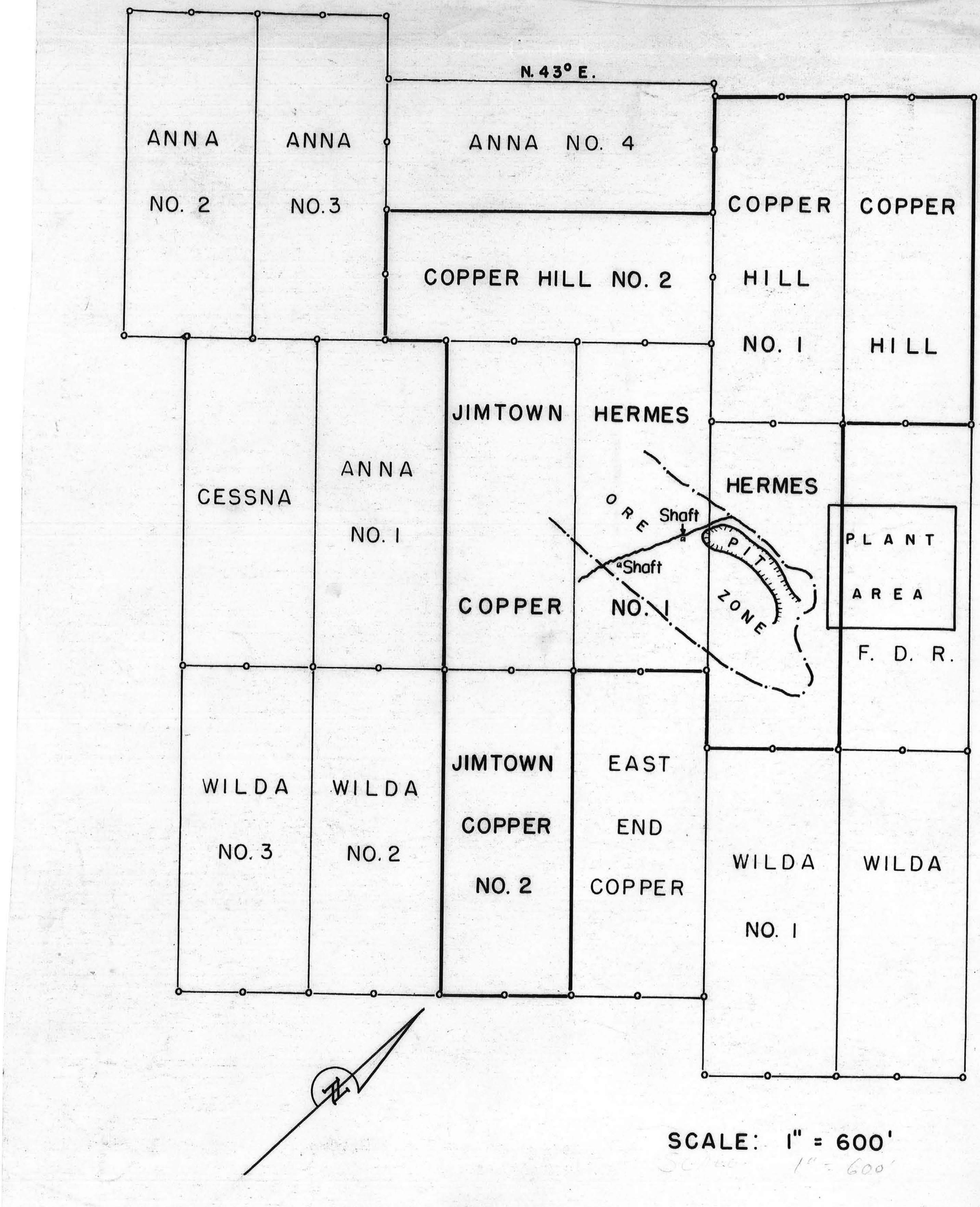
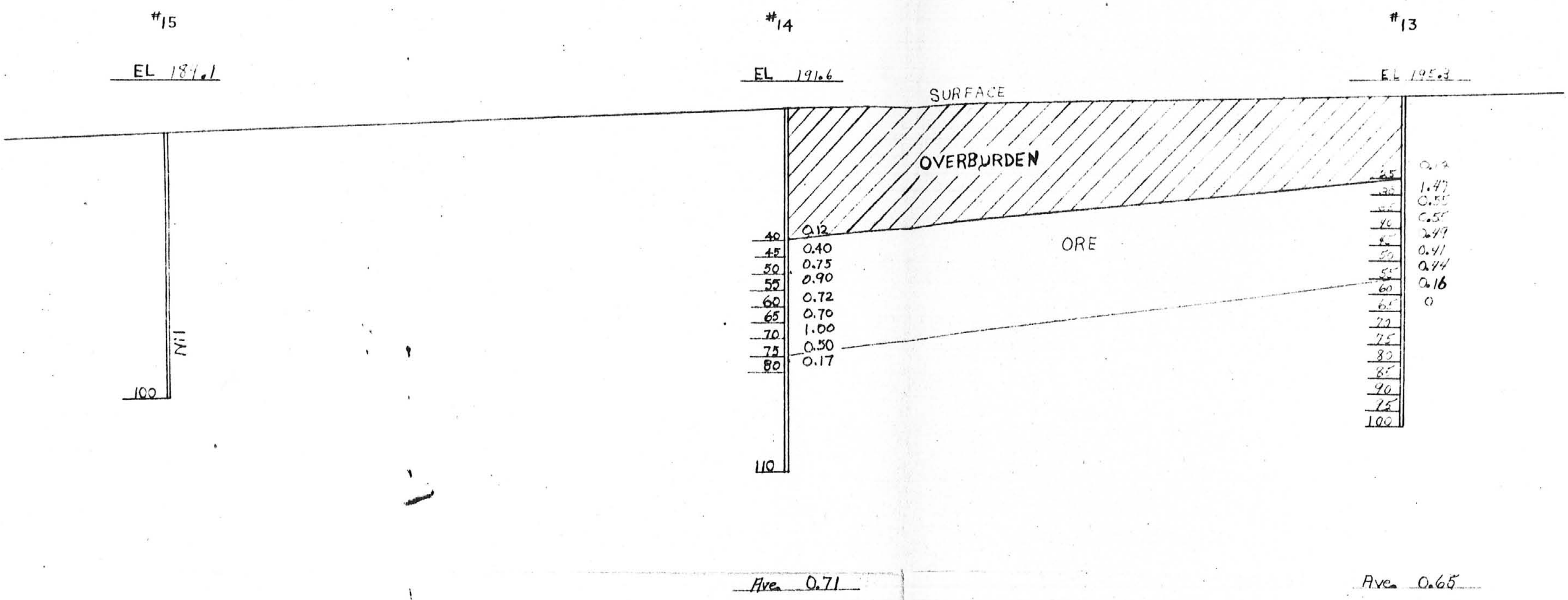


Figure 2.- Property map, Emerald Isle mine, Mohave Co., Arizona.

DMA 374



ASSAY MAP

SECTION ON

CHURN DRILL TEST HOLES

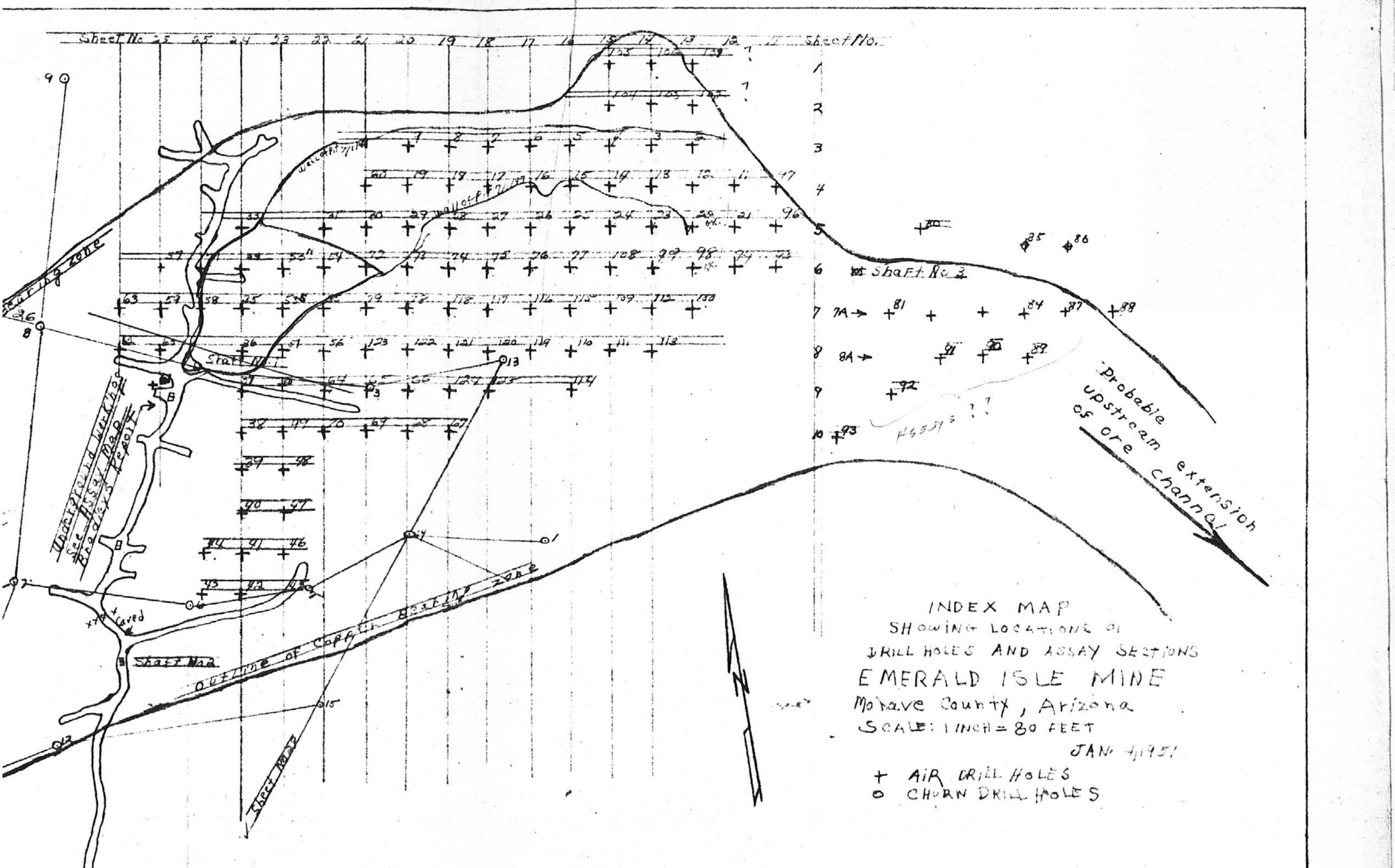
LEWIN MATHES MINING CO.

EMERALD ISLE DIV.

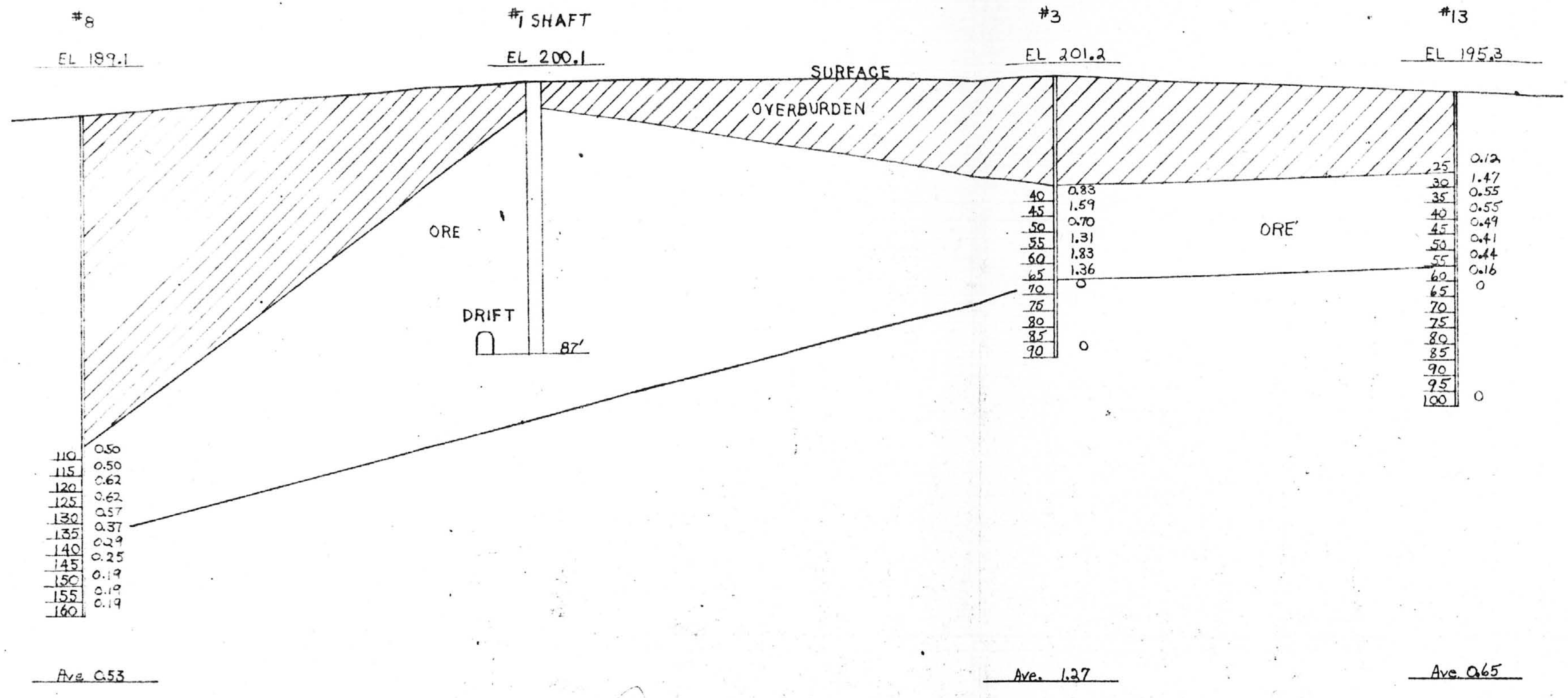
MOHAVE CO. ARIZONA

SCALE 1" = 20' NOV. 1, 1947

SHEET No. 29



INDEX MAP
SHOWING LOCATIONS OF
DRILL HOLES AND ASSAY SECTIONS
EMERALD ISLE MINE
Mohave County, Arizona
SCALE: 1 INCH = 80 FEET
JAN. 4, 1951
+ AIR DRILL HOLES
o CHURN DRILL HOLES
Sheet No.
Shaft No. 3
Shaft No. 1
Probable upstream extension of ore channel
Outline of Copper Bearing zone
Underground workings
See Assay Map
Bradley's Report
caved



ASSAY MAP

SECTION ON

CHURN DRILL TEST HOLES

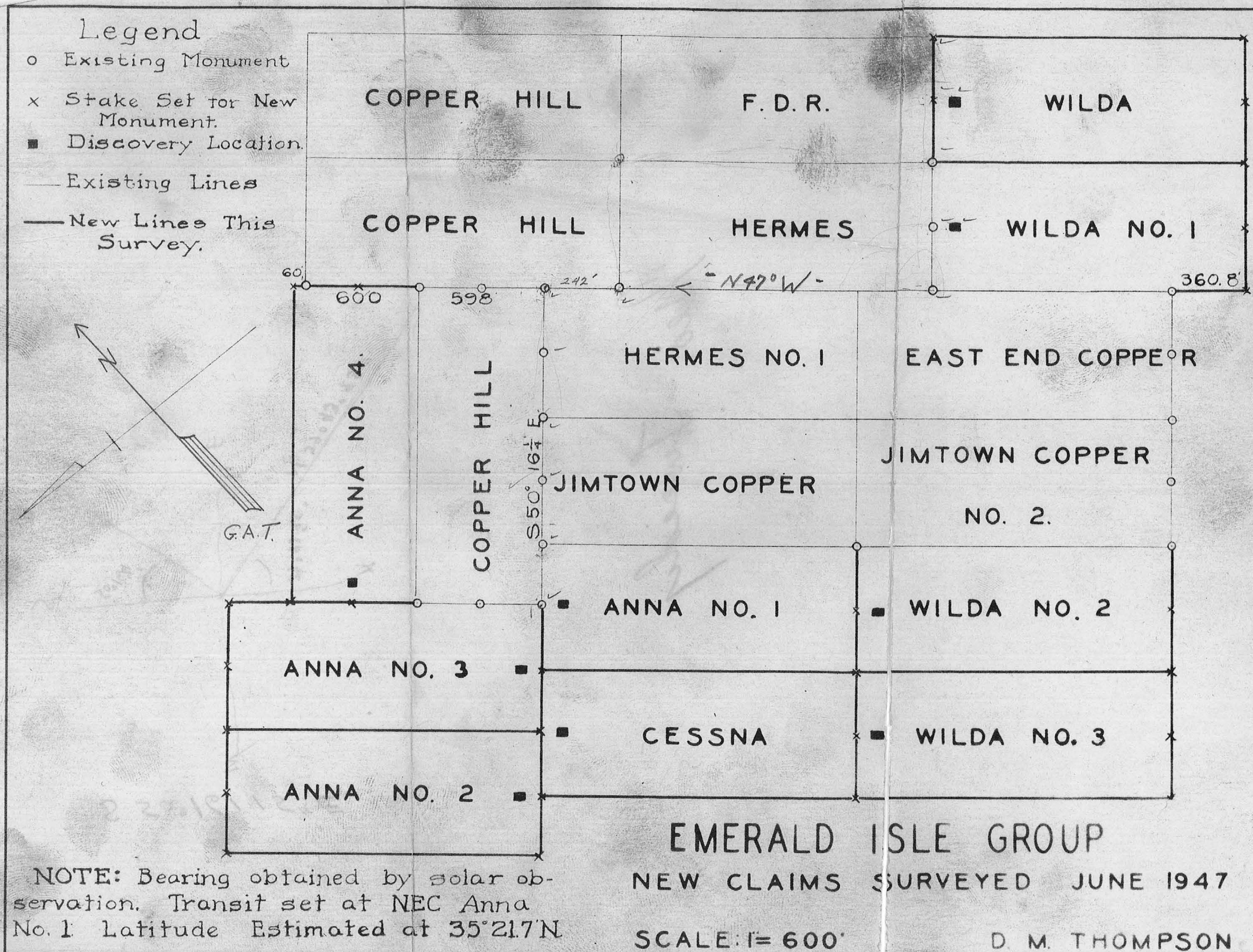
LEWIN MATHES MINING CO.

EMERALD ISLE DIV.

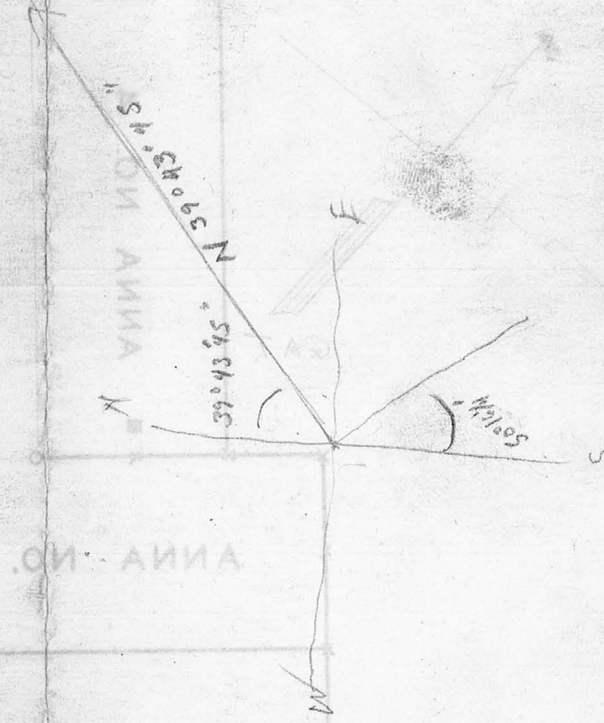
MOHAVE CO. ARIZONA

SCALE 1"=20' NOV. 1, 1947.

SHEET No. 26

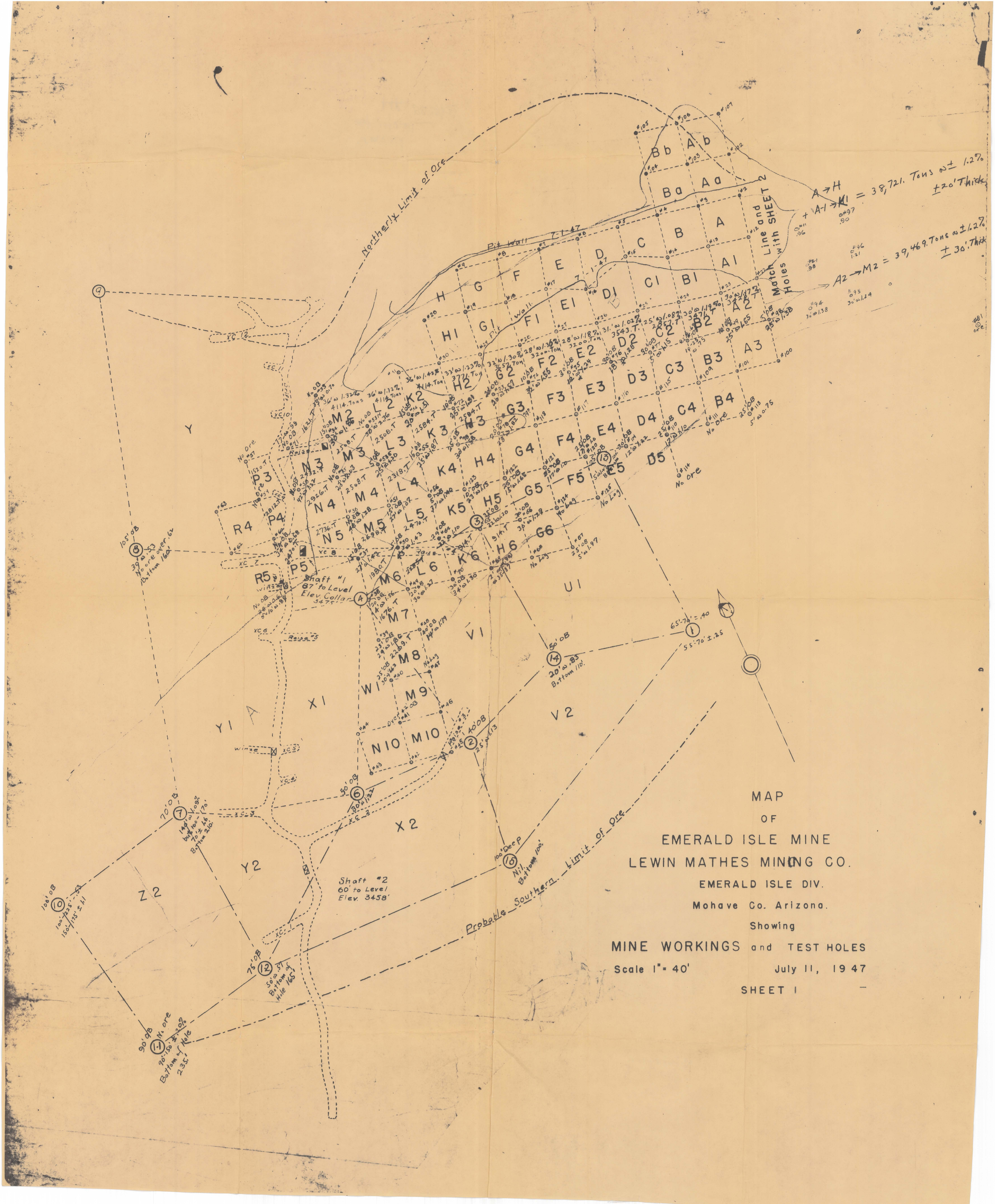

Legend
o Existing Monument
x Stake Set for New Monument.
■ Discovery Location.
Existing Lines
New Lines This Survey.
COPPER HILL
F. D. R.
WILDA
COPPER HILL
HERMES
WILDA NO. 1
60
600
598
242'
N47°W
360.8'
ANNA NO. 4.
COPPER HILL
S50° 16¼ E
HERMES NO. 1
EAST END COPPER
JIMTOWN COPPER
JIMTOWN COPPER NO. 2.
G.A.T.
ANNA NO. 1
WILDA NO. 2
ANNA NO. 3
CESSNA
WILDA NO. 3
ANNA NO. 2
EMERALD ISLE GROUP
NEW CLAIMS SURVEYED JUNE 1947
SCALE: 1"= 600'
D. M. THOMPSON
NOTE: Bearing obtained by solar observation. Transit set at NEC Anna No. 1. Latitude Estimated at 35°21.7'N.

Geo. Tweedy

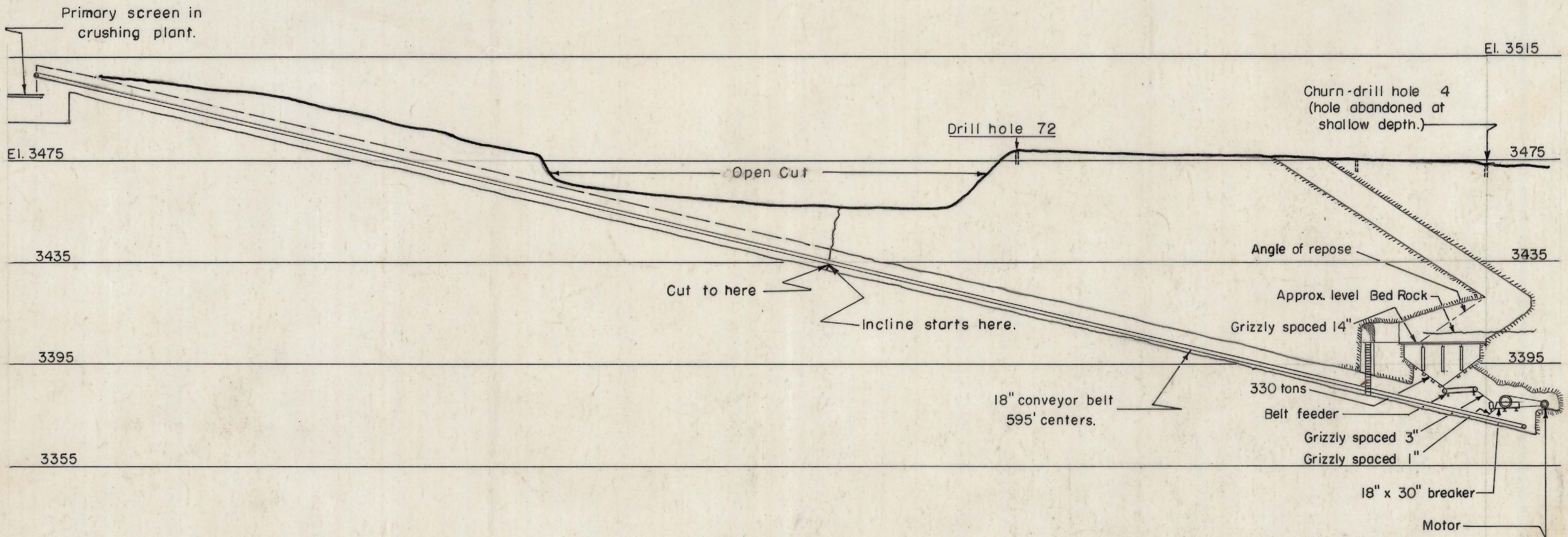


S 20°16'15" E

39°43'45"
20°16'15"



MAP
OF
EMERALD ISLE MINE
LEWIN MATHES MINING CO.
EMERALD ISLE DIV.
Mohave Co. Arizona.
Showing
MINE WORKINGS and TEST HOLES
Scale 1" = 40'
July 11, 1947
SHEET 1
A→H
A-1 → K1 = 38,721. Tons @ ± 1.2%
± 20' Thick
A2 → M2 = 39,469. Tons @ ± 1.2%
± 30' Thick
Northerly Limit of Ore
Probable Southern Limit of Ore
Pit Wall
Line and Match with SHEET 2
Holes
Shaft #1
87' to Level
Elev. Collar 3475'
Shaft #2
60' to Level
Elev. 3458'
Winze
Raise



From sketch
By: C. F. Weeks

D.M.A. 374

Scale: 1" = 40'

Figure 4 .- Proposed pocket and conveyor, Emerald Isle mine.

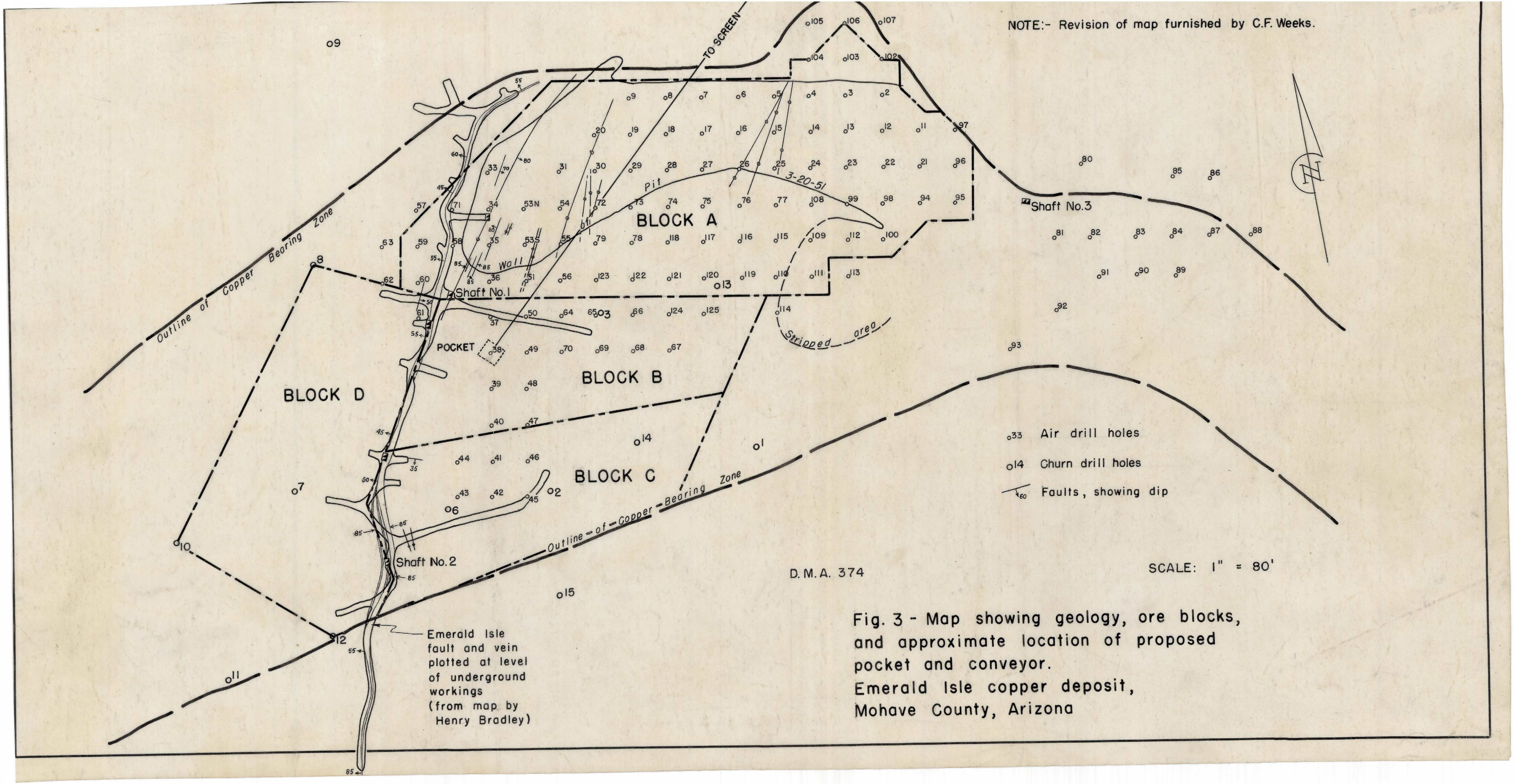
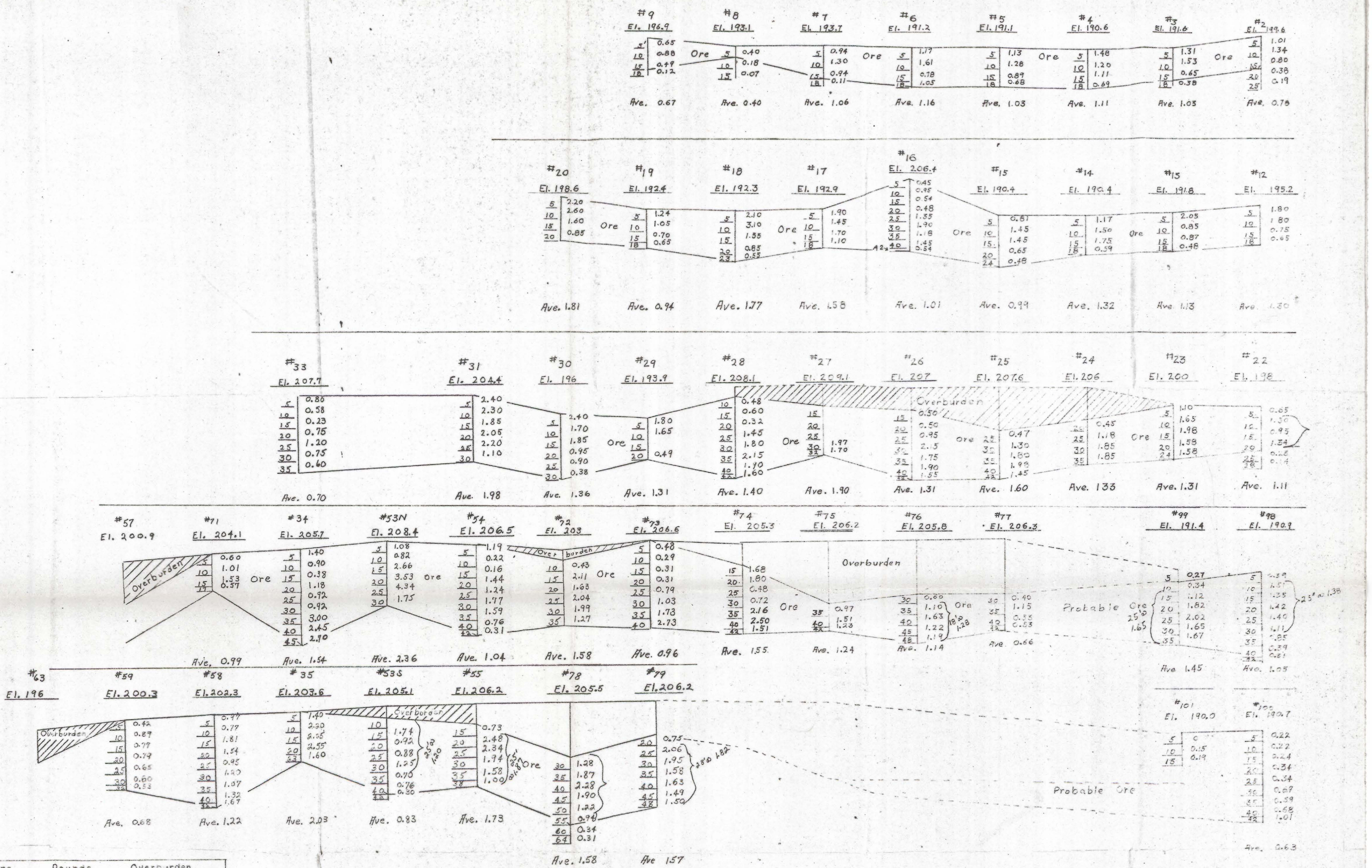


Fig. 3 - Map showing geology, ore blocks, and approximate location of proposed pocket and conveyor.
Emerald Isle copper deposit,
Mohave County, Arizona



Block Number	Average Copper %	Thickness of Ore	Tons of Ore	Pounds of Copper	Overburden Thickness	Overburden Tons
A	1.06	18.5	2114	44,816	0	
B	1.147	18	2057	47,187	0	
C	1.11	19.5	2228	49,461	0	
D	1.05	25	2857	59,997	0	
E	1.20	23	2628	63,072	0	
F	1.20	15	1714	41,136	0	
G	0.945	15	1714	32,394	0	
H	1.14	17.7	1011	23,050	0	
A 1	1.21	20	2286	55,321	0	
B 1	1.27	20	2286	58,064	4	34
C 1	1.31	21	2400	62,880	9	685
D 1	1.23	23.5	2685	66,051	8	610
E 1	1.45	28.5	3257	94,453	5	381
F 1	1.66	24.7	2822	93,690	4	34
G 1	1.35	23	2628	70,956	0	
H 1	1.35	22	2514	67,878	0	
K 1	1.72	26.6	1520	52,288	0	

38,721 Tons
768

SHEET No. "A"

ASSAY MAP

EAST WEST SECTIONS of TEST HOLES

EMERALD ISLE MINE

LEWIN MATHES MINING CO.

EMERALD ISLE DIV.

Mohove Co. Arizona.

Scale 1" = 20'   July 11, 1947.