UNITED STATES
DEPARTMENT OF THE INTERIOR
DEFENSE MINERALS ADMINISTRATION
WASHINGTON 25, D. C.

224 New Customhouse
Denver 2, Colorado

Air Mail

October 4, 1951

Memorandum

To: Executive Secretary, Defense Minerals Administration

From: DMA Field Team, Region IV

Subject: Final Report, DMA Docket 374, C. W. Mains, Applicant, Emerald Isle Copper Deposit, Mohave County, Arizona

Enclosed herewith are the original and two copies of the report on the subject docket.

We concur with the conclusions and recommendations of the examining team that the development loan of approximately $600,000 be denied.

The brochure is enclosed.

J. H. East, Jr.

for Ogden Tweet
A. H. Koschmann

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

Mining Division
Region IV

224 NEW CUSTOMHOUSE
DENVER 2, COLORADO

October 8, 1951

Memorandum

To: DMA Field Team, Region IV

From: Chief, Mining Division, Region IV

Subject: Final Report, DMA Docket 374, C. W. Weeks, Applicant, Emerald Isle Copper Deposit, Mohave County, Arizona

The applicant of the subject docket submitted an elaborate plan to produce cement copper and copper sulphate by leaching chrysocolla and melanconite copper minerals occurring in a conglomerate resting on a granite pediment. This plan included an acid plant; rehabilitation of the Tennessee mill for the production of pyrite and lead zinc concentrates, and asponge iron plant. Details on the costs of the various phases of the project were not submitted, but the estimated total cost was $600,000.

As the proposal appeared to have some merit, detailed examinations were made by the U. S. Geological Survey and by the Mining and Metallurgical Divisions of the U. S. Bureau of Mines. Their conclusions are that supplemental drilling would be required to accurately determine the tonnage and grade of indicated reserves and to obtain representative metallurgical samples. Several phases of the proposal such as the manufacture of blue stone and purchase and rehabilitation of the Tennessee mill are not considered feasible.

The proposed production plan of the applicant is not attractive enough to warrant asking the applicant to submit a more fully integrated proposal. Production costs for cement copper at the Emerald Isle Mine would require accelerated amortization and a purchasing plan for the entire output for five years in order to insure repayment of the loan.

The only recommendation made is that consideration should be given to a revised proposal for rehabilitation and enlargement of the present plant with an adjunctive acid-leaching plant should the copper situation become more critical. We concur with this recommendation.

Paul T. Allsman

P. T. Allsman

Mining Division
Date Rec'd.
SEP 27 1951
BUREAU OF MINES
TUCSON, ARIZONA
Denver, Colorado

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
MINING DIVISION
REGION IV

Arizona-New Mexico Branch
[redacted]
Box 4097
UNIVERSITY STATION

September 25, 1951

Memorandum

To : Paul T. Allsman, Denver, Colorado

From : Chief, Arizona-New Mexico Br., Mining Div., Region IV
Tucson, Ariz.

Subject : Final Report, DMA Docket 374, Emerald Isle Copper Deposit, Mohave County, Ariz.

Enclosed are original and 7 copies of the Engineering Section by T. M. Romslo and W. T. Griswold of the Bureau of Mines, for the above report. The sections on Summary, Conclusions and Recommendations, and on Metallurgy will be submitted by J. B. Clemmer of the Bureau of Mines. The section on Geology will be submitted by A. F. Shride of the Geological Survey.

This report has my approval.

Enclosed is brochure.

Walter R. Storms

Walter R. Storms

CC P. T. Allsman (7)
C. A. Anderson
374
DF

FOR VICTORY
BUY
UNITED STATES SAVINGS BONDS AND STAMPS

UNITED STATES
DEPARTMENT OF THE INTERIOR
DEFENSE MINERALS ADMINISTRATION
WASHINGTON 25, D. C.

200 New Customhouse
Denver 2, Colorado

September 28, 1951

Memorandum

To: DMA Field Team, Region IV

From: A. H. Koschmann

Subject: DMA Docket 374, Emerald Isle Copper Deposit, Mohave County, Arizona

Enclosed are the original and 7 copies of the geologic report by A. F. Shride of the U. S. Geological Survey covering the above docket.

A. H. Koschmann
Supervising Geologist
Colorado-Wyoming

Enclosures (8)

UNITED STATES
DEPARTMENT OF THE INTERIOR
GEOLOGICAL SURVEY
P.O.Box 4126, University Station
Tucson, Arizona

September 25, 1951

Memorandum

To: Field Team, Region IV, Denver, Colorado

From: A. F. Shride, Tucson, Arizona

Subject: Geologic Report, DMA 374, Emerald Isle Copper Deposit, Mohave County, Arizona

Enclosed are the original and 7 copies of the above report by A. F. Shride of the Geological Survey. The Mining and Metallurgical Divisions of the Bureau of Mines will also submit reports.

A. F. Shride

A. F. Shride
Acting District Supervisor
Arizona-New Mexico

Enclosures 8

SEP 1951
RECEIVED
U.S. Geological Survey
Mineral Deposits Branch
Denver

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

SOUTHWEST EXPERIMENT STATION
Box 4097
UNIVERSITY STATION

TUCSON, ARIZONA

September 27, 1951

REGIONAL DIRECTOR'S
OFFICE
REGION IV
OCT 1 1951
BUREAU OF MINES
DENVER, COLORADO

Memorandum

Registered Mail

To : J.H. East, Jr., Denver, Colo.

From : J. Bruce Clemmer, Tucson, Ariz.

Subject : DMA-374, Emerald Isle

Transmitted herewith are original and seven copies of metallurgical report on Emerald Isle (DMA-374) together with summary, conclusions and recommendations.

J. Bruce Clemmer

Enc:

cc: JHEast, Jr. (7)
SRZimmerley
PTAllsman
CAAnderson
DMA-374

UNITED STATES
DEPARTMENT OF THE INTERIOR
OSCAR L. CHAPMAN, SECRETARY

DEFENSE MINERALS ADMINISTRATION

REPORT OF EXAMINATION BY FIELD TEAM
REGION IV

DMA-374, Emerald Isle Copper Deposit

Mohave County, Ariz.

A. F. Shride, Geologist
U. S. Geological Survey

T. M. Romslo, Mining Engineer
U. S. Bureau of Mines

J. B. Clemmer, Metallurgist
U. S. Bureau of Mines

F. N. Bender, Metallurgist
U. S. Bureau of Mines

September 1951

SUMMARY, CONCLUSIONS AND RECOMMENDATIONS

By A. F. Shride[1], T. M. Romslo[2], J. H. Clemmer[3], and F. N. Bender[3]

C. F. Weeks submitted to the Defense Minerals Administration an informal proposal for rehabilitating and enlarging the hydrometallurgical plant at the Emerald Isle mine. Because of its several novel features, the proposed plan appeared to have considerable merit and a field study was requested to determine whether Mr. Weeks should be encouraged to go to the expense of obtaining the detailed engineering costs and presenting the more complete description of the project needed to support a formal loan application and certificate of necessity.

The Emerald Isle copper deposit is at the base of a weak to moderately cemented conglomerate, that veneers a pediment surface on decomposed granite. Chrysocolla and malaconite are disseminated through the matrix of the conglomerate and are deposited as fillings in northwest-trending fissures and in voids in the conglomerate. The deposit is blanket-like in form and varies from 5 to 50 feet in thickness over a known area of approximately 7 acres. Conglomerate barren of, or essentially barren of, copper covers most of the deposit. This overburden ranges in thickness from a few inches to as much as 150 feet.

Total reserves, outlined by past drilling, are estimated as 623,800 tons of indicated ore averaging 1.12 percent total copper (13,990,000 pounds of total copper). The average ratio of overburden

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1/ Geologist, U. S. Geological Survey
2/ Mining Engineer, U. S. Bureau of Mines
3/ Metallurgists, U. S. Bureau of Mines

to ore is 0.67 yards to 1 ton of ore for the known area of the deposit. Not all of the copper in the ore is acid-soluble, but it is believed that acid-soluble copper reserves are of the above order of magnitude. For some areas the reserve estimates are more reliable than for other areas. Further drilling in the area already blocked out might indicate that reserves as estimated should be increased or decreased somewhat, but the change in reserve estimates would likely be small and would not significantly change the basis for the present appraisal.

Some supplemental drilling of the deposit should be done to collect data for use in considering a metallurgical plan. Before a mining operation is considered additional drilling also should be done to better determine the tonnage and grade of ore available in the southern and western portions of the deposit.

The deposit can be mined at a rate of 500 tons daily by Mr. Weeks' proposed open-pit method, at an estimated cost of $0.70 a ton of ore. This cost includes preparatory work and the equipping of the mine with a conveyor, primary crushing plant, and the necessary mining equipment.

The proposed metallurgical plan is for a fully integrated operation at the property involving the following features:

1. Heap, tank, and counter-current leaching of the crushed and screen-sized ore;

2. Production of a major part of the copper as blue stone and the remainder as cement copper to increase the operating return;

3. Manufacture of sulfuric acid for leaching by catalytic oxidation of sulfur dioxide from roasting pyrite recovered from local pyritic ores and tailings;

4. Manufacture of sponge iron for copper cementation from the pyritic roaster calcines, and

5. The purchase and operation of the Tennessee mill to recover pyrite from the tailing pond and from custom milling of pyritic ores of the Wallapai district to supply the sulfur and iron needed in steps 3 and 4.

Several of the features were described in considerable detail in the proposal, but others were only briefly mentioned. As success of the production plan was largely predicated upon making blue vitrol from the leach liquors, metallurgical tests seemed warranted to confirm and extend the findings reported in the proposal.

Leaching tests were made on samples of open-pit and underground ore taken during examination of the mine. The open-pit ore was readily amenable to leaching but the underground sample was refractory. Leaching the open-pit ore extracted 86 percent of the copper with an acid consumption of 2.86 pounds per pound of copper; the underground sample gave a 76 percent extraction and the acid consumption was more than 9 pounds per pound of copper. While these samples may not be representative ores, the tests suggest that ores from various parts of the deposit may differ greatly in their leaching deportment.

Laboratory tests also were made to determine the grade and yield of blue stone from primary and secondary crystallization of pregnant liquors from leaching the open-pit ore. Blue stone for the West Coast market should have a purity of 99.5 percent and contain not more than 0.1 percent zinc or iron and only a trace of arsenic, manganese, lead and other heavy metals. Although crystals of this purity were not produced, a double crystallization gave blue stone of near marketable grade; about 49 percent of the total copper in the sample of open-pit ore was recovered as blue stone acceptable as to copper and iron but containing 0.15 percent zinc. The crystallizer rejects and wash solutions from the ore which would be sent to cementation accounted for another 36 percent of the total copper. The production of high purity blue stone is much more involved than the proposal would indicate. The evaporation requirements were determined and estimates were made of the operating costs. Assuming that average ore would respond in a similar fashion to the open-pit material, the increase in operating profit over that of making cement copper is not enough to justify the more involved and highly experimental procedure.

The following conclusions were reached regarding the overall extraction plan and the separate features of the proposal:

1. The extraction plan as proposed is much too involved for a 500-ton per day operation. Several features in the proposal are not practical nor feasible at the present price of copper.

2. Although production of blue stone from impure leach solutions is metallurgically possible, installation of a crystallizing plant of the size required to process half or more of the copper is not warranted by present information. The process should be tried in a small pilot operation before venturing into commercial production.

3. The proposal to make acid from roasting pyrite is feasible and practical.

4. Leaching costs can be materially decreased by making hydrometallurgical acid at the plant. The pyrite can be purchased from other Arizona plants cheaper than it can be made at the Tennessee mill.

5. Manufacture of sponge iron for use as a copper precipitant from roaster calcines is not feasible. The material is a poor cementation agent.

6. Purchase and operation of the Tennessee mill to retreat tailings and custom mill pyritic ores of the Wallapai district is not feasible at the present price of lead and zinc.

The production plan as presently proposed is not attractive enough to warrant asking Mr. Weeks to spend additional time or money to develop a more fully integrated proposal. It is recommended that the application be denied.

Should the copper situation become critical, however, a revised proposal for rehabilitating and enlarging the present plant for leaching and cementation of copper from the ores, with an adjunctive acid-

making unit to supply plant needs, would merit careful consideration. As the production costs for cement copper at Emerald Isle would be high, accelerated amortization and a purchasing plan for the plant output for a 5-year period would be necessary to insure repayment of a loan.

EMERALD ISLE COPPER DEPOSIT
MOHAVE COUNTY, ARIZONA
DM# 374

Geology

By A. F. Shride
Geologist, U. S. Geological Survey

EMERALD ISLE COPPER DEPOSIT
MOHAVE COUNTY, ARIZONA
DMR 374

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Fig. 3 Map showing geology, ore blocks and approximate location of proposed pocket and conveyer, Emerald Isle copper deposit.	1

GENERAL GEOLOGY

The Emerald Isle copper deposit is in moderately consolidated alluvium, which appears to represent the fluviatile deposits that form a thin "skin" on a pediment surface. Topographically, the deposit exists as a small hummock in a low drainage basin that is in the line of junction between two alluvial fans. Therefore, if other copper deposits of the Emerald Isle type exist in the general vicinity, they would be covered by recent fanglomerates.

The bedrock in the vicinity of the deposit was seen in but three small areas and is medium to moderately coarse-grained granite which contains in one outcrop numerous quartz veinlets that contain limonite pseudomorphous after pyrite. The conglomerates that conceal the bedrock were laid down on a surface of small relief. C. F. Weeks, the applicant, reports that drilling indicated that there are numerous "knobs" and "hollows" on this surface. Such features he considers to be on the order of 20-30 feet apart and to have a relief of 15-20 feet. Probably the topographic expression is not as irregular in detail as visualized by Mr. Weeks; his idea probably results from the fact that a distinction between conglomerate and the underlying decomposed granite was not carefully made in logging the drill holes. Some relief of the conglomerate-bedrock surface is likely due to post-conglomerate faulting.

The alluvium, in the general vicinity of the deposit, varies from a few feet thick, just north of the present pit, to at least 210 feet thick in the area explored by churn drill hole 10 (see west end of map, fig. 3). From all available information, it seems likely that the alluvium thins over an approximately east-west bedrock ridge, the crest

of which is a few tens to perhaps 150 feet north of the northernmost line of wagon-drill holes (holes 105-107, fig. 3). There is some suggestion that the alluvium also thins toward the south, in the vicinity of and south of the line shown on the map as the southern limit of the copper bearing zone. Thus it has been suggested that the copper deposit is localized in a west-trending alluvium-filled valley. There is too little evidence to infer that this is the case.

The copper-bearing alluvium is exposed for examination only in the pit and the southern 200 feet of underground workings. The alluvium is composed of beds of poor to moderately well cemented conglomerate. Most of the conglomerate beds range from 6 inches to 3 feet in thickness, but some beds may be as much as 15 feet thick. No complete exposures of the thicker beds could be seen. Some beds exhibit better sorting than others but the materials of all beds are only very crudely sorted. Some beds are lenticular, and several show large-scale cross-lamination. Individual beds could be traced only approximately for as much as 100 feet laterally due to discontinuity of outcrop; possibly faulting in covered areas also makes correlation difficult. The pebbles of the conglomerate ranged in diameter from a granule size to boulders as much as 6 feet in diameter. Pebbles are angular to moderately rounded; due to foliation or lines of weakness along fractures few approach sphericity. The pebbles are dominately of a coarse-grained granite, a fine-grained granite, or diorite; some of the diorite contains much hornblende. Quartz and a fine-grained gneiss, some of which contains a high content of mafic minerals occur in moderate amounts as pebbles; minor pebble constituents were a fresh-appearing diabase, quartzite, and basalt or basaltic andesite.

In general, only the granite, the gneiss, and some of the diorite pebbles show much decomposition; these are the only pebble constituents likely on milling to furnish much fines to the leach plant feed.

Some conglomerate beds have a dominately sandy matrix, others dominately a silty clay matrix. Those beds with a matrix of high clay content contribute enough fines to the general run of ore to cause a serious problem in leaching the ore.

STRUCTURE

The conglomerate beds are sensibly horizontal. The only structures observed in the area were northeast-striking vertical, or nearly vertical, faults which are post-alluvium in age. These faults undulate in strike (see fig. 3), and with the exception of the main fault (hereafter referred to as the Emerald Isle fault, see fig. 3), which the underground workings follow, individual faults may not persist along strike for more than 200 feet. The throw of most of the faults is probably on the order of inches; some faults may have a throw of a few feet, but with the possible exception of the Emerald Isle fault, probably none have a throw on the order of tens of feet. Exposures are such that the throw of the larger faults cannot be accurately determined. Some conglomerate beds were well enough consolidated that boulders as much as 4 feet in diameter were faulted through.

The conglomerate-granite contact is said to dip steeply to the west in the vicinity of the northernmost underground workings. Whether this steepening of the contact is due to pre-conglomerate topography or to faulting is not known. A comparison of the logs of the drill holes east and west of the underground workings suggests that the Emerald Isle fault

may have a throw varying from 25 to 100 feet along the strike of the fault. If this is true, the downthrown block is the western block. Where the Emerald Isle fault contacts granite at depth, it is said to flatten and dip westward 45-55 degrees. This is said to be the attitude of the conglomerate-granite contact in part of the underground workings. The Emerald Isle fault may have granite on both walls at depth or in some areas as far as it has been followed down-dip may have a conglomerate hanging wall and a granite footwall. The descriptions available to the writer are not entirely clear on these points; exposures of the fault in granite were not accessible at the time of our examination. Whether the abrupt change in attitude of the conglomerate-granite contact is due to original topographic gradient or to faulting is of little concern. The significant point is that the copper-bearing conglomerate, just above bedrock, drops abruptly in elevation to the west and is covered by a greater thickness of overburden west of the Emerald Isle fault than east of the fault.

THE EMERALD ISLE COPPER DEPOSIT

The copper of the Emerald Isle deposit occurs as blue, green, brown, and black aphanitic material impregnating the matrix of the conglomerate and filling fractures in the conglomerate. The copper-bearing conglomerate is essentially confined to a zone 5 to 50 feet thick in the conglomerate immediately above the granite bedrock. Churn-drill hole 7 (fig. 3) is reported to penetrate a 135 foot thickness of ore; very likely this hole follows a copper-impregnated fault zone and the thickness is not typical of adjacent areas. The ore "blanket" is covered by a few inches to as much as 150 feet of barren conglomerate, hereafter referred to as "overburden."

Chrysocolla and melaconite are the copper minerals that have been identified. Dioptase has been reported 1/ as constituting part of the ore. No sulfides are known to exist or likely to exist in the deposit. The five check samples taken in this examination contain a fraction of one percent zinc (see metallurgical report); the zinc mineral was not identified. Some of the melaconite occurs as small inclusions in opaline silica and as grains coated by opaline silica. Therefore a part of the copper is not acid soluble.

Theories of the origin for the Emerald Isle deposit have been described recently by Searles 2/ and Thomas. 3/ Searles' postulation that the copper of the Emerald Isle deposit was derived from nearby disseminated cupriferous pyrite in granite seems the most feasible explanation of a copper source.

Obviously fracture filling has been an important factor in localizing this deposit. The greatest concentrations of chrysocolla are in the Emerald Isle fault and in a fractured zone adjacent to this fault; melaconite is prevalent in this portion of the deposit but not elsewhere in the deposit. Throughout the pit concentrations of chrysocolla can be observed in and near fractures. Chrysocolla-filled fractures do extend from the conglomerate into the granite bedrock. Such veins are reported to decrease in width and number a short distance below the conglomerate-granite contact. Therefore, little of the upper, decomposed portion of the granite will be mined as ore.

1/ Thomas, B. E., Ore deposits of the Wallapai district, Arizona: Econ. Geology, vol. 44, p. 701.
2/ Searles, Fred, Jr., The Emerald Isle Copper Deposit; Econ. Geology, vol. 45, pp. 175-176, 1950.
3/ Thomas, B. E., op. cit., pp. 700-703, 1949.

In general, no one conglomerate bed seems more favorable to chrysocolla deposition than others. However, locally a certain bed or lense definitely has a higher copper content than adjacent conglomerates. The beds, laminae, or lenses that have a sandy matrix were more favorable to chrysocolla deposition than those that have a silty or clayey matrix. Some planes between beds were very permeable and relatively high copper content layers of chrysocolla were deposited along these planes for a few tens of feet laterally. Conglomerate beds in which the material has been better sorted than average, and that in general contain pebbles less than 3 inches maximum diameter, usually contain the most copper. In places where numerous closely spaced fractures exist, the type of conglomerate has little bearing on copper content.

Conglomerates which include numerous large boulders tend to be low in copper content even though the matrix is highly mineralized. Dilution of the ore by the barren boulders is, of course, the cause of low general copper content in such beds. Some hand sorting to eliminate the large barren boulders in such conglomerates has been practiced and will probably be practiced to a certain degree in future mining. Favorable beds and unfavorable beds and fault zones are so distributed that selective mining of certain beds would not be feasible.

Chrysocolla impregnates the outside decomposed portion of the pebbles as much as 1 inch, for most pebbles the depth of chrysocolla penetration is less than ½ inch. Therefore, the reaction time on leaching should be fairly fast and recovery high on the crushed rock portion of the ore. Most commonly, only the gneiss, granite, and diorite pebbles are rimmed with a decomposed layer impregnated by chrysocolla. In a general

way the relative grades of ore can be visually estimated from the appearance of the ore. Beds which are poorly consolidated are usually low in copper content; these beds and the overburden are usually brownish in color as viewed from a distance. Grayish to blackish portions of the mineralized conglomerate are relatively high in copper content; the blackish material is well cemented. The "hard" ore breaks on primary blasting into blocks as much as 8 feet in diameter and requires secondary blasting.

The southwest face of the present pit is an exposure of highly fractured conglomerate that has a general blackish cast regardless of the character of the conglomerate. This ore is reported by the applicant to be relatively high in copper and the best samples collected by the examiners are ore of this type. The blackish coloration is due at least in part to the malachite content; manganese oxides are possibly also more abundant here.

Samples from the underground workings which follow the Emerald Isle vein that occupies the Emerald Isle fault assay as much as 8½ percent copper over a sample length of 4 feet. Approximately 14,400 tons of 5.63 percent copper ore are reported to have been mined from the Emerald Isle vein. However, this zone accounts for but a small part of the total tonnage in the ore deposit, and ore from this zone will not increase appreciably the general tenor of the ore. The average grade of that part of the conglomerate presumed to be ore from past exploration is 1.12 percent.

The matrix of the conglomerate contains most of the copper content of the ore. The matrix will also furnish most of the fines on milling. It should be noted that an appreciable percentage of the

copper content of the ore will be concentrated in these fines. The applicant states that past experience has shown that the fines contain 2½ to 3½ percent copper.

The applicant states that as the tenor of the ore decreases to the order of 0.5 percent copper, the lime content increases abruptly. This was confirmed in testing the various grades of ore with acid in the field; the hard higher grade ores effervesced only slightly or not at all, some of the soft low-grade ores and overburden effervesced vigorously. Therefore, care in mining to prevent dilution of the better ore by low grade ore or overburden is indicated as preferrable in order to decrease acid consumption in the leaching operation.

The ore zone is exposed at the surface in many places in the vicinity of the present pit. Elsewhere, the mineralized zone is in general concealed by unmineralized, poorly cemented conglomerate. There is no sharp line of demarkation between the mineralized and unmineralized conglomerates. In faulted areas the mineralized zone extends to higher horizons stratigraphically than in unfaulted areas. From the exposures seen in the pit it does not seem likely that the barren conglomerate overburden represents a covering of alluvium on top of previously mineralized conglomerate. However, it is possible that where the overburden is thickest, some of the overburden may represent post-mineral deposits.

The overburden near shaft No. 1 and east of the Emerald Isle fault is thin. To the south the amount of overburden increases so that in the vicinity of churn drill holes 2 and 6 (fig. 3), the overburden is 40-50 feet thick. West of the Emerald Isle fault the overburden

apparently thickens abruptly so that in churn drill holes 7, 8, 10, and 12 the thickness of overburden is 75, 105, 150, and 85 feet respectively.

In general, the overburden is loosely cemented; all of that overburden exposed in the present pit can probably be stripped without blasting. How well the thick overburden west of the Emerald Isle fault is cemented is not known; presumably that overburden is also poorly cemented at depth.

ORE RESERVES

Ore reserves of the Emerald Isle deposit are calculated on the basis of assay data made available by the applicant. Assays of large samples taken in the pit area checked reasonably well the data furnished by the applicant. Assays of samples taken underground did not agree with previous sampling data. However, as pointed out in the application, channel sampling of the material underground by different examiners is likely to give greatly varying results due to the nature of the material sampled. The present examiners believe that the assay data furnished are reliable data on which to base total copper content calculations and tonnage of ore.

Drill samples were taken by wagon drill and churn drill methods. Assay data for Block A (see fig. 3) are much more complete for the other blocks; there is less complete data for Block B. Calculations for Block C are based on only 3 churn drill holes; the wagon-drill holes in Block C did not penetrate through overburden to the ore zone. Reserve calculations for Block D are based on 4 churn drill holes which are so widely spaced and in which the ore varies in grade and thickness so much that these samples cannot be considered as necessarily representative of Block D.

All of the applicant's assay data record total copper and do not record the percentage of acid-soluble copper. Because copper will be extracted from this ore by acid-leaching methods, only acid-soluble copper should be considered in calculating reserves (further note of this factor will be made later). Because reserves can be calculated only on the basis of total copper, this writer considers the tonnage estimates for Block A to be the equivalent of "measured" tonnage but that estimates of grade and therefore copper content are more nearly the equivalent of data usually considered of "indicated" reliability. It will be noted (fig. 3) that the northwest corner of Block A is outside the sampled area. However, the writer is confident, on the basis of visual inspection and geologic probability, that averages of grade and depth of ore can be extended to include this unsampled area. Block B can be considered reasonably well "indicated" ore; Block C is "indicated" ore but estimates are not as reliable as for Block B; Block D, because of the nature of data available, should be considered on the borderline between "indicated" and "inferred" reserves. A minimum area has been included in the block designated "D" because the data are probably not representative. For the above reasons all reserves are considered "indicated" reserves but of different degrees of reliability.

The following table is a summation of reserves by blocks as shown on figure 3. Because overburden is a factor to be considered in estimating mining costs, the estimates of overburden are included.

	Block A	Block B	Block C	Block D	Total
Tons of ore	156,200	74,400	117,000	276,200	623,800
Pounds total copper	4,170,000	2,100,000	2,390,000	5,380,000	13,990,000
Average grade of ore	1.36	1.41	1.02	0.96	1.12
Yards of overburden	36,800	38,400	86,300	259,000	420,500
Yards of overburden/ton of ore	0.236	0.808	1.666	0.938	0.67

There is too little information to estimate what percentage of the above copper is acid-soluble. In one large sample of hard ore taken from the pit 94 percent of the copper was acid-soluble; in another sample of hard ore, nearly all of the copper was soluble. In one sample of soft ore as little as 22 percent of the copper was soluble. Meager data of this sort suggests that a very rough estimate of the ratio of acid-soluble copper to total copper could be made, if the ratios of hard to soft ore could be estimated. From present exposures too little data could be collected to make such an estimate. Lithologic logs do not accompany the drill data. The writer would guess from exposures seen that not less than 75 percent of the total copper content is soluble, and a larger percentage of the copper is probably as acid-soluble copper.

It should be emphasized that sampling to determine a ratio representative of the entire acid-soluble copper to total copper was not possible in this examination. Data obtained by drilling would be necessary for such a purpose. Therefore, the reserve estimates given in the above tables represent the best data available. The estimate of total copper likely represents the order of magnitude for acid-soluble copper.

The applicant furnished a detailed breakdown of his method of determining reserves. In general, the writer's estimates check closely those made by the applicant. For the area included in Blocks A, B, C, D,

the writer's estimates are approximately 90,000 tons (approximately 2,500,000 pounds of total copper) lower than those of the applicant. This discrepancy occurs because the writer believes it prudent not to allow certain churn drill data that shows unusual thicknesses of ore to influence too large an area in making reserve estimates. It is possible that a hole put down a fault zone would sample material that would represent but a small part of the mineralized material that is adjacent to that hole.

Many of the wagon drill holes "bottom" in ore. Therefore, the applicant added in Block A an additional 53,000 tons of "probable" ore averaging 1.47 percent copper. Such a tonnage may be reasonable but there is no basis for estimating grade, and 1.47 percent copper is a higher grade than the general run of the ore. Therefore, the writer has not included such ore in his estimates, but he does agree that some ore probably exists beyond the depth to which drilling was done. It should be pointed out that the applicant has not claimed similar "probable" ore estimates in Block B; therefore his estimate of inferred ore for the area included on figure 3 is probably conservative. The applicant infers an additional 16,000 tons of ore averaging 1.06 percent copper in the area outlined roughly by wagon-drill holes 100, 95, 81, and 87 (see east side of fig. 3). From the data available the writer does not believe that the copper-bearing materials in this area should be classified as ore. Further drilling to greater depths would be necessary before it can be inferred that minable ore exists in this area.

It will be noted that the grade of ore in Blocks C and D is considerably lower than that in Blocks A and B. Estimates of tonnage and grade in these areas are based on but a few drill holes, which may not

be representative. It is possible that if Blocks C and D were drilled more thoroughly, the grade of the ore would be found to be comparable to that in Blocks A and B. Many of the drill holes in the area encompassed by Blocks A, B, C, D, bottom in ore. Drilling to the bedrock horizon might prove additional comparatively small tonnages in each of Blocks A and B. There is too little data to suggest whether or not deeper drilling would add appreciably to the tonnages of ore in Blocks C and D. For Block B the assay data is so erratic that additional drilling might prove that less ore exists than the 276,000 tons estimated in this report.

The applicant believes that ore is likely to exist east and southwest of the presently drilled area. That faults have influenced chrysocolla deposition is certain. Assay data suggest that the grade of the ore decreases away from the Emerald Isle and subsidiary faults. There are no exposures to suggest that faulting exists at the extreme southwest and east edges of the presently drilled area. If faults do not exist, it is likely that the ore will grade out into material of too low a value to be mined; vice versa, if faulting does exist beyond presently explored ground, more ore may exist. However, without confirmatory data to suggest the existence of ore in the areas as yet unexplored, the writer prefers not to infer that ore may exist "upstream" (to the east) "300, 500, or possibly 800 feet," or that "we have 200,000 tons per hundred feet of downstream continuation" (to the southwest) as Mr. Weeks infers in his application.

In summary the writer believes that the tonnages of ore estimated by the applicant are of the correct order of magnitude; however, his over-all grade estimate may be somewhat high. The average grade of ore

estimated by this writer is 1.12 percent copper; the applicant estimates the average grade to be 1½ percent. The writer also believes that without further exploration work it would be risky to infer reserves in an eastward and/or a southwestward direction from the area at present explored.

CONCLUSIONS

The feasibility of this proposal is dependent on economic factors and will be discussed in the engineering and metallurgical portions of the report. It should be noted, however, that if government participation in this project appears feasible, some supplementary drilling should be done over the entire presently explored area in order to determine the percentage of copper that is acid soluble and to determine the zinc content and the lime content for use in metallurgical appraisals. Additional drilling should also be done in the areas designated C and D on figure 3 to define better the grade and tonnage of ore to be expected in these areas.

UNITED STATES
DEPARTMENT OF THE INTERIOR
OSCAR L. CHAPMAN, SECRETARY

DEFENSE MINERALS ADMINISTRATION

REPORT OF EXAMINATION BY FIELD TEAM
REGION IV

DMA-374 - Emerald Isle Copper Deposit

Mohave County, Ariz.

T. M. Romslo, Mining Engineer
U. S. Bureau of Mines

W. T. Griswold, Mining Engineer
U. S. Bureau of Mines

September 1951

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EMERALD ISLE COPPER DEPOSIT
MOHAVE COUNTY, ARIZ.
DMA DOCKET 374

Engineering Report

By T. M. Rumsle
Mining Engineer
U. S. Bureau of Mines

W. T. Griswold
Mining Engineer
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INTRODUCTION

This report covers an examination, from the mining aspect, of the Emerald Isle Copper property in the Wallapai Mining District, Mohave County, Ariz. The examination was made by a Defense Minerals Administration team in connection with a request by C. F. Weeks of the Emerald Isle Copper Co. for Governmental assistance under the Defense Production Act of 1950. The applicant's request carries Docket Number DMA 374.

Weeks submitted neither a formal loan application nor an application for a certificate of necessity for accelerated amortization. However, he did submit rather complete reports on the property and on a proposed method of treating low-grade copper oxide ore from the Emerald Isle mine to produce copper sulfate and cement-copper. He stated a firm estimate of the cost of the project, tentatively estimated at $600,000, would be prepared by reputable contractors upon recommendation of the Defense Minerals Administration. The Copper Commodity Committee and Operating Committee of the Defense Minerals Administration recommended serious consideration of the project as an economical venture and requested that necessary studies be made to determine if Weeks should be encouraged to go to the expense of obtaining detailed cost estimates and preparing a complete description of the proposed project.

Separate reports have been prepared on the geologic, mining, and metallurgical aspects of the proposed project. This report contains estimates of the ore reserves, cost of mining, and information regarding

costs and availability of water and power. These data are made available for possible consideration in a metallurgical report that will present conclusions concerning the feasibility of the overall project.

Field examinations were made March 20 to 22, 1951, by A. F. Shride, Geological Survey, and T. M. Romalo, Bureau of Mines. On March 22, 1951, these engineers accompanied C. F. Weeks, and J. Bruce Clemmer, Bureau of Mines, on an inspection trip of the company's mine and plant, and to the tailings dump at the Tennessee-Schuylkill mine, near Chloride, Ariz.

LOCATION AND ACCESSIBILITY

The Emerald Isle mine is about 13 air-miles northwesterly from Kingman, Ariz., the county-seat of Mohave County (Fig. 1). Kingman, the shipping point and supply center for the area, is a station on the main line of the Atchison, Topeka and Santa Fe railroad and is served by U. S. Highway 66. The village of Chloride, Ariz., is about 3½ miles north of the Emerald Isle Mine.

The mine may be reached from Kingman by traveling to the western outskirts of the town on paved U. S. Highway 66, thence 18.9 miles northwesterly on paved U. S. Highway 93, and thence 3.0 miles easterly on dirt roads to the property. The dirt roads are maintained by the county and are reported to be in good condition at all times. There are no excessive grades from Kingman to the mine on either the paved highways or dirt roads.

PHYSICAL FEATURES AND CLIMATE

The Emerald Isle property is at an average altitude of about 3,700 feet in an area of gently rolling hills on the western piedmont of the Cerbat Mountains. The Cerbat Mountains form a range about 10 miles wide that extends northward about 30 miles from Kingman, Ariz. The crestline of the range has an average altitude of about 6,000 feet, but contains a few peaks that rise to altitudes of about 7,000 feet. The range is bordered on the west by the Sacramento Valley and on the east by the Hualapai Valley. The land surface rises gradually from the Sacramento Valley to the foot of the mountains, about one-half mile east of the Emerald Isle property. On the property the range in relief is only a few tens of feet.

Drainage in the western piedmont of the Cerbat Mountains is to the southwest. Mineral Park Wash and Bismark Canyon are the principal drainages near the Emerald Isle property. A dam placed across Mineral Park Wash diverts the drainage from its normal course across the Emerald Isle property. There are no permanent streams in the area.

Vegetation on the piedmont consists of scattered brush and a sparse growth of forage grass.

The climatic conditions, characteristic of the intermediate altitudes of northern Arizona, are suitable for all-year mining operations. At Kingman the annual temperatures range from 6 to 117 degrees and the annual mean is 61.5 degrees. Precipitation is 11.1 inches annually, including 3.9 inches of snow.

HISTORY AND PRODUCTION

The Emerald Isle group of claims was located some years before 1914 and was first put into production during the period of World War I by a company headed by F. C. Jacobson of Kingman, Ariz. The company erected a 50-ton pilot plant and treated about 10,000 tons of ore by acid-leaching and electrolytic precipitation before closing down shortly after the war was terminated. A few years later a Mr. Woods took an option on the property and completed the underground workings. There are no available records on production from these operations that were probably terminated during the early 1920's.

In 1937, Cyrus F. Weeks and M. B. Lauzon of Kingman, Ariz., obtained a lease on the property. During November 1942 the property was examined for the Western-Knapp Engineering Co. by Henry Bradley. The examination included sampling of both underground and surface workings.

The next period of activity was during 1945-1946 when the Emerald Isle Copper Co. operated the property under the lease agreement obtained by Weeks and Lauzon in 1937. During this period the company produced 34,400 tons of ore. Of this total production 14,400 tons containing 5.63 percent copper was shipped to a smelter and the remainder, 20,000 tons containing 1.0 percent copper, was treated by leaching and precipitation to produce copper cement. These operations were terminated about the middle of 1946 and the property lay idle until 1947 when the Lewin-Mathes Mining Co. started operations under

a sublease agreement with Weeks and Lauzon. During 1947 and 1948 this company tested the property by drilling both churn-drill and wagon-drill holes, and produced about 46,000 tons of ore containing 1.65 percent copper. The ore also was treated by leaching and precipitation to produce cement copper. Production from this ore and the 20,000 tons treated in 1945-1946 (66,000 tons containing an average of 1.453 percent copper) yielded 1,140,000 pounds of cement copper.

PROPERTY AND OWNERSHIP

The Emerald Isle property consists of a group of 18 unpatented lode mining claims in Secs. 22 and 27, T. 23 N., R. 18 W., Gila and Salt River Base and Meridian, Wallapai Mining District, Mohave County, Ariz.

Seven of the 18 mining claims are owned by J. Bennett Downie and Edith F. Downie of Los Angeles, Calif. and have been leased to Cyrus F. Weeks and Maurice B. Lauzon, Kingman, Ariz. These seven claims are within the heavily outlined area on figure 2. A supplement to the original lease agreement states the lease is still in full force and effect and that Anna I. Lauzon has succeeded to all of the interests of her deceased husband, Maurice B. Lauzon. The original lease, dated May 17, 1937, is for a period of 99 years and calls for the payment of royalties on a sliding scale. The royalty to be paid varies from 2 percent on ore having a value of $5.00 a ton, to 50 percent on ore valued at $1,000 a ton.

The other 11 mining claims of the group were located by Cyrus F. Weeks and Maurice B. Lauzon and have been recorded in Mohave County, Ariz.

DESCRIPTION OF THE DEPOSIT

The Emerald Isle copper deposit is at the base of a conglomerate which veneers a pediment surface on decomposed granite. Chrysocolla and probably other (unidentified) oxide copper minerals are disseminated through the matrix of the conglomerate and are deposited as fillings in northeast-trending faults and in voids in the conglomerate. The deposit is blanket-like in form and varies from 5 to 50 feet in thickness over a known area of approximately 7 acres.

The deposit is covered with a poorly cemented conglomerate that is essentially barren of copper mineralization. The overburden ranges in thickness from a few inches to as much as 150 feet. The average ratio of overburden to ore is 0.67 yards to 1 ton ore for the known area of the deposit.

ORE RESERVES

The total reserves are estimated as 623,800 tons of indicated ore averaging 1.12 percent copper (13,990,000 pounds of total copper). Not all of the copper in the ore is acid-soluble, but it is believed that the acid-soluble copper reserves are of the above order of magnitude. The lime content of the ore is variable. Field tests indicated the lime content increases as the tenor of the ore decreases. Some of the reserves included in the above total probably are not based on representative data. Many of the drill holes do not bottom the ore-bearing zone; reserves have been calculated only on the basis of the thickness of zone that was penetrated. However, deeper drilling

in Blocks A and B would probably add only comparatively small tonnages of ore to the reserves. For Block D it is possible that additional drilling would decrease the calculated reserves. On the basis of present geologic knowledge, it would be unsound practice to infer additional reserves of ore beyond the areas already drilled. However, if additional information through drilling becomes available, it may be possible to infer additional reserves beyond the areas already drilled.

PRESENT STATUS

Development

The principal development work consists of underground workings on the vein, a large open-cut, 13 churn-drill holes, and 121 wagon-drill holes. The vein has been developed by two shafts, an 800-foot long drift on the vein with cross-cuts, raises and winzes as shown on figure 3. At present only shaft no. 2 and the workings south of this shaft are accessible.

The open-cut is approximately 600 feet long and averages about 20 feet wide and 10 feet deep. Figure 3 shows the surface outline of the open-cut and the locations of underground workings and drill holes.

The report on the Emerald Isle mine by C. F. Weeks gives the following information on methods used in test-drilling the property and his interpretation of the results in sampling both the drill holes and the underground workings:

"The air-drill holes were all drilled with an I.R. wagon-drill. These holes were all drilled dry. The hole depth was determined by the mud-collars that formed when the holes reached moist ground, and, as will be noticed, a good many of them were bottomed in ore, instead of being drilled on through to the underlying country rocks. The only casing used was a 2 ft. section on the collar. There was very little indication of sloughing, -and since the overburden and the low-grade ore are almost invariably very much softer than the good ore, any sloughing there was, would have had the effect of dilution on the samples. The drillings were blown from the holes with air in the usual manner and were caught in a metal sample collector that fitted over the casing. The sample was pretty well collected in this way, except that a part of the dust was carried away by the blast of air from the hole, and lost. As the dust carries from two to three times as much copper as the average ore, there was some down-grading of the sample here. However, since the weight of sample lost thus was quite small, I do not think this dust loss is very important.

"The churn-drill holes were 6" dia., and the only casing used was 20 ft. at the collar. They were, of course, drilled wet, and there was considerable sloughing in these holes. Some 5 ft. sections produced as much as twice the normal amount of sample. Most of this sloughing occurred in the ore.

"In a great many ores, the soft portions carry the best values. but here the exact reverse is the case. In the ore, the Chrysocolla is the cementing agent. The 2½% to 3% copper-bearing agglomerate is usually so hard that when blasted, the break will come right through the inclusions of quartz, gneiss, and granite, - while the low-grade portions (from .25% to .5%) can be readily disintegrated with a prospect pick. In moiling channel samples in the ore of grades from 0% to 1.75%, considerable difficulty is experienced, because the barren pebble s loosen and fall out of the copper-bearing cementing material. This is because the pebbles (up to an inch or two in diameter) are a good deal harder than the cementing material. How this problem is handled, will have considerable effect on results obtained by the sampling engineer. There will be a serious up-grading of the sample. Of course, there was no opportunity to evaluate this situation as it affected the samples from the churn drill holes. These samples were taken just as they came from the holes, and assayed. How much down-grading of the sample was caused by this condition would be impossible to say, but certainly it had a considerable effect. The sample was dumped into buckets from the bailer and from there into the splitter, wet."

Power

An electric power line connects the property with the transmission system of the Citizens Utility Co. of Kingman, Ariz. who secures most of its electricity from the Government power plant at Hoover Dam. Power from a 44,000 volt line is sold at rates ranging from 9 mills to 1.7 cents per KWH and the rate is determined by percentage demand. The lower rate of 9 mills a KWH is based on a 95 percent demand.

Water

The Lewin-Mathes Mining Co., during their operation of the property, obtained water from the White Horse and Ostrich mines located 2.3 miles and 2.8 miles, respectively, from the Emerald Isle mine. The water was pumped from shafts at these mines and delivered by gravity flow through a 3-inch pipeline to a 300,000 gallon steel storage tank located at the treatment plant. The complete water system is still in place and Weeks reports it is in operating condition at present. It was operated to deliver 35 gallons a minute but is designed to handle 50 gallons a minute at full capacity. Water was paid for on a monthly rental basis, $50 a month when pumping from the White Horse mine and $25 a month for water obtained from the Ostrich mine. It is possible that all water needed for the proposed plant might be obtainable from the present sources, but in order to assure an adequate supply Weeks planned to obtain additional water from a churn-drill hole on the Emerald Isle property. He reported that bailing of water from the churn-drill hole indicated a flow of from 30 to 35 gallons a minute. It is estimated a water system designed to supply water to the storage tank from a well in the area of the drill hole would cost $5,000.

It is Weeks opinion that the above water systems could supply more water than would be needed for the proposed treatment plant. However, in event the supply proves inadequate additional water could be obtained from the Tennessee-Schuylkill mine and at other nearby mines, all of which are located at altitudes higher than the Emerald Isle property.

Surface Plant and Equipment

The following list of buildings and major equipment on the property was supplied by Weeks. He stated that there are other necessary supplemental equipment, including electric motors, and that the estimated present installed value of the plant is $200,000.

Buildings

1 - Office & warehouse building, frame, 28 ft. by 14 ft.
1 - Store-house 12 ft. by 14 ft.
1 - Compressor house 18 ft. by 25 ft.
1 - Machine shop 32 ft. by 42 ft.
1 - Solution foreman's office 8 ft. by 10 ft.
1 - Transformer house 12 ft. by 16 ft.
1 - Assay office and drafting room 20 ft. by 30 ft.
1 - Hoist - house 12 ft. by 16 ft.
1 - Iron bin - 14 ft. by 24 ft. by 10 ft.
1 - Iron bin Trestle 12 ft. by 50 ft.
1 - Change house 12 ft. by 16 ft.
1 - Conveyor house 6 ft. by 90 ft.
1 - Dwelling house 12 ft. by 30 ft. by 12 ft.

Assay office - Equipped for wet assay work.

Office

2 - Desks
1 - Filing cabinet

Shop

1 - Power hack-saw
1 - Drill press
1 - Heavy grinder - 2 wheels
1 - 16 in. Lathe
1 - 300 amp. Elect. welder (gas-driven)

Power

3 - 75 KVA Transformers, 44,000/400

Water Supply

1½ mile 3½ in. O.D. Pipe line (to White Horse mine)
.8 mile 3 in. O.D. " " " " " "
.5 mile 3 in. O.D. " " " " " "
1 - 15 HP Pump jack and pump (White Horse)
1 - Steel water supply tank 47 ft. by 24 ft. - 320,000 gals.

Pumps

1 - 2 in. Wilfly sand-pump
3 - 350 g.p.m. Rubber-lined pumps
4 - " " Stainless steel pumps
2 - 500 " Stainless steel pumps
Rubber hose for pump-lines

Crushing Plant

2 - Symons Pod-deck screens X7-16
1 - Cyclone dust separator - 14 ft.
1 - 3 ft. Short-head Symons crusher
1 - 18 in. by 30 in. A.C. Blake-type Jaw Breaker
Belt Conveyors

Leaching Dep't.

4 - 10 ft. by 110 ft. by 7 ft. Concrete leaching vats
1 - 52 HP Elect. slusher hoist for vats
3 - Redwood sol. tanks 30 ft. dia. by 10 ft. deep.
1 - " " " 35 ft. " " " " "
1 - " " " 45.5 ft." " " " "
1 - Asphalt paved leaching base

Precipitation Dep t.

1 - 6 ft. by 50 ft. Revolving drum precipitator

Mine

1 - 100 HP slusher hoist and bucket
2 - 52 C.P. Jackhammers
2 - C.P. Demolition Tool
1 - Sullivan Stoper
1 - 500 ft. 2-stage compressor

PRODUCTION PLAN

General

The following information concerning mining operations of the Levin-Mathes Mining Co. was taken from the report on the Emerald Isle Mine by C. F. Weeks. The information serves to justify the proposed mining method and also supplies data pertinent for estimating mining costs.

At the beginning of the above mentioned operations the broken ore was slushed by bulldozer into a trench and then moved by a dragline-scraper to the crushing plant. This system proved unsatisfactory because the larger pieces of ore interferred with proper operation of the dragline and it was decided breaking of the ore to a size that could be handled efficiently by the scraper would be uneconomic. The system was discarded in favor of a power-shovel and truck operation.

A Diesel shovel, capacity 1½ cubic yards, was used for both mining and stripping operations. The shovel loaded into two dump trucks, each of 7 cubic yards capacity. This method proved unsatisfactory, in the opinion of Weeks, because of the uneven surface at the contact between ore and the bedrock. In some cases ore was left in the floor and in other instances the ore was diluted by the mining of soft bedrock that occurred as humps extending above the working surface of the shovel.

Weeks made the following statement regarding the character of the ore from the mining standpoint: The ore drills easily and little drilling is required to break the ground. Jackhammers of 50-pound weight were found to be more satisfactory for drilling than wagon-drills. Several tungsten-carbide drill-bits were used during the last part of the operations and none of them required sharpening. One of the bits drilled 455 feet of hole. Data concerning the total amount of explosives used in the open-pit operation are not available.

However, the explosives used in several test holes that were blasted amounted to 0.35 pound of 45 percent bag-powder per ton of ore for primary blasting and approximately 0.12 pound of powder was consumed in secondary blasting operations. These holes were drilled in a line, 18 feet from the mining face, on 27-foot centers. Each hole was 25 feet deep. In this test the overhead exceeded the break of 18 feet of burden at the front of the holes by 20 percent. The operations showed there was a considerable variation in the amount of explosives used in different parts of the orebody. The above tests were made in the most highly consolidated rock, the hardest type exposed in the open-pit. Primary blasting of the softer portions of the orebody requires more powder than used in these tests. However, the total powder consumption for each type of rock is about the same because very little powder is consumed in the secondary blasting of the softer type of rock.

Proposed Mining Method

The proposed method of mining the Emerald Isle orebody involves the installation of an underground crushing plant from which ore would be delivered by a belt conveyor to the screening plant (Fig. 4). The crushing plant would consist of a grizzly, ore bin, belt feeder, and crusher. After completion of this installation a raise would be driven from the level of the grizzly to connect on the surface in an area previously stripped of overburden. When this development work was completed, probably within 6 months after start of project, .

production on a scale of 500 tons a day could be started by using a "glory-hole" system of mining. Ore drilled and blasted into the raise at the surface would be passed through the grizzly into the ore bin, and then be crushed for transportation by the belt conveyor to the screening plant. This system of mining would evolve into an open pit operation with the broken ore being moved to the grizzly by a bulldozer. This method of moving the ore would be changed to haulage by a "carryal" when the open-pit expanded to the point where bulldozer operation would become inefficient due to the comparatively long distances the ore would have to be transported.

verburden would be moved to areas outside the boundary of the orebody and also into "mined-out" areas. The bulldozer used in moving ore also would be utilized in the stripping operations. Stripping of the thinner sections of overburden would be done with the bulldozer while a "carryall" pulled by the bulldozer would be used for removing waste from areas containing the thicker sections of overburden. Past operations indicate any loosening of the overburden that might be required for removal with the above equipment could be accomplished with a scarifier pulled by the bulldozer.

Ore produced by this system of mining will probably be diluted by a very small amount of waste from the bedrock and by a small amount of lower-grade material that overlies the ore. Because the percentage of dilution will probably not be over a few percent the writer does not believe it is a factor to be considered in a report of this type.

Costs

Preparatory Work - The sketch of the proposed development work and the following estimates of the costs of the development work were supplied during these investigations by Weeks. He said the data, submitted as a rush job, would serve to give a general idea of the proposed work and that the cost estimates "will undoubtedly vary somewhat one way or the other, from the facts."

The submitted estimates of development costs follow:

Cut, 6 ft. by 15 ft. by 300 ft.; 1,000 cu. yds. @ $1.00	$1,000
Incline, 6 ft. by 8 ft.; 285 ft. @ $15.00	4,275
Manway, 4 ft. by 6 ft.; 27 ft. @ $12.00	320
Pocket to ore-line; 650 cu. yds. @ $1.50	975
Raise to surface; 120 ft. @ $10.00	1,200
Total	$7,770

The writer estimated that some of the unit costs of the above work were too low and that the total cost would probably be $10,000. Weeks agreed that the cost as revised was acceptable and was probably more reasonable than his original estimate.

Thus the total cost of preparing the mine for production and installing a primary crushing plant with facilities to deliver ore to the screening plant would be:

Excavation	$10,000
New materials; grizzly, ore bin, conveyor	14,160
Installation	3,600
Contingencies, 10 percent	2,776
Total	$30,536

Mining - The following are estimated costs of mining and stripping operations as submitted by Weeks. The estimates are based on a production of 500 tons of ore daily.

	Cost a ton
Labor (6 man, $73.00), social security tax included	$ 0.146
Blasting supplies	.150
Bulldozer fuel and maintenance	.100
Compressed air	.050
Total	$ 0.446

This total has been revised by the writer to a total cost of $0.601 a ton by the addition of $0.11 for royalty, $0.015 for employment and compensation insurance, $0.015 for drill-bits and maintenance of drilling equipment, and $0.015 for administrative costs.

Some of the factors considered in arriving at the above estimated mining costs are as follows:

1. The deposit will contain approximately 0.6 yard of overburden to one ton of ore;
2. Two drillers can be expected to do all drilling for both primary and secondary ore-breaking operations. Much of the secondary breaking of ore can be done efficiently with pavement-breakers;
3. One bulldozer operator, working one shift daily, will be able to handle all overburden and ore during the first part of the operations. Operation of a 2-shift basis probably will be necessary when the ore has to be moved comparatively long distances.

New Equipment

The following new equipment, with costs, will be required to prepare the mine for production and for mining operations. These data were supplied by Weeks.

Grizzly, ore-bin, and conveyer

Rails, 120-pound; 22 tons @ $80	$ 1,760
I-beams, 3 - 10 in. by 33 ft.	300
Timber, 5M @ $100	1,000
Conveyer materials	12,000
Total	$14,160

Mining

Caterpillar type tractor, D-8 or similar model, with dozer blade	$20,000
Scarifier	3,500
Carry-all	5,000
Total	$28,500

Manpower

The following information regarding available manpower and living accommodations is quoted from the report by Weeks:

"It is unnecessary to provide living accommodations at the property, since Chloride, 3½ miles distance, and Kingman, 18 miles distant, are within easy commuting distance. Since this is a mining community of long standing, and little mining is in progress, ample experienced help is available and in residence here."

SUMMARY: MINING COSTS

Capital Requirements

The estimated capital for the mining phase of the project includes the cost of preparatory work and mining equipment, a total of $59,036. In addition, capital would be required to defray mine operation expenses at the rate of $300.00 daily until operations were self-substaining.

Mining Costs

The following summary indicates a primary crushing plant and conveyor could be installed and the deposit mined at a cost of about 70 cents a ton.

Item	Cost a ton
Preparatory work, $30,536	$0.049
Mining equipment, $28,500	0.046
Mining	0.601
Total	$0.696

CONCLUSIONS

The Emerald Isle deposit contains measured and indicated reserves totalling 623,800 tons of ore containing 1.12 percent copper (13,990,000 pounds of copper). Copper mineralization probably extends beyond the zone of the above reserves but the tonnage and grade of inferred reserves cannot be estimated from available data.

Supplementary drilling and sampling is necessary for the following reason:

1. To more accurately determine the tonnage and grade of ore where previous drilling has indicated the greatest thicknesses of ore;
2. To obtain a metallurgical sample that would be representative of the percentages of acid-soluble copper, lime, and zinc in the ore.

A primary crushing plant and conveyer can be installed and the deposit mined at an estimated cost of $0.70 a ton of ore

RECOMMENDATIONS

The recommendations are dependent upon the findings of the metallurgical investigations and are included in the section that summarizes the findings of the overall investigation.

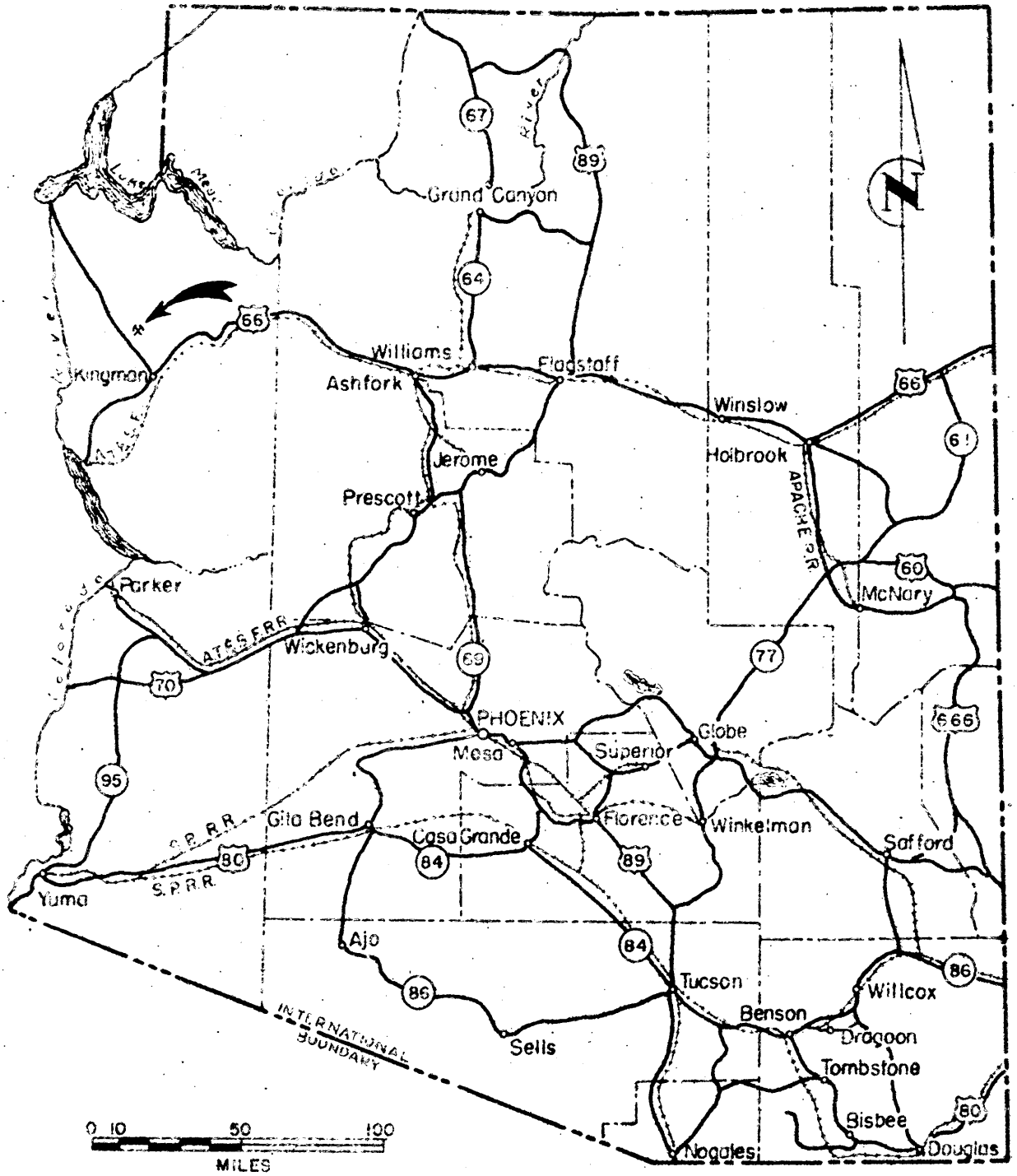


FIG I - LOCATION MAP,
EMERALD ISLE COPPER MINE
MOHAVE COUNTY, ARIZONA
DMA 374

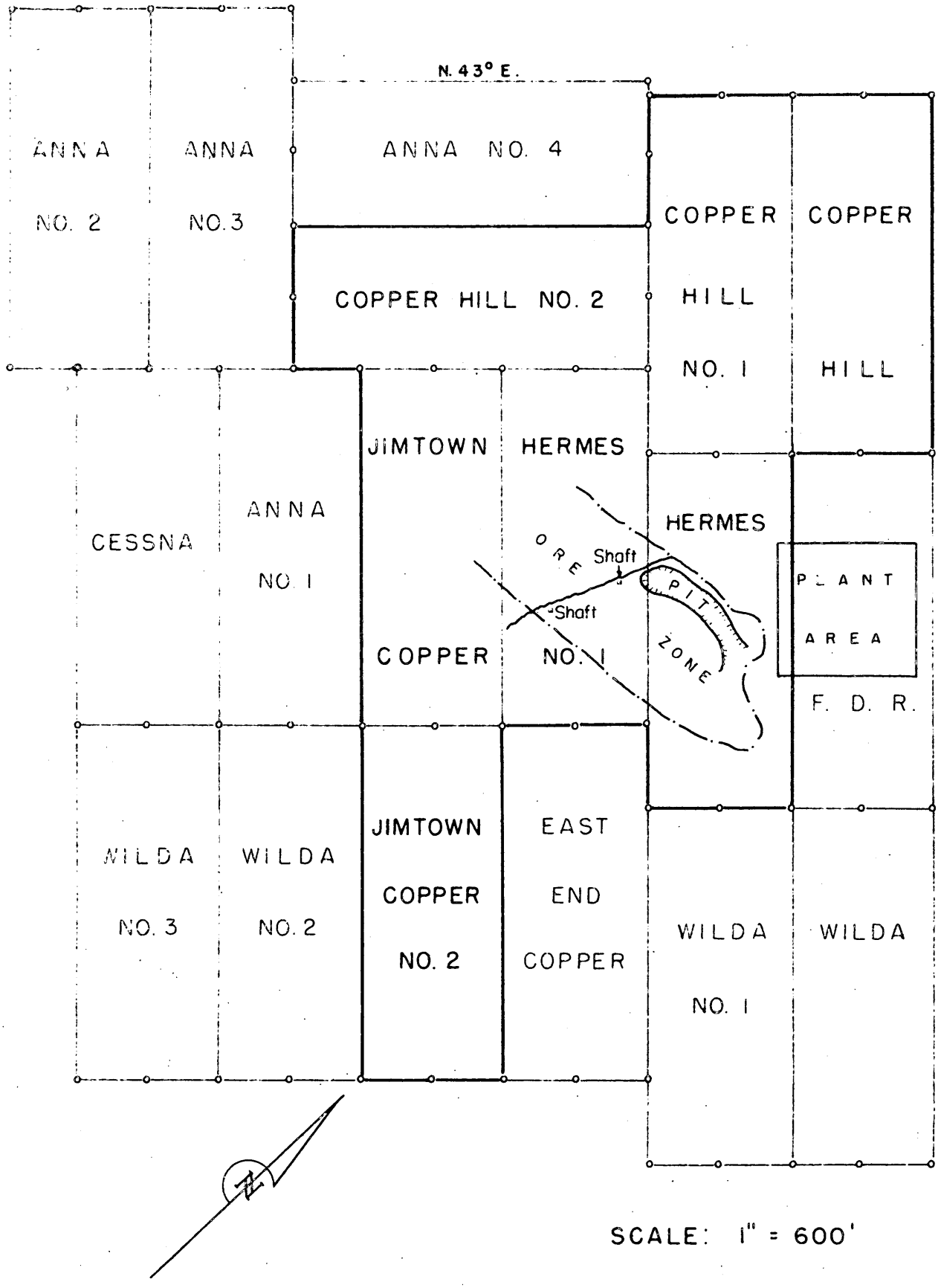
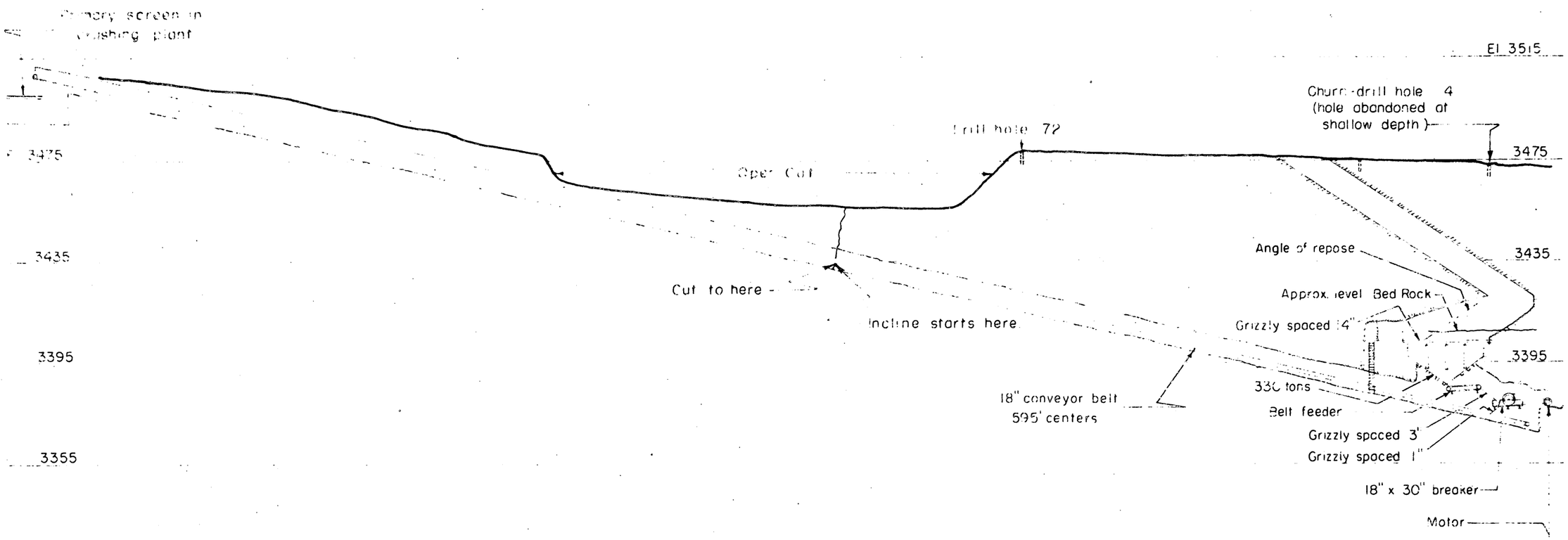


Figure 2.- Property map, Emerald Isle mine, Mohave Co., Arizona.

DMA 374



From sketch
By: C. F. Weeks

D.M.A. 374

Scale: 1" = 40'

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

DMA-374

Emerald Isle Copper Proposal

Wallapai Mining District

Mohave County, Arizona

METALLURGICAL REPORT

by: J.B. Clemmer

and

F.N. Bender

Metallurgists

U.S. BUREAU OF MINES

1951

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INTRODUCTION

This report discusses the metallurgical aspects of a plan proposed by C.F. Weeks, Kingman, Ariz., for rehabilitating and enlarging the hydrometallurgical plant at the Emerald Isle Mine for processing oxide copper ores. The proposal contains several unusual features including production of a major part of the copper as blue stone, and the manufacture of sulfuric acid for leaching and sponge iron for copper cementation from pyrite recovered from local pyritic ores and tailings.

Although the information submitted included rather complete reports on the property and on several steps in the treatment process, Mr. Weeks did not detail the costs of the various phases of the project other than a tentative lump-sum estimate of $600,000. Because of its several novel features, the proposal appeared to have some merit and a field study was requested to determine whether Mr. Weeks should be encouraged to obtain the detailed cost estimates required and to present the more complete description of the project needed to support a formal loan application and a certificate of necessity.

A trip was made to inspect the plant at Emerald Isle and to discuss the economics and metallurgical aspects of the proposed treatment methods with Mr. Weeks. Laboratory tests subsequently were made on samples of open pit and underground ore. A number of leaching tests were made on the open pit ore and crystallization tests were made on the pregnant liquors to obtain additional information on the yield and grade of blue stone which could be expected by careful processing of the ore. The data and observation from these tests will be considered before discussing the feasibility of the proposed production plan.

METALLURGICAL TESTS ON EMERALD ISLE ORE

Mineralogy

Although the mineralogy of the Emerald Isle ore has been described in Mr. Shride's geological report, a few additional remarks regarding the ore from the standpoint of metallurgical processing seem indicated.

Chemical analyses and microscopic examinations were made on several underground and open-pit samples taken by Messrs. Semple and Shride. The analyses of the samples are given in Table 1.

Table 1 - Analyses of Samples

Sample Number	Source	Assay, percent				
		Cu			Zn	$CaCO_3$
		Total	Acid-soluble	Oxide		
11464	Underground	0.73	0.58	0.70	-0.1	0.35
65	"	0.57	0.32	0.54	0.1	0.45
66	Open-pit, low grade	0.36	0.08	0.31	-0.1	0.98
67	" " high grade	1.63	1.53	1.62	0.1	0.35
68	" " " "	1.58	1.56	1.56	0.5	0.70

Petrographic examination of the higher grade open-pit samples showed that they are similar and consisted of an arkosic sandstone cemented with an intergrowth of opaline silica, calcite, wad, hematite and chrysocolla, with sporadic grains of malaconite. The sand grains of the arkose were quartz, orthoclase, hornblende, biotite and magnetite. No sulfide minerals were observed in the samples. Although the zinc undoubtedly occurs in the oxidized form, the zinc minerals were not positively identified.

The low-grade open pit and underground samples also were similar. As compared to the higher grade open-pit samples, they contained a higher proportion of copper as malaconite. Many of the malaconite grains are coated with opaline silica and this copper would not be recovered by

ordinary leaching procedures. In order to obtain a correct assay for oxide copper on these samples, the minus 100-mesh ore pulps were digested in strong caustic soda to dissolve the opaline coating before making the conventional acid treatment for an oxide copper determination. The difference between the oxide and total copper determinations is due to incomplete dissolution of the low-grade copper silicates and oxides disseminated in silica as a careful microscopic examination failed to reveal any copper sulfides in these samples.

Although these few samples may not be representative of the entire ore body, they are typical types of ore which must be processed without resorting to expensive selective mining. Subsequent leaching tests demonstrated that the higher grade open-pit ore containing chrysocolla was amenable to leaching. The underground ore containing a higher proportion of opaline-coated copper minerals was more refractory and the acid consumption was much higher even though the samples contained less calcite than the open pit material. All of the samples contained clay from weathering of feldspars.

As the proposed treatment plan calls for production of blue vitriol which must be substantially free of zinc to meet trade specifications, the zinc content of the Emerald Isle ore also must be considered. Unfortunately, all prior mine and drill hole samples were analyzed only for total copper and nothing is known about the zinc, calcium carbonate, and acid-soluble copper contents of ores in various parts of the deposit. Mr. Skeide has estimated the ore reserves at 633,000 tons containing 1.12 percent total copper. The open-pit samples tested in this study was high grade and the underground sample was lower grade than "average" ore.

Leaching Tests

Although plant and laboratory leaching tests on various open pit ores are reported in the docket, supplementary tests on the samples obtained by Messrs. Remsle and Shride seem warranted to compare deportment of the open pit and underground ores and to prepare enough pregnant solution for subsequent crystallization tests.

A composite of the open pit samples (Nos. 11467-68) was stage-crushed to pass 0.5 inch and sized on 28 mesh. Flood and bottle-agitation leaching tests were made on the coarse fraction which represented 89 percent of the ore by weight. In flood leaching, the column was agitated at frequent intervals to circulate the liquor. The fines were leached by bottle-agitation on rolls. Flood leaching tests also were made on a minus 0.5-inch plus 28-mesh portion of the underground sample (No. 11464) for comparison with open pit ore. The results obtained with various acid strengths and leaching periods are summarized in Table 2.

Bottle-agitation and flood-leaching gave moderately good extractions on the coarse open pit ore. A 48-hour agitation with 5 and 10 percent acid dissolved 81 and 86 percent of the copper with an average acid consumption of 2.9 pounds per pound of copper. For a similar recovery, flood leaching requires use of stronger acid and longer leaching periods.

The fine fraction of open-pit ore is readily amenable to leaching. A 1-hour contact with 2.5 percent acid gave an 85 percent extraction of the copper with an acid consumption of 2.4 pounds per pound of copper. Leaching with small volumes of 5 and 10 percent acid gave slightly higher extractions and the copper content of the pregnant liquors was proportion-

Table 2 – Summary of Leaching Tests on Open-pit and Underground Samples

Method	Time	Acid Strength Gm/liter	Solution/ore Ratio	Pounds of Acid per ton of ore Added	Pounds of Acid per ton of ore Consumed	Pounds Acid Consumed per pound copper extracted	Analysis of Leach liquor-gms per liter Cu	Fe	Fe	Al	Residue Assay % Cu	Distribution of copper, percent Leach Liquor	Wash Liquor	Total	Residue
Open pit sample -1/2-inch + 28 mesh fraction; 1.46% Total Cu, 1.38% Acid Soluble Cu, 0.3% Fe, 0.38% CaCO3															
Flood	6 hrs	125	0.34	84.4	58.6	2.84	(Pregnant liquors				0.47	65.7*	3.8	69.6	30.4
"	12 "	125	"	84.4	64.2	2.65	(and first wash solutions				0.36	69.9*	7.1	77.0	23.0
"	24 "	125	"	84.4	63.4	2.66	(combined for crystallization				0.34	70.2*	7.8	78.0	22.0
"	48 "	125	"	84.4	70.5	2.51	(tests				0.26	77.3*	6.1	83.4	16.6
"	96 "	125	"	84.4	78.5	2.23					0.19	78.6*	8.8	87.8	12.2
"	144 "	125	"	90.6	81.8	2.35					0.20	83.1*	4.2	87.3	12.7
Bottle	48 "	104	0.425	88.4	79.1	2.80	29.88	4.65	0.79	7.87	0.31	68.0	18.4	86.4	13.6
"	48 "	52.1	0.85	88.3	74.4	2.95	14.35	2.30	0.22	3.94	0.39	70.0	10.3	81.1	18.9
"	48 "	25.3	1.70	86.0	70.2	2.88	7.18	1.65	0.085	1.82	0.33	73.2	5.2	78.4	21.6
Open pit sample -28 mesh fraction; 3.73% Total Cu, 3.38% Acid Soluble Cu, 0.72% Fe, 1.13% CaCO3															
Bottle	30 min.	104	0.75	156.8	132.8	2.60	28.70	5.1	0.42	7.65	0.86	50.8	27.8	78.6	21.4
"	30 "	52.1	1.50	156.2	150.0	2.63	17.80	2.4	0.13	3.66	0.98	60.0	16.3	76.3	23.7
"	30 "	25.3	3.0	153.8	148.4	2.68	9.08	1.2	0.07	1.82	0.93	66.1	11.6	77.7	22.3
"	60 "	104	.875	182.0	170.0	2.68	34.34	4.7	0.36	7.60	0.60	56.5	20.0	85.5	14.5
"	60 "	104	1.0	207.8	173.0	2.72	38.46	4.3	0.40	7.73	0.60	62.4	24.4	86.8	13.2
"	60 "	52.1	1.75	182.1	167.8	2.65	17.35	2.4	0.16	3.82	0.62	65.3	20.2	85.5	14.5
"	60 "	52.1	2.0	208.2	172.4	2.76	18.00	2.2	0.19	4.03	0.54	71.6	15.0	86.6	13.4
"	60 "	25.3	3.5	177.8	138.4	2.42	8.66	1.15	0.085	1.83	0.56	74.1	10.8	84.9	15.1
"	60 "	25.3	4.0	202.0	156.0	2.43	7.70	1.15	0.08	1.90	0.61	77.3	8.3	85.5	14.5
Underground sample -1/2 inch + 28 mesh fraction; 0.72% Total Cu, 0.52% Acid Soluble Cu, -0.1 Fe, 0.35% CaCO3															
Flood	96 hrs	125	0.34	84.4	60.7	4.07	10.15	–	–	–	0.19	41.8*	24.2	75.7	24.3
"	144 "	125	0.34	84.4	66.4	3.81	10.35	nil	7.86	0.67	0.18	4.15*	35.1	76.7	23.3

*Assays and Distribution values represent the pregnant liquor plus the first wash to make up the liquor volume to that of the acid added.

ately increased. As the volume of leach liquor retained in the residues is nearly the same for either weak or strong acid, the percentage of copper recovered in the pregnant liquor falls off with increasing strength of acid.

The underground sample is refractory toward leaching. A long contact with rather strong acid is necessary to extract 75 percent of the copper. The acid consuming capacity of the sample is high; considerable iron and aluminum dissolves with the copper and about 9 pounds of acid are required per pound of copper extracted. Should the deposit contain any appreciable tonnage of this type of ore, leaching costs would be excessive. It is regrettable that more is not known about the tonnage and leaching deportment of the underground ores. Without this information, it is impossible to predict what the treatment cost would be or to decide upon the best combination of methods for economic processing of the deposit. Any plan proposed for treating the ore body based solely upon open pit material should be looked upon with skepticism.

The analyses of the pregnant liquors from drainage of the flood-leached samples and filtration of the bottle-leached residues are recorded in the table. After removing the pregnant liquors, the leach residues were thoroughly washed to recover the remaining dissolved copper. The distribution of copper in the pregnant and wash solutions are recorded and will be referred to later.

Crystallization Tests

A proposal to make blue stone at Emerald Isle is described in H.R. Hanley's report (Engineering Report No. 5) in the docket. The cursory tests reported are not conclusive as regards either the

yield or grade of bluestone which could be produced by recrystallization methods. As much of the copper and zinc were unaccounted for and assumptions were made regarding the copper recovery, additional tests seem warranted to confirm and extend Hanley's work before making a decision as to the feasibility of the proposal.

The pregnant liquors and first wash solutions from the 6, 12, 24 and 48-hour flood leaching tests made on the open-pit ore with 84 pounds of sulfuric acid per ton were combined for the tests. Enough fresh ore was added to the solution to consume most of the remaining free acid and establish a solution pH of about 1.8. The filtered mother liquor contained 43.4 grams of copper, 0.56 grams of iron, 4.9 grams of zinc and 5.3 grams of aluminum per liter, and was somewhat more than half saturated with copper sulfate at laboratory temperature. Four crops of crystals were recovered by successive evaporation of 1500 ml. of the solution to 750, 500, 250 and 125 mls. The liquors were evaporated to the desired volume at 100°C. and then slowly cooled to 20°C. and agitated to promote crystallization. The crystals were filtered under vacuum and air dried for assay. The results of the tests are given in Table 3. The first crop of crystals contained considerable gypsum but they otherwise were of good grade. The second and third crops were the best recovered but they contained more zinc than desired. The fourth crop crystals were badly contaminated and contained about 38 percent of impurities. These and the residual liquor enriched in iron, zinc, and aluminum would be sent to cementation.

Table 3 - Crystallization tests on Leach Liquors

Product	Weight or Volume	Analyses: Assay, Percent or Grams per liter						Percent $CuSO_4 \cdot 5H_2O$ (Gels)	Distribution: Percent in mother liquor				Percent of Total Cu in Ore	Cumulative Percent Total Cu
		Cu	Zn	Fe	Al	Ca	H_2O		Cu	Zn	Fe	Al		
Mother Liquor	1500 ml	43.4	4.90	0.56	5.30	-	-	-	100.0	100.0	100.0	100.0	77.3*	-
First Crystals	19.2 g	23.05	0.30	0.04	0.18	3.0	30.0	90.6	6.8	0.8	1.0	0.4	5.3	5.3
Second "	127.4 g	25.25	0.50	0.05	0.12	0.17	33.55	99.25	49.3	8.8	8.6	1.9	38.1	43.4
Third "	67.8 g	23.70	0.75	0.06	0.21	0.53	34.51	93.15	24.6	7.1	5.5	1.8	19.0	62.4
Fourth "	41.9 g	15.80	1.70	0.18	2.23	0.25	36.80	62.1	10.1	9.9	10.1	11.7	7.8	70.2
Residual Liquor	112 ml	53.63	47.25	4.97	59.80	-	-		9.2	73.4	74.8	84.2	7.1	77.3

*Value for liquor recovered from 48 hour leach with 125 gm/L acid

Recrystallization tests were made on a composite of the first three crops of primary crystals which contained 80.7 percent of the copper in the mother liquor and 62.4 percent of the total copper in the ore. The crystals were dissolved in hot water, filtered to remove gypsum and then recrystallized in successive steps by evaporation and cooling. The data are recorded in Table 4. The first and second crops accounted for 78.4 percent of copper in the primary crystals, 63.3 percent of copper in the leach liquor, and 48.9 percent of the total copper in the ore. These crystals were low in iron, aluminum and calcium but still contained 0.15 percent zinc. Although the third crop in crystals contained much zinc, more careful processing should permit recovery of about half of the copper as secondary crystals and increase the over-all yield to 54 percent of the copper in the ore.

As the solubility of zinc sulfate is about twice that of copper sulfate at 25° and 100°C. and since the crystallizer solutions are unsaturated in zinc, Hanly assumes (Engineering Report No. 5, page 4) that there is no possibility of zinc sulfate crystallizing with the blue stone. He attributes zinc contamination to adhering tail solution. A simple test shows that this assumption is not valid. A 50-gram portion of C. P. copper sulfate crystals was added to a saturated solution prepared from the copper sulfate. The solution was heated to dissolve the crystals and then cooled to 25°C. to recover the recrystallized blue stone; the zinc contents of the initial and final crystals were nil. A similar recrystallization test was made from a solution containing 20 grams of C.P. zinc sulfate per liter. The crystallized blue stone was filtered under vacuum and split into two portions; one fraction was air dried and the other was washed with 100 ml of the saturated C.P. copper sulfate solution and then air dried. The zinc contents of the two portions were

Table 4 - Recrystallization of Primary Blue Stone

Product	Weight or Volume	Analyses: Assay, percent or grams per liter					Percent $CuSO_4 \cdot 5H_2O$	Distribution, percent Cu		
		Cu	Zn	Fe	Al	Ca		Primary Crystals	Leach Liquor	Ore
Primary Crystals	150 gm	24.46	0.58	0.05	0.15	0.29	-	100.0	80.7	62.4
First Crop	85.1	25.2	0.15	0.01	0.045	0.11	99.1	58.7	47.4	36.6
Second "	28.9	25.0	0.15	0.15	.04	0.11	98.3	19.7	15.9	12.3
Third "	24.0	24.9	0.45	0.35	.08	0.14	97.5	16.4	13.2	10.2
Residual liquor	29 ml	66.15	21.70	1.75	5.9	0.61		5.2	4.2	3.3

the same, 0.10 percent. Somewhat similar results were obtained from washing primary crystals from Emerald Isle leach solutions. While washing with saturated C.P. copper sulfate solution replaces the residual tail solution and dissolves any discrete zinc sulfate crystals thrown down with the blue stone, repeated washing failed to remove all of the zinc. The crystals invariably contained 0.15 percent or more zinc which must be inherent in the blue stone. The recovery of pure blue stone from contaminated solutions is much more involved than the docket reports indicates.

The liquors used in the crystallization tests were from leaching the open pit ore with relative strong (approximately 12.5 percent) acid. Although it seemed likely that leaching with weaker acid might dissolve less zinc, analyses of the pregnant liquors recorded in table 2 shows that such is not the case. While the zinc content of the liquors decreases with acid strength, the volume is larger and the quantity of zinc leached was almost identical in the various tests.

A discussion of crystallization would be incomplete without mentioning evaporation requirements. The minimum quantity of water which would have to be evaporated to recovery primary, secondary or tertiary crystals from feed solutions containing from 20 to 45 grams of copper per liter were calculated from the crystallization tests on Emerald Isle leach liquors. The rounded-out values are given in Table 5. These data show that evaporation requirements become excessive for dilute solutions. The cost of evaporation in single, double, and triple-effect units is generally estimated at $3, $2 and $1 per ton, respectively. The anticipated minimal cost for single and triple-effect evaporation of 20, 30 and 40 grams per liter feeds for primary, secondary and tertiary crystals are shown in Table 6.

Table 5 - Evaporation of Copper Sulfate Solutions

Cu in Evaporator Feed gm/L.	Pounds water evaporated per pound of crystals recovered		
	Primary	Secondary	Tertiary
45	5	7	8.5
40	6	8	10
35	7	9	11
30	8.5	11	13
25	11	14	16.5
20	14.5	18	21

Table 6 - Cost of Evaporating Solutions of Different Copper Contents

Cu in feed gm./L	Type of Evaporation	Cents per pound		
		Primary Crystals	Secondary Crystals	Tertiary Crystals
20	Single-effect all stages	2.175	2.70	3.15
	Triple-effect for primary and single effect for others	0.725	1.25	1.70
30	Single-effect all stages	1.275	1.65	1.95
	Triple-effect for primary and single-effect for others	0.425	0.80	1.1
40	Single-effect all stages	0.9	1.2	1.5
	Triple-effect for primary and single-effect for others	0.3	0.6	0.9

The pregnant solution analyses recorded in table 2 indicates that the open-pit ore would have to be leached with about 10 percent acid to obtain a pregnant liquor containing 30 to 35 grams of copper per liter. A part of an advanced wash could be sent to evaporation, but dilute washes would have to be diverted to cementation along with the crystallizer residuals. If the underground ore tested is typical, both pregnant and wash liquors would go to cementation. Although solar evaporation of wash solutions could be resorted to, the plant water requirements would then exceed the assured supply.

The cost of crystallizing blue stone was not estimated but it would be low. It is anticipated that atmospheric evaporative-type crystallizers would be employed.

METALLURGICAL PRACTICE AT EMERALD ISLE

The history and production record of the Emerald Isle property are summarized in Mr. Romslo's engineering report. The up-and-down record of operation is not impressive and indicates that past operation has been marginal.

Mr. Weeks attributes the uneconomic operation to several factors. These deserve brief mention as a background for the new production plan. Although some 66,000 tons of ore have been processed by the Emerald Isle Copper Company, the operation was of a pilot plant nature. Initially, not enough was known about handling the ore to prepare a proper feed for tank leaching. Spiral and rake classification failed to remove enough of the clay and fine sand and the charges were too impervious for leaching; only about 57 percent of the leachable copper was recovered. Leaching of screen-sized feed was used later and was more successful. Heap

leaching of a somewhat coarser feed also was tried on an experimental scale and gave moderately good results. A description of the leaching plant and the results obtained from tank and heap leaching of sized ore are given in C.F. Weeks' Engineering Report No. 2, included in the docket. It should be noted that Mr. Weeks refers to a flood leaching test made on a 50-ton batch of classifier overflow. The sizing analysis shows, that this was a segrated product free of slime rather than true classifier overflow.

Difficulties in classification and lack of equipment precluded leaching the entire ore and added to the production costs. Another contributing factor for the uneconomic operation was the high cost of reagents. According to Mr. Weeks, the average cost of sulfuric acid and shredded iron delivered at the plant was about $30 per ton. Mr. Weeks stated in private conversation that the net smelter return for cement copper was seldom more than 17 cents per pound. The operating profits during the best periods of operation must have been small.

PRODUCTION PLAN

The new production plan proposed by Mr. Weeks is for a fully integrated metallurgical operation at Emerald Isle involving the following features:

1. Heap, tank and counter-current decantation for leaching the coarse, intermediate and fine ore from screen-sizing of the crushed feed;
2. Production of a major portion of the leached copper as blue stone and the remainder as cement copper to obtain a higher return;
3. Manufacture of hydrometallurgical strength sulfuric acid at the property by catalytic oxidation of sulfur dioxide from roasting pyrite recovered from local mill tailings and ores;

4. Production of sponge iron for the cement copper precipitation from reduction of the pyrite roaster calcines, and

5. Purchase and operation of the Tennessee mill to recover pyrite from the tailing pond and from custom milling of pyritic lead-zinc of the Wallapai District to supply the sulfur and iron needed in features 3 and 4.

The production plan is much too elaborate for a 500-ton per day operation. It seems apparent that several features were included in the proposal only to complete the picture as no attempt was made to justify them on the basis of operating economy. Although general reports on leaching the ore and the production of blue stone and acid were included in the docket, the other features of the proposal were only briefly mentioned. A discussion of the various proposals in light of metallurgical tests previously reported will show that several of the steps are either metallurgically unfeasible or economically unsound at the present price of copper.

Leaching Plant

The difficulty of preparing a proper feed for leaching by mechanical classification has been mentioned. In the new plan, wet screens would replace the classifiers to alleviate the difficulty. The ore would be crushed dry to pass 5/8-inch and wet screened at 1/8-inch and 30 mesh. Mr. Weeks' estimated distribution of feed for leaching is as follows:

Size	Weight, percent	Tons per day	Treatment
-5/8 + 1/8 inch	80	400	Leach in heaps
-1/8-inch + 30 mesh	10	50	Leach in heap or tanks
-30 mesh	10	50	Leach by counter-current decantation after thickening

While this plan is undoubtedly workable, the desirability of heap leaching the coarse ore is questionable. Heap leaching lacks the nicety of control of flood or percolation leaching. It is slower for an equivalent recovery, and ties up too much copper in process. As the open pit ore must be crushed to about 1/2-inch before it is amenable, leaching in tanks seems indicated. The laboratory leaching tests show that percolation leaching the 1/2-inch to 28-mesh fraction of open pit ore with about 10 percent acid will extract 85 percent of the copper in 48 hours. The higher copper recovery would soon pay for the four additional tanks needed to process 450 tons per day on a 96-hour cycle. Heap leaching with weak acid should be restricted to low-grade ore and strippings.

The results of bottle-agitation leaching tests on a minus 28-mesh open pit ore have been discussed. The General Engineering and American cyanamid reports in the docket (Metallurgical Reports 1 and 2) give results of similar tests on other ores. Except for variation in acid consumption particular to the various ores, the results of the three laboratories were not greatly different. Agitation-leaching of fine ore in the plant should give good results. Unfortunately, no data are available on thickening of the screen underside nor on the washing characteristics of the fine material by counter-current decantation. Counter-current leaching and washing tests

should be made before considering design of a commercial plant. Leach-precipitation-flotation may have to be used if the leach residues prove difficult to handle.

The W.S. Tyler report in the docket (Metallurgical Report No. 5) adequately covers wet screening of the fine ore. As the step-tandem screens require considerable spray water, the undersize will have to be thickened to recover the water for reuse and give a thickened overflow for counter-current leaching.

Blue Stone Plant

One of the main features of the proposed plan is the production of a major part of the copper as blue stone which could be marketed at a higher price than cement copper and thereby increase the operating return. While such a proposal is metallurgically possible, the crystallization tests on leach solutions demonstrated that blue stone production is more involved than the docket reports indicate.

In discussing blue stone production with Mr. Weeks it was found that he had very little information on the chemical engineering involved or the equipment requirements, the production costs, or what tonnage he could market of different crystal grades and sizes.

As a result of our discussion, Mr. Weeks contacted the Mafford Chemical Company, one of the largest distributors of copper sulfate on the West Coast, to obtain information on marketable grades of copper sulfate. Copies of their letters are included in the Appendix. Their specifications call for crystals of about 99.5 percent purity containing less than 0.1 percent zinc, 0.1 percent iron, and only traces of arsenic,

manganese, lead and other metals. Blue stone of this quality would be difficult to produce from impure leach solutions. Although off-grade crystals could undoubtedly be marketed by taking a negotiated penalty, the laboratory tests on open pit ore show that one recrystallization at least would be necessary to keep the zinc content within reasonable limits, and double recrystallization might be needed on high-zinc ores.

The laboratory leaching and crystallization tests on open pit ore showed that 49 percent of the copper could be recovered as secondary blue stone of near marketable grade and 36 percent would pass to cementation. If it be assumed that average Emerald Isle ore assaying 1.12 percent copper will respond in a similar fashion, a 500-ton per day operation should yield 11 tons of secondary blue stone crystals and 2 tons of cement copper. While the idea of producing blue stone is intriguing, no information is available at the moment which would warrant installation of an experimental crystallizing plant of this size. The process should be tried in a small pilot operation before venturing into commercial production.

In his summary reports, pages 2 and 3, Weeks has attempted to estimate the operating costs and profits when producing cement copper and a combination of blue stone (75%) and cement copper (25%) from various grades of ores. His estimate for producing cement copper are based upon past plant operation and appear to be of the proper magnitude. The estimated costs for blue stone are guesses blended with some wishful thinking. Interpolation of the figures will show that the cost of producing blue stone was estimated at 22.7 cents per pound of contained copper,

or 5.7 cents per pound of crystals. The production cost of blue stone is only 0.65 cents more than that of the contained cement copper. These estimates for blue stone are unrealistic. The cost of evaporation and crystallization alone would exceed this amount. The cost of extra labor, power, filtration, drying, screening and bagging of the crystals were overlooked. When these are added, it is doubtful that marketable grade blue stone could be produced for much less than 7.75 to 8.25 cents per pound depending upon the number of crystallizing steps needed for a low-zinc product. If these estimates are correct, the operating profit from marketing blue stone is nearly the same as for cement copper and the processing is much more involved. This feature of the proposal is metallurgically possible but economically unfeasible.

Cementation Plant

No difficulty is anticipated in producing a good grade of cement copper from the impure leach solutions. The drum precipitator developed by Mr. Weeks has proved highly successful in plant operation. According to Mr. Weeks, the precipitator has ample capacity to handle all of the leach and wash solutions from processing 500 tons of average ore per day when leaching with 5 to 10 percent acid.

Acid Plant

The proposal to make weak sulfuric acid by catalytic oxidation of pyrite roaster gas appears sound. The report by D.N. Vedensky (Metallurgy Report No. 4) describes in considerable detail operation of an acid-making unit in the Manganese Ore Company's plant near Las Vegas, Nev. The Bureau of Mines also investigated the possibility of making hydrometallurgical strength acid by catalytic oxidation methods. The study (1) made in small

(1) Carl Rampacek and J.B. Clemmer: Sulfuric Acid from Sulfur Dioxide by Catalytic Oxidation in Mechanical Cells. A.I.M.E., San Francisco Meeting 1949.

laboratory cells has been extended to larger units. As a result of the investigation commercial-size acid-making cells will be incorporated in the manganese pilot plant at Boulder City. Pyrite flotation concentrates containing about 48 percent sulfur will be roasted in a Flue-Solids roaster to give a gas containing 10 to 15 percent sulfur dioxide; about 90 percent of the sulfur can be recovered in roasting. The gas will then be fed to the acid cells under self-induced conditions using 5 to 10 grams of manganese per liter as the catalyst. Based upon laboratory data, it is believed that acid can be made at a cost of $25 per ton as a 10 percent solution. While this is not cheap acid, it can be made as needed from roaster gas at about half the cost of concentrated acid now in short supply.

As the acid required for leaching average ore for an 85 percent extraction will be about 70 pounds per ton, 17.5 tons of acid would be required per day for a 500-ton plant. It is estimated that an acid plant of this size, including the roaster and gas scrubber, would cost about $108,500. Some 13.5 ton of pyrite containing 48 percent sulfur would be needed per day for acid making.

This feature of the proposal is metallurgically feasible and economically sound.

Sponge Iron Plant

Another feature of the proposal is the production of sponge iron for copper cementation from reduction of the pyrite roaster calcine. This would not be feasible. As only 5 tons of iron per day would be required for cementation, it would not be practical to build a sponge iron plant of this capacity. Furthermore, sponge iron produced from

pyritic calcines is not a good cementation agent. Such a sponge iron invariably contains some ferrous sulfide that decomposes in acidic leach solutions giving hydrogen sulfide which precipitates a part of the copper as unrecoverable colloidal copper sulfide. In addition to being free of sulfide, the sponge also would have to be as chunks or nodules for use in a drum-type precipitator.

This feature of the proposal is not practical.

The Tennessee Mill

In order to complete the overall production plan, the proposal includes purchase and operation of the Tennessee-Schuylkill mill near Chloride, Ariz. Enough pyrite would be produced by custom milling pyritic lead-zinc ores of the Wallapai district and by retreating impounded tailings, to supply sulfur for the acid plant and calcines for the sponge iron plant.

In view of the fact that lead-zinc ores of the Wallapai district are marginal or sub-marginal at present metal prices, there is little incentive for rehabilitating the Tennessee mill. The Tennessee Metals Corp. recently applied for a production loan to reopen the mines and rehabilitate the mill. A field survey (2) revealed that because of high operating costs in the district no substantial tonnage of ore could be mined at a profit. As metal prices have not advanced, the Weeks' proposal to operate the mill on a custom basis also is unfeasible at the present time.

It would not be practical to operate the mill on tailings to recover pyrite. The 14 tons of pyrite needed per day could be purchased from other Arizona plants. The Shattuck-Denn Mining Corp., Iron King Branch, at

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(2) DMA Docket 21-50; Tennessee-Schuylkill Project

Humboldt, Ariz., for example, is now producing 3,000 tons of auriferous pyrite for smelter flux per month. Another 1,000 tons barren pyrite concentrates assaying about 46 percent sulfur could be recovered per month by installing additional equipment. The company officials have signified a willingness to enlarge the plant if they could contract for their output at $2.50 per ton, f.o.b. Humboldt. A large tonnage of pyrite also could be recovered from a number of copper mills for contracted sale.

SUMMARY AND CONCLUSIONS

C.F. Weeks submitted to the Defense Minerals Administration an informal proposal for rehabilitating and enlarging the hydrometallurgical plant at the Emerald Isle mine. Because of its several novel features, the proposed plan appeared to have considerable merit and a field study was requested to determine whether Mr. Weeks should be encouraged to go to the expense of obtaining the detailed engineering costs and presenting a more complete description of the project needed to support a formal loan application and certificate of necessity.

The proposed plan is for a fully integrated metallurgical operation at the property involving the following features:

1. Heap, tank, and counter-current leaching of the crushed and screen-sized ore;
2. Production of a major part of the copper as blue stone and the remainder as cement copper to increase the operating return;
3. Manufacture of sulfuric acid for leaching by catalytic oxidation of sulfur dioxide from roasting pyrite recovered from local pyritic ores and tailings;

4. Manufacture of sponge iron for copper cementation from the pyritic roaster calcines, and

5. Purchase and operation of the Tennessee mill to recover pyrite from the tailing pond and from custom milling of pyritic ores of the Wallapai district to supply the sulfur and iron needed in steps 3 and 4.

Several of the features were described in considerable detail in the proposal, but others were only briefly mentioned. As success of the production plan was largely predicated upon making blue vitriol from the leach liquors, metallurgical tests seemed warranted to confirm and extend the findings reported in the proposal.

Leaching tests were made on samples of open-pit and underground ore taken during examination of the mine. The open pit ore was readily amenable to leaching but the underground sample was refractory. Leaching the open pit ore extracted 86 percent of the copper with acid consumption of 2.86 pounds per pound of copper; the underground sample gave a 76 percent extraction and the acid consumption was more than 9 pounds per pound of copper. While these samples may not be representative ores, the tests suggest that ores from various parts of the deposit may differ greatly in their leaching deportment. Too little is known about the ore body to predict what "average" ore will be or how it will respond in leaching. Prior mine and drill-hole samples were analyzed for total copper only and the pulps are no longer available. Some supplementary drilling of the deposit should be done to determine the percentage of acid-soluble copper, lime, and zinc in the ore before any metallurgical treatment plan is adopted. Assumptions as to grade or amenability are a poor basis for developing a flowsheet or for predicting operating costs and profits.

Laboratory tests were made to determine the grade and yield of blue stone from primary and secondary crystallization of pregnant liquors from leaching the open pit ore. Blue stone for the West Coast market should have a purity of 99.5 percent and contain not more than 0.1 percent zinc or iron and only a trace of arsenic, manganese, lead and other heavy metals. Although crystals of this purity were not produced, a double crystallization gave blue stone of near marketable grade; about 49 percent of the total copper in the ore was recovered as blue stone acceptable as to copper and iron, but containing 0.15 percent zinc. The crystallizer rejects and wash solutions from the ore which would be sent to cementation accounted for another 36 percent of the total copper. The production of high purity blue stone is much more involved than the proposal would indicate. The evaporation requirements were determined and estimates were made of the operating costs. Assuming that average ore would respond in a similar fashion to the open pit material; the increase in operating profit over that of making cement copper is not enough to justify the more involved and highly experimental procedure.

Based upon the laboratory tests, the following conclusions were reached regarding the overall production plan and the separate features of the proposal:

1. The production plan as proposed is much too involved for a 500-ton per day operation. Several features in the proposal are not practical nor feasible at the present price of copper.
2. Although production of blue stone from impure leach solutions is metallurgically possible, installation of a crystallizing plant

of the size required to process half or more of the copper is not warranted by present information. The process should be tried in a small pilot operation before venturing into commercial production.

3. The proposal to make acid from roasting pyrite is feasible and practical.

4. Leaching costs could be materially decreased by making hydrometallurgical acid at the plant. The pyrite can be purchased from other Arizona plants cheaper than it can be produced at the Tennessee mill.

5. Manufacture of sponge iron from the roaster calcines is not feasible. The material is a poor cementation agent.

6. Purchase and operation of the Tennessee mill to retreat tailings and custom mill pyritic ores of the Wallapai district is not feasible at the present price of lead and zinc.

RECOMMENDATIONS

The production plant as proposed is not attractive enough to warrant asking Mr. Weeks to spend additional time or money to develop a more fully integrated proposal.

Should the copper situation become critical, however, a revised proposal for rehabilitating and enlarging the present plant for leaching and cementation of copper from the ores, with an adjunctive acid-making unit to supply plant needs, would merit careful consideration.

As the production costs for cement copper at Emerald Isle would be high, accelerated amortization and a purchasing plan for the plant output for a 3-year period would be necessary to insure repayment of a loan.

MEFFORD CHEMICAL COMPANY

1026 Santa Fe Avenue
Los Angeles 21,
California
April 25, 1951

Mr. Cyrus Foss Weeks
Kingman
Arizona

C
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Dear Mr. Weeks:

We acknowledge your letter of the 24th with reference to the specifications we gave you on the 17th on Copper Sulphate. We have asked Phelps Dodge for an accurate analysis of their current production - in the meantime we have analysis from Mountain Copper Company as follows:

Cu	25.46
Fe	.097
Zn	.040
As	.00047
$CuSO_4$ $5H_2O$	100.01
$FeSO_4$ $7H_2O$	.48
$ZnSO_4$ $7H_2O$	.176

You will see from this that the specification that we gave you is rather liberal, however, we do not think you would run into any difficulty producing Copper Sulphate Crystal of equal purity, although your problem might be slightly more complex than if you were working with pure Copper as a starter instead of ore. We think that by a controlled processing your impurities could be held at a low level.

When we receive the Phelps Dodge analysis we will send it on to you.

Very truly yours,

MEFFORD CHEMICAL COMPANY

By

H. Percival Chase
Vice President in Charge
of Sales and Purchases

HPC:R
C.C. Mr. Bruce Clemmer

MEFFORD CHEMICAL COMPANY

1026 Santa Fe Avenue
Los Angeles 21,
California
April 17, 1951

C
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Y

Mr. C.F. Weeks
P.O. Box 227
Kingman, Arizona

Dear Mr. Weeks:

Referring to our several conversations on the subject of your proposed Copper Sulphate plant, we wish to give you our ideas as to the possibility of marketing the output of this plant.

While we, unfortunately, do not have accurate figures as to the total use of Copper Sulphate in the West in terms of tonnage we, in our experience, feel that an output of 20 to 25 tons per day could be marketed in the West and by that we mean that we would expect to cover the entire California market, the Northwest including Oregon, Washington, Idaho and Montana. Additional markets are available in Utah, Colorado, Arizona and even Nevada. We are well setup with contracts to the agricultural, mining and water treatment fields and other special uses where Copper Sulphate is of particular interest.

As to the purity of the product usually demanded we will require Copper Sulphate running approximately 99-1/2% pure with Iron less than .1% and Zinc also less than .1%. Other impurities such as Arsenic, Manganese, Lead and other metals should occur in only trace amounts. The trade demands various specifications as to particle size including the large - medium - snowform and powdered form, so it would be necessary for you to provide proper equipment to screen your product to these various grades.

We are interested in arranging with you to handle, exclusively, the entire sales program governing the distribution of your Copper Sulphate and we believe that with our past experience and knowledge of the markets that we will be able to give you a satisfactory, continuous and desirable market not only in the domestic fields, but also for export.

Very truly yours,

MEFFORD CHEMICAL COMPANY

By
H. Percival Chase
Vice President in Charge
of Sales and Purchases

HPC:R
C.C. Mr. Bruce Clemmer
U.S. Bureau of Mines

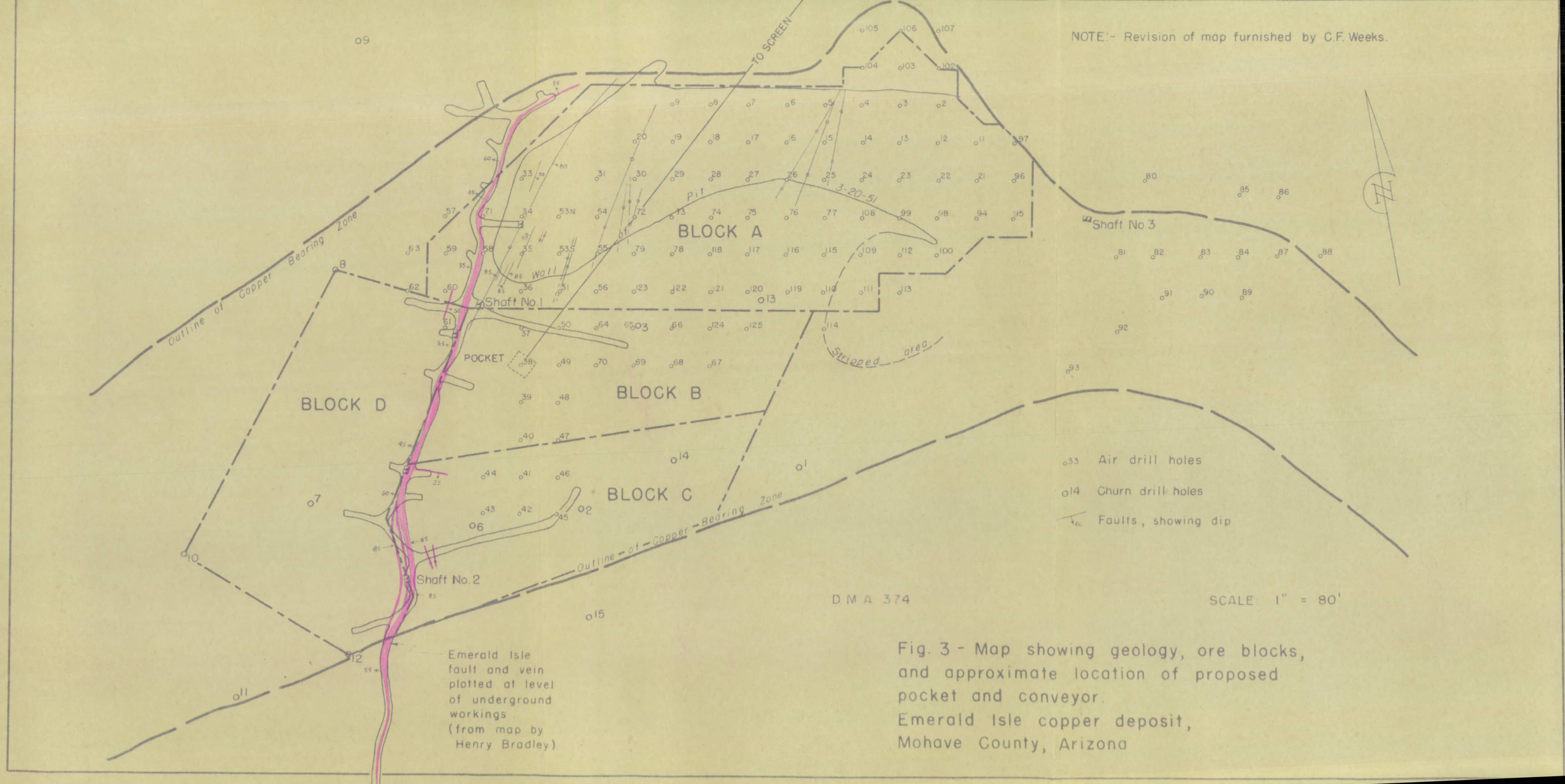

Fig. 3 - Map showing geology, ore blocks, and approximate location of proposed pocket and conveyor.
Emerald Isle copper deposit, Mohave County, Arizona