Bureau of Mines
Report of Investigations 4706

INVESTIGATION OF THE LAKE SHORE COPPER DEPOSITS, PINAL COUNTY, ARIZ.

BY T. M. ROMSLO

United States Department of the Interior — July 1950

INVESTIGATION OF THE LAKE SHORE COPPER DEPOSITS, PINAL COUNTY, ARIZ.

BY T. M. ROMSLO

* * * * * * * * * Report of Investigations 4706

UNITED STATES DEPARTMENT OF THE INTERIOR
Oscar L. Chapman, Secretary
BUREAU OF MINES
James Boyd, Director

Work on manuscript completed March 1950. The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is made: "Reprinted from Bureau of Mines Report of Investigations 4706."

July 1950

INVESTIGATION OF THE LAKE SHORE COPPER DEPOSITS, PINAL COUNTY, ARIZ.

by

T. M. Romslo[1]

CONTENTS

1/ Mining engineer, U. S. Bureau of Mines, Tucson Branch, Minerals Division, Tucson, Ariz.

Report of Investigations 4706

TABLES

ILLUSTRATIONS

INTRODUCTION AND SUMMARY

The Lake Shore property, located in the early 1880's, contains copper-bearing deposits that have been developed by surface excavations, underground workings, and churn-drill holes. Intermittent operation of the property ended in 1929 with a total recorded production of 280,000 pounds of copper.

The property is near the foot of the Slate Mountains, which are made up mainly of schist, probably the Pinal formation of pre-Cambrian age. In the mine area there are a few outcrops of granite, which is exposed over a large area east of the property. Other outcropping rocks on the property are limestone, quartzite, and diabase. The limestone and quartzite probably are the Mescal and Troy formations of pre-Cambrian and Cambrian age, respectively.

The predominant copper mineral is chrysocolla, a hydrous silicate that occurs mainly as fracture filling in bedded schist. It is also the principal copper mineral in the shear zone at the schist-granite contact and in limestone southeast of the main workings.

Investigation of the Lake Shore property by the Bureau of Mines included both topographic and geologic mapping, exploratory drilling, and metallurgical test work. One diamond-drill hole and five churn-drill holes were completed for a total of 2,872.5 feet. Drilling started January 19 and was completed May 13, 1949.

ACKNOWLEDGMENTS

These investigations were initiated in 1942 when O. M. Bishop, formerly a mining engineer of the Bureau of Mines, examined the property with the object of determining ore reserves and obtaining samples for metallurgical tests. Appreciation is extended to Frank M. Leonard, Jr., one of the owners of the property, for accompanying the engineer during the examination, for relating the history of the property, and for supplying an assay map of the mine workings and assay graphs of the churn drill holes. Later in the same year, T. C. Denton, also a former mining engineer of the Bureau, obtained additional samples for metallurgical tests.

The Bureau wishes to thank Nels P. Peterson of the U. S. Geological Survey for mapping both the surface and the underground geology during brief visits to the property in January and March 1949.

The investigations made during the Bureau's drilling program were supervised by J. H. Hedges, Chief, Tucson Branch, Mining Division, and analytical work was by Ray Stiles, under J. Bruce Clemmer, chief, Tucson Branch, Metallurgical Division. Metallurgical tests by the Bureau in 1942 and 1943 were made at the Salt Lake City station with H. G. Poole in charge. Clemmer and

Carl Rampacek conducted the tests at Tucson in 1949 and prepared the text on metallurgical tests. Transit surveys of the surface and underground workings, started by the author, were completed by M. H. Berliner, mining engineer of the Tucson Branch, Bureau of Mines.

Acknowledgment is made to the Indian Service of the Department of the Interior for grading an entry road to the mine and for providing a source of domestic and drilling water from a well at the nearby Indian Village of Komelik.

LOCATION AND ACCESSIBILITY

The Lake Shore mine is in the Papago Indian Reservation and the Casa Grande mining district, Gila and Salt River Base Line and Meridian, secs. 25 and 36, T. 10 S., R. 4 E., Pinal County, Ariz. (fig. 1). It may be reached from Casa Grande, a town on the Southern Pacific Railroad and State Highway 80, by traveling southwestward 28.2 miles on a well-maintained dirt road and thence 2.6 miles east on a desert road to the property.

PHYSICAL FEATURES AND CLIMATE

The Lake Shore mine is on the southwest piedmont of the Slate Mountains at an altitude of about 1,800 feet. The mountain range trends northwestward and reaches its maximum altitude of 3,330 feet at Prieta Peak, about 2 miles north of the mine.

Vegetation is of the desert variety, typical of the lower altitudes of southern Arizona. Palo Verde trees and Saguaro cactus are prominent.

Winters are mild and summers are hot. At Ajo, about 60 miles west of the property, the annual mean temperature is 71°, with a range from 17° to 115°. The annual precipitation averages about 9.3 inches.

PROPERTY AND OWNERSHIP

The Lake Shore property consists of three patented lode mining claims: the Arizona, Copper Bell, and Isabella (fig. 2). N. Frank Leonard, Butte, Mont., owns 96 percent of the stock of the Hidden Treasure Mining Co., which is the holder of the property.

There are no buildings or equipment on the property.

HISTORY AND PRODUCTION

The mine was located early in the 1880's by Trout and Atchinson. A shaft was sunk, and some drifting was done before 1884, when the property was abandoned because of failure of the copper market. In 1905, B. S. Wilson relocated the mine and shipped some ore sorted from the dump. In 1914 he sold the property to Frank M. and Charles Leonard. A new shaft was sunk to the 225-foot level, and development of the ore body was started on three levels. In 1917 the Atlas Development Co., Chicago, Ill., leased the mine and shipped 850 tons of 5.2 percent copper ore to a smelter at Sasco, Ariz. In 1919,

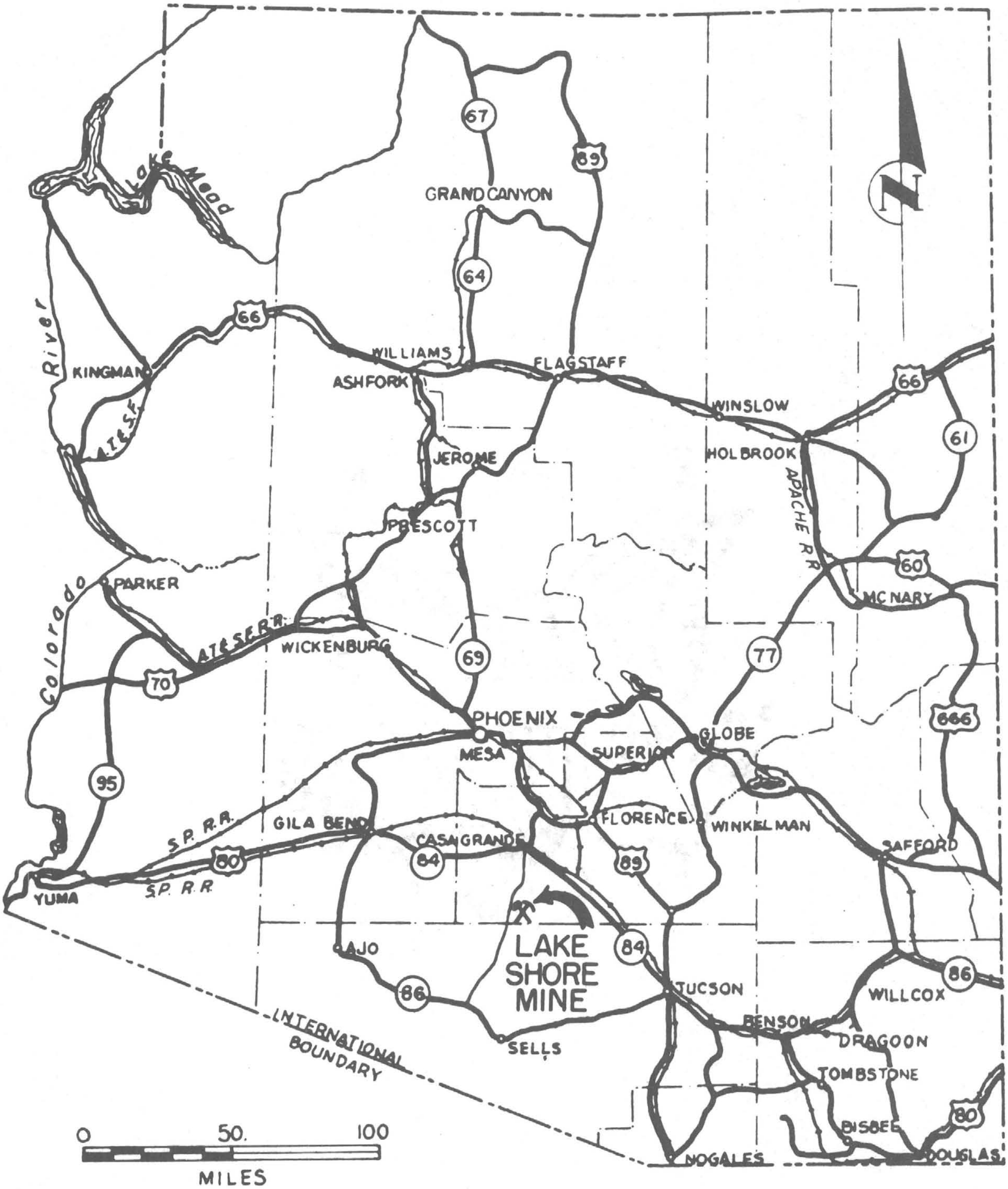


Figure I. - Location map, Lake Shore copper project, Pinal County, Ariz.

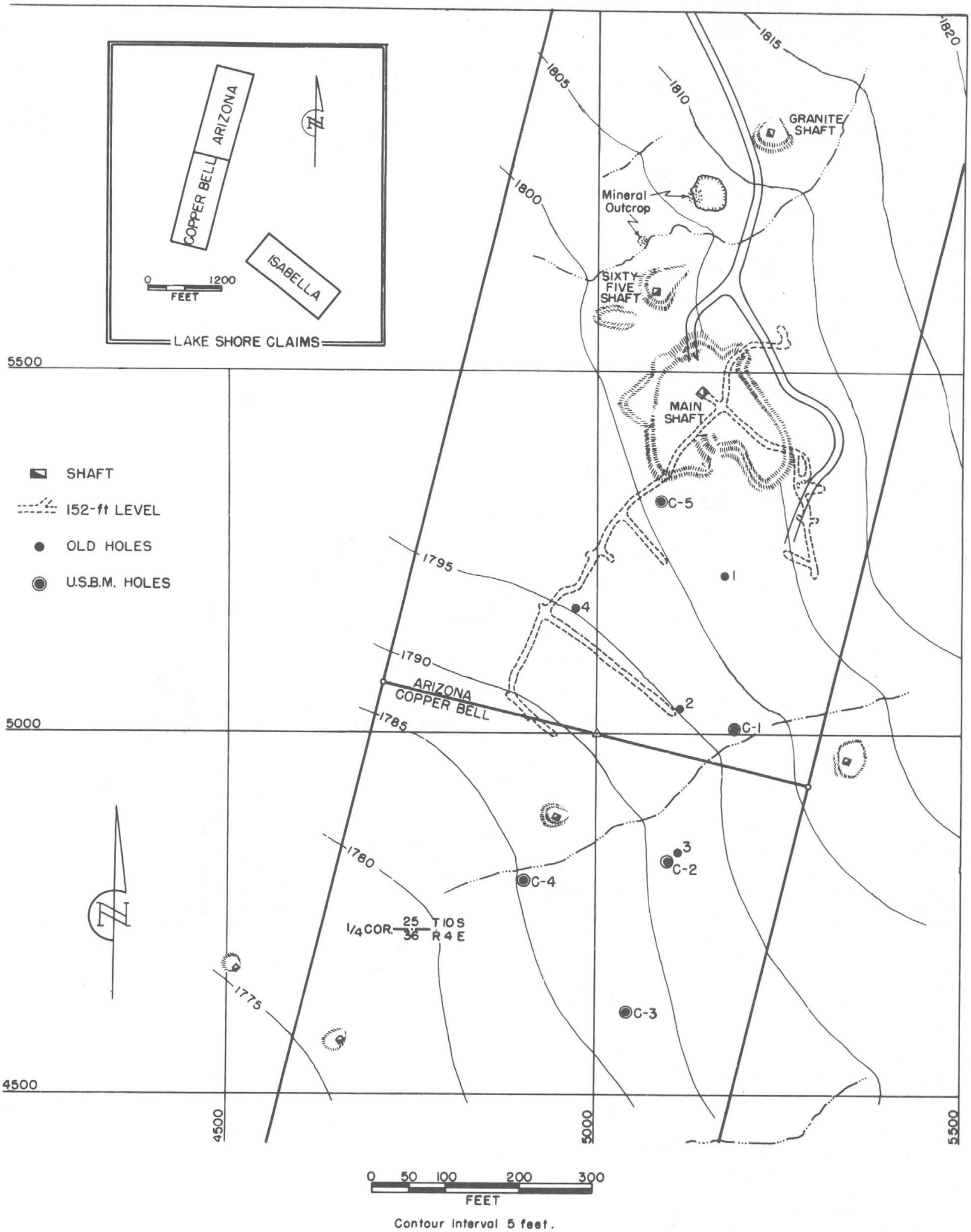


Figure 2. - Surface map, Lake Shore copper deposits, Pinal County, Ariz.

after terminating the lease, the Leonards drilled 5 churn drill holes and sank two winzes. During this period 12 tons of 15 percent copper ore in sulfide form was mined from the schist-granite contact zone on the 285-foot level. The last reported production was in 1929, when ore was trucked from the mine dump to Casa Grande for shipment.

Total production from the property is reported to have been 280,000 pounds of copper.[2]

GEOLOGY

General

The Slate Mountains are composed mainly of schist, tentatively identified as the Pinal formation of pre-Cambrian age. Biotite granite has intruded the schist near the southwest end of the mountain range. It crops out over a very small area on the Isabella claim and is prominently exposed east of the Lake Shore property. Other rock exposures on the property are confined to a small area of altered schist on the Arizona claim and to limestone, quartzite, and diabase on the Isabella claim. The limestone and quartzite are probably the Mescal and Troy formations of pre-Cambrian and Cambrian age, respectively.

Deposits

Copper mineralization is associated with a fault that has an average strike of about S. 11° W. and a dip of 60° to 70° west (figs. 3 and 4). Granite, probably an integral part of the intrusive mass, forms the block east of the fault. On the west side of the fault is a bed of highly altered, intensely fractured, fine-grained rock that has been classified as schist. A thin bed of quartzite is spottily present near the base of the schist. Underlying the schist is an intensely altered mass of rock tentatively classified as andesitic lava or tuff. Part of this formation can be identified megascopically as andesite. Spottily present in the andesite is a very fine-grained unidentified rock of light color and stony appearance. Of similar occurrence and texture is a dark-colored rock tentatively identified as basalt. The schist strikes about S. 37° W. and dips 37° to 45° east. South of the main shaft, a comparatively small body of granite is in contact, on the west, with the fault.

Copper mineralization occurs sparingly throughout the bedded rocks but is concentrated mainly at the base of the schist and in the fault zone. The planes of the fault and the planes of the bedded rocks diverge to form a trough that plunges to the southwest at an angle of about 24°.

Mineralogy

The following is an analysis of a 158-pound sample submitted to the Salt Lake City Station for metallurgical testing in 1942.

2/ Elsing, M. J., and Heineman, R.E.S., Arizona Metal Production: University of Ariz. Bull. 140.

Insol.	Oxide								
	SiO_2	Fe	CaO	S	Cu	Cu*	Al_2O_3	Zn	Pb
49.4	37.1	17.5	5.1	Nil	2.3	2.15	6.5	Nil	Nil

*Soluble in dilute H_2SO_4 saturated with sulfur dioxide.

The late R. E. Head,[3] of the Bureau of Mines, stated:

> Examination of thin sections prepared from representative pieces of the ore indicate that basically two types of copper association are represented. In addition to the copper-bearing material, there appears to be also an indeterminate quantity of rock that is virtually free of copper.
>
> In the one type of copper occurrence, the ground mass is almost entirely quartzitic. Chrysocolla, the copper silicate, occurs in this type of rock as a filling in fractures both in the rock itself and in the quartz particles.
>
> In some of these fracture fillings the chrysocolla occurs as masses of hairlike fibers intermixed with calcite and claylike material. In addition to this type of association, the chrysocolla is also present as a shell or coating on many of the quartz particles. In some cases, aggregates of very small quartz particles are cemented together with chrysocolla, which occurs as films so thin as to amount to scarcely more than stains.
>
> In the other type of association, the chrysocolla is distributed uniformly through the claylike ground mass in the form of minute veinlets and also as fracture fillings. This association of chrysocolla with the gangue is very intimate, and examination of thin sections showed that the individual clay particles were ringed with copper carbonate.
>
> The ore contains an appreciable quantity of magnetic iron oxide, magnetite.

Subsequent investigation of other samples of the ore in connection with metallurgical testing showed the copper to be present mainly in the silicate form as chrysocolla and some dioptase. Also present is a yellowish copper mineral, which is probably a silicate. A trace of sulfide-copper is present mainly as chalcocite.

A little pyrite and a small amount of native copper were seen in the cuttings from the fault zone at churn-drill hole C-2.

3/ Head, R. E. (deceased), Preliminary Microscopic Examination of oxidized ore from the Lake Shore Mines, Arizona: August 1942.

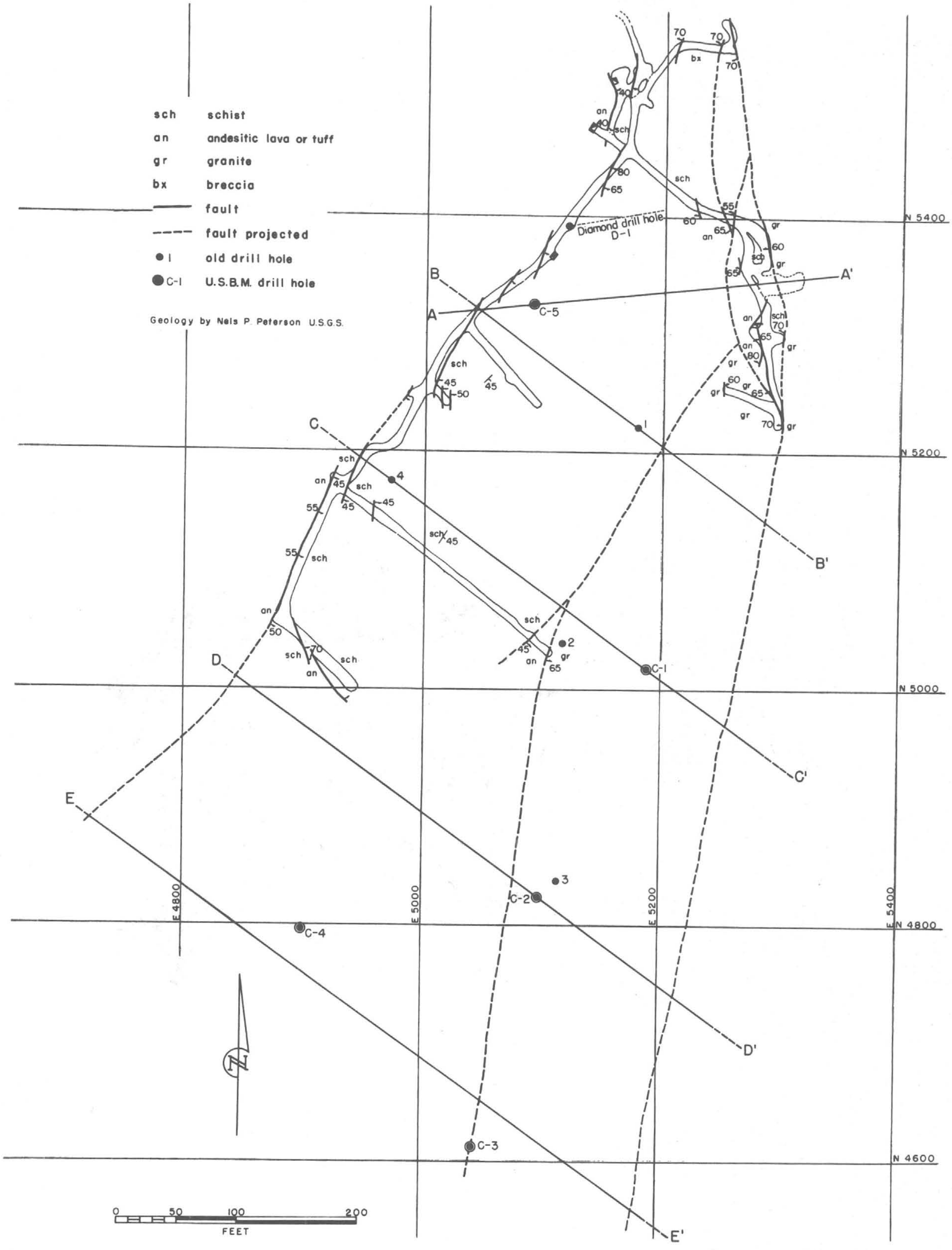


Figure 3. - Geologic map, 152-foot level, Lake Shore copper deposits, Pinal County, Ariz.

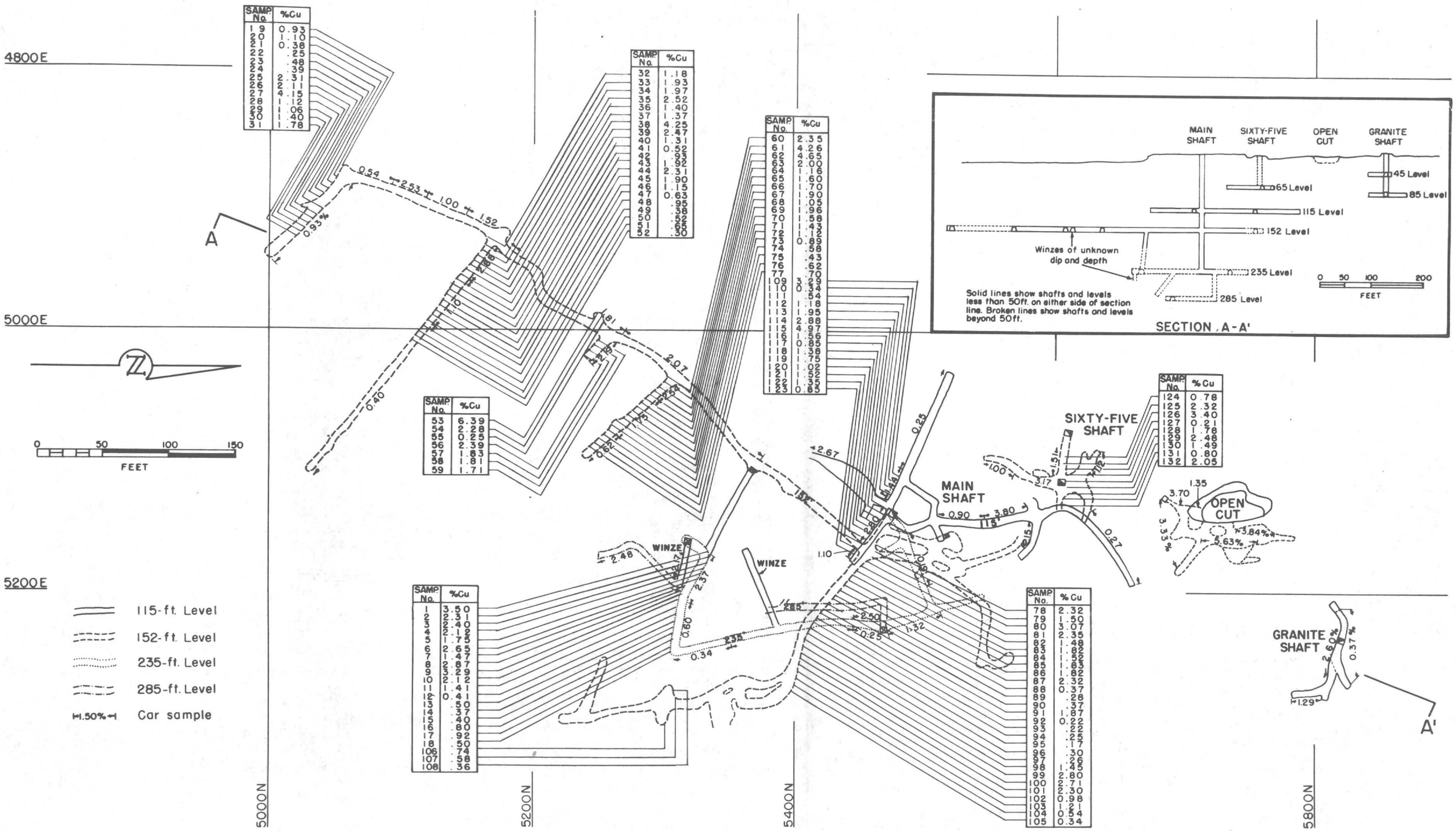


Figure 4. - Assay map, Lake Shore copper deposits, Pinal County, Ariz.

MINE WORKINGS (figs. 2, 3, and 4)

The main shaft is vertical and fully timbered into a 4-foot square hoisting compartment and a 2-1/2- by 4-foot manway compartment. It is 235 feet deep and at present is accessible to the water that stands at 221 feet below the collar of the shaft. Levels at depths from the surface of 115, 152, and 235 feet have been opened from the shaft, whereas the bottom or 285-foot level has been developed from two winzes sunk from the 235-foot level. Lineal development on the four levels consists of over 2,700 feet of drifts and crosscuts. Near the footwall of the bedded deposit are two small stopes on the 115-foot level and two on the 152-foot level (fig. 3). Another small stope on the 152-foot level is in the schist-granite contact zone.

The Sixty-Five shaft and the Granite shaft, both inaccessible, are situated 130 feet northwest and 350 feet northeast of the main shaft, respectively. The Sixty-Five shaft, 65 feet deep, has one level at its bottom. The Granite shaft has two levels - one at a depth of 45 feet and the other at its bottom of 83 feet. About midway between the two shafts is an open cut in the only surface exposure of ore on the property. It was the source of several cars of ore.

A longitudinal section through the main workings is shown on the assay map (fig. 4).

In addition to the above workings, there are several shallow shafts and pits.

WORK BY THE BUREAU OF MINES

Field Work

During examination of the mine by the Bureau of Mines in 1942, sampling was confined to the 115- and 152-foot levels, because the lower workings were flooded with the water, which stood at 228 feet below the collar of the shaft. Seven channel samples were cut to duplicate corresponding samples that are similarly numbered on figure 4. In addition, six samples, each weighing 25 to 55 pounds, were cut from six crosscuts. These, also, were channel samples and, with the exception of sample 100, were cut from channels that carry similar numbers. Sample 100 represents the material exposed in a section of the crosscut on the 115-foot level. Analyses of the samples are shown in table 1.

TABLE 1. - Analyses of channel samples

Level	Sample	Width, feet	Percent copper
115.............	114	5	3.54
115.............	115	5	4.39
115.............	116	5	1.90
152.............	61	5	2.69
152.............	62	5	2.18
152.............	63	5	2.81
152.............	64	5	1.28
115.............	100	38	2.69
115.............	113-116	20	2.17
152.............	25-31	35	1.90
152.............	32-39	40	2.27
152.............	60-63	20	2.56
152.............	78-87	50	1.74

A 158-pound sample was made of the six large samples for metallurgical tests. Later in the same year four additional samples were taken for metallurgical testing. Each of these represented 50 continuous feet of crosscut and ranged in weight from 272 to 619 pounds. They were taken from crosscuts at the shaft on the 115- and 152-foot levels and from the first and second crosscuts south of the shaft on the 152-foot level.

Active work on the exploratory project started November 22, 1948. The first truck loads of equipment and supplies, after being assembled and conditioned in Tucson, were hauled to the mine on December 6. While a complete camp to accommodate 25 to 30 men was being built and equipped, work was started on rehabilitation of the main shaft. Shaft work consisted of replacing the collar and second sets of timbers and making minor repairs to both the hoisting and manway compartments. A tripod was placed over the shaft, and a hoist was installed. Two 210-c.f.m. compressors were placed near the shaft, and an air line was installed to the site of diamond drill hole D-1. Track was laid, the drill station was drilled and blasted, and the muck was trammed to the shaft and hoisted to the surface in buckets. While the diamond drill hole was being drilled, the air line and track were advanced, and two more drill stations were drilled and blasted. The muck from these stations was hoisted to the surface after diamond drilling was completed. A total of about 100 tons of broken rock was removed from the mine.

A transit survey of the surface and underground workings started while the camp was being built showed that available maps could be used for laying out the drilling program. This work, as completed, included plumbing the main shaft, transit surveys of the 115- and 152-foot levels, and topographic surveys of the area shown in this report, the Isabella claim, and a 25-acre area adjoining the Isabella claim on the east.

Diamond drilling, consisting of one hole completed at a depth of 203.5 feet, was started January 19 and completed March 18. A vertical section through the hole is shown in figure 5, and the assays of samples are given in the log of the hole that is appended to this report. Original plans included diamond-drilling 6 or 8 holes from underground stations, each designed to

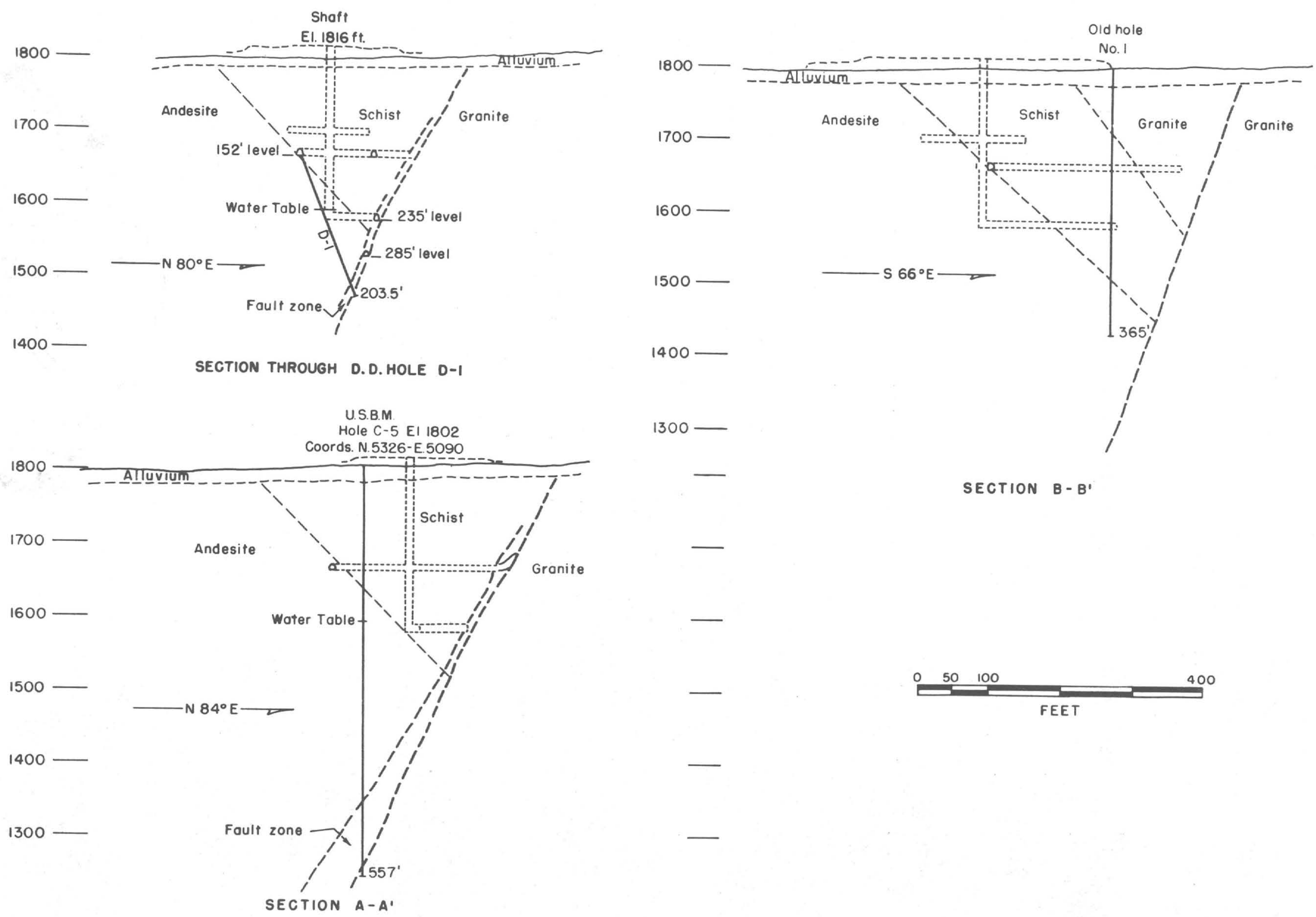


Figure 5. - Geologic sections, Lake Shore copper deposits, Pinal County, Ariz.

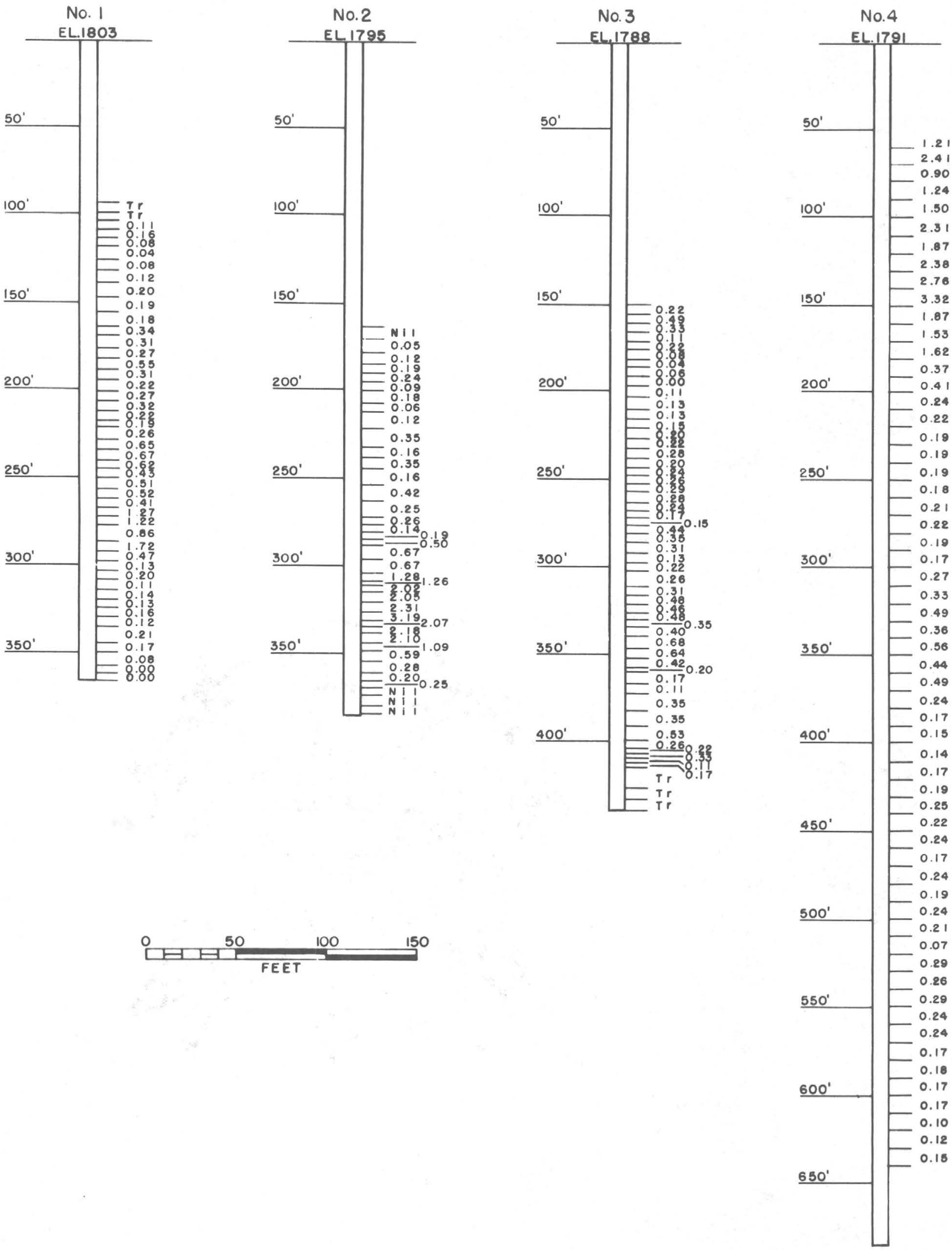


Figure 6. - Assay graphs, old churn drill holes, Lake Shore copper deposits.

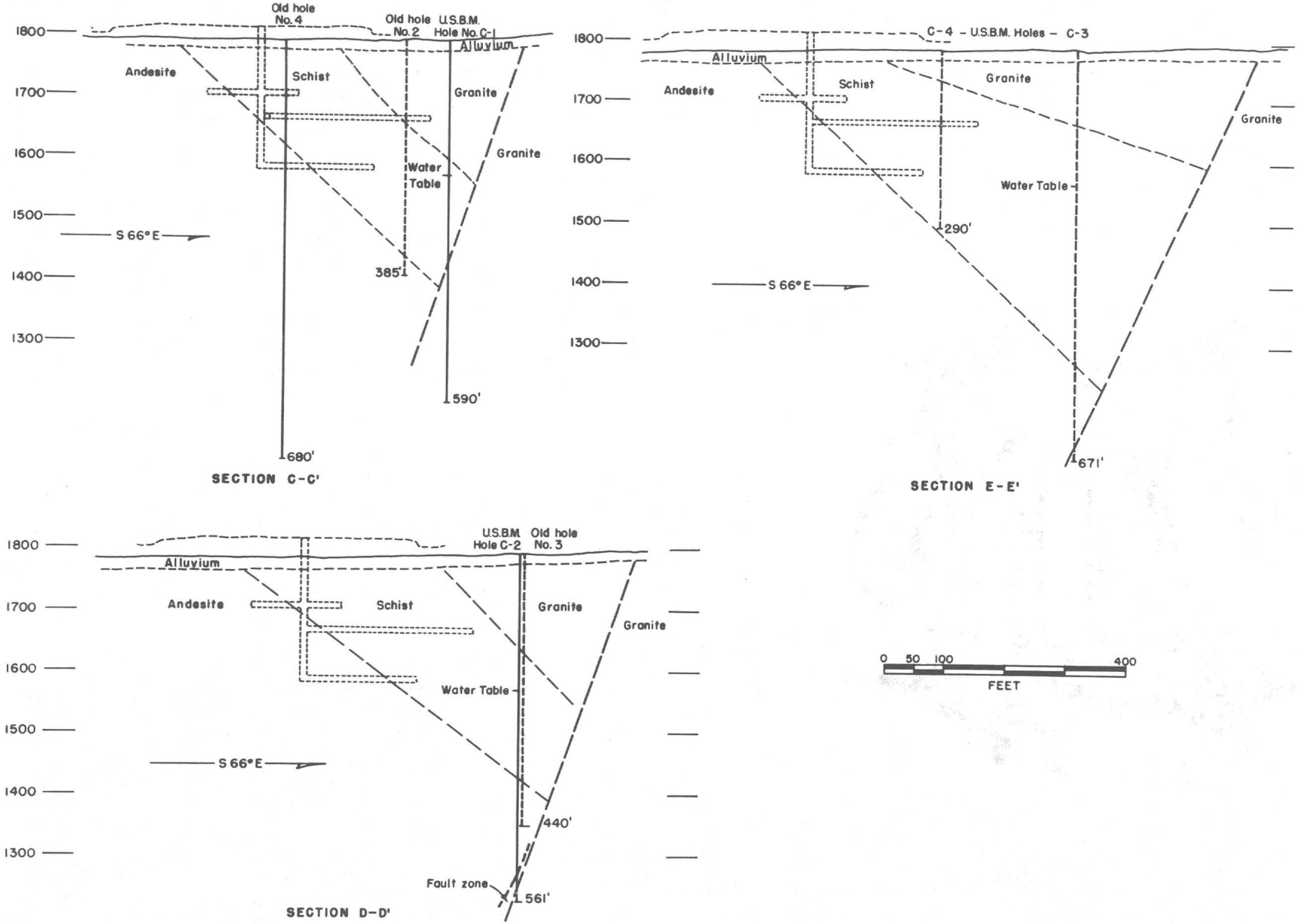


Figure 7. - Geologic sections, Lake Shore copper deposits, Pinal County, Ariz.

intersect the fault zone below the water table at intervals along the strike of the fault. Diamond drilling was terminated upon completion of one hole because costs were excessive to both the contractor and the Government. From the collar of the hole to the fault zone the rock is intensely fractured, and core recovery averaged about 6 percent. In general, after drilling a section of the rock the hole would close in as soon as the core barrel was removed.

Repeated cement jobs on portions of the hole failed, and in these cases it was necessary to drive the casing ahead. Attempts to advance the hole by blasting also failed. The contractor also tried unsuccessfully to keep the hole open and to consolidate the ground ahead of the bit by freezing. This operation consisted in using fuel oil cooled by dry ice as the circulating medium. Little trouble was experienced in penetrating the fault zone, where core recovery averaged 3.6 percent. The diamond drill was operated two shifts daily for 6 days a week. Double-tube core barrels 5 and 10 feet long were used. Drilling data for the hole, which was numbered D-1, follow:

Diamond-drilling data

Hole	Depth, ft.	Stand-pipe (3-inch)	Feet							
			Drilled			Reamed		Cased		
			NX	BX	AX	BX to NX	AX to BX	BX	AX	Cemented
D-1	203.5	11.0	34.0	94.0	64.5	33.0	18.0	78.0	157.0	146.5

Churn drilling, consisting of five holes for a total depth of 2,669 feet, was started January 13 and completed May 13, 1949. The rock was easy to drill, but, being ravelly, it was generally necessary to carry casing close to the bottom of the hole. The drill was operated two or three shifts daily, mainly on a two-shift basis, for 6 days a week.

Pertinent drilling data are given in table 2, and the logs of holes drilled by the Bureau are appended. Assay graphs of four of the churn-drill holes put down by the owners in 1919 are shown in figure 6.

Sections through the churn drill holes are shown on figure 7.

Drill-hole samples for analysis totaled 295, of which 56 were from the diamond-drill hole and 239 were from the churn-drill holes. Drill cuttings were dried, weighed, and reduced in size with a Jones splitter, and core samples were weighed and split. One half of each core sample and the samples of drill cuttings were sent to Tucson for analysis. The other half of the core was placed in core boxes, which were stored in the Bureau core house in Tucson. Two large samples of muck from the diamond-drill stations also were sent to Tucson for metallurgical tests.

Three thousand lineal feet of road work was done. This consisted of repairs to existing roads and building new roads to drilling sites but does not include 2.6 miles of 20-foot-wide road from the Casa Grande-Ajo road to the mine, which was built by the Indian Service of the Interior Department.

All drill holes were capped with a Bureau marker showing project number, hole number, and date of completion.

TABLE 2. - Churn-drilling data

Churn-drill hole	Feet											
		Drilled, bit size (inches)				Cased, pipe size (inches)				Reamed, bit size (inches)		
	Depth	12	10	8	6	12	10	8	6	10-12	8-10	6 - 8
C-1...	590.0	250.0	115.0	225.0		20.0	382.0			50.0	6.0	
C-2...	561.0	155.0	105.0	200.0	101.0	12	205.0	452.5	546.0	20.0	31.0	15.0
C-3...	671.0	250.0	90.0	160.0	171.0	194.0	321.0	477.0	641.0	67.0	46.0	111.0
C-4...	290.0	250.0	40.0			31.0						
C-5...	557.0	175.0	250.0	105.0	27.0	155.0	412.5	466.0	526.0	169.0		
	2,669.0	1,080.0	600.0	690.0	299.0	412.0	1,320.5	1,395.5	1713.0	306.0	83.0	126.0

Copper Analyses

The Lake Shore samples were analyzed for copper by conventional procedures. Total copper was determined by the long iodide method, using a mixture of hot concentrated hydrochloric, nitric, and sulfuric acid for decomposition of the minus 100-mesh samples. Samples that contained 0.5 percent or more of copper were reassayed for acid-soluble copper with a 5-percent solution of sulfuric acid saturated with sulfur dioxide to dissolve the copper silicates, oxides, and carbonates. Common practice is to report the acid-soluble assay as "oxide" copper, and the difference between the total and oxide assays is reported as "sulfide" copper.

Although such analyses would indicate that many of the Lake Shore samples contain 0.5 percent or more of sulfide copper, microscopic examination failed to reveal more than a trace of copper sulfides. Furthermore, the sulfur content of the samples was too small to account for this quantity of copper. Subsequent examination and microchemical tests on sink-float fractions of the Lake Shore ore indicated that this copper is associated with the gangue minerals as minute inclusions of an unidentified copper mineral that is somewhat more refractory toward leaching than chrysocolla.

The total and acid-soluble copper contents of samples from holes drilled by the Bureau are shown in the logs.

Metallurgical Tests[4]

The five samples from an examination of the mine in 1942 were submitted to the Salt Lake City Station for metallurgical tests. An analysis of a 158-pound character sample is shown in the section on mineralogy of the ore. The analyses of the other samples are given in table 3. The samples from crosscuts Nos. 1 to 3 on the 152-foot level, numbered to the south from the crosscut at the shaft, were identified as Nos. Ar-4.1, Ar-4.2, and Ar-4.3, respectively, and the sample from the 115-foot level was numbered Ar-4.4.

The Salt Lake City metallurgical tests revealed that the mineral association in the samples was too intimate for beneficiation by ore-dressing methods. Acid leaching of the ore was not attractive owing to the presence of lime, which caused excessive acid consumption. Tests employing the reducing-roast and ammonia-leach process extracted as much as 86 percent of the copper. In these tests, minus 20-mesh material was roasted with coke in an atmosphere of natural gas for 1 hour at 500° to 600° C. to reduce the copper. The samples were then cooled to 180° C. and quenched in water. Leaching was carried out at 25 percent solids in a combination air-mechanical agitation tank for 4 hours, using a 10 percent solution of ammonium hydroxide and ammonium carbonate in equal parts, containing the equivalent of 0.3 pound potassium cyanide per ton of ore. The leach residues were filter-washed with ammonia and water.

4/ Prepared by Carl Rampacek and J. Bruce Clemmer, metallurgists, Bureau of Mines, Tucson Branch, Metallurgical Division, Tucson, Ariz.

TABLE 3. - Analyses of metallurgical samples

Sample	Insol.	Percent										Oz./ton		Cu soluble in 10% solution (24 hr.)	
		SiO_2	Fe	CaO	S	Zn	Pb	Cu	Ox Cu*/	Al_2O_3	MgO	Au	Ag	H_2SO_4	NH_4OH
4.1	45.7	31.6	17.1	7.9	.08	Nil	Nil	1.71	1.60	7.9	11.3	Nil	Tr.	1.33	Nil
4.2	62.9	41.6	5.35	10.7	.07	0.15	Nil	1.29	1.28	9.9	10.6	Nil	Tr.	1.25	Nil
4.3	35.4	25.8	29.2	5.2	<.05	Nil	Nil	2.18	1.79	3.6	11.3	Nil	Tr.	1.75	Nil
4.4	66.2	54.6	7.25	4.5	<.05	Nil	Nil	1.66	1.59	5.7	9.7	Nil	Tr.	1.32	Nil

*/ Copper soluble in dilute sulfuric acid saturated with sulfur dioxide.

Metallurgical tests were made subsequently at the Tucson station on a composite sample taken from drill stations 1, 2, and 3 on the 152-foot level. Analysis of the sample gave 3.51 percent total copper, 2.96 percent acid-soluble copper, 8.25 percent iron, 1.73 percent calcium carbonate, 0.04 percent sulfate-sulfur, and 0.01 percent sulfide-sulfur. The copper was present predominately as chrysocolla and dioptase, with only traces of sulfides and carbonates.

Batch flotation of the ore ground to pass 65 or 200 mesh made with conventional sulfide and nonsulfide collecting agents failed to effect separation. The trace of sulfides, largely chalcocite, floated readily, but recovery of the chrysocolla and dioptase was poor, regardless of the conditions employed.

Acid leaching and leach-precipitation-flotation of the sample also were investigated. The results of a number of bottle leaching tests are summarized in table 4. The tests on portions of the ore ground to pass 10, 20, and 65 mesh were made at 50 percent solids with different quantities of acid and various contact periods.

The leaching tests revealed that about 375 pounds of acid, 4.1 times the theoretical based on the acid-soluble copper content of the feed, were required for a good extraction of copper from the 10, 20, and 65-mesh feeds. Although the finer material leached more rapidly, a 24-hour contact was essential for an 88 to 90 percent extraction of the total copper. The acid consumed varied from 4.1 to 4.4 pounds per pound of copper extracted. Neither longer leaching nor use of more acid materially improved copper extraction. Cursory tests on charges of the ore ground to 200 mesh gave slightly higher copper extractions but not enough to justify the added cost of finer grinding.

Although the chrysocolla in the ore is amenable to leaching, long contact with excessive acid is required to dissolve the 0.5 percent or more of copper that is intimately associated with the gangue. Tests were made to determine if the refractory copper could be extracted within a reasonable period by employing stronger acid solutions. The dry ore was mixed with the desired quantity of acid and enough water to give an agglomerated or pasty charge containing about 75 percent solids. A 50 percent acid solution proved adequate, but more concentrated acid was used in some of the tests. The agglomerated charges were permitted to stand at room temperature for various lengths of time and then were leached 15 minutes with water to extract the solubilized copper. Tests were made on 10-, 20-, and 65-mesh feeds with 375 pounds of acid per ton and varying the contact period from 1 to 24 hours. The stagnant leaching of the agglomerated charges gave copper extractions almost identical to those of bottle leaching at 50 percent solids, as recorded in table 4.

Although stagnant leaching of the acid-agglomerated charges at room temperature failed to improve extraction of the refractory copper, supplementary tests revealed that moderate heating of the agglomerules expedited solution of the copper for an improved recovery. The results of several tests on 10-, 20-, and 65-mesh portions of the ore are summarized in table 5. The charges were mixed for about 5 minutes with the quantity of acid shown and just enough water to form agglomerules. These were heated in a muffle furnace to give a substantially dry sulfated product, which was subsequently leached with water for 15 minutes to extract the copper. For convenience, the sulfated products were leached at 33 percent solids. In other tests, however, leaching at 50 percent solids gave equally good results, and it seems likely that adequate leaching could be obtained in even thicker pulps.

TABLE 4. - Bottle leaching of Lake Shore ore.

Leaching time, hr.	Mesh feed	H_2SO_4 added, lb./ton	H_2SO_4 consumed Lb./ton	H_2SO_4 consumed Lb/lb of copper extracted	Extraction, percent of total copper
1	65	105	102	3.8	38.5
1	65	155	151	3.7	58.7
1	65	205	180	3.4	75.5
1	65	260	198	3.6	78.3
1	65	310	209	3.7	80.9
1	65	360	210	3.6	82.3
1	65	410	219	3.8	82.3
1	65	205	180	3.4	75.5
2	65	205	183	3.3	78.1
4	65	205	195	3.4	80.3
1	65	375	201	3.5	82.3
4	65	375	231	3.8	87.3
8	65	375	239	3.8	88.6
12	65	375	247	4.0	89.2
24	65	375	259	4.1	90.0
1	20	375	198	3.6	77.8
4	20	375	232	3.9	84.3
8	20	375	250	4.2	84.9
12	20	375	264	4.3	87.2
24	20	375	275	4.4	89.7
1	10	375	171	3.4	72.6
4	10	375	212	3.7	82.3
8	10	375	228	3.8	84.9
12	10	375	236	3.9	86.0
24	10	375	258	4.2	88.3

TABLE 5. - Results of acid-sulfating tests.

Mesh of feed	Sulfating treatment			H_2SO_4 consumed		Extraction, percent of total copper
	H_2SO_4 added, lb./ton	Furnace temp., °C.	Time, Min.	Lb./ton	Lb/lb of copper extracted	
10	375	25	60	195	3.7	75.4
10	375	250	7.5	328	5.7	81.8
10	375	250	15	352	5.9	84.9
10	375	250	30	366	6.1	85.2
20	375	25	60	224	4.0	79.5
20	375	250	7.5	344	5.6	87.7
20	375	250	15	364	5.9	87.5
20	375	250	30	375	6.0	88.6
65	375	25	60	233	4.0	83.6
65	375	75	7.5	264	4.2	89.5
65	375	75	15	300	4.6	92.0
65	375	75	30	318	4.9	92.6
65	375	250	7.5	351	5.3	94.0
65	375	250	15	375	5.7	94.0
65	375	250	30	375	5.8	94.2
65	375	400	7.5	368	5.6	93.7
65	375	400	15	375	5.8	92.3
65	375	400	30	375	5.9	90.6
65	105	250	15	105	2.9	50.7
65	155	250	15	155	3.0	72.9
65	205	250	15	205	3.5	84.9
65	310	250	15	310	4.8	92.0
65	375	250	15	375	5.7	94.0
65	410	250	15	410	6.2	94.3

The tests demonstrated that moderate heating of an agglomerated or pasty charge converts the copper to the sulfate form, which is amenable to rapid leaching with water. Provided enough acid was used, a 7.5- to 15-minute heat at temperatures between 75° and 400° C. permitted good extraction of the copper from the 10-, 20-, and 65-mesh feeds. The optimum temperature for sulfating the Lake Shore ore appears to be about 250° C. Although a temperature of 400° C. is permissible, a higher temperature dehydrates the sulfate and necessitates leaching of the calcine with weak acid. Virtually all of the acid employed in the sulfating procedure is consumed. No free acid, or only minor quantities, was found in the leach liquors. The moderate heat treatment increases solution of the clay and iron minerals in the ore, and the acid consumed per pound of copper dissolved is higher than in bottle leaching. The greater consumption of acid, however, is offset by the higher extraction of copper and the shorter treatment period required. The sulfated charges from tests made at 250° C. were compact and dry, regardless of the quantity of acid used. The calcines produced at lower sulfating temperatures were slightly moist. No difficulty was experienced in leaching the calcines, as they slaked readily upon addition of water, and the copper sulfate dissolved rapidly. The leached residues thickened readily and were much easier to filter than those from the bottle leaching tests. The mild heat treatment apparently dehydrates the colloidal silica and increases the filtration rate.

Copper extraction in the acid-sulfating tests decreased with increasing coarseness of the feed. Incomplete extraction of the copper in the 10- and 20-mesh feeds may be attributed to slow diffusion of acid through the particles during the short agglomerating and heating periods. Acid-sulfating gave somewhat lower extractions on coarse feeds than bottle leaching. As regards the time required for comparable copper extractions, however, acid-sulfating is superior. The results of several tests by the two procedures are given in table 6.

TABLE 6. - Comparison of acid sulfating and bottle leaching of 10-, 20-, and 65-mesh ore.

Mesh of feed	Method	H_2SO_4 added, lb./ton	H_2SO_4 consumed		Extraction, percent of total copper
			Lb./ton	Lb./lb. of copper extracted	
10......	15-min. acid sulfating and 15-min. water leach.	375	352	5.9	84.9
10......	8-hour bottle leach.	375	228	3.8	84.9
20......	15-min. acid sulfating and 15-min. water leach.	375	364	5.9	87.5
20......	12-hour bottle leach.	375	264	4.3	87.2
65......	15-min. acid sulfating and 15-min. water leach.	375	375	5.7	94.0
65......	72-hour bottle leach.	412	377	5.6	93.8

Supplementary tests were made to observe the deportment of the ore toward leaching-precipitation-flotation. The results of a typical leach-float test employing bottle leaching are given in table 7. The ore was ground in a rod mill to pass 65 mesh and leached for 2 hours at 50 percent solids, 205 pounds of sulfuric acid being used per ton of ore. Part of the free acid remaining in the pulp was neutralized with hydrated lime, and the cement copper was then precipitated with iron nails. After neutralization of substantially all the remaining free acid, the cement copper was floated, Minerac A being used as the collector. Single-cleaning of the rougher froth yielded a cement copper concentrate that assayed 71.42 percent copper and represented a recovery of 73.2 percent. Leach-flotation of 200-mesh portions of the ore gave almost identical results. Depending on the reagents employed, 76 to 80 percent of the copper was recovered as a rougher product assaying 35 to 40 percent copper. Inability to obtain a higher copper recovery by leach-flotation can be attributed to incomplete dissolution of the refractory copper silicate rather than to inferior flotation of the cement copper. The copper content of flotation tailings and of residues from comparable leaching tests were almost identical.

TABLE 7. - Bottle leaching-precipitation-flotation of 65-mesh ore.

Product	Weight, percent	Assay, percent total copper	Distribution, percent total copper
Copper concentrate.......	3.5	71.42	73.2
Middling.................	9.3	1.77	4.8
Rougher froth............	12.8	20.82	78.0
Tailing..................	87.2	0.86	22.0
Composite................	100.0	3.42	100.0

Reagent	Pounds per ton						
				Flotation			
	Leaching	Precipitation		Conditioner		Rougher	Cleaner
H_2SO_4........	206	-		-		-	-
$Ca(OH)_2$.......	-	16.0		24.0		-	-
Minerac A.....	-	-			0.2	-	-
Pine oil......	-	-			-	0.04	0.02
			Iron nails				
Time (min.)...	120	15	30	5	2.5	5	2.5
pH............	1.75	2.85	3.40	4.90	-	4.5	5.0

Other tests were made with more acid in the leaching step in an effort to obtain more complete extraction of the copper. The tests were not successful. The large quantity of free acid remaining in the leached pulp vitiated both precipitation and flotation of the cement copper. Prohibitive quantities of lime were required to neutralize the acid, and the pulps became so contaminated with salts that flotation of the cement copper was incomplete. When neutralizing steps were omitted, precipitation of the copper was incomplete, and much iron was dissolved by the free acid. The iron salts and residual acid inhibited subsequent flotation of the copper. These and other tests demonstrated that excess acid must be avoided in conventional leach-float procedures.

Precipitation-flotation tests also were made on acid-sulfated charges. The results of a typical test made on the 65-mesh feed and employing 375 pounds of acid per ton for sulfating are given in table 8. The acid-agglomerated ore was heated 15 minutes at 250° C. and then leached 15 minutes with water at 50 percent solids. As the leach pulp was substantially free of acid, the neutralizing steps before copper precipitation and flotation were not necessary. Single-stage cleaning of the rougher froth yielded a cement copper concentrate that assayed 69.7 percent copper and represented a recovery of 89.7 percent; the rougher concentrate accounted for 90.7 percent of the copper. Flotation of the cement was excellent and copper losses in the tailings were due primarily to presence of undissolved silicates.

Excellent results also were obtained on acid-sulfated charges of the ore by precipitating the copper during the water-leaching step. Simultaneous leaching and precipitation gave a somewhat finer and darker-colored cement copper than two-stage treatment, but it was readily amenable to flotation.

TABLE 8. - Precipitation-flotation of acid-sulfated ore.

Product	Weight, percent	Assay, percent total copper	Distribution, percent total copper
Copper concentrate.....	4.4	69.70	89.7
Middling...............	4.9	0.71	1.0
Rougher froth..........	9.3	33.35	90.7
Tailing................	90.7	0.35	9.3
Composite..............	100.0	3.42	100.0

Reagent	Pounds per ton: Sulfating treatment	Water extraction[1/]	Precipitation	Flotation: Conditioner	Rougher	Cleaner
H_2SO_4.............	375		-	-	-	-
Minerec A........			-	0.30	-	-
Pine oil..........			-	0.04	0.04	-
			Iron nails			
Temperature, $^{\circ}$C. .	250					
Time, minutes.....	15		30	2.5	5	2.5
pH................		3.15	3.15	3.5	3.5	3.6

1/ 15-minute agitation with water at 50 percent solids at room temperature.

Summary and Conclusions of Metallurgical Tests

The Lake Shore ore is refractory toward leaching. A long contact period with a large excess of acid is necessary to obtain a high copper extraction. Acid-sulfating at temperatures between 75° and 400° C. is superior to conventional leaching. Acid-sulfating requires more acid than flood or trickle leaching but is offset by the higher copper extraction and the shorter treatment period required. On other less refractory ores, the quantities of acid required for acid sulfating and bottle leaching were almost identical.

The leach liquors from conventional leaching of the Lake Shore ore contain much free acid, whereas, those from acid-sulfated charges were virtually free of acid. In leaching-precipitation or leaching-precipitation-flotation procedures, where free acid in the leach liquor or ore pulp is objectionable, acid sulfating should have merit.

Flotation of the Lake Shore ore by usual sulfide and nonsulfide collectors was ineffective. Leach-precipitation-flotation gave good copper recoveries. In conjunction with the leach-float procedure, acid-sulfating was superior to bottle leaching. When using flood or trickle leaching, the excess acid remaining in the pulp must be partly neutralized before precipitation and flotation of the cement copper. As virtually no free acid remains in the acid-sulfated pulps, the neutralizing steps before precipitation and flotation are unnecessary, thus simplifying the procedure. Simultaneous leaching and precipitation of the copper from acid-sulfated charges also gave good results.

DRILL-HOLE LOGS

Hole D-1

Location: N. 5391, E. 5119
Elevation of collar: 1,664 ft.
Depth: 203.5 ft.

Dip: -73°
Bearing: N. 80° E.
Date: 1/19 to 3/18/49

Footage			Percent copper		Oz./ton		
From-	To-	Feet	Total	Acid-soluble	Au	Ag	Description and remarks
0	11.0	11.0	1.48	1.48	......	....	Schist.
11.0	16.0	5.0	.26				Andesite.
16.0	21.0	5.0	.28				Do.
21.0	26.0	5.0	.25				Do.
26.0	32.0	6.0	.44				Do.
32.0	35.0	3.0	.40				Do.
35.0	40.0	5.0	.33				Do.
40.0	45.0	5.0	.34				Do.
45.0	50.0	5.0	.25				Do.
50.0	53.0	3.0	.20				Do.
53.0	58.0	5.0	.21				Do.
58.0	61.5	3.5	.20				Do.
61.5	65.5	4.0	.24				Do.
65.5	70.5	5.0	.20				Do.
70.5	75.5	5.0	.17				Do.
75.5	78.0	2.5	.18				Do.
78.0	81.5	3.5	.17				Do.
81.5	86.2	4.7	.18				Do.
86.2	88.2	2.0	.18				Do.
88.2	90.7	2.5	.18				Do.
90.7	94.6	3.9	.28				Do.
94.6	99.6	5.0	.14				Do.
99.6	104.9	5.3	.17				Do.
104.9	110.0	5.1	.17				Do.
110.0	115.0	5.0	.18				Do.
115.0	120.0	5.0	.19				Do.
120.0	125.0	5.0	.17				Do.
125.0	127.0	2.0	.10				Do.
127.0	132.0	5.0	.17				Do.
132.0	137.0	5.0	.13				Do.
137.0	140.0	3.0	.19				Do.
140.0	145.0	5.0	.19				Do.
145.0	148.3	3.3	.19				Do.
148.3	153.3	5.0	.22				Do.
153.3	156.7	3.4	.16				Do.
156.7	161.7	5.0	.14				Do.
161.7	163.7	2.0	.13				Do.
163.7	168.5	4.8	.13				Do.
168.5	173.5	5.0	.13				Do.
173.5	178.5	5.0	1.45	1.20	Tr	0.1	Shear zone.
178.5	180.5	2.0	.46	.25			Do.
180.5	185.5	5.0	.83	.57			Do.
185.5	189.2	3.7	.89	.60			Do.
189.2	193.5	4.3	.68	.42			Do.
193.5	203.5	10.0					Granite.

Hole C-1

Location: N. 5007, E. 5188 — Depth: 590.0 ft.
Elevation of collar: 1,796 ft. — Date: 1/13 to 2/4/49

Footage			Percent copper		Description and remarks
From-	To-	Feet	Total	Acid-soluble	
0	20	20.0			Sand and gravel.
20	193	173.0			Weathered granite.
193	195	2.0	0.31		Schist and clay.
195	205	10.0	.33		Schist.
205	215	10.0	.36		Schist, water table at 211.0 ft.
215	225	10.0	.32		Schist.
225	235	10.0	.31		Do.
235	245	10.0	.27		Do.
245	250	5.0	.34		Do.
250	255	5.0	.26		Do.
255	265	10.0	.36		Do.
265	275	10.0	.41		Do.
275	285	10.0	.39		Do.
285	295	10.0	.27		Do.
295	305	10.0	.35		Do.
305	310	5.0	.39		Do.
310	315	5.0	.42		Do.
315	320	5.0	.43		Do.
320	325	5.0	.61	0.36	Do.
325	330	5.0	.57	.29	Do.
330	335	5.0	.51	.28	Do.
335	340	5.0	.34		Quartzite and schist.
340	345	5.0	.36		Do.
345	350	5.0	.28		Do.
350	355	5.0	.38		Do.
355	360	5.0	.39		Do.
360	365	5.0	.25		Contact - schist and granite.
365	370	5.0	.16		Schist and granite.
370	375	5.0	.14		Granite and schist.
375	380	5.0	.10		Do.
380	385	5.0	.14		Do.
385	550	165.0			Granite.
550	555	5.0			Shear zone, clay.
555	590	35.0			Granite.

Hole C-2

Location: N. 4813, E. 5098 Depth: 561.0 ft.
Elevation of collar: 1,792 ft. Date: 2/11 to 3/5/49

Footage			Percent copper		
From-	To-	Feet	Total	Acid-soluble	Description and remarks
0	20	20.0			Sand and gravel.
20	155	135.0			Weathered granite.
155	165	10.0	0.23		Schist.
165	175	10.0	.22		Do.
175	185	10.0	.22		Do.
185	195	10.0	.23		Do.
195	205	10.0	.33		Do.
205	215	10.0	.32		Do.
215	225	10.0	.32		Schist, water table at 220 ft.
225	235	10.0	.31		Schist.
235	245	10.0	.48		Do.
245	255	10.0	.53	0.06	Do.
255	260	5.0	.51	.06	Do.
260	270	10.0	.62	.06	Do.
270	280	10.0	.46		Do.
280	290	10.0	.54	.06	Do.
290	300	10.0	1.03	.22	Do.
300	305	5.0	.57	.19	Do.
305	310	5.0	.50	.15	Do.
310	315	5.0	1.25	.55	Do.
315	320	5.0	.89	.40	Do.
320	325	5.0	.90	.33	Do.
325	330	5.0	.91	.31	Do.
330	335	5.0	.80	.28	Do.
335	340	5.0	.54	.17	Do.
340	345	5.0	.98	.29	Do.
345	350	5.0	.63	.19	Do.
350	355	5.0	.62	.18	Do.
355	360	5.0	.68	.21	Do.
360	365	5.0	.34		Do.
365	370	5.0	.31		Andesite.
370	375	5.0	.29		Do.
375	380	5.0	.26		Do.
380	385	5.0	.28		Do.
385	390	5.0	.62	0.09	Do.
390	395	5.0	.31		Do.
395	405	10.0	.24		Do.
405	415	10.0	.22		Do.
415	425	10.0	.22		Do.
425	435	10.0	0.26		Do.
435	445	10.0	.19		Do.
445	455	10.0	.15		Do.
455	460	5.0	.19		Do.
460	470	10.0	.18		Quartzite.
470	480	10.0	.18		Do.
480	490	10.0	.14		Do.

Hole C-2, Cont'd.

Footage			Percent copper		
From-	To-	Feet	Total	Acid-soluble	Description and remarks
490	500	10.0	0.16		Andesite.
500	510	10.0	.16		Do.
510	520	10.0	.23		Do.
520	525	5.0	.51	0.19	Shear zone.
525	530	5.0	.79	.48	Do.
530	535	5.0	.89	.63	Do.
535	540	5.0	1.43	.91	Shear zone. Little pyrite and native copper.
540	545	5.0	1.27	.57	Shear zone.
545	547	2.0	.74	.28	Schist.
547	550	3.0	.72	.30	Schist and granite.
550	555	5.0	.65	.27	Do.
555	561	6.0			Granite.

Hole C-3

Location: N. 4610, E. 5045 Depth: 671.0 ft.
Elevation of collar: 1,788 ft. Date: 3/9 to 4/2/49

Footage			Percent copper		
From-	To-	Feet	Total	Acid-soluble	Description and remarks
0	10	10			Sand and gravel.
10	125	115.0			Weathered granite.
125	135	10.0	0.33		Schist.
135	145	10.0	.22		Do.
145	155	10.0	.31		Do.
155	165	10.0	.40		Do.
165	175	10.0	.32		Do.
175	185	10.0	.27		Do.
185	195	10.0	.23		Do.
195	205	10.0	.16		Do.
205	215	10.0	.10		Do.
215	225	10.0	.16		Schist, water table at 225 ft.
225	235	10.0	.17		Schist.
235	245	10.0	.22		Do.
245	255	10.0	.17		Do.
255	265	10.0	.16		Do.
265	275	10.0	.25		Schist, shear - much Fe oxide.
275	285	10.0	.25		Schist.
285	295	10.0	.27		Do.
295	305	10.0	.30		Schist, shear - much Fe oxide.
305	315	10.0	.32		Schist.
315	325	10.0	.25		Do.
325	335	10.0	.27		Schist, shear - much Fe oxide.
335	345	10.0	.22		Schist.
345	355	10.0	.20		Do.

Hole C-2

Location: N. 4813, E. 5098 Depth: 561.0 ft.
Elevation of collar: 1,792 ft. Date: 2/11 to 3/5/49

Footage			Percent copper		
From-	To-	Feet	Total	Acid-soluble	Description and remarks
0	20	20.0			Sand and gravel.
20	155	135.0			Weathered granite.
155	165	10.0	0.23		Schist.
165	175	10.0	.22		Do.
175	185	10.0	.22		Do.
185	195	10.0	.23		Do.
195	205	10.0	.33		Do.
205	215	10.0	.32		Do.
215	225	10.0	.32		Schist, water table at 220 ft.
225	235	10.0	.31		Schist.
235	245	10.0	.48		Do.
245	255	10.0	.53	0.06	Do.
255	260	5.0	.51	.06	Do.
260	270	10.0	.62	.06	Do.
270	280	10.0	.46		Do.
280	290	10.0	.54	.06	Do.
290	300	10.0	1.03	.22	Do.
300	305	5.0	.57	.19	Do.
305	310	5.0	.50	.15	Do.
310	315	5.0	1.25	.55	Do.
315	320	5.0	.89	.40	Do.
320	325	5.0	.90	.33	Do.
325	330	5.0	.91	.31	Do.
330	335	5.0	.80	.28	Do.
335	340	5.0	.54	.17	Do.
340	345	5.0	.98	.29	Do.
345	350	5.0	.63	.19	Do.
350	355	5.0	.62	.18	Do.
355	360	5.0	.68	.21	Do.
360	365	5.0	.34		Do.
365	370	5.0	.31		Andesite.
370	375	5.0	.29		Do.
375	380	5.0	.26		Do.
380	385	5.0	.28		Do.
385	390	5.0	.62	0.09	Do.
390	395	5.0	.31		Do.
395	405	10.0	.24		Do.
405	415	10.0	.22		Do.
415	425	10.0	.22		Do.
425	435	10.0	0.26		Do.
435	445	10.0	.19		Do.
445	455	10.0	.15		Do.
455	460	5.0	.19		Do.
460	470	10.0	.18		Quartzite.
470	480	10.0	.18		Do.
480	490	10.0	.14		Do.

Hole C-2, Cont'd.

Footage			Percent copper		Description and remarks
From-	To-	Feet	Total	Acid-soluble	
490	500	10.0	0.16		Andesite.
500	510	10.0	.16		Do.
510	520	10.0	.23		Do.
520	525	5.0	.51	0.19	Shear zone.
525	530	5.0	.79	.48	Do.
530	535	5.0	.89	.63	Do.
535	540	5.0	1.43	.91	Shear zone. Little pyrite and native copper.
540	545	5.0	1.27	.57	Shear zone.
545	547	2.0	.74	.28	Schist.
547	550	3.0	.72	.30	Schist and granite.
550	555	5.0	.65	.27	Do.
555	561	6.0			Granite.

Hole C-3

Location: N. 4610, E. 5045 Depth: 671.0 ft.
Elevation of collar: 1,788 ft. Date: 3/9 to 4/2/49

Footage			Percent copper		Description and remarks
From-	To-	Feet	Total	Acid-soluble	
0	10	10			Sand and gravel.
10	125	115.0			Weathered granite.
125	135	10.0	0.33		Schist.
135	145	10.0	.22		Do.
145	155	10.0	.31		Do.
155	165	10.0	.40		Do.
165	175	10.0	.32		Do.
175	185	10.0	.27		Do.
185	195	10.0	.23		Do.
195	205	10.0	.16		Do.
205	215	10.0	.10		Do.
215	225	10.0	.16		Schist, water table at 225 ft.
225	235	10.0	.17		Schist.
235	245	10.0	.22		Do.
245	255	10.0	.17		Do.
255	265	10.0	.16		Do.
265	275	10.0	.25		Schist, shear - much Fe oxide.
275	285	10.0	.25		Schist.
285	295	10.0	.27		Do.
295	305	10.0	.30		Schist, shear - much Fe oxide.
305	315	10.0	.32		Schist.
315	325	10.0	.25		Do.
325	335	10.0	.27		Schist, shear - much Fe oxide.
335	345	10.0	.22		Schist.
345	355	10.0	.20		Do.

Hole C-3, Cont'd.

Footage			Percent copper		
From-	To-	Feet	Total	Acid-soluble	Description and remarks
355	365	10.0	0.23		Schist.
365	375	10.0	.20		Do.
375	385	10.0	.25		Do.
385	395	10.0	.15		Schist, shear - much Fe oxide.
395	405	10.0	.15		Schist.
405	415	10.0	.24		Do.
415	425	10.0	.18		Do.
425	435	10.0	.13		Do.
435	445	10.0	.12		Do.
445	455	10.0	.19		Do.
455	465	10.0	.14		Do.
465	475	10.0	.12		Do.
475	485	10.0	.13		Do.
485	495	10.0	.15		Do.
495	500	5.0	.18		Do.
500	510	10.0	.26		Do.
510	520	10.0	.23		Do.
520	530	10.0	.12		Andesite, shear - much Fe oxide.
530	540	10.0	.15		Andesite.
540	550	10.0	.14		Do.
550	560	10.0	.13		Do.
560	570	10.0	.12		Do.
570	580	10.0	.10		Do.
580	590	10.0	.14		Do.
590	600	10.0	.23		Andesite, shear - much Fe oxide.
600	605	5.0	.20		Andesite.
605	610	5.0	.24		Do.
610	615	5.0	.20		Do.
615	620	5.0	.14		Do.
620	625	5.0	.13		Do.
625	630	5.0	.18		Do.
630	635	5.0	.15		Do.
635	640	5.0	.14		Do.
640	645	5.0	.18		Do.
645	650	5.0	.26		Do.
650	655	5.0	.42		Andesite and granite.
655	660	5.0	.23		Granite and andesite.
660	671	11.0			Granite.

Hole C-4

Location: N. 4795, E. 4900
Elevation of collar: 1,786 ft.

Depth: 290.0 ft.
Date: 4/15 to 4/19/49

Footage			Percent copper		Description and remarks
From-	To-	Feet	Total	Acid-soluble	
0	10	10.0			Sand and gravel.
10	50	40.0			Weathered granite.
50	60	10.0	0.35		Schist.
60	70	10.0	.33		Do.
70	80	10.0	.32		Do.
80	90	10.0	.35		Do.
90	100	10.0	.30		Do.
100	110	10.0	.36		Do.
110	120	10.0	.35		Do.
120	130	10.0	.17		Do.
130	140	10.0	.58	0.11	Do.
140	150	10.0	.33		Do.
150	160	10.0	.27		Do.
160	170	10.0	.23		Do.
170	180	10.0	.47		Do.
180	190	10.0	.82	0.52	Do.
190	200	10.0	1.23	.76	Do.
200	210	10.0	1.50	.65	Do.
210	220	10.0	1.55	.90	Do.
220	225	5.0	1.75	1.16	Do.
225	230	5.0	1.02	.66	Schist, water table at 225 feet.
230	235	5.0	1.31	1.00	Schist.
235	240	5.0	3.05	2.80	Do.
240	245	5.0	2.31	1.95	Do.
245	250	5.0	1.94	1.27	Do.
250	255	5.0	1.51	.96	Do.
255	260	5.0	1.54	.98	Do.
260	265	5.0	1.75	.97	Do.
265	270	5.0	.61	.36	Schist and quartzite.
270	290	20.0			Quartzite.

Hole C-5

Location: N. 5326, E. 5090
Elevation of collar: 1,801 ft.

Depth: 557.0 feet
Date: 4/20 to 5/13/49

Footage			Percent copper		Description and remarks
From-	To-	Feet	Total	Acid-soluble	
0	25	25.0			Sand and gravel.
25	35	10.0	0.49		Schist.
35	45	10.0	.41		Do.
45	55	10.0	.40		Do.
55	65	10.0	.45		Do.
65	75	10.0	.87	0.41	Do.

Hole C-5 Cont'd.

Footage			Percent copper		Description and remarks
From-	To-	Feet	Total	Acid-soluble	
75	85	10.0	1.05	0.52	Schist.
85	95	10.0	1.18	.65	Do.
95	105	10.0	1.55	.96	Do.
105	115	10.0	2.10	1.19	Do.
115	125	10.0	1.69	1.00	Do.
125	135	10.0	1.76	1.10	Do.
135	140	5.0	2.21	1.14	Do.
140	145	5.0	2.08	1.16	Do.
145	150	5.0	1.87	1.13	Do.
150	155	5.0	2.15	1.41	Do.
155	160	5.0	1.65	1.39	Do.
160	165	5.0	.70	.39	Andesite.
165	175	10.0	.32		Do.
175	185	10.0	.26		Do.
185	195	10.0	.15		Do.
195	205	10.0	.15		Do.
205	215	10.0	.16		Do.
215	225	10.0	.10		Do.
225	235	10.0	.18		Andesite. Water table at 230 ft.
235	245	10.0	.16		Andesite.
245	255	10.0	.10		Do.
255	265	10.0	.14		Do.
265	275	10.0	.16		Do.
275	285	10.0	.16		Do.
285	295	10.0	.16		Do.
295	305	10.0	.15		Do.
305	315	10.0	.13		Do.
315	325	10.0	.18		Do.
325	335	10.0	.18		Do.
335	345	10.0	.20		Do.
345	355	10.0	.29		Do.
355	365	10.0	.19		Do.
365	375	10.0	.23		Do.
375	385	10.0	.18		Do.
385	395	10.0	.28		Do.
395	405	10.0	.20		Do.
405	415	10.0	.15		Do.
415	425	10.0	.15		Do.
425	430	5.0	.26		Do.
430	435	5.0	.08		Do.
435	440	5.0	.08		Do.
440	445	5.0	.95	0.67	Do.
445	450	5.0	.34		Do.
450	455	5.0	.19		Do.
455	460	5.0	1.88	1.73	Shear zone.
460	465	5.0	2.61	2.53	Do.

Hole C-5, Cont'd.

Footage			Percent copper		Description and remarks
Feet-	To-	Feet	Total	Acid-soluble	
465	470	5.0	1.78	1.55	Shear zone.
470	475	5.0	1.17	1.07	Do.
475	480	5.0	1.16	1.06	Do.
480	485	5.0	1.40	1.16	Do.
485	490	5.0	2.48	2.25	Do.
490	495	5.0	1.97	1.73	Do.
495	500	5.0	.91	.73	Do.
500	505	5.0	2.41	1.92	Do.
505	510	5.0	.50	.33	Do.
510	515	5.0	.73	.50	Do.
515	520	5.0	.73	.49	Do.
520	525	5.0	.18		Do.
525	530	5.0	1.54	1.26	Do.
530	535	5.0	2.98	2.31	Do.
535	540	5.0	3.06	2.51	Do.
540	545	5.0	2.09	1.65	Do.
545	550	5.0	.53	.38	Andesite and granite.
550	557	7.0			Granite.

INVESTIGATION OF THE LAKE SHORE COPPER DEPOSITS, PINAL COUNTY, ARIZ.

BY T. M. ROMSLO

* * * * * * * * * **Report of Investigations 4706**

UNITED STATES DEPARTMENT OF THE INTERIOR
Oscar L. Chapman, Secretary
BUREAU OF MINES
James Boyd, Director

Work on manuscript completed March 1950. The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is made: "Reprinted from Bureau of Mines Report of Investigations 4706."

July 1950

INVESTIGATION OF THE LAKE SHORE COPPER DEPOSITS, PINAL COUNTY, ARIZ.

by

T. M. Romslo[1]

CONTENTS

1/ Mining engineer, U. S. Bureau of Mines, Tucson Branch, Minerals Division, Tucson, Ariz.

Report of Investigations 4706

TABLES

ILLUSTRATIONS

INTRODUCTION AND SUMMARY

The Lake Shore property, located in the early 1880's, contains copper-bearing deposits that have been developed by surface excavations, underground workings, and churn-drill holes. Intermittent operation of the property ended in 1929 with a total recorded production of 280,000 pounds of copper.

The property is near the foot of the Slate Mountains, which are made up mainly of schist, probably the Pinal formation of pre-Cambrian age. In the mine area there are a few outcrops of granite, which is exposed over a large area east of the property. Other outcropping rocks on the property are limestone, quartzite, and diabase. The limestone and quartzite probably are the Mescal and Troy formations of pre-Cambrian and Cambrian age, respectively.

The predominant copper mineral is chrysocolla, a hydrous silicate that occurs mainly as fracture filling in bedded schist. It is also the principal copper mineral in the shear zone at the schist-granite contact and in limestone southeast of the main workings.

Investigation of the Lake Shore property by the Bureau of Mines included both topographic and geologic mapping, exploratory drilling, and metallurgical test work. One diamond-drill hole and five churn-drill holes were completed for a total of 2,872.5 feet. Drilling started January 19 and was completed May 13, 1949.

ACKNOWLEDGMENTS

These investigations were initiated in 1942 when O. M. Bishop, formerly a mining engineer of the Bureau of Mines, examined the property with the object of determining ore reserves and obtaining samples for metallurgical tests. Appreciation is extended to Frank M. Leonard, Jr., one of the owners of the property, for accompanying the engineer during the examination, for relating the history of the property, and for supplying an assay map of the mine workings and assay graphs of the churn drill holes. Later in the same year, T. C. Denton, also a former mining engineer of the Bureau, obtained additional samples for metallurgical tests.

The Bureau wishes to thank Nels P. Peterson of the U. S. Geological Survey for mapping both the surface and the underground geology during brief visits to the property in January and March 1949.

The investigations made during the Bureau's drilling program were supervised by J. H. Hedges, Chief, Tucson Branch, Mining Division, and analytical work was by Ray Stiles, under J. Bruce Clemmer, chief, Tucson Branch, Metallurgical Division. Metallurgical tests by the Bureau in 1942 and 1943 were made at the Salt Lake City station with H. G. Poole in charge. Clemmer and

Carl Rampacek conducted the tests at Tucson in 1949 and prepared the text on metallurgical tests. Transit surveys of the surface and underground workings, started by the author, were completed by M. H. Berliner, mining engineer of the Tucson Branch, Bureau of Mines.

Acknowledgment is made to the Indian Service of the Department of the Interior for grading an entry road to the mine and for providing a source of domestic and drilling water from a well at the nearby Indian Village of Komelik.

LOCATION AND ACCESSIBILITY

The Lake Shore mine is in the Papago Indian Reservation and the Casa Grande mining district, Gila and Salt River Base Line and Meridian, secs. 25 and 36, T. 10 S., R. 4 E., Pinal County, Ariz. (fig. 1). It may be reached from Casa Grande, a town on the Southern Pacific Railroad and State Highway 80, by traveling southwestward 28.2 miles on a well-maintained dirt road and thence 2.6 miles east on a desert road to the property.

PHYSICAL FEATURES AND CLIMATE

The Lake Shore mine is on the southwest piedmont of the Slate Mountains at an altitude of about 1,800 feet. The mountain range trends northwestward and reaches its maximum altitude of 3,330 feet at Prieta Peak, about 2 miles north of the mine.

Vegetation is of the desert variety, typical of the lower altitudes of southern Arizona. Palo Verde trees and Saguaro cactus are prominent.

Winters are mild and summers are hot. At Ajo, about 60 miles west of the property, the annual mean temperature is 71°, with a range from 17° to 115°. The annual precipitation averages about 9.3 inches.

PROPERTY AND OWNERSHIP

The Lake Shore property consists of three patented lode mining claims: the Arizona, Copper Bell, and Isabella (fig. 2). N. Frank Leonard, Butte, Mont., owns 96 percent of the stock of the Hidden Treasure Mining Co., which is the holder of the property.

There are no buildings or equipment on the property.

HISTORY AND PRODUCTION

The mine was located early in the 1880's by Trout and Atchinson. A shaft was sunk, and some drifting was done before 1884, when the property was abandoned because of failure of the copper market. In 1905, B. S. Wilson relocated the mine and shipped some ore sorted from the dump. In 1914 he sold the property to Frank M. and Charles Leonard. A new shaft was sunk to the 225-foot level, and development of the ore body was started on three levels. In 1917 the Atlas Development Co., Chicago, Ill., leased the mine and shipped 850 tons of 5.2 percent copper ore to a smelter at Sasco, Ariz. In 1919,

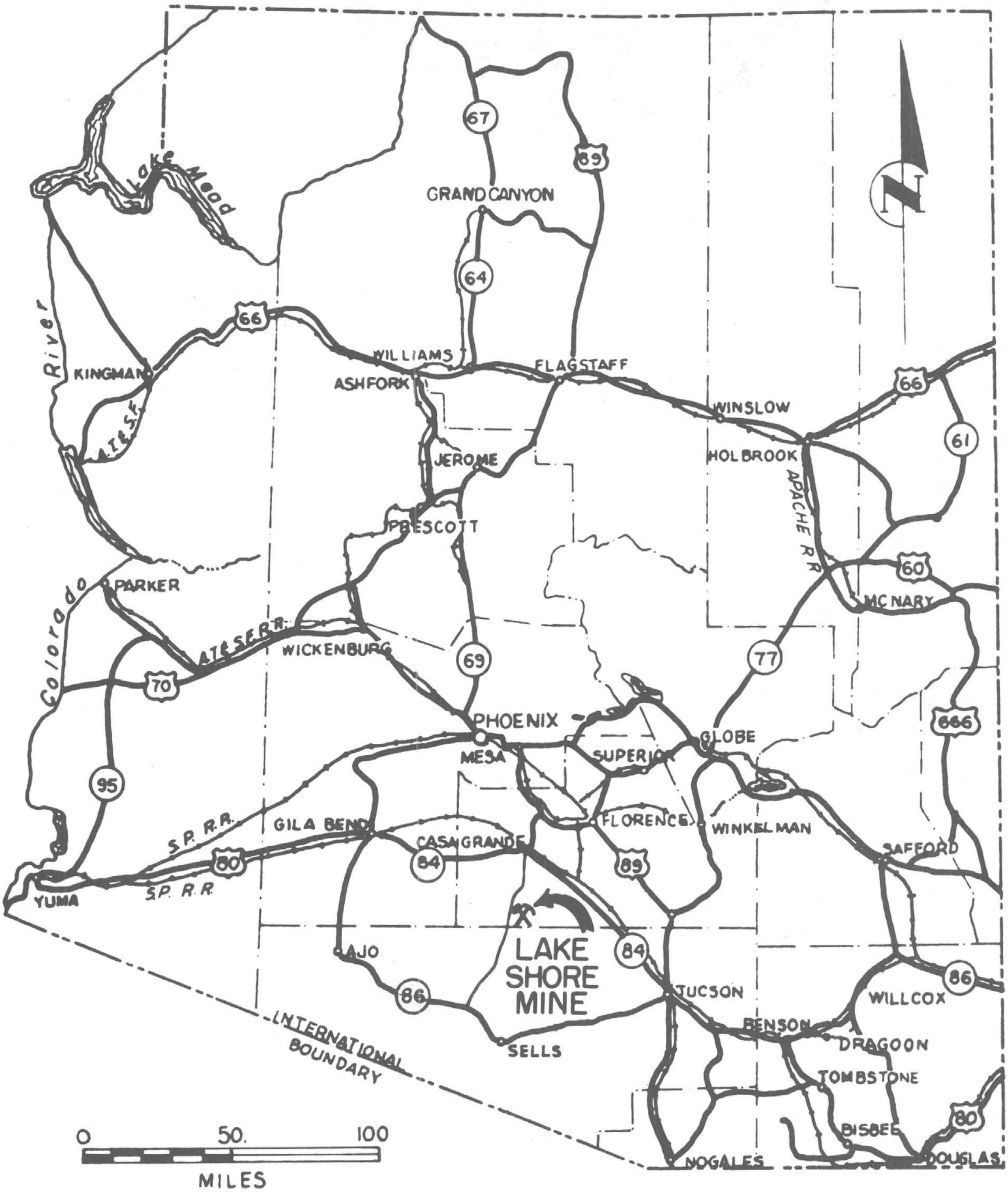


Figure 1. - Location map, Lake Shore copper project, Pinal County, Ariz.

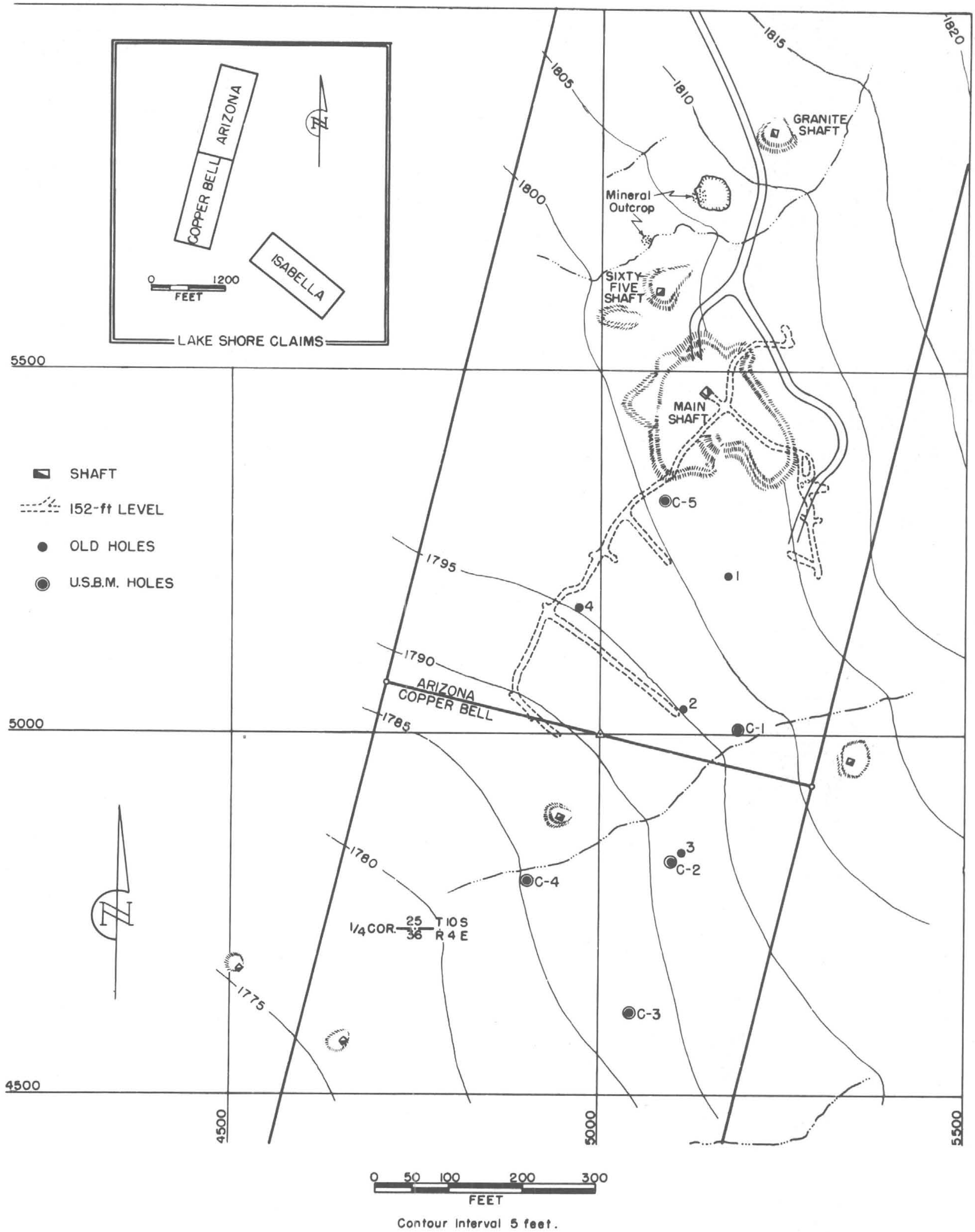


Figure 2. - Surface map, Lake Shore copper deposits, Pinal County, Ariz.

after terminating the lease, the Leonards drilled 5 churn drill holes and sank two winzes. During this period 12 tons of 15 percent copper ore in sulfide form was mined from the schist-granite contact zone on the 285-foot level. The last reported production was in 1929, when ore was trucked from the mine dump to Casa Grande for shipment.

Total production from the property is reported to have been 280,000 pounds of copper.[2]

GEOLOGY

General

The Slate Mountains are composed mainly of schist, tentatively identified as the Pinal formation of pre-Cambrian age. Biotite granite has intruded the schist near the southwest end of the mountain range. It crops out over a very small area on the Isabella claim and is prominently exposed east of the Lake Shore property. Other rock exposures on the property are confined to a small area of altered schist on the Arizona claim and to limestone, quartzite, and diabase on the Isabella claim. The limestone and quartzite are probably the Mescal and Troy formations of pre-Cambrian and Cambrian age, respectively.

Deposits

Copper mineralization is associated with a fault that has an average strike of about S. 11° W. and a dip of 60° to 70° west (figs. 3 and 4). Granite, probably an integral part of the intrusive mass, forms the block east of the fault. On the west side of the fault is a bed of highly altered, intensely fractured, fine-grained rock that has been classified as schist. A thin bed of quartzite is spottily present near the base of the schist. Underlying the schist is an intensely altered mass of rock tentatively classified as andesitic lava or tuff. Part of this formation can be identified megascopically as andesite. Spottily present in the andesite is a very fine-grained unidentified rock of light color and stony appearance. Of similar occurrence and texture is a dark-colored rock tentatively identified as basalt. The schist strikes about S. 37° W. and dips 37° to 45° east. South of the main shaft, a comparatively small body of granite is in contact, on the west, with the fault.

Copper mineralization occurs sparingly throughout the bedded rocks but is concentrated mainly at the base of the schist and in the fault zone. The planes of the fault and the planes of the bedded rocks diverge to form a trough that plunges to the southwest at an angle of about 24°.

Mineralogy

The following is an analysis of a 158-pound sample submitted to the Salt Lake City Station for metallurgical testing in 1942.

2/ Elsing, M. J., and Heineman, R.E.S., Arizona Metal Production: University of Ariz. Bull. 140.

	Oxide								
Insol.	SiO_2	Fe	CaO	S	Cu	Cu*	Al_2O_3	Zn	Pb
49.4	37.1	17.5	5.1	Nil	2.3	2.15	6.5	Nil	Nil

*Soluble in dilute H_2SO_4 saturated with sulfur dioxide.

The late R. E. Head,[3] of the Bureau of Mines, stated:

> Examination of thin sections prepared from representative pieces of the ore indicate that basically two types of copper association are represented. In addition to the copper-bearing material, there appears to be also an indeterminate quantity of rock that is virtually free of copper.
>
> In the one type of copper occurrence, the ground mass is almost entirely quartzitic. Chrysocolla, the copper silicate, occurs in this type of rock as a filling in fractures both in the rock itself and in the quartz particles.
>
> In some of these fracture fillings the chrysocolla occurs as masses of hairlike fibers intermixed with calcite and claylike material. In addition to this type of association, the chrysocolla is also present as a shell or coating on many of the quartz particles. In some cases, aggregates of very small quartz particles are cemented together with chrysocolla, which occurs as films so thin as to amount to scarcely more than stains.
>
> In the other type of association, the chrysocolla is distributed uniformly through the claylike ground mass in the form of minute veinlets and also as fracture fillings. This association of chrysocolla with the gangue is very intimate, and examination of thin sections showed that the individual clay particles were ringed with copper carbonate.
>
> The ore contains an appreciable quantity of magnetic iron oxide, magnetite.

Subsequent investigation of other samples of the ore in connection with metallurgical testing showed the copper to be present mainly in the silicate form as chrysocolla and some dioptase. Also present is a yellowish copper mineral, which is probably a silicate. A trace of sulfide-copper is present mainly as chalcocite.

A little pyrite and a small amount of native copper were seen in the cuttings from the fault zone at churn-drill hole C-2.

3/ Head, R. E. (deceased), Preliminary Microscopic Examination of oxidized ore from the Lake Shore Mines, Arizona: August 1942.

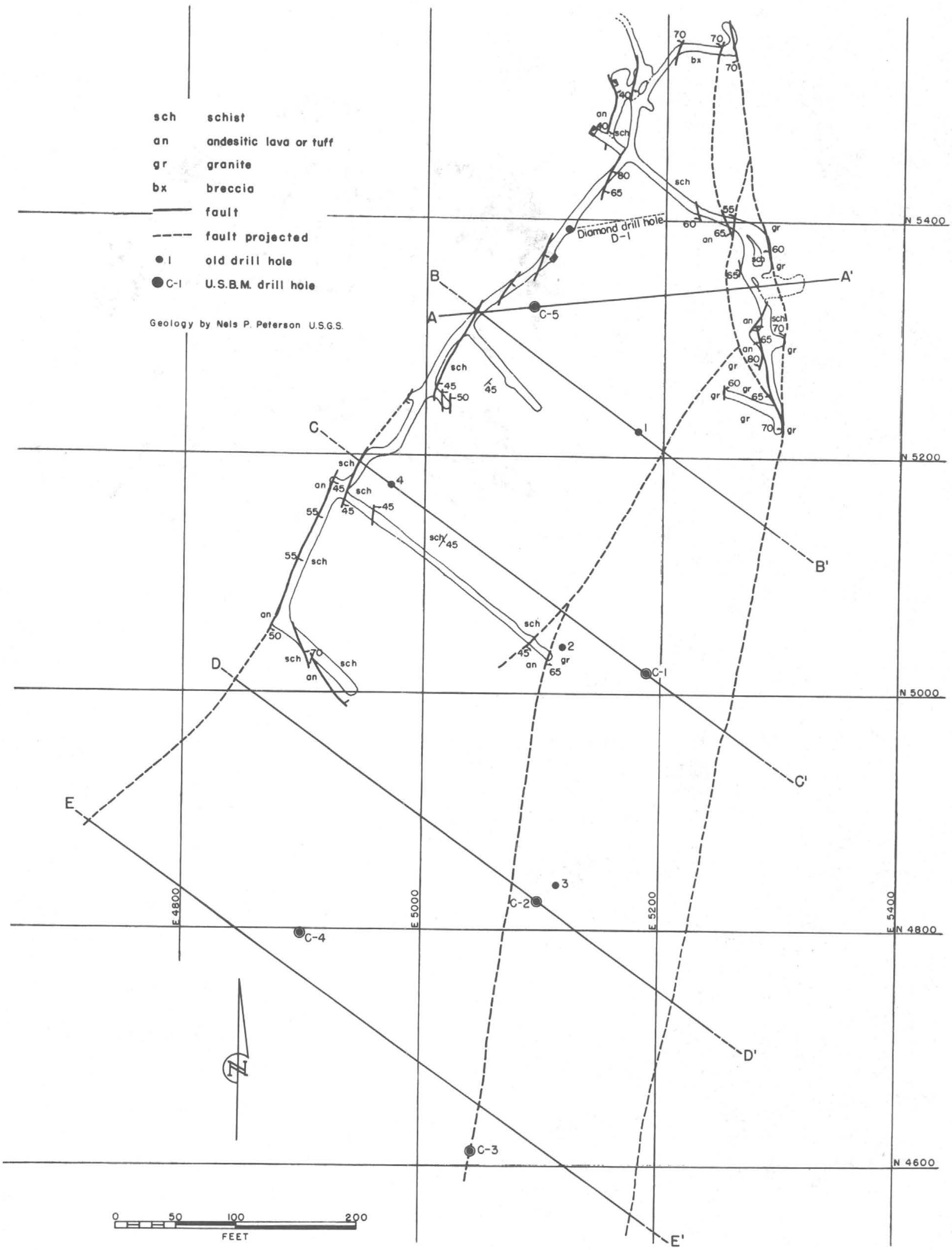


Figure 3. - Geologic map, 152-foot level, Lake Shore copper deposits, Pinal County, Ariz.

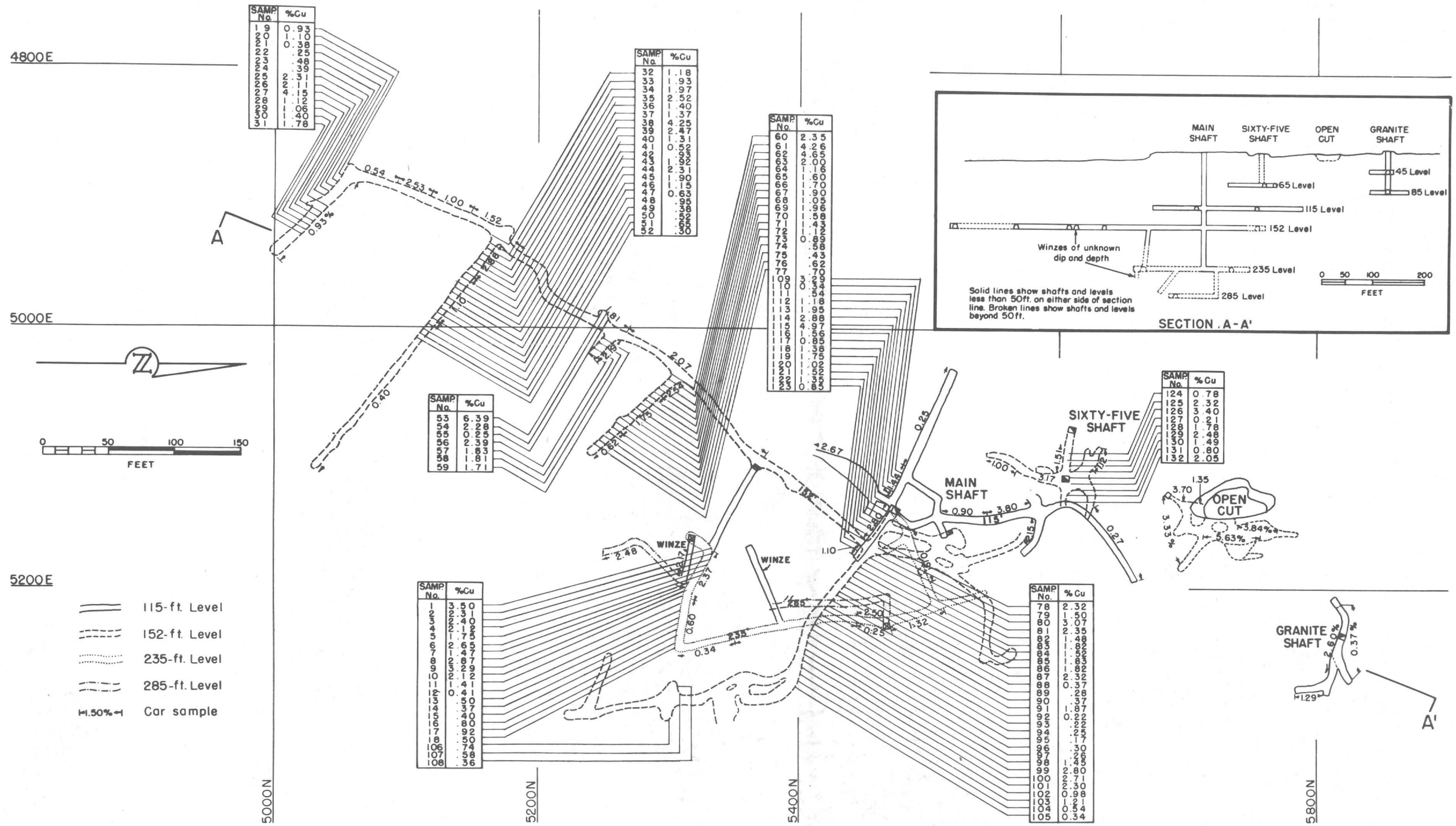


Figure 4. - Assay map, Lake Shore copper deposits, Pinal County, Ariz.

MINE WORKINGS (figs. 2, 3, and 4)

The main shaft is vertical and fully timbered into a 4-foot square hoisting compartment and a 2-1/2- by 4-foot manway compartment. It is 235 feet deep and at present is accessible to the water that stands at 221 feet below the collar of the shaft. Levels at depths from the surface of 115, 152, and 235 feet have been opened from the shaft, whereas the bottom or 285-foot level has been developed from two winzes sunk from the 235-foot level. Lineal development on the four levels consists of over 2,700 feet of drifts and crosscuts. Near the footwall of the bedded deposit are two small stopes on the 115-foot level and two on the 152-foot level (fig. 3). Another small stope on the 152-foot level is in the schist-granite contact zone.

The Sixty-Five shaft and the Granite shaft, both inaccessible, are situated 130 feet northwest and 350 feet northeast of the main shaft, respectively. The Sixty-Five shaft, 65 feet deep, has one level at its bottom. The Granite shaft has two levels - one at a depth of 45 feet and the other at its bottom of 83 feet. About midway between the two shafts is an open cut in the only surface exposure of ore on the property. It was the source of several cars of ore.

A longitudinal section through the main workings is shown on the assay map (fig. 4).

In addition to the above workings, there are several shallow shafts and pits.

WORK BY THE BUREAU OF MINES

Field Work

During examination of the mine by the Bureau of Mines in 1942, sampling was confined to the 115- and 152-foot levels, because the lower workings were flooded with the water, which stood at 228 feet below the collar of the shaft. Seven channel samples were cut to duplicate corresponding samples that are similarly numbered on figure 4. In addition, six samples, each weighing 25 to 55 pounds, were cut from six crosscuts. These, also, were channel samples and, with the exception of sample 100, were cut from channels that carry similar numbers. Sample 100 represents the material exposed in a section of the crosscut on the 115-foot level. Analyses of the samples are shown in table 1.

TABLE 1. - Analyses of channel samples

Level	Sample	Width, feet	Percent copper
115...............	114	5	3.54
115...............	115	5	4.39
115...............	116	5	1.90
152...............	61	5	2.69
152...............	62	5	2.18
152...............	63	5	2.81
152...............	64	5	1.28
115...............	100	38	2.69
115...............	113-116	20	2.17
152...............	25-31	35	1.90
152...............	32-39	40	2.27
152...............	60-63	20	2.56
152...............	78-87	50	1.74

A 158-pound sample was made of the six large samples for metallurgical tests. Later in the same year four additional samples were taken for metallurgical testing. Each of these represented 50 continuous feet of crosscut and ranged in weight from 272 to 619 pounds. They were taken from crosscuts at the shaft on the 115- and 152-foot levels and from the first and second crosscuts south of the shaft on the 152-foot level.

Active work on the exploratory project started November 22, 1948. The first truck loads of equipment and supplies, after being assembled and conditioned in Tucson, were hauled to the mine on December 6. While a complete camp to accommodate 25 to 30 men was being built and equipped, work was started on rehabilitation of the main shaft. Shaft work consisted of replacing the collar and second sets of timbers and making minor repairs to both the hoisting and manway compartments. A tripod was placed over the shaft, and a hoist was installed. Two 210-c.f.m. compressors were placed near the shaft, and an air line was installed to the site of diamond drill hole D-1. Track was laid, the drill station was drilled and blasted, and the muck was trammed to the shaft and hoisted to the surface in buckets. While the diamond drill hole was being drilled, the air line and track were advanced, and two more drill stations were drilled and blasted. The muck from these stations was hoisted to the surface after diamond drilling was completed. A total of about 100 tons of broken rock was removed from the mine.

A transit survey of the surface and underground workings started while the camp was being built showed that available maps could be used for laying out the drilling program. This work, as completed, included plumbing the main shaft, transit surveys of the 115- and 152-foot levels, and topographic surveys of the area shown in this report, the Isabella claim, and a 25-acre area adjoining the Isabella claim on the east.

Diamond drilling, consisting of one hole completed at a depth of 203.5 feet, was started January 19 and completed March 18. A vertical section through the hole is shown in figure 5, and the assays of samples are given in the log of the hole that is appended to this report. Original plans included diamond-drilling 6 or 8 holes from underground stations, each designed to

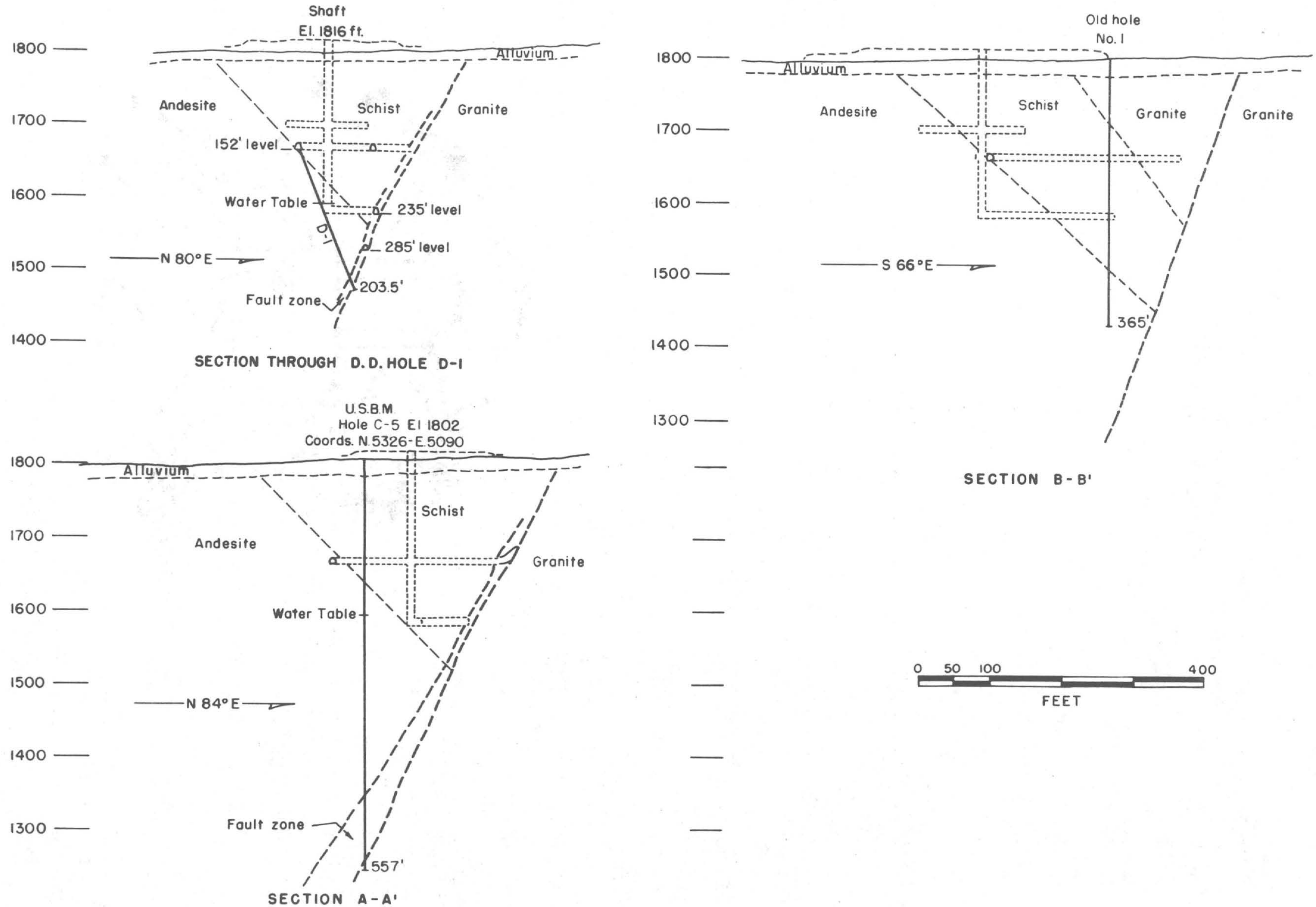


Figure 5. - Geologic sections, Lake Shore copper deposits, Pinal County, Ariz.

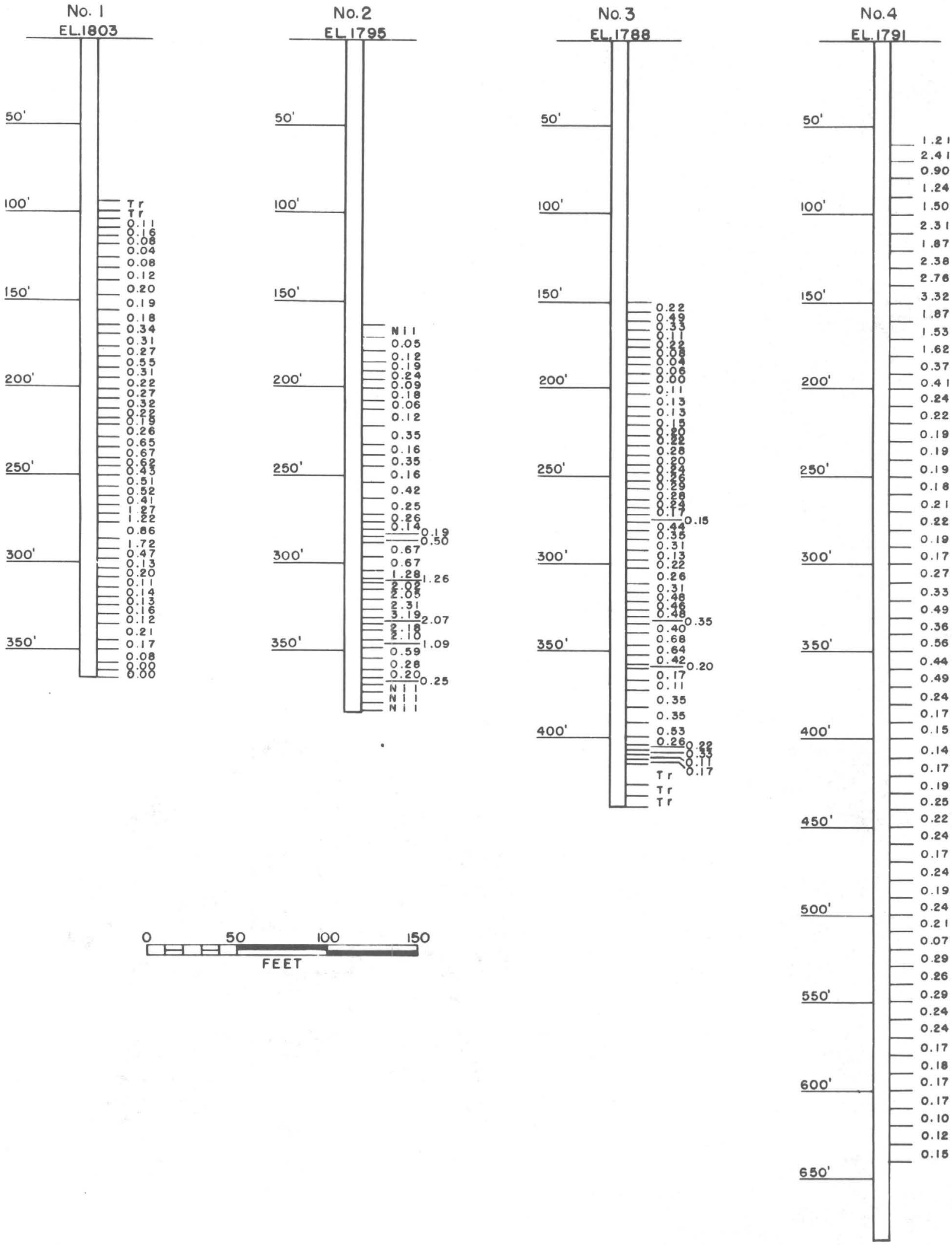


Figure 6. - Assay graphs, old churn drill holes, Lake Shore copper deposits.

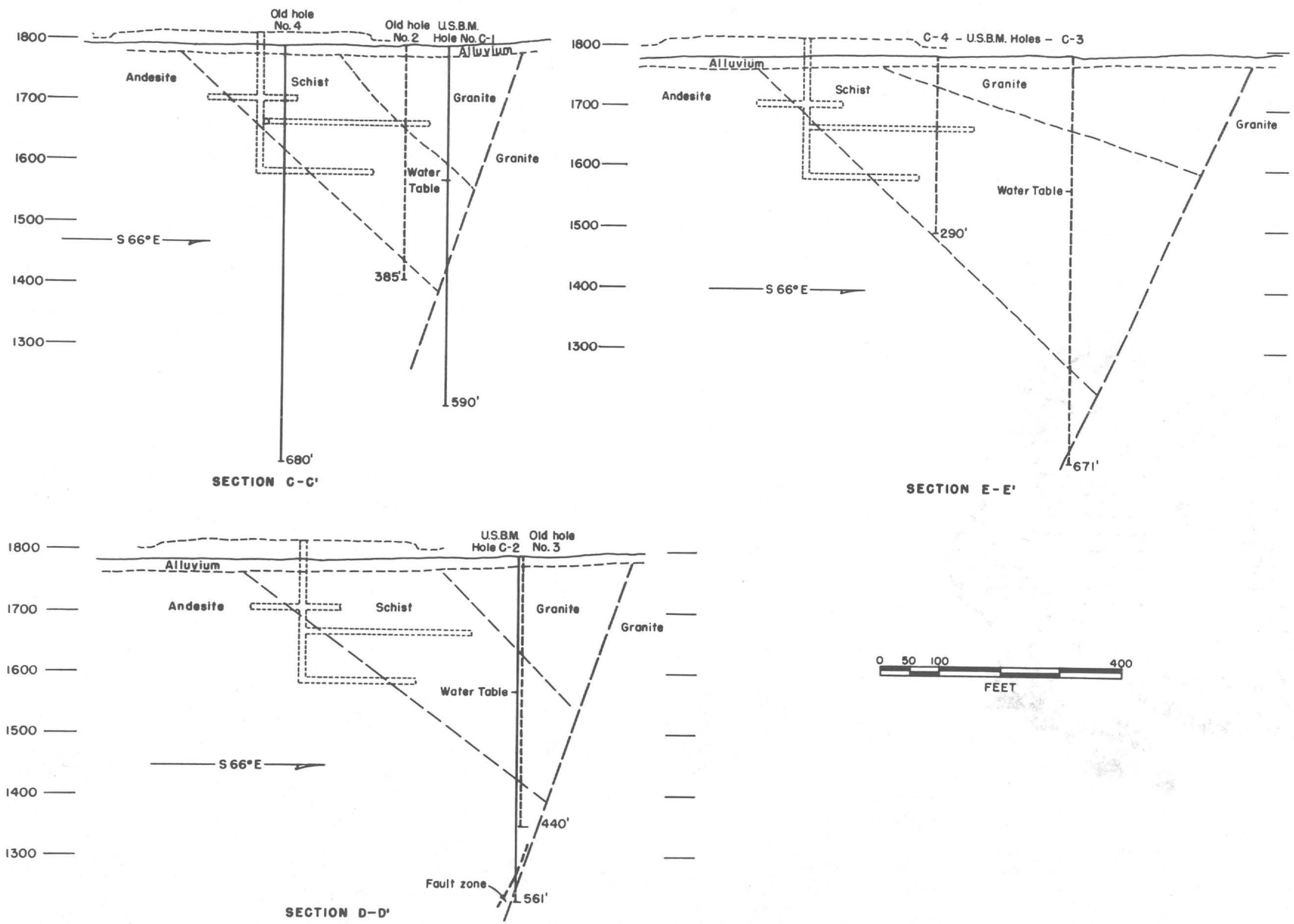


Figure 7. - Geologic sections, Lake Shore copper deposits, Pinal County, Ariz.

intersect the fault zone below the water table at intervals along the strike of the fault. Diamond drilling was terminated upon completion of one hole because costs were excessive to both the contractor and the Government. From the collar of the hole to the fault zone the rock is intensely fractured, and core recovery averaged about 6 percent. In general, after drilling a section of the rock the hole would close in as soon as the core barrel was removed.

Repeated cement jobs on portions of the hole failed, and in these cases it was necessary to drive the casing ahead. Attempts to advance the hole by blasting also failed. The contractor also tried unsuccessfully to keep the hole open and to consolidate the ground ahead of the bit by freezing. This operation consisted in using fuel oil cooled by dry ice as the circulating medium. Little trouble was experienced in penetrating the fault zone, where core recovery averaged 3.6 percent. The diamond drill was operated two shifts daily for 6 days a week. Double-tube core barrels 5 and 10 feet long were used. Drilling data for the hole, which was numbered D-1, follow:

Diamond-drilling data

		Stand-pipe	Feet							
			Drilled			Reamed		Cased		
Hole	Depth, ft.	(3-inch)	NX	BX	AX	BX to NX	AX to BX	BX	AX	Cemented
D-1	203.5	11.0	34.0	94.0	64.5	33.0	18.0	78.0	157.0	146.5

Churn drilling, consisting of five holes for a total depth of 2,669 feet, was started January 13 and completed May 13, 1949. The rock was easy to drill, but, being ravelly, it was generally necessary to carry casing close to the bottom of the hole. The drill was operated two or three shifts daily, mainly on a two-shift basis, for 6 days a week.

Pertinent drilling data are given in table 2, and the logs of holes drilled by the Bureau are appended. Assay graphs of four of the churn-drill holes put down by the owners in 1919 are shown in figure 6.

Sections through the churn drill holes are shown on figure 7.

Drill-hole samples for analysis totaled 295, of which 56 were from the diamond-drill hole and 239 were from the churn-drill holes. Drill cuttings were dried, weighed, and reduced in size with a Jones splitter, and core samples were weighed and split. One half of each core sample and the samples of drill cuttings were sent to Tucson for analysis. The other half of the core was placed in core boxes, which were stored in the Bureau core house in Tucson. Two large samples of muck from the diamond-drill stations also were sent to Tucson for metallurgical tests.

Three thousand lineal feet of road work was done. This consisted of repairs to existing roads and building new roads to drilling sites but does not include 2.6 miles of 20-foot-wide road from the Casa Grande-Ajo road to the mine, which was built by the Indian Service of the Interior Department.

All drill holes were capped with a Bureau marker showing project number, hole number, and date of completion.

TABLE 2. - Churn-drilling data

Churn-drill hole	Feet											
	Depth	Drilled, bit size (inches)				Cased, pipe size (inches)				Reamed, bit size (inches)		
		12	10	8	6	12	10	8	6	10-12	8-10	6 - 8
C-1...	590.0	250.0	115.0	225.0		20.0	382.0			50.0	6.0	
C-2...	561.0	155.0	105.0	200.0	101.0	12	205.0	452.5	546.0	20.0	31.0	15.0
C-3...	671.0	250.0	90.0	160.0	171.0	194.0	321.0	477.0	641.0	67.0	46.0	111.0
C-4...	290.0	250.0	40.0			31.0						
C-5...	557.0	175.0	250.0	105.0	27.0	155.0	412.5	466.0	526.0	169.0		
	2,669.0	1,080.0	600.0	690.0	299.0	412.0	1,320.5	1,395.5	1713.0	306.0	83.0	126.0

Copper Analyses

The Lake Shore samples were analyzed for copper by conventional procedures. Total copper was determined by the long iodide method, using a mixture of hot concentrated hydrochloric, nitric, and sulfuric acid for decomposition of the minus 100-mesh samples. Samples that contained 0.5 percent or more of copper were reassayed for acid-soluble copper with a 5-percent solution of sulfuric acid saturated with sulfur dioxide to dissolve the copper silicates, oxides, and carbonates. Common practice is to report the acid-soluble assay as "oxide" copper, and the difference between the total and oxide assays is reported as "sulfide" copper.

Although such analyses would indicate that many of the Lake Shore samples contain 0.5 percent or more of sulfide copper, microscopic examination failed to reveal more than a trace of copper sulfides. Furthermore, the sulfur content of the samples was too small to account for this quantity of copper. Subsequent examination and microchemical tests on sink-float fractions of the Lake Shore ore indicated that this copper is associated with the gangue minerals as minute inclusions of an unidentified copper mineral that is somewhat more refractory toward leaching than chrysocolla.

The total and acid-soluble copper contents of samples from holes drilled by the Bureau are shown in the logs.

Metallurgical Tests[4/]

The five samples from an examination of the mine in 1942 were submitted to the Salt Lake City Station for metallurgical tests. An analysis of a 158-pound character sample is shown in the section on mineralogy of the ore. The analyses of the other samples are given in table 3. The samples from crosscuts Nos. 1 to 3 on the 152-foot level, numbered to the south from the crosscut at the shaft, were identified as Nos. Ar-4.1, Ar-4.2, and Ar-4.3, respectively, and the sample from the 115-foot level was numbered Ar-4.4.

The Salt Lake City metallurgical tests revealed that the mineral association in the samples was too intimate for beneficiation by ore-dressing methods. Acid leaching of the ore was not attractive owing to the presence of lime, which caused excessive acid consumption. Tests employing the reducing-roast and ammonia-leach process extracted as much as 86 percent of the copper. In these tests, minus 20-mesh material was roasted with coke in an atmosphere of natural gas for 1 hour at 500° to 600° C. to reduce the copper. The samples were then cooled to 180° C. and quenched in water. Leaching was carried out at 25 percent solids in a combination air-mechanical agitation tank for 4 hours, using a 10 percent solution of ammonium hydroxide and ammonium carbonate in equal parts, containing the equivalent of 0.3 pound potassium cyanide per ton of ore. The leach residues were filter-washed with ammonia and water.

4/ Prepared by Carl Rampacek and J. Bruce Clemmer, metallurgists, Bureau of Mines, Tucson Branch, Metallurgical Division, Tucson, Ariz.

TABLE 3. - Analyses of metallurgical samples

Sample	Insol.	Percent										Oz./ton		Cu soluble in 10% solution (24 hr.)	
		SiO_2	Fe	CaO	S	Zn	Pb	Cu	Ox Cu*/	Al_2O_3	MgO	Au	Ag	H_2SO_4	NH_4OH
4.1	45.7	31.6	17.1	7.9	.08	Nil	Nil	1.71	1.60	7.9	11.3	Nil	Tr.	1.33	Nil
4.2	62.9	41.6	5.35	10.7	.07	0.15	Nil	1.29	1.28	9.9	10.6	Nil	Tr.	1.25	Nil
4.3	35.4	25.8	29.2	5.2	<.05	Nil	Nil	2.18	1.79	3.6	11.3	Nil	Tr.	1.75	Nil
4.4	66.2	54.6	7.25	4.5	<.05	Nil	Nil	1.66	1.59	5.7	9.7	Nil	Tr.	1.32	Nil

*/ Copper soluble in dilute sulfuric acid saturated with sulfur dioxide.

Metallurgical tests were made subsequently at the Tucson station on a composite sample taken from drill stations 1, 2, and 3 on the 152-foot level. Analysis of the sample gave 3.51 percent total copper, 2.96 percent acid-soluble copper, 8.25 percent iron, 1.73 percent calcium carbonate, 0.04 percent sulfate-sulfur, and 0.01 percent sulfide-sulfur. The copper was present predominately as chrysocolla and dioptase, with only traces of sulfides and carbonates.

Batch flotation of the ore ground to pass 65 or 200 mesh made with conventional sulfide and nonsulfide collecting agents failed to effect separation. The trace of sulfides, largely chalcocite, floated readily, but recovery of the chrysocolla and dioptase was poor, regardless of the conditions employed.

Acid leaching and leach-precipitation-flotation of the sample also were investigated. The results of a number of bottle leaching tests are summarized in table 4. The tests on portions of the ore ground to pass 10, 20, and 65 mesh were made at 50 percent solids with different quantities of acid and various contact periods.

The leaching tests revealed that about 375 pounds of acid, 4.1 times the theoretical based on the acid-soluble copper content of the feed, were required for a good extraction of copper from the 10, 20, and 65-mesh feeds. Although the finer material leached more rapidly, a 24-hour contact was essential for an 88 to 90 percent extraction of the total copper. The acid consumed varied from 4.1 to 4.4 pounds per pound of copper extracted. Neither longer leaching nor use of more acid materially improved copper extraction. Cursory tests on charges of the ore ground to 200 mesh gave slightly higher copper extractions but not enough to justify the added cost of finer grinding.

Although the chrysocolla in the ore is amenable to leaching, long contact with excessive acid is required to dissolve the 0.5 percent or more of copper that is intimately associated with the gangue. Tests were made to determine if the refractory copper could be extracted within a reasonable period by employing stronger acid solutions. The dry ore was mixed with the desired quantity of acid and enough water to give an agglomerated or pasty charge containing about 75 percent solids. A 50 percent acid solution proved adequate, but more concentrated acid was used in some of the tests. The agglomerated charges were permitted to stand at room temperature for various lengths of time and then were leached 15 minutes with water to extract the solubilized copper. Tests were made on 10-, 20-, and 65-mesh feeds with 375 pounds of acid per ton and varying the contact period from 1 to 24 hours. The stagnant leaching of the agglomerated charges gave copper extractions almost identical to those of bottle leaching at 50 percent solids, as recorded in table 4.

Although stagnant leaching of the acid-agglomerated charges at room temperature failed to improve extraction of the refractory copper, supplementary tests revealed that moderate heating of the agglomerules expedited solution of the copper for an improved recovery. The results of several tests on 10-, 20-, and 65-mesh portions of the ore are summarized in table 5. The charges were mixed for about 5 minutes with the quantity of acid shown and just enough water to form agglomerules. These were heated in a muffle furnace to give a substantially dry sulfated product, which was subsequently leached with water for 15 minutes to extract the copper. For convenience, the sulfated products were leached at 33 percent solids. In other tests, however, leaching at 50 percent solids gave equally good results, and it seems likely that adequate leaching could be obtained in even thicker pulps.

TABLE 4. - Bottle leaching of Lake Shore ore.

Leaching time, hr.	Mesh feed	H_2SO_4 added, lb./ton	H_2SO_4 consumed		Extraction, percent of total copper
			Lb./ton	Lb/lb of copper extracted	
1	65	105	102	3.8	38.5
1	65	155	151	3.7	58.7
1	65	205	180	3.4	75.5
1	65	260	198	3.6	78.3
1	65	310	209	3.7	80.9
1	65	360	210	3.6	82.3
1	65	410	219	3.8	82.3
1	65	205	180	3.4	75.5
2	65	205	183	3.3	78.1
4	65	205	195	3.4	80.3
1	65	375	201	3.5	82.3
4	65	375	231	3.8	87.3
8	65	375	239	3.8	88.6
12	65	375	247	4.0	89.2
24	65	375	259	4.1	90.0
1	20	375	198	3.6	77.8
4	20	375	232	3.9	84.3
8	20	375	250	4.2	84.9
12	20	375	264	4.3	87.2
24	20	375	275	4.4	89.7
1	10	375	171	3.4	72.6
4	10	375	212	3.7	82.3
8	10	375	228	3.8	84.9
12	10	375	236	3.9	86.0
24	10	375	258	4.2	88.3

TABLE 5. - Results of acid-sulfating tests.

Mesh of feed	Sulfating treatment: H_2SO_4 added, lb./ton	Furnace temp., °C.	Time, Min.	H_2SO_4 consumed: Lb./ton	Lb/lb of copper extracted	Extraction, percent of total copper
10	375	25	60	195	3.7	75.4
10	375	250	7.5	328	5.7	81.8
10	375	250	15	352	5.9	84.9
10	375	250	30	366	6.1	85.2
20	375	25	60	224	4.0	79.5
20	375	250	7.5	344	5.6	87.7
20	375	250	15	364	5.9	87.5
20	375	250	30	375	6.0	88.6
65	375	25	60	233	4.0	83.6
65	375	75	7.5	264	4.2	89.5
65	375	75	15	300	4.6	92.0
65	375	75	30	318	4.9	92.6
65	375	250	7.5	351	5.3	94.0
65	375	250	15	375	5.7	94.0
65	375	250	30	375	5.8	94.2
65	375	400	7.5	368	5.6	93.7
65	375	400	15	375	5.8	92.3
65	375	400	30	375	5.9	90.6
65	105	250	15	105	2.9	50.7
65	155	250	15	155	3.0	72.9
65	205	250	15	205	3.5	84.9
65	310	250	15	310	4.8	92.0
65	375	250	15	375	5.7	94.0
65	410	250	15	410	6.2	94.3

The tests demonstrated that moderate heating of an agglomerated or pasty charge converts the copper to the sulfate form, which is amenable to rapid leaching with water. Provided enough acid was used, a 7.5- to 15-minute heat at temperatures between 75° and 400° C. permitted good extraction of the copper from the 10-, 20-, and 65-mesh feeds. The optimum temperature for sulfating the Lake Shore ore appears to be about 250° C. Although a temperature of 400° C. is permissible, a higher temperature dehydrates the sulfate and necessitates leaching of the calcine with weak acid. Virtually all of the acid employed in the sulfating procedure is consumed. No free acid, or only minor quantities, was found in the leach liquors. The moderate heat treatment increases solution of the clay and iron minerals in the ore, and the acid consumed per pound of copper dissolved is higher than in bottle leaching. The greater consumption of acid, however, is offset by the higher extraction of copper and the shorter treatment period required. The sulfated charges from tests made at 250° C. were compact and dry, regardless of the quantity of acid used. The calcines produced at lower sulfating temperatures were slightly moist. No difficulty was experienced in leaching the calcines, as they slaked readily upon addition of water, and the copper sulfate dissolved rapidly. The leached residues thickened readily and were much easier to filter than those from the bottle leaching tests. The mild heat treatment apparently dehydrates the colloidal silica and increases the filtration rate.

Copper extraction in the acid-sulfating tests decreased with increasing coarseness of the feed. Incomplete extraction of the copper in the 10- and 20-mesh feeds may be attributed to slow diffusion of acid through the particles during the short agglomerating and heating periods. Acid-sulfating gave somewhat lower extractions on coarse feeds than bottle leaching. As regards the time required for comparable copper extractions, however, acid-sulfating is superior. The results of several tests by the two procedures are given in table 6.

TABLE 6. - Comparison of acid sulfating and bottle leaching of 10-, 20-, and 65-mesh ore.

Mesh of feed	Method	H_2SO_4 added, lb./ton	H_2SO_4 consumed: Lb./ton	H_2SO_4 consumed: Lb./lb. of copper extracted	Extraction, percent of total copper
10......	15-min. acid sulfating and 15-min. water leach.	375	352	5.9	84.9
10......	8-hour bottle leach.	375	228	3.8	84.9
20......	15-min. acid sulfating and 15-min. water leach.	375	364	5.9	87.5
20......	12-hour bottle leach.	375	264	4.3	87.2
65......	15-min. acid sulfating and 15-min. water leach.	375	375	5.7	94.0
65......	72-hour bottle leach.	412	377	5.6	93.8

Supplementary tests were made to observe the deportment of the ore toward leaching-precipitation-flotation. The results of a typical leach-float test employing bottle leaching are given in table 7. The ore was ground in a rod mill to pass 65 mesh and leached for 2 hours at 50 percent solids, 205 pounds of sulfuric acid being used per ton of ore. Part of the free acid remaining in the pulp was neutralized with hydrated lime, and the cement copper was then precipitated with iron nails. After neutralization of substantially all the remaining free acid, the cement copper was floated, Minerac A being used as the collector. Single-cleaning of the rougher froth yielded a cement copper concentrate that assayed 71.42 percent copper and represented a recovery of 73.2 percent. Leach-flotation of 200-mesh portions of the ore gave almost identical results. Depending on the reagents employed, 76 to 80 percent of the copper was recovered as a rougher product assaying 35 to 40 percent copper. Inability to obtain a higher copper recovery by leach-flotation can be attributed to incomplete dissolution of the refractory copper silicate rather than to inferior flotation of the cement copper. The copper content of flotation tailings and of residues from comparable leaching tests were almost identical.

TABLE 7. - Bottle leaching-precipitation-flotation of 65-mesh ore.

Product	Weight, percent	Assay, percent total copper	Distribution, percent total copper
Copper concentrate........	3.5	71.42	73.2
Middling..................	9.3	1.77	4.8
Rougher froth.............	12.8	20.82	78.0
Tailing...................	87.2	0.86	22.0
Composite.................	100.0	3.42	100.0

	Pounds per ton						
				Flotation			
Reagent	Leaching	Precipitation		Conditioner		Rougher	Cleaner
H_2SO_4........	206	-		-		-	-
$Ca(OH)_2$.......	-	16.0		24.0		-	-
Minerac A.....	-	-			0.2	-	-
Pine oil......	-	-			-	0.04	0.02
			Iron nails				
Time (min.)...	120	15	30	5	2.5	5	2.5
pH...........	1.75	2.85	3.40	4.90	-	4.5	5.0

Other tests were made with more acid in the leaching step in an effort to obtain more complete extraction of the copper. The tests were not successful. The large quantity of free acid remaining in the leached pulp vitiated both precipitation and flotation of the cement copper. Prohibitive quantities of lime were required to neutralize the acid, and the pulps became so contaminated with salts that flotation of the cement copper was incomplete. When neutralizing steps were omitted, precipitation of the copper was incomplete, and much iron was dissolved by the free acid. The iron salts and residual acid inhibited subsequent flotation of the copper. These and other tests demonstrated that excess acid must be avoided in conventional leach-float procedures.

Precipitation-flotation tests also were made on acid-sulfated charges. The results of a typical test made on the 65-mesh feed and employing 375 pounds of acid per ton for sulfating are given in table 8. The acid-agglomerated ore was heated 15 minutes at 250° C. and then leached 15 minutes with water at 50 percent solids. As the leach pulp was substantially free of acid, the neutralizing steps before copper precipitation and flotation were not necessary. Single-stage cleaning of the rougher froth yielded a cement copper concentrate that assayed 69.7 percent copper and represented a recovery of 89.7 percent; the rougher concentrate accounted for 90.7 percent of the copper. Flotation of the cement was excellent and copper losses in the tailings were due primarily to presence of undissolved silicates.

Excellent results also were obtained on acid-sulfated charges of the ore by precipitating the copper during the water-leaching step. Simultaneous leaching and precipitation gave a somewhat finer and darker-colored cement copper than two-stage treatment, but it was readily amenable to flotation.

TABLE 8. - Precipitation-flotation of acid-sulfated ore.

Product	Weight, percent	Assay, percent total copper	Distribution, percent total copper
Copper concentrate.....	4.4	69.70	89.7
Middling..............	4.9	0.71	1.0
Rougher froth..........	9.3	33.35	90.7
Tailing...............	90.7	0.35	9.3
Composite.............	100.0	3.42	100.0

Reagent	Sulfating treatment	Water extraction[1]	Precipitation	Flotation: Conditioner	Flotation: Rougher	Flotation: Cleaner
H_2SO_4............	375		-	-	-	-
Minerec A..........			-	0.30	-	-
Pine oil.........			-	0.04	0.04	-
			Iron nails			
Temperature, °C. .	250					
Time, minutes.....	15		30	2.5	5	2.5
pH...............		3.15	3.15	3.5	3.5	3.6

1/ 15-minute agitation with water at 50 percent solids at room temperature.

Summary and Conclusions of Metallurgical Tests

The Lake Shore ore is refractory toward leaching. A long contact period with a large excess of acid is necessary to obtain a high copper extraction. Acid-sulfating at temperatures between 75° and 400° C. is superior to conventional leaching. Acid-sulfating requires more acid than flood or trickle leaching but is offset by the higher copper extraction and the shorter treatment period required. On other less refractory ores, the quantities of acid required for acid sulfating and bottle leaching were almost identical.

The leach liquors from conventional leaching of the Lake Shore ore contain much free acid, whereas, those from acid-sulfated charges were virtually free of acid. In leaching-precipitation or leaching-precipitation-flotation procedures, where free acid in the leach liquor or ore pulp is objectionable, acid sulfating should have merit.

Flotation of the Lake Shore ore by usual sulfide and nonsulfide collectors was ineffective. Leach-precipitation-flotation gave good copper recoveries. In conjunction with the leach-float procedure, acid-sulfating was superior to bottle leaching. When using flood or trickle leaching, the excess acid remaining in the pulp must be partly neutralized before precipitation and flotation of the cement copper. As virtually no free acid remains in the acid-sulfated pulps, the neutralizing steps before precipitation and flotation are unnecessary, thus simplifying the procedure. Simultaneous leaching and precipitation of the copper from acid-sulfated charges also gave good results.

DRILL-HOLE LOGS

Hole D-1

Location: N. 5391, E. 5119
Elevation of collar: 1,664 ft.
Depth: 203.5 ft.

Dip: -73°
Bearing: N. 80° E.
Date: 1/19 to 3/18/49

Footage			Percent copper		Oz./ton		
From-	To-	Feet	Total	Acid-soluble	Au	Ag	Description and remarks
0	11.0	11.0	1.48	1.48			Schist.
11.0	16.0	5.0	.26				Andesite.
16.0	21.0	5.0	.28				Do.
21.0	26.0	5.0	.25				Do.
26.0	32.0	6.0	.44				Do.
32.0	35.0	3.0	.40				Do.
35.0	40.0	5.0	.33				Do.
40.0	45.0	5.0	.34				Do.
45.0	50.0	5.0	.25				Do.
50.0	53.0	3.0	.20				Do.
53.0	58.0	5.0	.21				Do.
58.0	61.5	3.5	.20				Do.
61.5	65.5	4.0	.24				Do.
65.5	70.5	5.0	.20				Do.
70.5	75.5	5.0	.17				Do.
75.5	78.0	2.5	.18				Do.
78.0	81.5	3.5	.17				Do.
81.5	86.2	4.7	.18				Do.
86.2	88.2	2.0	.18				Do.
88.2	90.7	2.5	.18				Do.
90.7	94.6	3.9	.28				Do.
94.6	99.6	5.0	.14				Do.
99.6	104.9	5.3	.17				Do.
104.9	110.0	5.1	.17				Do.
110.0	115.0	5.0	.18				Do.
115.0	120.0	5.0	.19				Do.
120.0	125.0	5.0	.17				Do.
125.0	127.0	2.0	.10				Do.
127.0	132.0	5.0	.17				Do.
132.0	137.0	5.0	.13				Do.
137.0	140.0	3.0	.19				Do.
140.0	145.0	5.0	.19				Do.
145.0	148.3	3.3	.19				Do.
148.3	153.3	5.0	.22				Do.
153.3	156.7	3.4	.16				Do.
156.7	161.7	5.0	.14				Do.
161.7	163.7	2.0	.13				Do.
163.7	168.5	4.8	.13				Do.
168.5	173.5	5.0	.13				Do.
173.5	178.5	5.0	1.45	1.20			Shear zone.
178.5	180.5	2.0	.46	.25			Do.
180.5	185.5	5.0	.83	.57	Tr	0.1	Do.
185.5	189.2	3.7	.89	.60			Do.
189.2	193.5	4.3	.68	.42			Do.
193.5	203.5	10.0					Granite.

Hole C-1

Location: N. 5007, E. 5188 Depth: 590.0 ft.
Elevation of collar: 1,796 ft. Date: 1/13 to 2/4/49

Footage			Percent copper		
From-	To-	Feet	Total	Acid-soluble	Description and remarks
0	20	20.0			Sand and gravel.
20	193	173.0			Weathered granite.
193	195	2.0	0.31		Schist and clay.
195	205	10.0	.33		Schist.
205	215	10.0	.36		Schist, water table at 211.0 ft.
215	225	10.0	.32		Schist.
225	235	10.0	.31		Do.
235	245	10.0	.27		Do.
245	250	5.0	.34		Do.
250	255	5.0	.26		Do.
255	265	10.0	.36		Do.
265	275	10.0	.41		Do.
275	285	10.0	.39		Do.
285	295	10.0	.27		Do.
295	305	10.0	.35		Do.
305	310	5.0	.39		Do.
310	315	5.0	.42		Do.
315	320	5.0	.43		Do.
320	325	5.0	.61	0.36	Do.
325	330	5.0	.57	.29	Do.
330	335	5.0	.51	.28	Do.
335	340	5.0	.34		Quartzite and schist.
340	345	5.0	.36		Do.
345	350	5.0	.28		Do.
350	355	5.0	.38		Do.
355	360	5.0	.39		Do.
360	365	5.0	.25		Contact - schist and granite.
365	370	5.0	.16		Schist and granite.
370	375	5.0	.14		Granite and schist.
375	380	5.0	.10		Do.
380	385	5.0	.14		Do.
385	550	165.0			Granite.
550	555	5.0			Shear zone, clay.
555	590	35.0			Granite.

Hole C-2

Location: N. 4813, E. 5098 Depth: 561.0 ft.
Elevation of collar: 1,792 ft. Date: 2/11 to 3/5/49

Footage			Percent copper		
From-	To-	Feet	Total	Acid-soluble	Description and remarks
0	20	20.0			Sand and gravel.
20	155	135.0			Weathered granite.
155	165	10.0	0.23		Schist.
165	175	10.0	.22		Do.
175	185	10.0	.22		Do.
185	195	10.0	.23		Do.
195	205	10.0	.33		Do.
205	215	10.0	.32		Do.
215	225	10.0	.32		Schist, water table at 220 ft.
225	235	10.0	.31		Schist.
235	245	10.0	.48		Do.
245	255	10.0	.53	0.06	Do.
255	260	5.0	.51	.06	Do.
260	270	10.0	.62	.06	Do.
270	280	10.0	.46		Do.
280	290	10.0	.54	.06	Do.
290	300	10.0	1.03	.22	Do.
300	305	5.0	.57	.19	Do.
305	310	5.0	.50	.15	Do.
310	315	5.0	1.25	.55	Do.
315	320	5.0	.89	.40	Do.
320	325	5.0	.90	.33	Do.
325	330	5.0	.91	.31	Do.
330	335	5.0	.80	.28	Do.
335	340	5.0	.54	.17	Do.
340	345	5.0	.98	.29	Do.
345	350	5.0	.63	.19	Do.
350	355	5.0	.62	.18	Do.
355	360	5.0	.68	.21	Do.
360	365	5.0	.34		Do.
365	370	5.0	.31		Andesite.
370	375	5.0	.29		Do.
375	380	5.0	.26		Do.
380	385	5.0	.28		Do.
385	390	5.0	.62	0.09	Do.
390	395	5.0	.31		Do.
395	405	10.0	.24		Do.
405	415	10.0	.22		Do.
415	425	10.0	.22		Do.
425	435	10.0	0.26		Do.
435	445	10.0	.19		Do.
445	455	10.0	.15		Do.
455	460	5.0	.19		Do.
460	470	10.0	.18		Quartzite.
470	480	10.0	.18		Do.
480	490	10.0	.14		Do.

Hole C-2, Cont'd.

Footage			Percent copper		
From-	To-	Feet	Total	Acid-soluble	Description and remarks
490	500	10.0	0.16		Andesite.
500	510	10.0	.16		Do.
510	520	10.0	.23		Do.
520	525	5.0	.51	0.19	Shear zone.
525	530	5.0	.79	.48	Do.
530	535	5.0	.89	.63	Do.
535	540	5.0	1.43	.91	Shear zone. Little pyrite and native copper.
540	545	5.0	1.27	.57	Shear zone.
545	547	2.0	.74	.28	Schist.
547	550	3.0	.72	.30	Schist and granite.
550	555	5.0	.65	.27	Do.
555	561	6.0			Granite.

Hole C-3

Location: N. 4610, E. 5045 — Depth: 671.0 ft.
Elevation of collar: 1,788 ft. — Date: 3/9 to 4/2/49

Footage			Percent copper		
From-	To-	Feet	Total	Acid-soluble	Description and remarks
0	10	10			Sand and gravel.
10	125	115.0			Weathered granite.
125	135	10.0	0.33		Schist.
135	145	10.0	.22		Do.
145	155	10.0	.31		Do.
155	165	10.0	.40		Do.
165	175	10.0	.32		Do.
175	185	10.0	.27		Do.
185	195	10.0	.23		Do.
195	205	10.0	.16		Do.
205	215	10.0	.10		Do.
215	225	10.0	.16		Schist, water table at 225 ft.
225	235	10.0	.17		Schist.
235	245	10.0	.22		Do.
245	255	10.0	.17		Do.
255	265	10.0	.16		Do.
265	275	10.0	.25		Schist, shear - much Fe oxide.
275	285	10.0	.25		Schist.
285	295	10.0	.27		Do.
295	305	10.0	.30		Schist, shear - much Fe oxide.
305	315	10.0	.32		Schist.
315	325	10.0	.25		Do.
325	335	10.0	.27		Schist, shear - much Fe oxide.
335	345	10.0	.22		Schist.
345	355	10.0	.20		Do.

Hole C-3, Cont'd.

Footage			Percent copper		Description and remarks
From-	To-	Feet	Total	Acid-soluble	
355	365	10.0	0.23		Schist.
365	375	10.0	.20		Do.
375	385	10.0	.25		Do.
385	395	10.0	.15		Schist, shear - much Fe oxide.
395	405	10.0	.15		Schist.
405	415	10.0	.24		Do.
415	425	10.0	.18		Do.
425	435	10.0	.13		Do.
435	445	10.0	.12		Do.
445	455	10.0	.19		Do.
455	465	10.0	.14		Do.
465	475	10.0	.12		Do.
475	485	10.0	.13		Do.
485	495	10.0	.15		Do.
495	500	5.0	.18		Do.
500	510	10.0	.26		Do.
510	520	10.0	.23		Do.
520	530	10.0	.12		Andesite, shear - much Fe oxide.
530	540	10.0	.15		Andesite.
540	550	10.0	.14		Do.
550	560	10.0	.13		Do.
560	570	10.0	.12		Do.
570	580	10.0	.10		Do.
580	590	10.0	.14		Do.
590	600	10.0	.23		Andesite, shear - much Fe oxide.
600	605	5.0	.20		Andesite.
605	610	5.0	.24		Do.
610	615	5.0	.20		Do.
615	620	5.0	.14		Do.
620	625	5.0	.13		Do.
625	630	5.0	.18		Do.
630	635	5.0	.15		Do.
635	640	5.0	.14		Do.
640	645	5.0	.18		Do.
645	650	5.0	.26		Do.
650	655	5.0	.42		Andesite and granite.
655	660	5.0	.23		Granite and andesite.
660	671	11.0			Granite.

Hole C-4

Location: N. 4795, E. 4900
Elevation of collar: 1,786 ft.

Depth: 290.0 ft.
Date: 4/15 to 4/19/49

Footage From-	To-	Feet	Percent copper Total	Acid-soluble	Description and remarks
0	10	10.0			Sand and gravel.
10	50	40.0			Weathered granite.
50	60	10.0	0.35		Schist.
60	70	10.0	.33		Do.
70	80	10.0	.32		Do.
80	90	10.0	.35		Do.
90	100	10.0	.30		Do.
100	110	10.0	.36		Do.
110	120	10.0	.35		Do.
120	130	10.0	.17		Do.
130	140	10.0	.58	0.11	Do.
140	150	10.0	.33		Do.
150	160	10.0	.27		Do.
160	170	10.0	.23		Do.
170	180	10.0	.47		Do.
180	190	10.0	.82	0.52	Do.
190	200	10.0	1.23	.76	Do.
200	210	10.0	1.50	.65	Do.
210	220	10.0	1.55	.90	Do.
220	225	5.0	1.75	1.16	Do.
225	230	5.0	1.02	.66	Schist, water table at 225 feet.
230	235	5.0	1.31	1.00	Schist.
235	240	5.0	3.05	2.80	Do.
240	245	5.0	2.31	1.95	Do.
245	250	5.0	1.94	1.27	Do.
250	255	5.0	1.51	.96	Do.
255	260	5.0	1.54	.98	Do.
260	265	5.0	1.75	.97	Do.
265	270	5.0	.61	.36	Schist and quartzite.
270	290	20.0			Quartzite.

Hole C-5

Location: N. 5326, E. 5090
Elevation of collar: 1,801 ft.

Depth: 557.0 feet
Date: 4/20 to 5/13/49

Footage From-	To-	Feet	Percent copper Total	Acid-soluble	Description and remarks
0	25	25.0			Sand and gravel.
25	35	10.0	0.49		Schist.
35	45	10.0	.41		Do.
45	55	10.0	.40		Do.
55	65	10.0	.45		Do.
65	75	10.0	.87	0.41	Do.

Hole C-5 Cont'd.

Footage			Percent copper		Description and remarks
From-	To-	Feet	Total	Acid-soluble	
75	85	10.0	1.05	0.52	Schist.
85	95	10.0	1.18	.65	Do.
95	105	10.0	1.55	.96	Do.
105	115	10.0	2.10	1.19	Do.
115	125	10.0	1.69	1.00	Do.
125	135	10.0	1.76	1.10	Do.
135	140	5.0	2.21	1.14	Do.
140	145	5.0	2.08	1.16	Do.
145	150	5.0	1.87	1.13	Do.
150	155	5.0	2.15	1.41	Do.
155	160	5.0	1.65	1.39	Do.
160	165	5.0	.70	.39	Andesite.
165	175	10.0	.32		Do.
175	185	10.0	.26		Do.
185	195	10.0	.15		Do.
195	205	10.0	.15		Do.
205	215	10.0	.16		Do.
215	225	10.0	.10		Do.
225	235	10.0	.18		Andesite. Water table at 230 ft.
235	245	10.0	.16		Andesite.
245	255	10.0	.10		Do.
255	265	10.0	.14		Do.
265	275	10.0	.16		Do.
275	285	10.0	.16		Do.
285	295	10.0	.16		Do.
295	305	10.0	.15		Do.
305	315	10.0	.13		Do.
315	325	10.0	.18		Do.
325	335	10.0	.18		Do.
335	345	10.0	.20		Do.
345	355	10.0	.29		Do.
355	365	10.0	.19		Do.
365	375	10.0	.23		Do.
375	385	10.0	.18		Do.
385	395	10.0	.28		Do.
395	405	10.0	.20		Do.
405	415	10.0	.15		Do.
415	425	10.0	.15		Do.
425	430	5.0	.26		Do.
430	435	5.0	.08		Do.
435	440	5.0	.08		Do.
440	445	5.0	.95	0.67	Do.
445	450	5.0	.34		Do.
450	455	5.0	.19		Do.
455	460	5.0	1.88	1.73	Shear zone.
460	465	5.0	2.61	2.53	Do.

Hole C-5, Cont'd.

Footage			Percent copper		
Feet-	To-	Feet	Total	Acid-soluble	Description and remarks
465	470	5.0	1.78	1.55	Shear zone.
470	475	5.0	1.17	1.07	Do.
475	480	5.0	1.16	1.06	Do.
480	485	5.0	1.40	1.16	Do.
485	490	5.0	2.48	2.25	Do.
490	495	5.0	1.97	1.73	Do.
495	500	5.0	.91	.73	Do.
500	505	5.0	2.41	1.92	Do.
505	510	5.0	.50	.33	Do.
510	515	5.0	.73	.50	Do.
515	520	5.0	.73	.49	Do.
520	525	5.0	.18		Do.
525	530	5.0	1.54	1.26	Do.
530	535	5.0	2.98	2.31	Do.
535	540	5.0	3.06	2.51	Do.
540	545	5.0	2.09	1.65	Do.
545	550	5.0	.53	.38	Andesite and granite.
550	557	7.0			Granite.

FRANK M. LEONARD, JR.
MINING ENGINEER
935 N. Olsen,
Tucson, Ariz.

Mr. Roland Mulchay,
Cananea, Sonora, Mexico.

Dear Mr. Mulchay:-

It is certainly good news that you are coming to Tucson before long, and even better that Mrs. Mulchay is coming with you. Paco is still with the Highway Department but is generally at home Wednesday, and always over the week end from Saturday ~~af~~ at noon to Monday morning. I am spending a great deal of my time in Sonora, and never know where I will be at any given time. Our house is at the corner of East Second Street and Olsen Avenue, about four blocks east of the University, and we are in the phone book. We will hope to see much of you and your family when in Tucson.

As to Lake Shore I do not believe it is of any use to make an effort under present circumstances. When you come I will show you complete maps with assays and you can tell exactly what the property is without going there as it has been repeatedly examined by well known engineers. I gave the data to Moehlman of the Inspiration a year or so ago, and Perry turned it down on the theory that a secondary deposition could not exist under the oxidized ore on account of its composition. My having discovered such an ore body and having corroberative testimony from other engineers was entirely disregarded. Neither Moehlman nor Perry ever saw the property. It is the future of copper and not the possibilities of the mine under developmet that discourage me. I would like to hang on to it, howver, and when you come will introduce you to my grandson Charles Bonham Leonard, who may realize on it as he is only ten years old, and the Leonards are a long lived bunch.

With best regards,

Yours sincerely,

Frank M Leonard Sr

Cananea, Sonora, Mex.,
Mar. 4, 1941

Mr. Frank Leonard,
935 N. Olsen Ave.,
Tucson, Arizona.

Dear Mr. Leonard:

We received your note and the enclosed delayed Christmas card yesterday. Very kind of you to remember us, and we both hope that 1941 will prove a happy and successful one for you.

You have no doubt about given my intentions to visit the Lakeshore acritical going over in the past few months. However, I still have hopes, and if you haven't done anything with it yet, I may be able to see it this spring. Since the strike we have been very busy here, and I haven't been able to get away anywhere. Mrs. Mulchay hopes to get to Tucson for some new clothes in the not too distant future, and I may be able tocome with her. If she does, and I do, I hope we can arrange for a quick trip to the mine.

With best personal regards to your family and yourself,

Yours very truly,

Roland B. Mulchay,
Cananea, Sonora, Mex.

Cananea, Sonora, Mex.,
April 27, 1940

Mr. Frank Leonard, Jr.,
935 N. Olsen Ave.,
Tucson, Arizona.

Dear Mr. Leonard:

I have recently returned from Butte, Montana where I had several short talks with Mr. E. I. Renouard, Jr. at the Mt. Con Mine. I believe you know Mr. Renouard.

He suggected that he would like to have me review the maps of the Lakeshore Mine, and to visit the property if possible for him, if you were agreeable. On Wednesday last as we passed through Tucson I called your residence with this idea in mind.

As the Cananea operation is at present stopped by a strike, I have some free time for my own interests. I would therefore be very pleased if you would inform me when I could see the Lakeshore maps and perhaps visit the property. I would like to suggest next weekend, May 4-5 as a convenient time for me if that would suit your time.

I have thought of the Lakeshore property many times since I discussed it with you and your father in 1931 when you visited Cananea. I hope this time I may actually get to see it.

With best personalregards and hopes that I may have a letter from you confirming the above suggested dates, I am

Yours very truly,

Roland B. Mulchay,
% 4 C Co.,
Cananea, Sonora, Mex.

3/3/41

935 N. Olsen,
Tucson, Arizona.
Feb. 28, 1941.

Mr. Roland Mulchay and family,
Cananea, Sonora, Mexico.

Dear Mr. Mulchay:-

It is a little late to wish you a Happy New Year, but anyhow I hope the remaining ten months of it will turn out all right.

I sent you the greeting in time all right as you will see by the postmark on the enclosed, but I sent it to the wrong place and it has laid right there and did not get back till right now. Things in Mexico may be a little slow, but apparently they arrive at last.

With best regards,

Sincerely,

Frank M. Leonard

FRANK M. LEONARD, JR.
MINING ENGINEER
935 N. Olsen Ave,
Tucson, Arizona.

Mr. R. D. Mulchay,
Cananea Copper Company,
Cananea, Sonora, Mexico.

Dear Mr. Mulchay:-

Do you still take any interest in seeing Lake Shore? If so I can now arrange a trip most any wek end. I can not see why anyone should wish to acquire a copper development property in the United States at this time, but would lend a willing ear to any proposition with me on the selling end.

Am just back from two months spent in Mexico City where I renewed some old friendships and made some new connections and was present at an election reminiscent of the old scraps in Butte in the days of the copper war. It looks to me as if the Cárdenas government is about to collapse, so am hoping your bosses will stand pat a little longer, but am sure they know a whole lot more about it than I do.

I had some business with the Mines Department in connection with the La Brisca placers, and to my surprise received a whole lot better treatment than I had expected, due probably to the revival of an old friendship with Enrique Ortiz..

Hoping that you and your family are well, and with best regards,

Yours sincerely,

Frank M. Leonard

BY DIRECT WIRE FROM

WESTERN UNION

CLASS OF SERVICE

This is a full-rate Telegram or Cablegram unless its deferred character is indicated by a suitable symbol above or preceding the address.

R. B. WHITE, PRESIDENT — NEWCOMB CARLTON, CHAIRMAN OF THE BOARD — J. C. WILLEVER, FIRST VICE-PRESIDENT

1223

SYMBOLS
DL = Day Letter
NL = Night Letter
LC = Deferred Cable
NLT = Cable Night Letter
Ship Radiogram

BE1 28 4 EXTRA NT=TUCSON ARIZ 27 VIA INSPIRATION ARIZ 28

ROLAND B MULCHAY=

CARE T E PASCHAL NACO ARIZ=

JUST RETURNED FROM SONORA. IF YOU STILL WISH TO SEE LAKESHORE CAN ARRANGE TRIP EITHER FOR THURSDAY OF THIS WEEK OR NEXT SUNDAY. REGARDS=

FRANK M LEONARD.

833A..

COPY

New York City	San Francisco	Salt Lake City,	Pachuca, Mexico
120 B'way	1504 Hobart Bldg.	900 Newhouse Bldg	Las Cajas.

UNITED STATES SMELTING, REFINING & MINING EXPLORATION COMPANY

55 Congress Street
Boston, Mass.

Salt Lake City, Utah, May 17, 1921

Mr. Frank M. Leonard,
3543 Third Street,
San Diego, California

Dear Mr. Leonard:-

Your letter of the 12th was forwarded to me here. As I remember the sulphide ore in the drift from the winze, was partially oxidized, indicating that you are about on the top of the sulphides. I do not reaall the exact copper content, but think it was about 6% or 8%. It occurred in bunches and streaks in the bottom of the drift, and was about 5 or 6 feet wide. The crossout back to the west from this drift was all in broken, crushed mineralized limestone, which gave strong indications of faulting and no replacement. If this is not correct I wish you would put me right in this matter. That is my memory of it. I shall be glad to give such a description to anyone interested.

I am glad to hear that you and the family are well and getting along nicely. I note that you expect to move to Berkeley in the fall. Only the youngest of my children will be in the University from now on, as all the others have graduated and taken their post graduate work. I am not sure whether we shall be in Berkeley during the winter, but I would be glad to have you keep me informed of your address when you do move to Berkeley, so that I can look you up when I am there.

With kinests regards to yourself and family, I remain,

Very truly yours,

(Signed) F.B. Weeks

Forma. 1007

ESTADOS UNIDOS MEXICANOS

DIRECCION GENERAL DE CORREOS Y TELEGRAFOS

SERVICIOS ECONOMICOS

Para el interior de la República, Europa, Estados Unidos de Norte América, Canadá, Guatemala, Honduras, Costa Rica, El Salvador, Cuba, etc.

Sírvase Ud. anotar la hora en que reciba este mensaje.

Forma 1002

ESTADOS UNIDOS MEXICANOS

CORREOS Y TELEGRAFOS

TELEGRAMA

Sello de la Oficina

Núm. ______ De ______ el ___ de ______ de 193__

Recibido en ______

H. D.	H. R.	T
		R

Vía 11 Nogales Son. 4 mayo de 40
21 w 70 ord. pd d 9.55

Sr. Ing. Roland B Mulchay
Cananea Copper Co.

Will be here until tuesday at san Carlos hotel can make appointment any time after that have mine date with me

Frank M Leonard

Cxgr 10.15

Todo telegrama debe llevar el sello de la Oficina.

Lea usted el reverso; le interesa conocer los diferentes servicios que le ofrece el Telégrafo.

935 N. Olsen Ave,
Tucson, Arizona.
May 31, 1940.

Mr. Roland B. Mulchay,
Inspiration Mining Co.
Inspiration, Arizona.

Dear Mr. Mulchay:-

Sorry you did not have time to come up to the house today when you passed through. Paco and I are both away most of the time but his wife and her mother and young Charles Leonarrd are always here and would be glad to see you or your family at any time, and week ends we generally manage to get here.

When I told you about the sulphides at Lake Shore, now under water, I mentioned a letter I had from F. B. Weeks, who came to the mine several times when these sulphides were in sight. Weeks was at that time at the head of the U. S. S. & R. exploration outfft, and was trying to negotiate with me. You said you would like to see this letter, and I am enclosing a copy of it.

My son could probably go with you to Lake Shore any Sunday, and if I am here I will go along.

With best regards,

Sincerely,

Frank M. Leonard

Battle Mountain Ne. Aug. 3, 1923.

COPPER CANYON MINING CO.)
F. Sommer Schmidt, Mfg.)
Battle Mountain, Nevada.)

Lake Shore Preliminary Report

Mr. L. R. Whicher
25 Broad Street
New York City, N. Y.

Dear Sir:

Following in my preliminary report on the Lake Shore Mine, Ariz.

Introduction

The following report is the result of several hours examination of the Lake Shore Mine in May 1923. It must be distinctly understood that it does not represent long careful study. It is simply the way I have "sized up" the proposition but my opinion is nevertheless a positive one. I think the property offers unusual promise for good grade sulfide tonnage. I went through all the workings but I had no maps at the time. I doubt whether final detailed examination would change my ideas as to the future possibilities in depth. What the property needs is work below the present deepest level (265) and have seen enough to take a very positive stand. I dont see how the proposition can be turned down on the deal that can be obtained even if it is recommended in a more qualified way.

Location

Township 10 S Range 4 E in the southwest corner of Pinal county Ariz. - 30 miles south west of Casa Grande a station on the S.P.R.R. The mine is located on the edge of the valley at the base of the foothills about 800 feet above the valley bottom. Very gentle grade to the mine. High gear all the way in any auto. The road to Casa Grande is very good.

Claims

3 patented and 30 unpatented lode claims. See Map.

Working Conditions

Drinking water must be hauled a few miles at present. Water for engines can be obtained from the mine. In the valley bottom there are wells with exceptionally large flows which could be used as a source of water for any ultimate reduction plant. Working conditions in general are favorable.

Equipment

Office-laboratory-boarding house- several miners shacks hoist house with hoist-engine-compressor apparently all ready to go.

History

Chas. Leonard of Butte, Mont. and Frank Leonard now in Mexico bought the property in 1913. During the war prices they gave lease and bond for $500,00 for 80% interest to representative of the Potter Palmer interests of Chicago. Later the same deal was made with representatives of the Gerard family former Ambassador to Germany. Both outfits asked for extension of lease and bond when their time was up. Leonard agreed to the option contingent on their developing the property but balked on the lease. He was through with that business and disgusted. He was right. The Gerard interests represented by Crawford drilled 4 churn drill holes. 3 of them ran into the granite footwall before reaching sulfides-while the 4th apparently did the same thing, though it attains a depth 684? of 624 ft before reaching the granite. The 4th hole is the only one that had any chance. The first 3 were impossible. The fact is that this is not a drilling proposition. These holes have hurt the property.

The maps however show that there is no occasion for this and I regard the 2 oxidised zones in hole 4 as very interesting. During this period many engineers have examined the property, and made application for just a deal as I have obtained. It is known by me that the following engineers have thought well of the property. Seeley Mudd and Wiseman-F.W. Weeks of the U.S. Smelting Co.-Jerolemon-Magma engineers-Kruttschnitt of A.S.R.Co.-Bjorge associated with Locke of the Calumet and Hecla geological dept. Phillip Wilson of the C&A- Carl Lindberg L.D. Adams of San Francisco and McCracken Mine. No deal could be obtained by any of the above better than the one mentioned $500,000 for 80% interest with payments coming early in the operating period.

Weeks spent 2 months examining the property but could do nothing with the owners. Mudd has also tried several times to do business and there are others. Personally I have my own opinion and formed it before I had any information about this past history--but I mention this because these matters naturally should have a bearing..

The mine has been well sampled and the data is in good shape as a result of the work done by these leasing outfits.

Production

Shipments have been made from the property probably from its first discovery down to 1917. I have no good data as to total production through shipment but I have the actual returns for 852 tons 5.30% cu. shipped in 1917 as a result of leaser-like operations.

Development Work

Total 4,000 ft of underground work. Levels 65-115-152-225-265 or winze level. On vertical main shaft down 225 ft. This shaft is cared though the mine workings are accessible through another shaft. Another shaft 115 ft depth is known as the 65 shaft. Another "Granite" shaft 85 ft depth. It was clear that practically all present workings were intended to produce shipping ore from the oxidised zone. They were looking for the high grade bunches of cuprite and carbonates and with the war prices of copper it was attractive from a leasing standpoint. The workings indicate this condition.

The most interesting development work is inaccessible being the 265 or winze level where the high grade sulfides were struck. The water now stands immediately under the 225 ft level. Only a little water was encountered. Part of the time they bailed it and then put in a small pump. The fact that this level is under water has prevented intelligent examination. The ore has been drifted on for 680 feet with 4,000 feet of workings above the 265 level-all done by operators who apparently cared little for sulfide possibilities. I regard this situation as peculiar especially as the sulfides encountered on the 265 justified further work. This I have not seen myself but shipments of the high grade chalcopyrite and bornite and covellite were made averaging 16%-which I discuss later.

I regard the present workings as important insofar as they disclose the geology and block out 660,000 tons of 2.08% cu. from which the higher grade could be smelted direct if there were a smelter on the job.

The main shaft is caved and while I am told quite definitely that it cannot be saved I am not so sure about this and would not rest satisfied with the previous attempt so far as my present knowledge goes.

Present Developed Tonnage

The workings in the oxidized zone are deprived of such high grade ore as has been shipped. I have not sampled the tonnage developed but there are good assay maps available and from these I figure a total of 661,000 tons of 2,08% cu. developed. To increase this tonnage rapidly by extending these upper workings, would apparently be easy as the ore continues in both directions. It would not require much leniency to figure this tonnage at 1,000,000 tons but I have held the tonnage to the actual extent of each level instead of allowing the whole length of the ore exposes on the 152 level to go down to the 225 level. I personally believe that this full 680 ft length of ore on the 152 level could be easily proven on the 225 giving 1,000,000 tons but the fact is the work has not been done. Assuming that this 680 ft in length yield 1,000,000 tons the 3,000 ft of length of vein exposed one way or another, would figure 4,400,000 tons. The grade would probably be higher that 2,08% for an average due to the fact that considerable high grade was extracted before sampling in arrivingat the 2.08%. From this tonnage a certain proportion of 3%-4%-5% ore could be selected and mined for direct smelting. Precise figures are impossible at this time. I offer this train of thought for consideration because in any case this tonnage would be a very interesting item when sulfide tonnage is developed.

These values might be concentrated or filmed and floated ore even leached. Concerning this I do not know but believe it would not take long to definitely decide. The character of this ore is carbonates silicate and oxides - both red and black.

Geology

A 70 ft wide shear zone 3,000 ft long on contact between granite footwall and shist hanging wall-is the mineral feeder. The limits of the zone laterally are not well defined because in crosscutting the shist but there are always a few tenths percent copper present even at considerable distance from the actual vein. I do not think there is a working in the mine that is actually barren. The mineralization also makes into the granite in disseminated form but of negligible value. It is not a disseminated deposit.

The original conception of the property was that a vein in the shist work striking N.E. and dipping 60 deg E descended to the 225 level where it encountered the granite which is dipping 60 deg W. and then reversed its dip to follow the granite down as its footwall. This intersection with the granite is shown on the 225 level also the change in dip. The 265 or winze level that disclosed the sulfides is below the turn and proves that the continuation of the vein is on the new dip. I think the explanation is that ascending primary ore solutions were floowing the shear zone upward along the granite contact but found a line of least resistance- a line of shistosity or original bedding or fracturing-forking off and following this upward. There may be more than one of these. Drill Hole 4 indicates 2 of them.

Whether the vein would continue to have granite footwall in depth or actually make into the granite I do not know. Also whether the other formation would continue to be shist is provlematical. There is limestone on the property according to the maps though I did not see any and should this powerful vein intersect a limestone there would be no doubt be an excellent replacement.

The surface is covered with float. The mine is at the base of the hills. Considerabe erosion has removed a large part of the original deposit.

In places the vein is tighter than I would like to see it. I do not regard the conditions for secondar enrichment as ideal everywhere in the vein. The oxidized values are greater than one would judge by inspection due to the cuprite being indistinguishable from the iron oxide. Again at many places the soft and mineralized conditions are perfect and no duubt the facilities for secondary enrichment.have varied. I imagine that such engineers as have hesitated to recommend the property base their ideas on the questionable degree of secondary enrichment. This is one of the most improtant considerations with any copper mine and at this point I take the following definite stand.

First-There is such a large proportion of thevein that is favorable to secondary enrichment that I believe considerable chalcocite and covellite will be found especially in connection with fissurings.

Second-My hopes for the future of the mine are based entirely on high grade <u>primary</u> ore - chalcopyrite and bornite etc. If the mine depended upon secondary values as we do in many properties I would still think as I do but I believe the primary ore is high grade and not dependent on enrichment to be valuable for the following reasons.

First-There is much cuprite, a brown smear on every crack, which represents the oxidation of chalcocite which in turn represents the replacement of primary, sulfide. There was an insufficiency of sulfuric acid and ferric sulfate to make a good leaching job, caused by their being no excess pyrite in the primary ore.

Second The sulfides encountered on the 265 were partly primary and this chalcopyrite was high grade.

The maps show the vein to be 3,000 ft long as ~~px~~ exposed by surface diggings beyond the limite of underground work. I did not check this. Possibly even this does not represent the full length. The surfaced showing that appealed so much to me is the one shown on the map as the open cut north of the fault which is also shown on the map. This taken in connection with the "granite shaft" workings near it indicates a width of 150 ft on the map. All of this is north of the fault and has never been developed by any of the mine workings though this would be a comparatively simple thing to do.

Some of the maps call the contact a granite-andesite contact. I made no careful examination of this formation. It was called shist by my guide and it looked like shist to me. The log of Hole 4 calls it shist and it behaves like shist near the vein.

The question of greatest interest concerns the showing on the 265 level in the sulfides. This level was first attained by Frank Leonard who found enough 24% sulfide ore to make shipment. There was considerable oxidised ore mixed with the sulfide. No assay map of this level is available, which arouses suspicion, were it not for corroborating evidence. They actually did have something sensational at least at times but immediately struck a fault that cut off the ore. They had trouble holding the ground, and hauling the muck and water twice through the winze. Leonard was paying the bills and the production by shipping did not pay its way. The fact at times averaged 5% and then again 3% with boulders running 24% which is probably what they shipped. They did very little work. Jerolemon and Weeks I know to be familiar with this showing. Later another winze was sunk by the last leasing outfit.

This winze also attained a depth of 40 ft below the 225 level. They struck sulfides which looked so favorable that they decided they had a much bigger thing than they probably did have and accordingly decided to churn drill the deposit. The accounts for the 4 drill holes.

The maps show the location of the drill holes also log of hole 4 which is the only important one. I have seen the logs of the first 3 holes but they encountered the granite before reaching sulfide depth. I take it from the log of hole 4 that the second leached and caving zone shown thereon indicates that this hole also did not have any sulfides. There are always some low grade values no matter where they drill nor how far from the vein - even in the granite. Having done so much drilling myself - here and at the Inspiration and Canada Copper I feel qualified to condem this work as having been ill-advised.

The vein is undoubtedly as claimed 3,000 ft long if not longer. It is about 70 ft wide. The mineralization has been thrust into the country rock considerable distances. The intensity of the mineralization is good. The gold values which are negligible in the upper levels increases with depth amounting to several dollars per ton in the sulfide at times. The district and geological province is a very healthy one for copper mines. These conditions have more or less favorable bearing on the continuation of ore in depth.

The question arises whether the present oxidized zone may not represent the same values as the sulfide zone will if there is no secondary enrichment and the values have been held up in the upper levels. I think this is to far-fetched and I further believe that the fate of a property that has a strong evidence in its favor as this one should not depend upon any very fine-spun geological theories. The possibilities are too great. One cannot afford to overlook them.

Recommendations

Under the terms of the option we would have ample time to develop before making the payments which are just and proper.

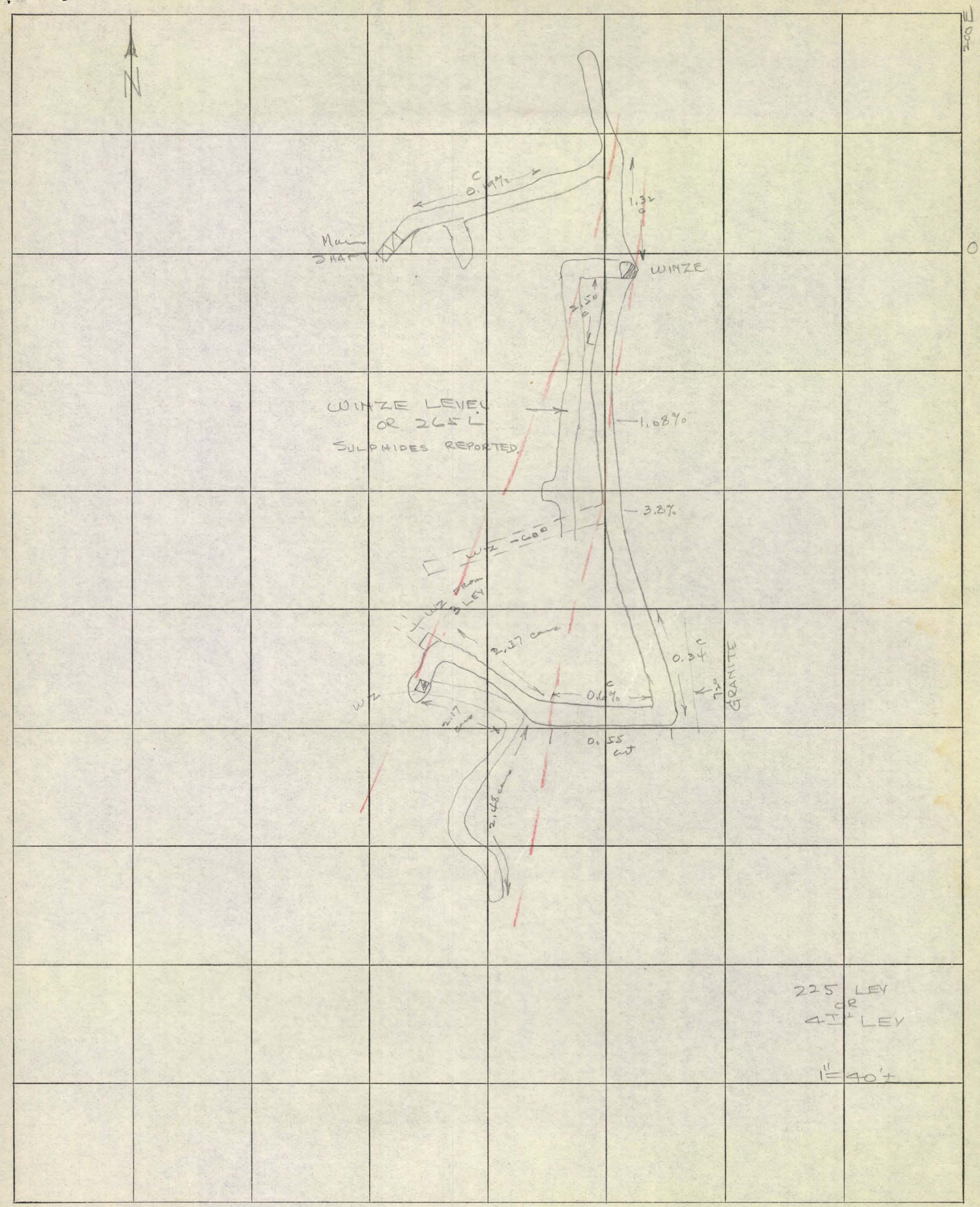
I would try to save the old shaft because its location is good and everything in installed. It would be cheaper than a new one and quicker. It would be intended only for prospecting and if after the results were satisfactory another larger one needed - this shaft would always be handy for air etc. Sink this shaft to 400 ft which takes it close to the granite wall. Explore at this level for sulfides, if not deep enough continue sinking until sufficient depth is attained to develope sulfides.

It may be necessary to enter the granite and crosscut west to the shist vein. Dont stop until sulfides have been found and explored and do it from present shaft if possible. Once in the sulfides I would not neglect going north of the fault which country has a beautiful surface showing and has never been explored at depth. The upper levels already assure 680 ft of shoot to be followed in the sulfides. I think 400 ft depth should be partly sulfide in any case.

Spend very little money if any in the oxidised zone unless there are exceptionally high prices for copper metal in which case it might pay to ship. I recognise the possibility of water increasing with depth though for the present I dont consider it any item of importance.

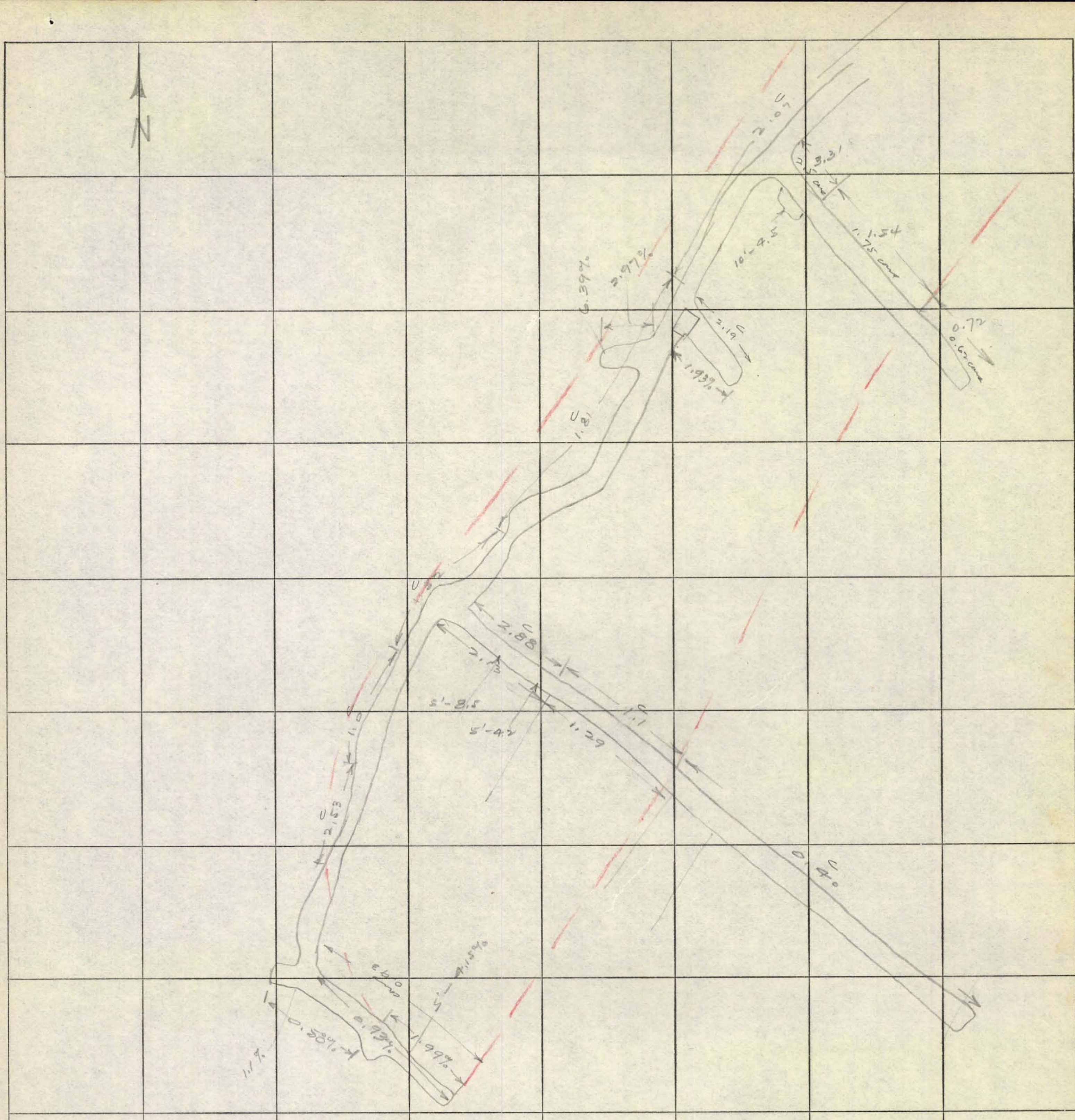
Respectfully submitted

(signed) F. SOMMER SCHMIDT.

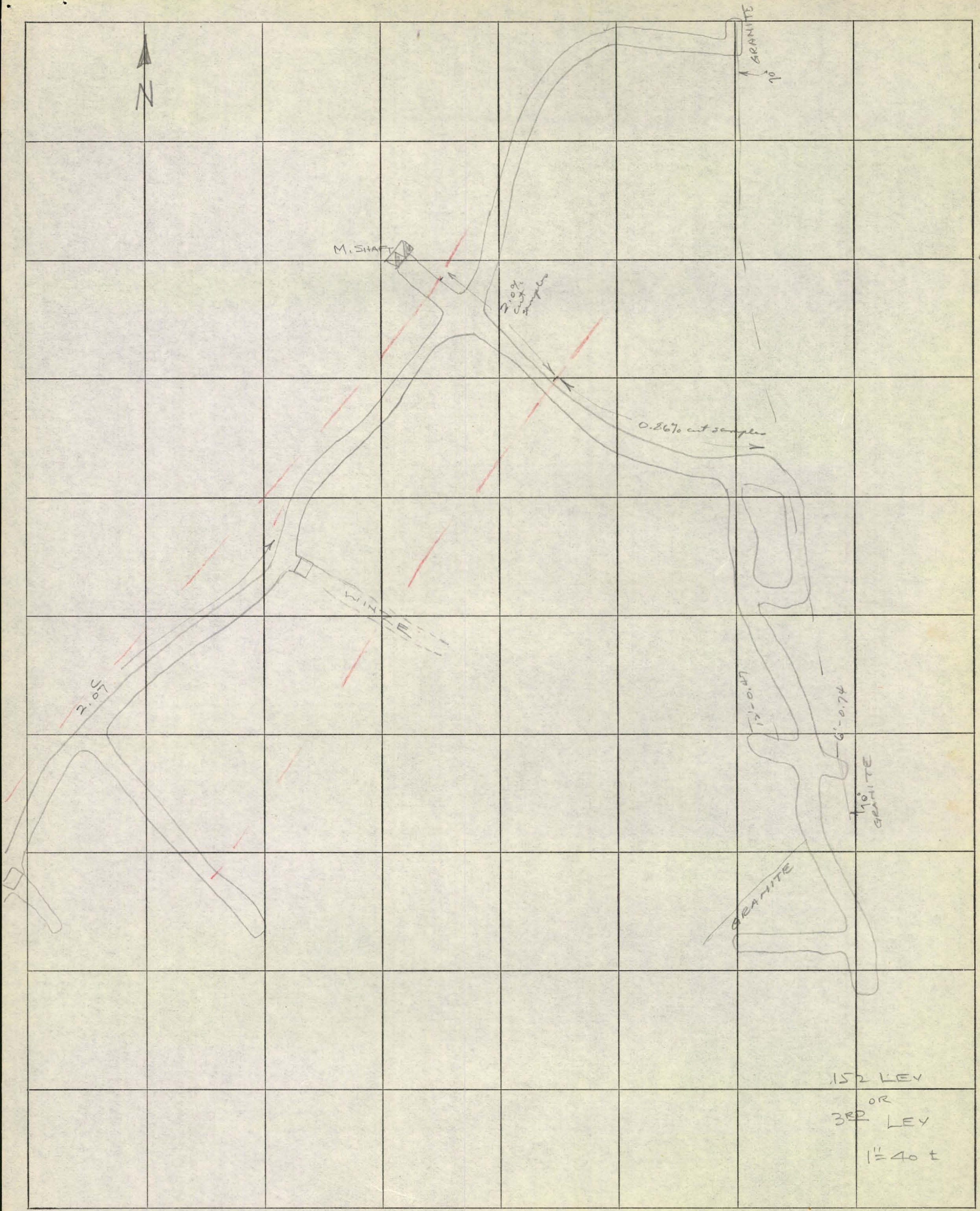


N
200 E
0.N
Main
SHAFT
1.32
WINZE
WINZE LEVEL
OR 265 L
SULPHIDES REPORTED
1.08%
3.8%
WZ
GRANITE
0.55
cut
225 LEV
OR
4th LEV
1" = 40' ±

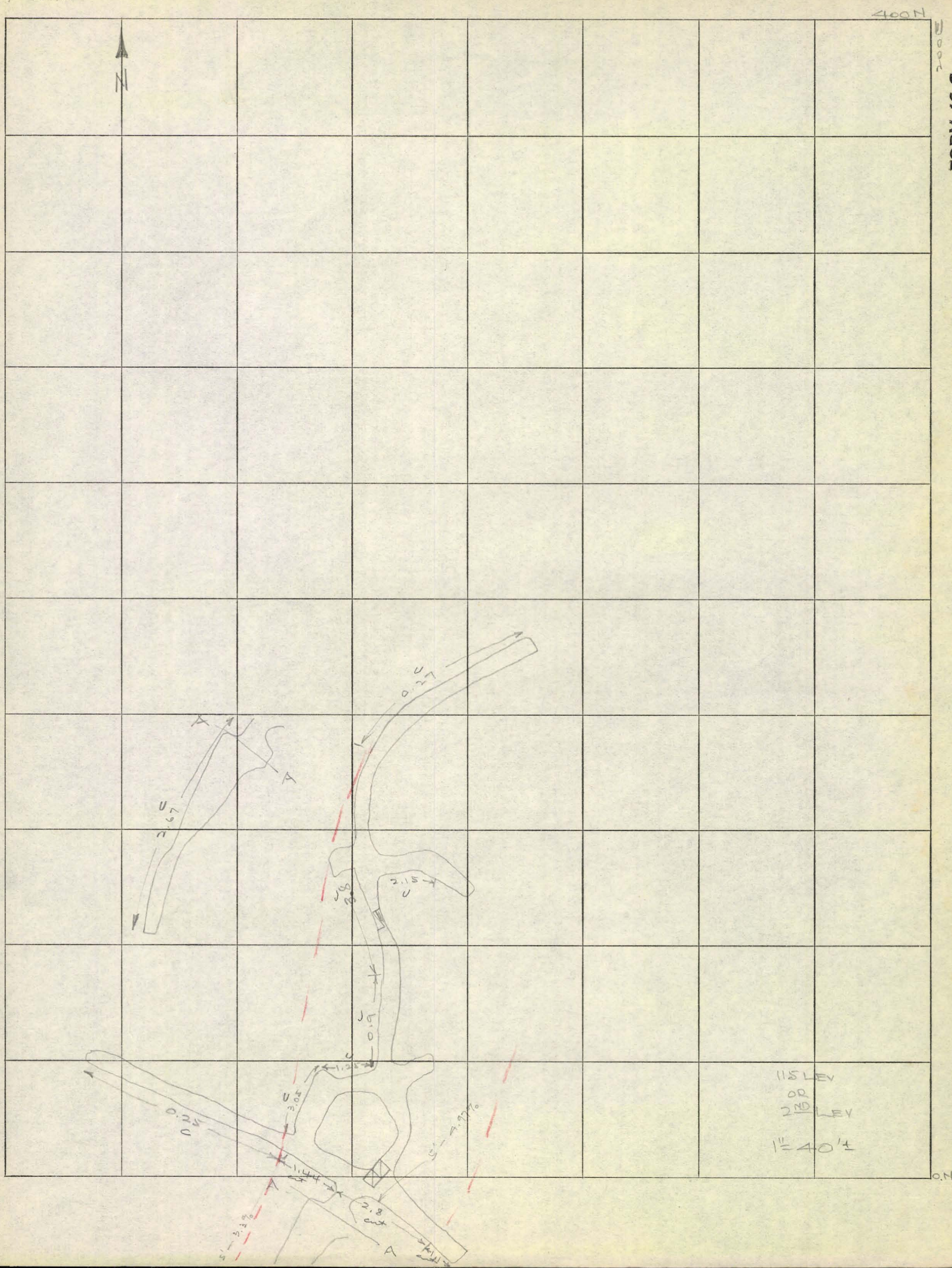
FORM 16 G



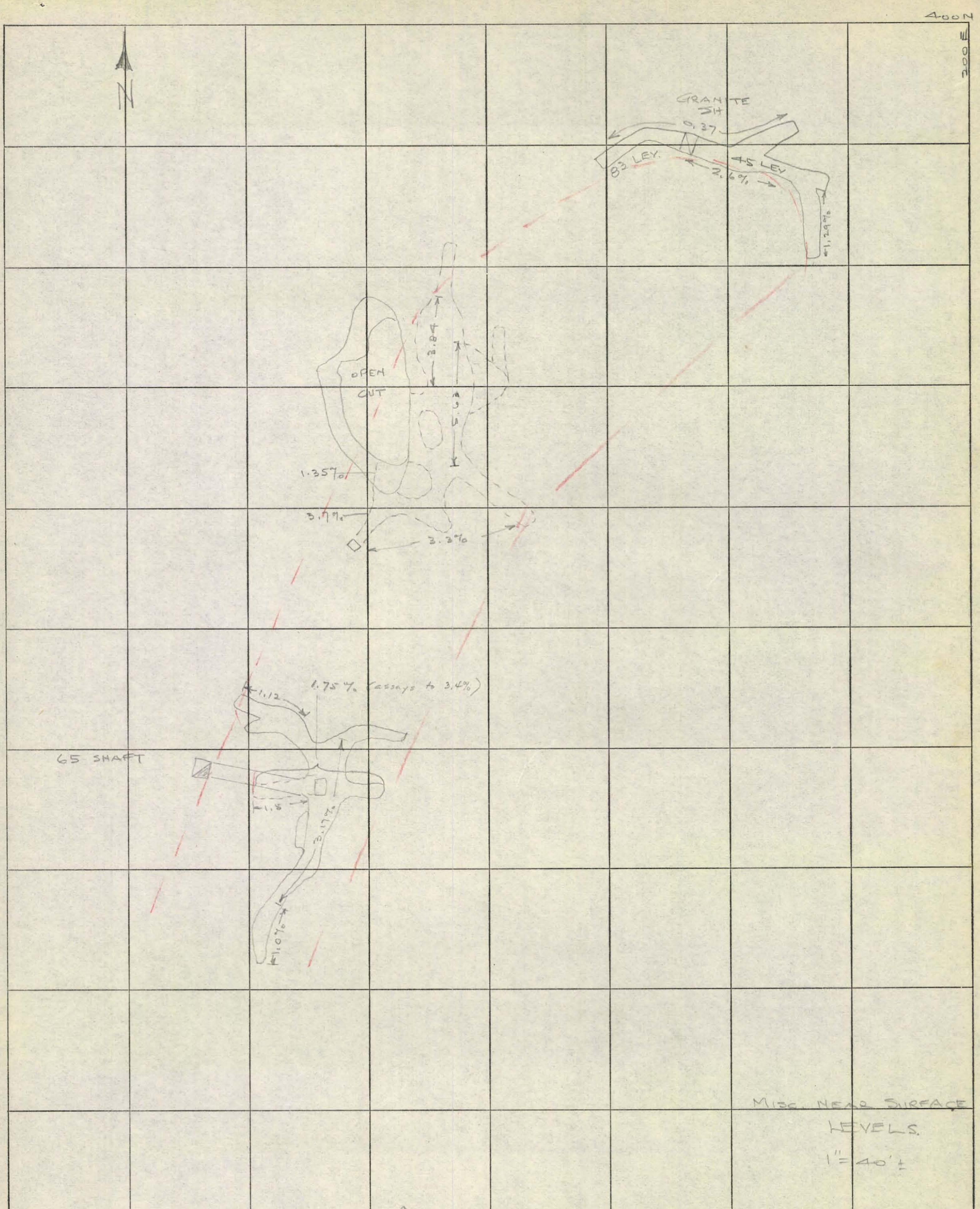
15 ½ LEV
OR.
3 RD LEV
SOUTH
1" = 40' ±


N
M. SHAFT
2.0% cut samples
0.86% cut samples
GRANITE
10°
200 E
0,H
WINZE
2.07
15'-0.47
6'-0.74
10°
GRANITE
GRANITE
152 LEV
OR
3RD LEV
1"=40 ft

FORM 16 G

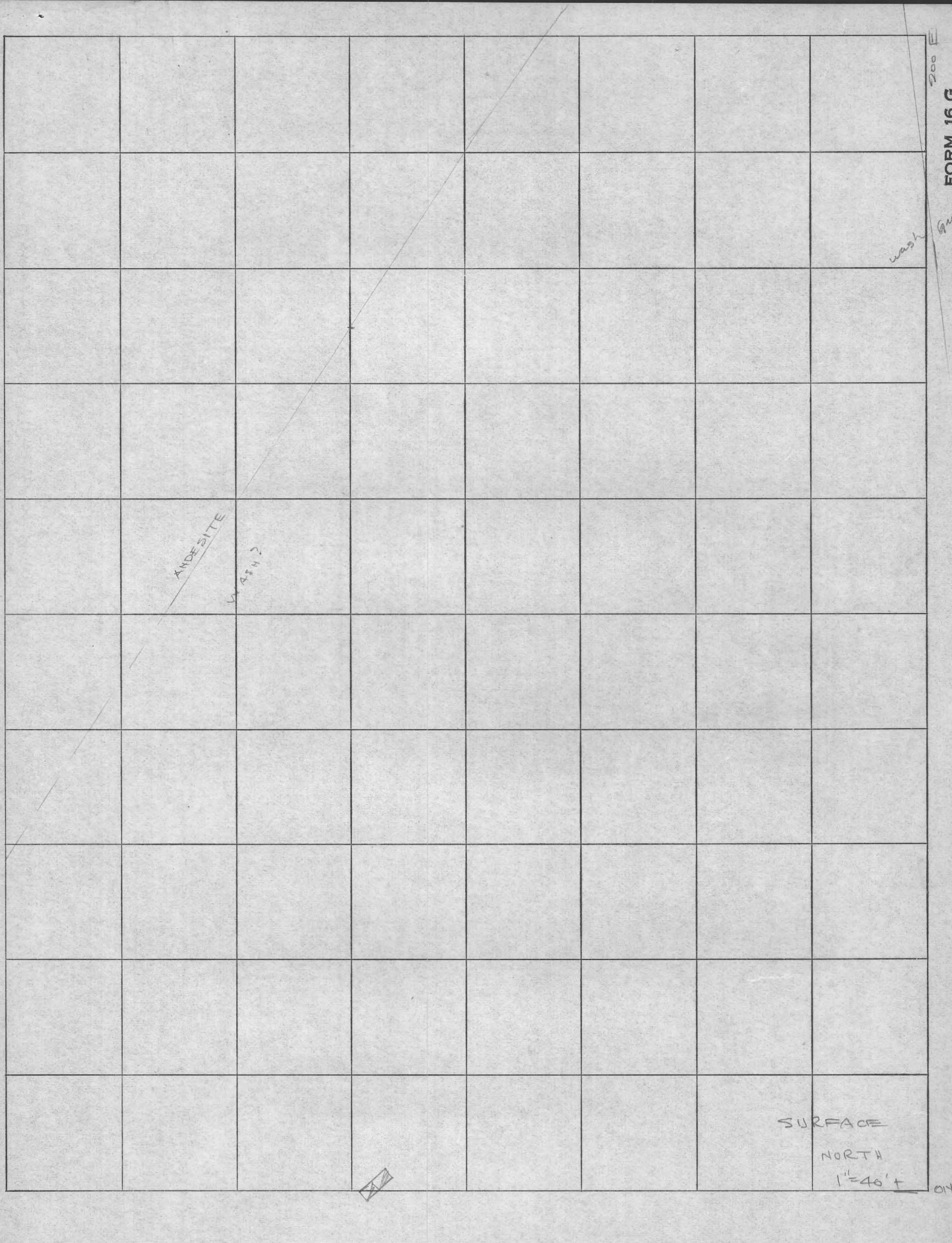

N
400 N
200 E
0 N
A
2.67
0.27
2.15
1.25
3.05
0.25
1.44
2.8 cut
115 LEV
OR
2ND LEV
1" = 40'

FORM 16 G



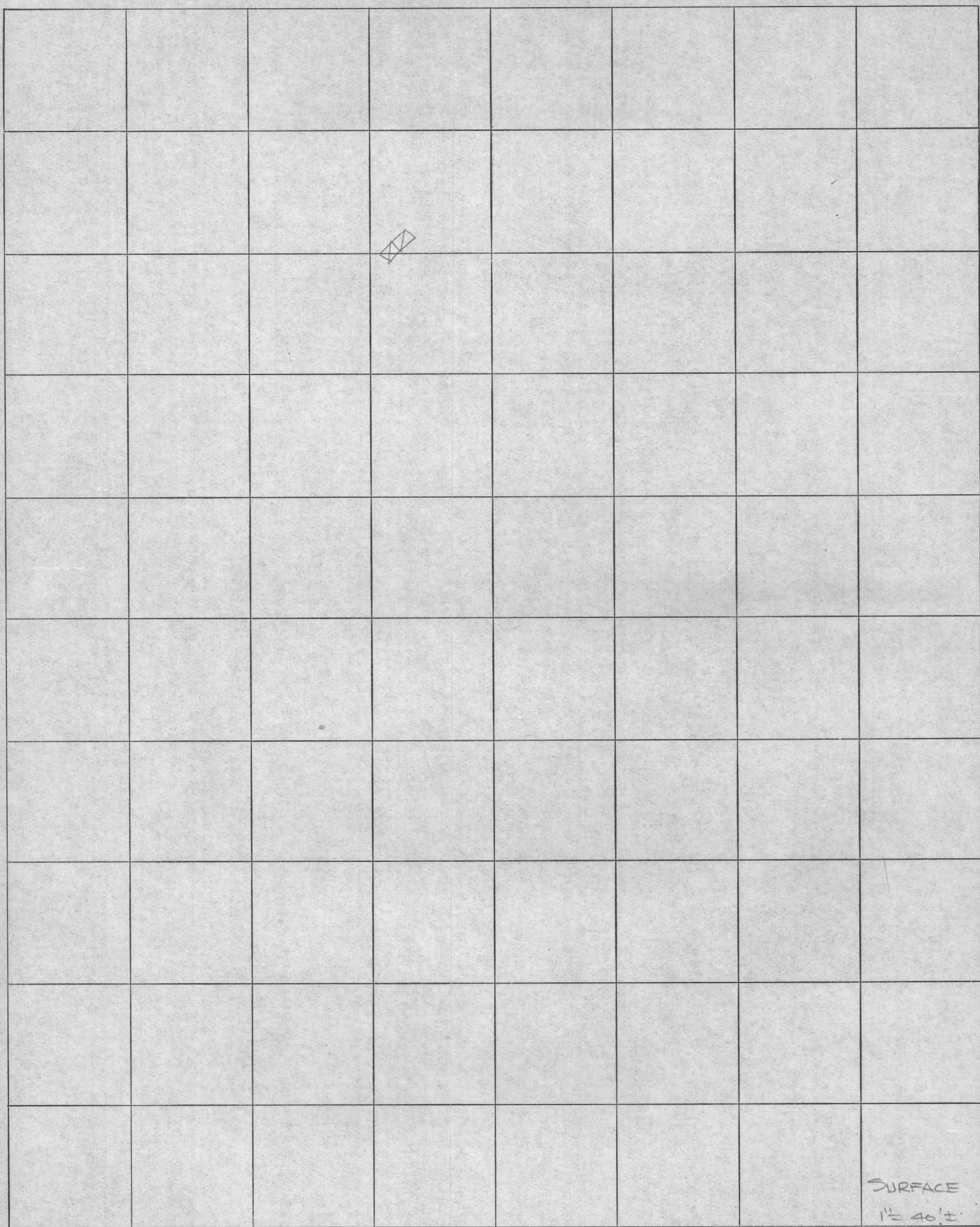
FORM 16 G

ADJUSTED - PRINT BADLY SHRUNK



FORM 16 G

FORM 16 G



FROM
1"=600' map
By F.M.Leonard Jr.