009611

3902

JULY 1946

UNITED STATES
DEPARTMENT OF THE INTERIOR
J. A. KRUG, SECRETARY

BUREAU OF MINES
R. R. SAYERS, DIRECTOR

REPORT OF INVESTIGATIONS

INVESTIGATION OF THE LOST RIVER TIN DEPOSIT
SEWARD PENINSULA, ALASKA

BY

H. E. HEIDE

c09613

REPORT OF INVESTIGATIONS

UNITED STATES DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

INVESTIGATION OF THE LOST RIVER TIN DEPOSIT SEWARD PENINSULA, ALASKA[1]

By H. E. Heide[2]

CONTENTS

INTRODUCTION

Before the war, the United States was the world's leading consumer of tin, the industrial applications of which had become widespread. Domestic production was negligible, and requirements were met almost entirely by imports, principally from British Malaya, United Kingdom, and China. Average annual imports for the years 1925-1929 amounted to 78,009 long tons.

With the advent of war in Europe and signing of the Japanese-German pact, it was realized that foreign sources of supplies were threatened, and steps were taken by Congress to forestall critical shortages of tin and other minerals. These steps in regard to tin included stock-piling, improving

1/ The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Report of Investigations 3902."

2/ Mining engineer, Bureau of Mines, Juneau, Alaska.

reclamation and conservation, increasing South American production, constructing a smelter in the United States for treatment of tin ores, and intensifying the search for domestic tin deposits. Annual imports were raised to 140,873 long tons by 1941. In 1943, annual imports had shrunk to 37,675 long tons and consumption had been reduced to 52,100 long tons of primary and secondary tin but not tin content of scrap alloy.

The Strategic Minerals Act, passed by the Congress in 1939, authorized the investigation of domestic tin deposits.

As Alaska had supplied virtually all of the domestic output of tin, an investigation of Alaska deposits was undertaken in 1940 and 1941 by J. B. Mertie, Jr., and Robert R. Coats of the Geological Survey.

The Lost River lode tin mine on the Seward Peninsula was examined by H. E. Heide of the Bureau of Mines in 1942. Possibilities for developing a significant amount of tin appeared favorable, and exploration of the ore bodies was started in August 1942.

Because of sub-arctic conditions, trenching and diamond drilling from the surface could only be carried on during the summer months, and the project was not completed until October 1944. The Federal Geological Survey maintained a field party on the project each season for the purpose of correlating geological data. Much of the geologic material used in this report has been freely abstracted from the reports of the Federal Geological Survey.

Feverish military activities on the Seward Peninsula handicapped Bureau of Mines operations, but the work was greatly facilitated by the information and cooperation obtained from residents and businessmen of Nome and Teller.

Robert L. Thorne, mining engineer of the Bureau of Mines, was in direct charge of exploration in 1944 and assisted greatly in the collecting and assembling of technical data.

Beneficiation tests of Lost River ores were made in the Bureau of Mines laboratories at Rolla, Missouri.

Acknowledgments

With respect to this report, special acknowledgment is due to J. B. Mertie, Jr., and to Robert R. Coats of the Federal Geological Survey for making available their unpublished report on geology; to Robert L. Thorne, Bureau of Mines, for his assistance in collecting and assembling data; to Robert S. Sanford, district engineer, Juneau, Alaska, and, to Lowell B. Moon, chief, Mining Branch, Washington, D. C., for their aid during project work and for revision of this report. Acknowledgment is also due to C. Travis Anderson, chief, Rolla Experiment Station, Bureau of Mines, Rolla, Mo., for conducting metallurgical studies.

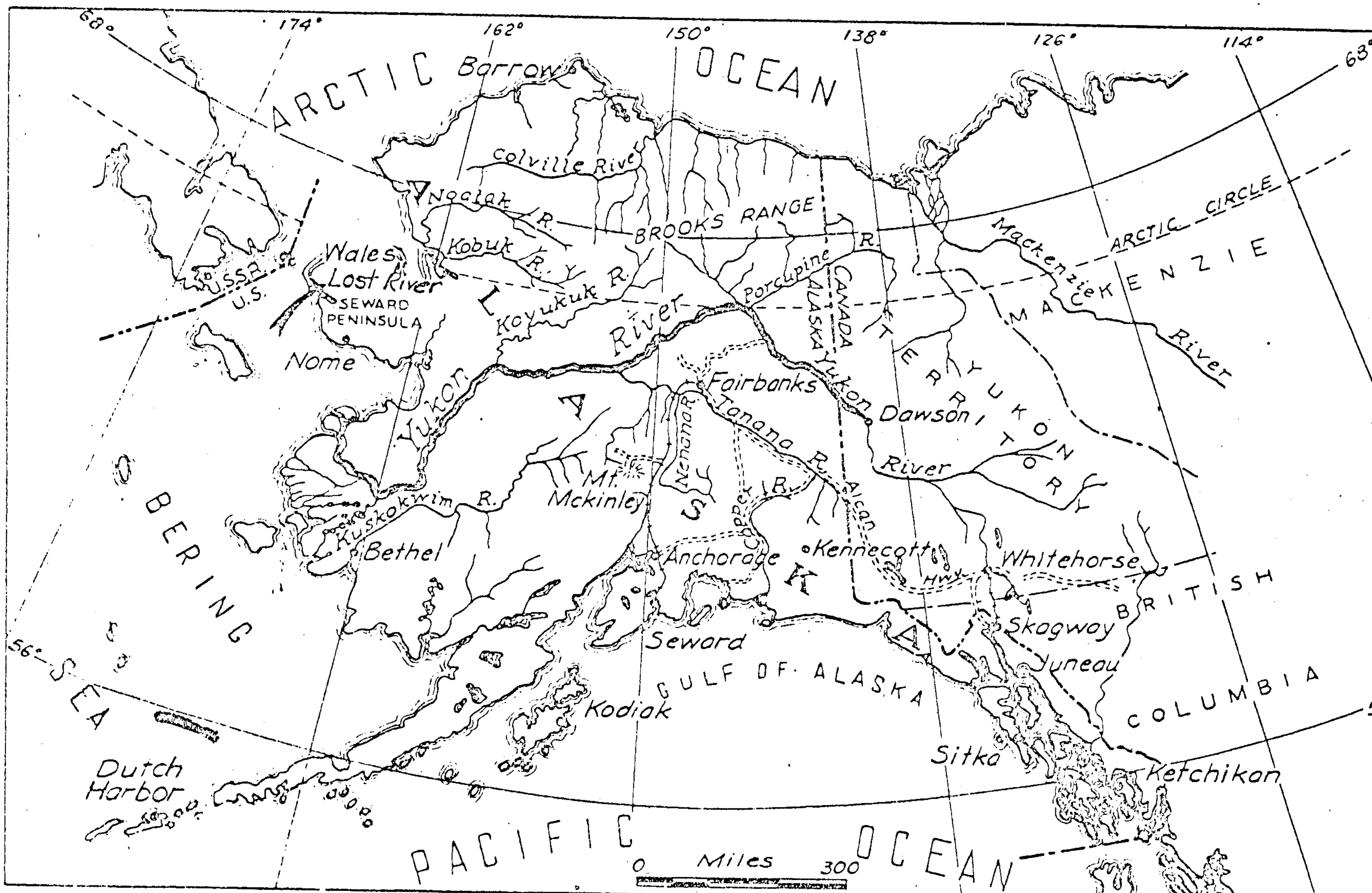


FIG. 1 INDEX MAP SHOWING LOCATION OF LOST RIVER AREA

LOCATION AND ACCESSIBILITY

The Lost River tin mine is on western Seward Peninsula, Alaska, at latitude 65° 28' N. and longitude 167° 8' E., about 90 miles northwest of Nome, which is the principal distributing center for the Seward Peninsula. The mine is approximately 6 miles inland from the Bering Sea, on Cassiterite Creek, a tributary of Lost River. It is situated in the Second Judicial Division and in the Port Clarence Mining Precinct. Mining records, formerly kept at Teller, Alaska, are now maintained at Nome (see fig. 1).

The property is isolated, and the commoner means of communication are entirely lacking. A natural landing field for planes exists on the east side of Lost River about 1.5 miles below the mine and near the mouth of Tin Creek. Another natural field at the beach, on the west side of Lost River, can be used if necessary. Both fields are only suitable for small planes but could be improved to land heavy loads. Weather conditions make flying an uncertain means of transportation. Plane fare from Nome to Lost River is $35 one way, and air express rates are 10 cents a pound for ordinary cargoes.

A coastwise steamer makes two or three trips between Seattle and Kotzebue during the season when the Bering Sea is open to navigation. The first boat leaves Seattle about the 20th of May and the last boat leaves in late August or early September. All boats are scheduled to depart from the Arctic Ocean and Bering Straits before October 6. Heavy equipment and supplies should be shipped by this route. There is no harbor at Lost River, but the vessels must anchor offshare and freight can be lightered ashore. A tug and lighter must be chartered and come from Teller for this purpose.

The average freight rate from Seattle is $30 a ton plus a 16-percent surcharge during the war. Lighterage rates are 55 percent of the freight rate. For small articles Eskimos with umiaks, or skin boats, can be hired to lighter at about $10 a ton.

Climatic conditions are severe, and as lightering operations can only be carried on in fair weather, demurrage may be high.

A small motorship carries mail and light cargoes from St. Michael to Kotzebue every two weeks during the open season. This boat will also tow a barge on request. Stops are made at Nome, Teller, and other wayside points. The freight rate from Nome to Lost River is $25 a ton, shore to shore.

In June 1945, the Alaska Steamship Co. announced postwar plans to enlarge its shipping facilities in these waters. If a large organization could guarantee return freight to Seattle, agreements could probably be made with the Steamship Co. for substantial reduction in freight rates.

The only harbor close to Lost River is at Teller, about 25 miles distant. Supplies now have to be lightered ashore at Teller, but the harbor has been surveyed and plans have been made by Army engineers to construct docks and dredge channels. It is believed that the plans have been suspended indefinitely, but any proposal to operate the Lost River mine should include a thorough investigation of the practicability of a port terminal at Teller. A good road for trucking could easily be constructed over favorable terrain from Teller to Lost River. The advantages of loading and discharging cargoes in a sheltered harbor would far outweigh the cost of road construction and additional haulage distances.

PHYSICAL FEATURES, CLIMATE, AND LABOR

The approach to the Lost River mine from Bering Sea is an area of generally low topographic relief. Lost River Valley is wide and open, and the surrounding hills are low and rounded. The river heads in the York Mountains, and the immediate vicinity of the deposits is more mountainous, with slopes of 1,000 feet at steep angles. Brooks Mountain, several miles north of the Lost River mine, rises to an elevation of nearly 3,000 feet and is the highest peak in the York Mountains.

The district is virtually barren of vegetation. Moss, grass, and occasionally scrub willows are found locally in valley floors, but the mountains are bare gray rocks, and no trees are found in the area. Timber for mining purposes would have to be shipped from southeastern Alaska or Seattle, with freight rates of $25 to $30 a thousand board feet.

Domesticated reindeer were at one time abundant in the Teller, Igloo, and Shishmaref areas, but the herds have been greatly depleted. There is no wild game or domesticated animal to supply meat for an extensive mining camp.

Climatic conditions are sub-arctic. No local records are available but conditions are best estimated from the records kept at Nome. Average annual precipitation at Nome is 17.82 inches. Over half of the rain falls during the four summer months, or from June to September, inclusive. Probably, precipitation is slightly greater at Lost River. The annual mean temperature is 25.7°. Winters are severe, and temperatures as low as -40° are recorded frequently. The winds blow continually in the Lost River Valley. They are very turbulent and frequently of high velocity.

Freezing stops the flow of water in Cassiterite Creek in winter. Lost River becomes covered with ice, but there is a perennial flow near its juncture with Rapid River. It is possible that water continues to flow under the ice or, perhaps, subsurface in Lost River in the vicinity of the mine. This might have an important bearing on the question of a winter supply of mill water and will be discussed later in the report under the subject of Metallurgy.

Nome, with a population of approximately 1,200, is the nearest source of supplies. Several trading stores are operated at Teller, but only a limited stock of goods is carried. Lode-mining equipment and supplies are not stocked by merchants and agents, because little such mining is done on the Seward Peninsula. In general, these supplies will have to be brought from Seattle.

Experienced labor, both underground and otherwise skilled, is virtually nonexistent in this part of Alaska, but it is possible that an adequate supply of unskilled Eskimo labor might be recruited. The Eskimos are fairly intelligent and industrious, and the skilled-labor problem could be solved by importing a skeleton crew of miners and mechanics with foremen to instruct and train the natives. In considering the Eskimos for underground labor, it should be mentioned that a high percentage of them are tubercular and only a small proportion might be acceptable for underground work.

HISTORY AND PRODUCTION

Cassiterite was discovered in this area in 1903 by Leslie Crim, Charles Randt, and W. J. O'Brien, and claims were staked along the Cassiterite dike on both sides of Cassiterite Creek.

In 1907 the Lost River Tin Mining Co. was organized and incorporated by the owners at Nome, Alaska, under the laws of the Territory of Alaska. The owners performed all assessment work and eventually obtained patents to the ground.

From 1904 to 1911, intermittent placer operations on Cassiterite Creek are said to have produced 20 tons of tin concentrates. Much of this production was from residual debris at the foot of the Greenstone lode.

The Jamme Syndicate of Seattle took a lease on the property in 1912 and started systematic development. A small pilot mill was erected and intermittently operated until 1915. About 8 tons of tin concentrates are reported to have been produced. In 1916 the lease was abandoned.

In 1918 the James F. Halpin interests took a lease on the property and continued development until 1920, when the lease was dropped. A complete examination and report on the Lost River mine was made by F. C. Fearing at this time. Some information given in the report by F. C. Fearing has been used in this present report.

The mine was idle until 1928, when the National Tin Mining Co., a Nevada corporation, took a lease on the property and started development. Operations were continued until 1930, when the lease was terminated.

Ownership of the Lost River mine now resides in the Lost River Tin Mining Co., which, in turn, is controlled by W. J. O'Brien and the Crim and Randt estates. The corporation was stricken from the files of the Territorial auditor on January 2, 1942, however, for nonpayment of the annual corporation tax, and the period allowed for reinstatement expired on January 2, 1944.

W. J. O'Brien resides in Seattle, Wash. Both Crim and Randt are deceased, and their estates are handled by the firm Ballinger, Hutson & Boldt, lawyers, whose offices are in the Hoge Building, Seattle, Wash. Originally, Crim and Randt each owned one-third of the issued stock, and their estates still control more than a majority of the issued stock.

Mineral Survey No. 1234 lists the following patented lode claims:

Engineer and Surveyor, on Camp Creek;
Shon Rue, Klondike, and Bald Eagle, on Greenstone lode;
Carry Gow, Three Prospectors, and Collier, on Cassiterite lode;
Mars, Jupiter, and Green, on the Ida Bell lode;
Rob Roy and Jenney Lyn, on the North Star lode;
Triangle and Lincoln, on Reaf lode.

The patent to above claims was issued on July 19, 1922, and is now of record in volume 99, page 64, of Miscellaneous Records of the Port Clarence Mining Precinct, Territory of Alaska.

ORE DEPOSITS

Since the discovery of tin in 1903, the Geological Survey has published several bulletins dealing with the area. Recently, J. B. Mertie, Jr., and Robert R. Coats studied the deposits and their unpublished report was made available to the Bureau of Mines. Although the report was written before a large granite mass had been discovered by Bureau drilling, the presence of granite was suspected, and their report includes a comprehensive description of the general geology.

Most of the following discussion of the general geology has therefore been abstracted or quoted directly from their report. A few additions and modifications have been caused by subsequent discoveries, but acknowledgment is due to the Geological Survey for most of the geological information.

General Geology of Lost River Area

The country rock consists mainly of Ordovician limestone, which has been intruded by two masses of granite and by numerous acidic and basic dikes of several ages. The older faults, which are later than the granitic intrusion, trend somewhat north or south of east. The dikes were later intruded along these faults, but the basic dikes are younger than the acidic dikes. Both the basic and acidic dikes are displaced by north-south faults, and one great fault of this system, which is followed by the valley of Lost River, displaces certain of them horizontally between 3,000 and 4,000 feet.

The tin ores are genetically related to the granitic rocks but are localized chiefly in and along the acidic dikes. Not all of these dikes, however, are mineralized, nor is any one dike mineralized throughout its entire length. Instead, the tin ores are localized in zones of intense contact metamorphism, which are caused by the presence of underlying cupolas or bosslike protuberances that project upward from a larger underlying mass of granite.

The principal workings of the Lost River mine are located on one of the acidic dikes known as Cassiterite Dike, which is mineralized for about 1,500 feet along its outcrop.

The beds of Ordovician limestone strike generally east-west and dip at moderate angles to the north. No general sequence of beds is apparent. Jointing is everywhere apparent. Three systems of faults have been recognized, of which two antedated the formation of the dikes and were the channels subsequently followed by the dikes. The general strikes of these earlier faults are N. 70° E. and N. 60° W. The third system of faults, striking somewhat west of north, postdated both the dikes and the tin mineralization. As no distinctive beds are present in the country rock, and the earlier faults were nonrotatory, their displacements are not generally apparent.

Granite occurs in two bosses, which outcrop about 3-1/2 miles apart. The larger of these, which forms the ridge south-southeast of Brooks Mountain, has an area of 1.1 square miles. The second boss lies in the headwater part of the valley of Tin Creek, west of the north fork of that stream, where it crops over an area of about 0.1 square mile. Tin minerals have been found in the two granite bosses and at other localities, but the Lost River mine is the principal site of tin ores in this area.

The Cassiterite dike begins along the west wall of the north fork of Tin Creek and extends N. 60° W. for 0.9 mile, then S. 85° W. for 0.45 mile. These major changes in direction probably occur at the intersections of faults. The Cassiterite dike intersects the Ida Bell dike near the top of the low spur between Cassiterite Creek and Lost River and, like the latter, it extends westward to or nearly to the Lost River fault. It has not been found west of the fault. In its medial part, between the two points where major changes occur in its strike, the Cassiterite dike dips steeply southward.

The Ida Bell is the most persistent of all the acidic dikes, but it is offset by numerous small faults and by the great fault that follows the valley of Lost River. As a source of tin it is of interest only in the vicinity of its intersection with the Cassiterite dike.

The acidic dikes are porphyritic rocks composed essentially of quartz, orthoclase, and albite in varying proportions. All three of these minerals occur as phenocrysts, but albite is not common in the groundmass. At the sites of tin mineralization, many secondary minerals also are present. These dikes are classified as rhyolite porphyry.

The two granitic bosses were the earliest of the igneous rocks to be intruded. During and after their intrusion, these bosses were faulted along lines ranging from N. 75° E. to N. 60° W. The acidic dikes were next intruded, following these lines of fracture. There next developed a system of north-south faulting, which ruptured both the bosses and the acidic dikes.

The basic dikes were then intruded, likewise, following mainly the older system of fault lines. After the intrusion of the basic dikes, the north-south faulting was renewed on a much larger scale.

The know ores of tin and tungsten are localized mainly along dikes and faults of the older system and were the earliest phases of the metallization. Most of these ores were formed before the emplacement of the basic dikes, and all of this metallization terminated before the last or most intense phase of the north-south faulting. Other types of mineralization, involving ores of lead, zinc, copper, iron, and certain low-grade ores of tin, were deposited not only along the acidic and basic dikes and older fault lines, but also to some extent along the north-south faults. All of the ores are considered to have come from cupolas or bosslike masses protruding upward from a larger mass of granitic rocks.

The report cited above was completed before exploration had been started by the Bureau of Mines, and new information gained during the exploration requires certain modifications of the original report.

Diamond drilling has confirmed the geologist's conclusion that the tin-bearing area is underlain by a granite protuberance. The most significant discovery is that the granite is strongly mineralized and may eventually become as important a source of tin as the acidic dikes. This granite apexes several hundred feet below the surface as a narrow ridge or spur trending roughly north-south. Drill holes around the boundary of the granite indicate that it slopes rather gently on its east flank and falls off abruptly on the west. The steep west slope is probably due to north-south faulting with considerable vertical displacement (see fig. 3, 29, 30, 31, and 32).

Among the fracture systems described in Geological Survey reports, only casual mention has been made of the east-west fractures, which apparently dip gently to the south. Many of these fractures are tin-bearing, and as they have been observed to cut the Ida Bell dike it is suggested that some of them may have caused local primary enrichment of the dikes. North of Cassiterite dike, and even north of Ida Bell dike, they appear to be little more then mineralized tension fractures dipping from about 10 to 30 degrees south.

South of Cassiterite dike, in the area overlying the granite, there is a flat-lying dike of fairly large proportions. (See fig. 4.) This dike outcrops in trenches in several places, and its exposures in old workings were described by Adolph Knopf and by Edward Steidtmann and S. H. Cathcart of the Geological Survey. According to their reports, the dike showed a slight dip south of 10 to 15 degrees. The dike has been encountered in Bureau of Mines drill holes 150 feet north and 200 feet west of the old workings. Its location in the drill holes indicates an apparent west or southwest dip, although this may be due to faulting.

Underlying the flat-dipping dike is a mineralized, brecciated zone over 100 feet thick, which has been intruded by numerous acidic and fewer basic dikes. The relationship of the breccia to the flat-lying dike is not known,

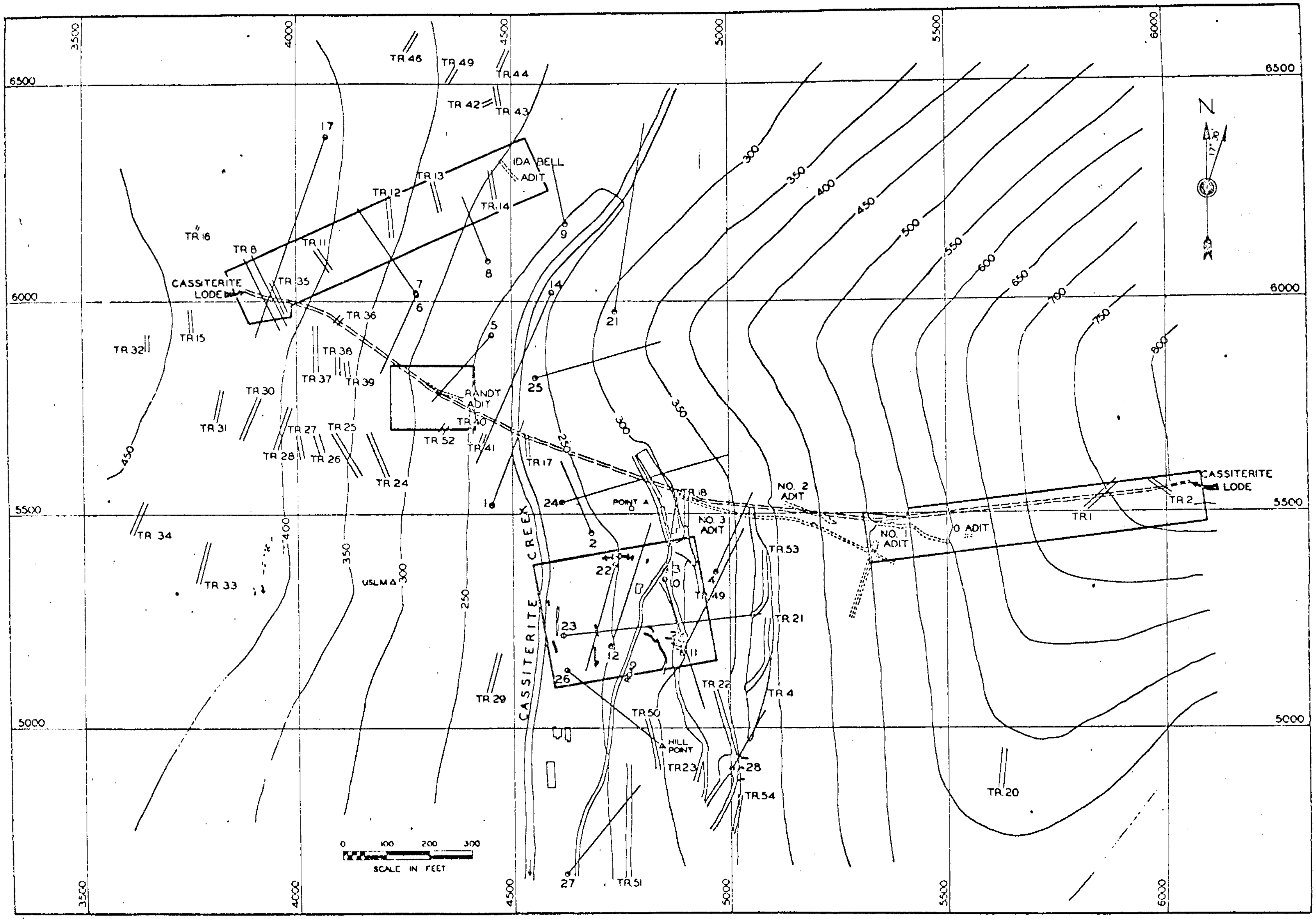


FIG. 2 GENERAL PLAN AND INDEX MAP

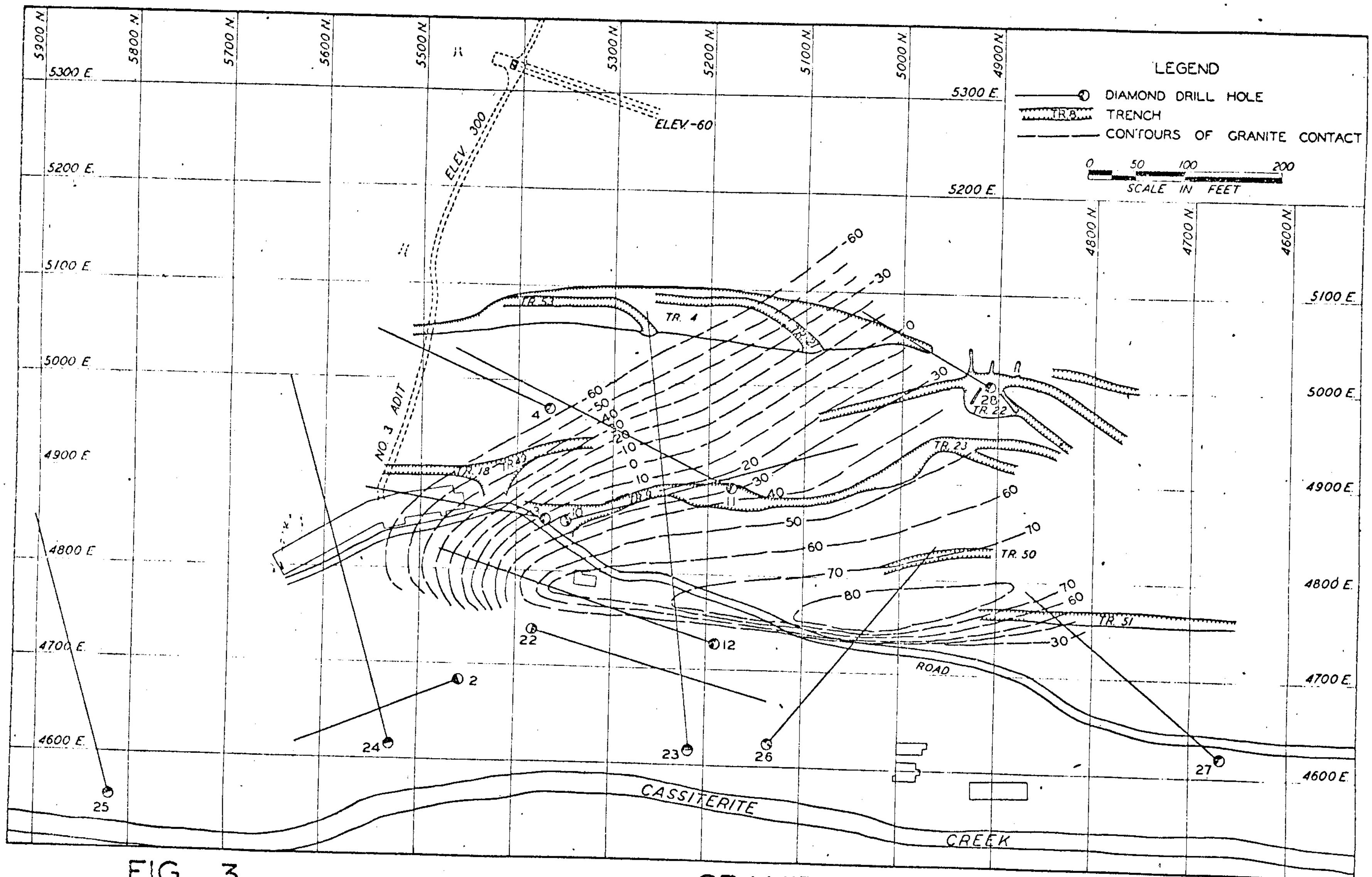


FIG. 3

GRANITE CONTACT AREA

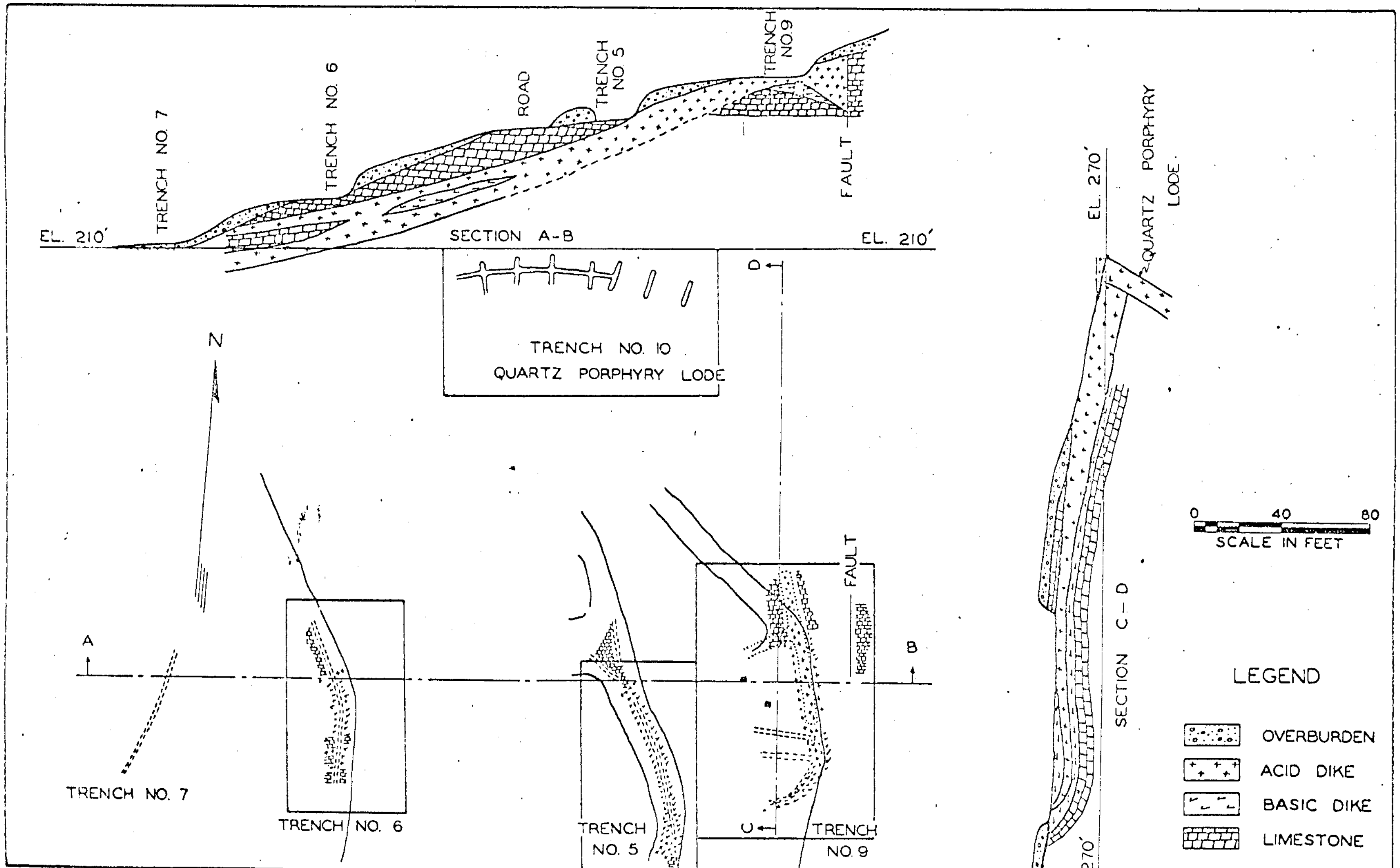


FIG. 4 SURFACE GEOLOGY — GREENSTONE AND QUARTZ PORPHYRY LODES

009637

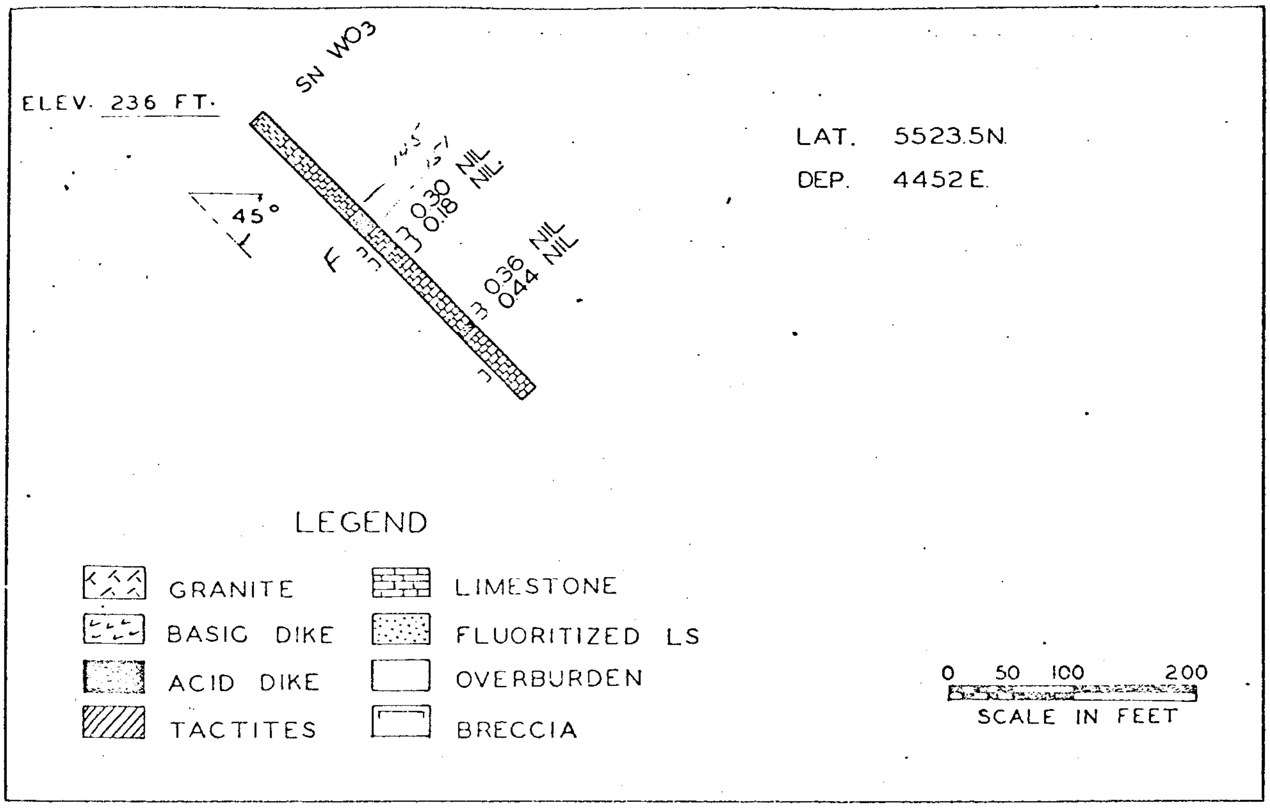


FIG. 5 GEOLOGIC SECTION D. D. HOLE NO. 1

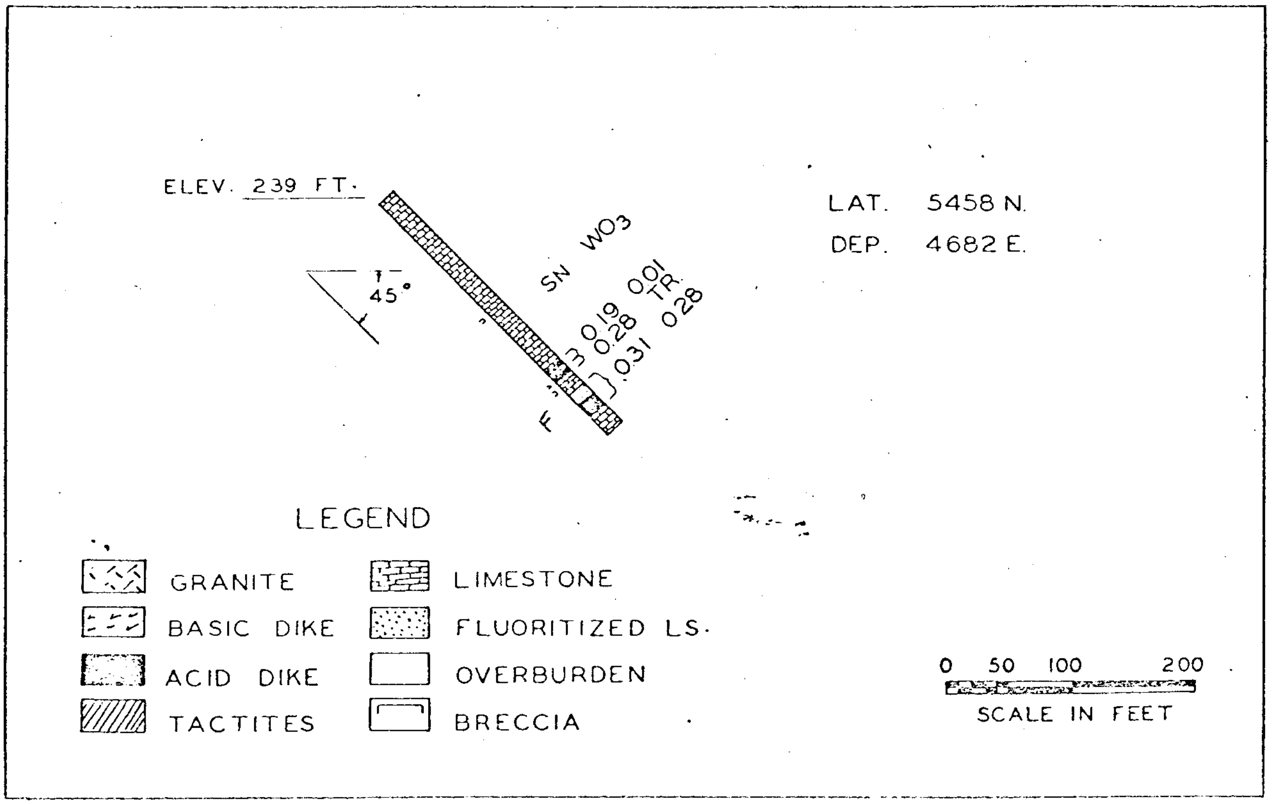


FIG. 6 GEOLOGIC SECTION D. D. HOLE NO. 2

069639

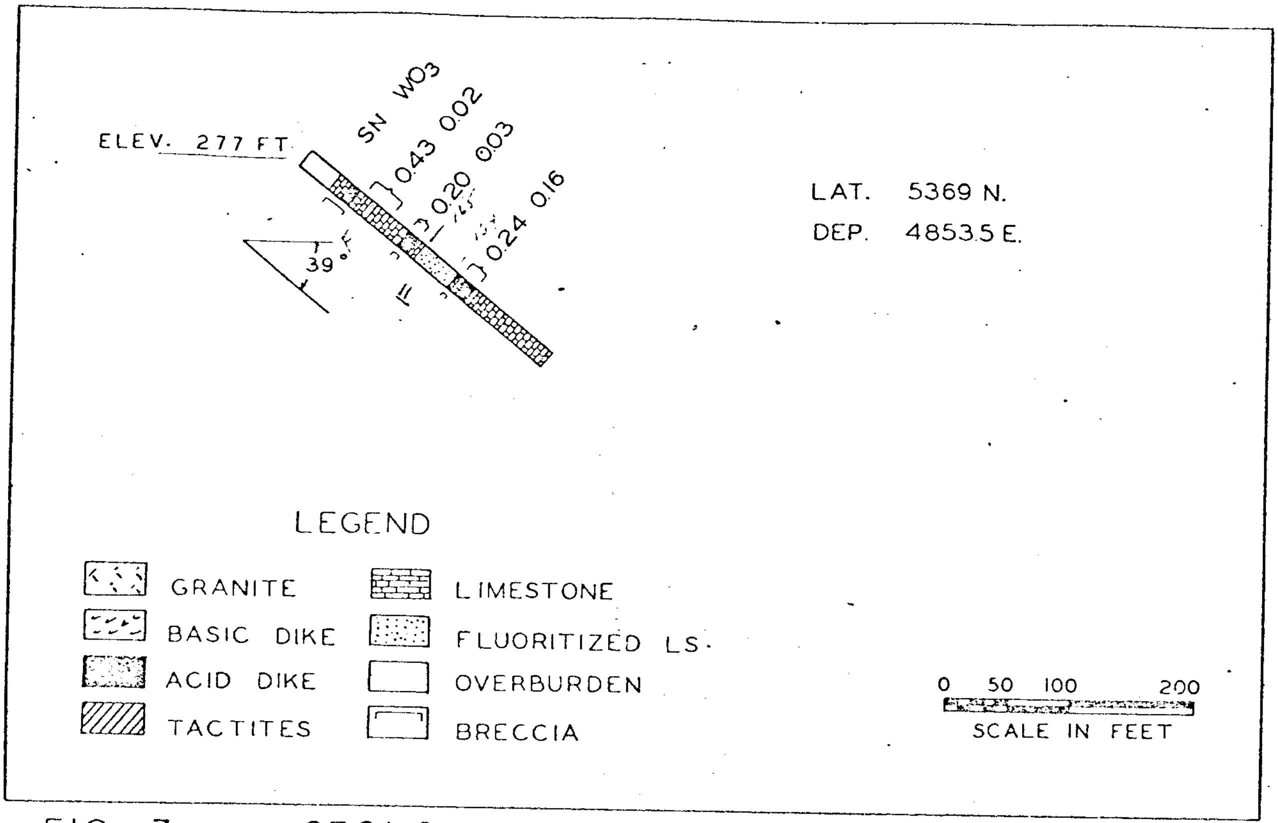


FIG. 7 — GEOLOGIC SECTION D. D. HOLE NO. 3

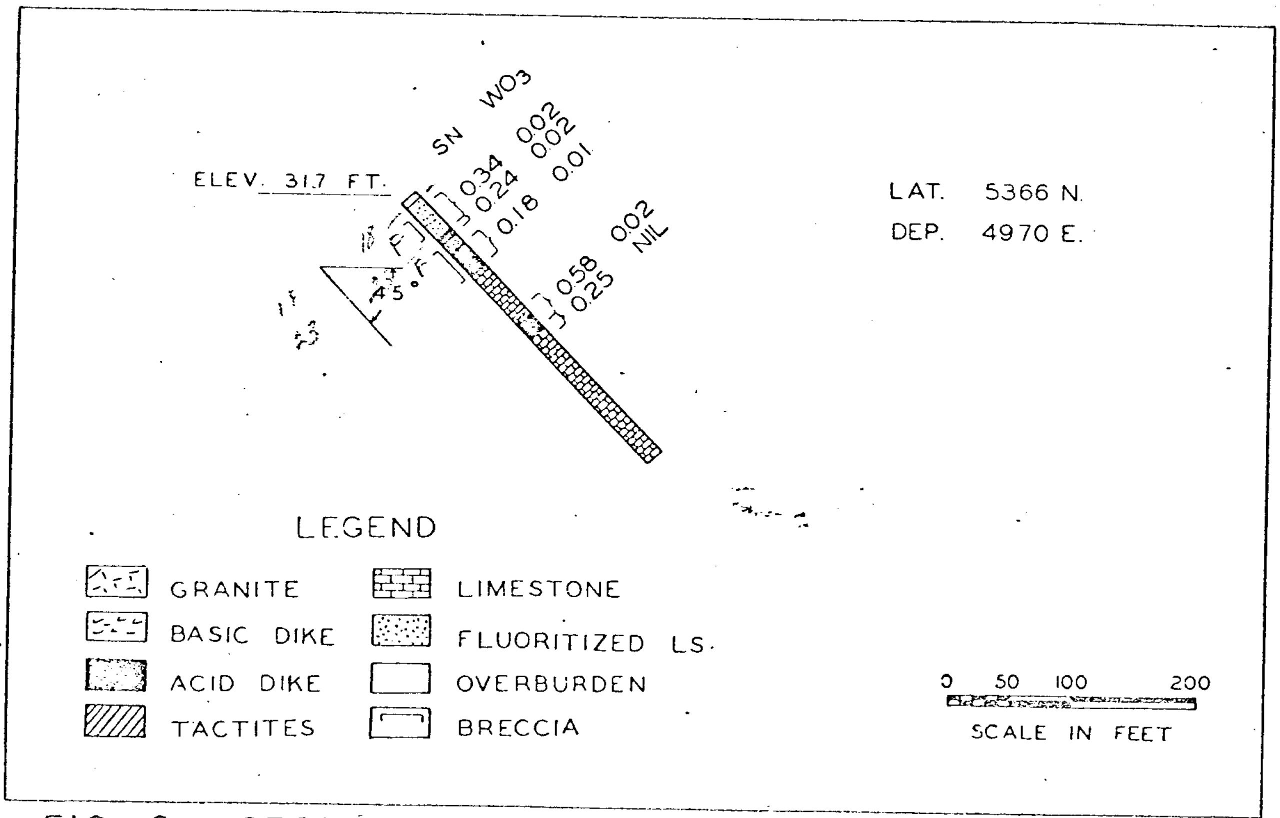


FIG. 8 GEOLOGIC SECTION D. D. HOLE NO. 4

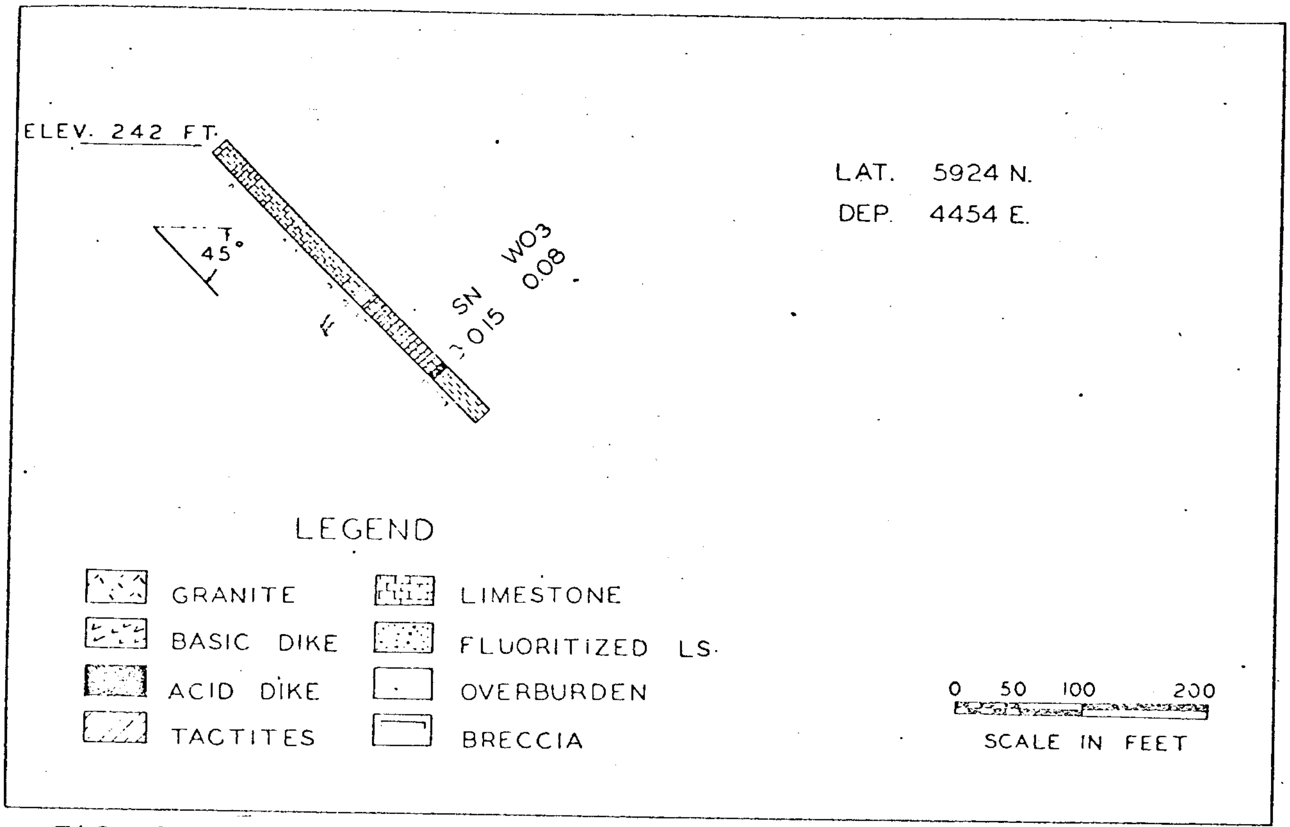


FIG. 9 GEOLOGIC SECTION D.D. HOLE NO. 5

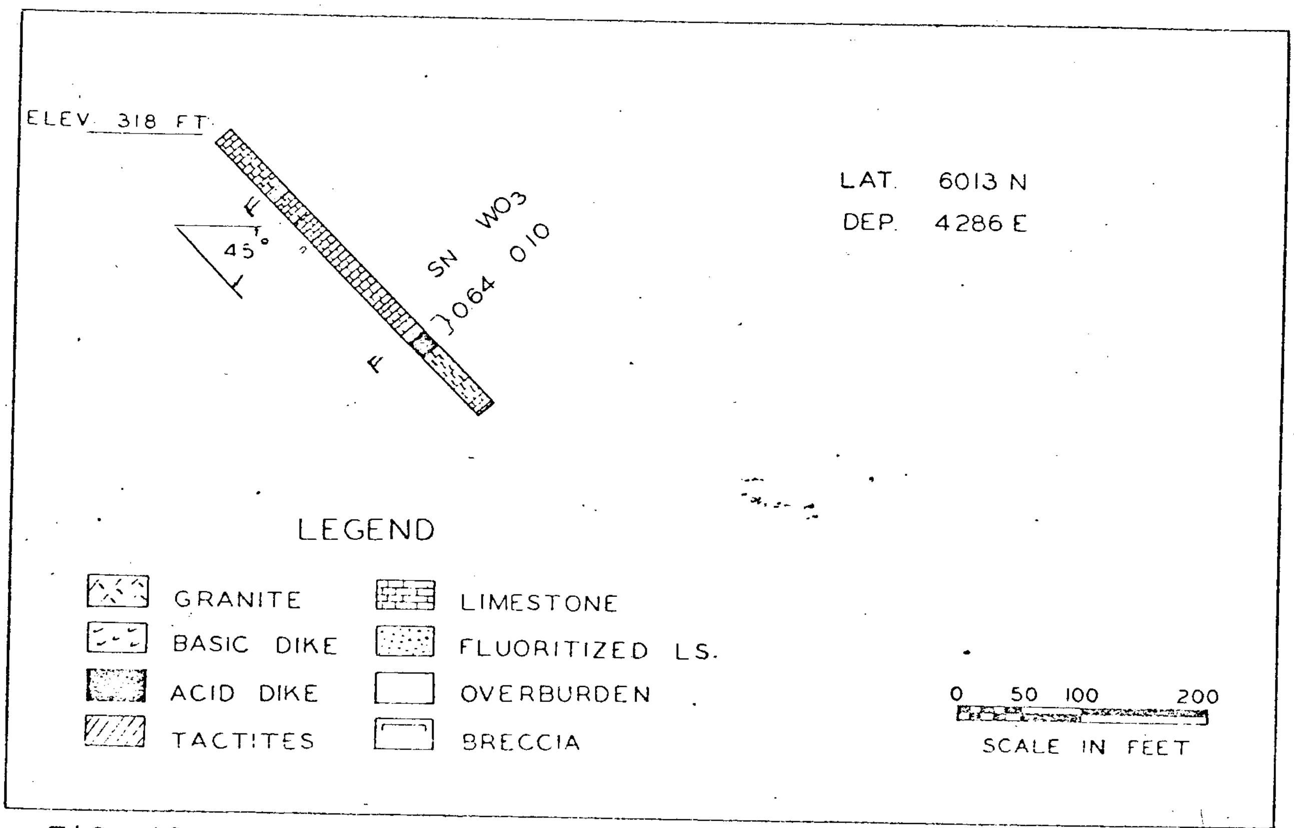


FIG. 10 GEOLOGIC SECTION D.D. HOLE NO. 6

009643

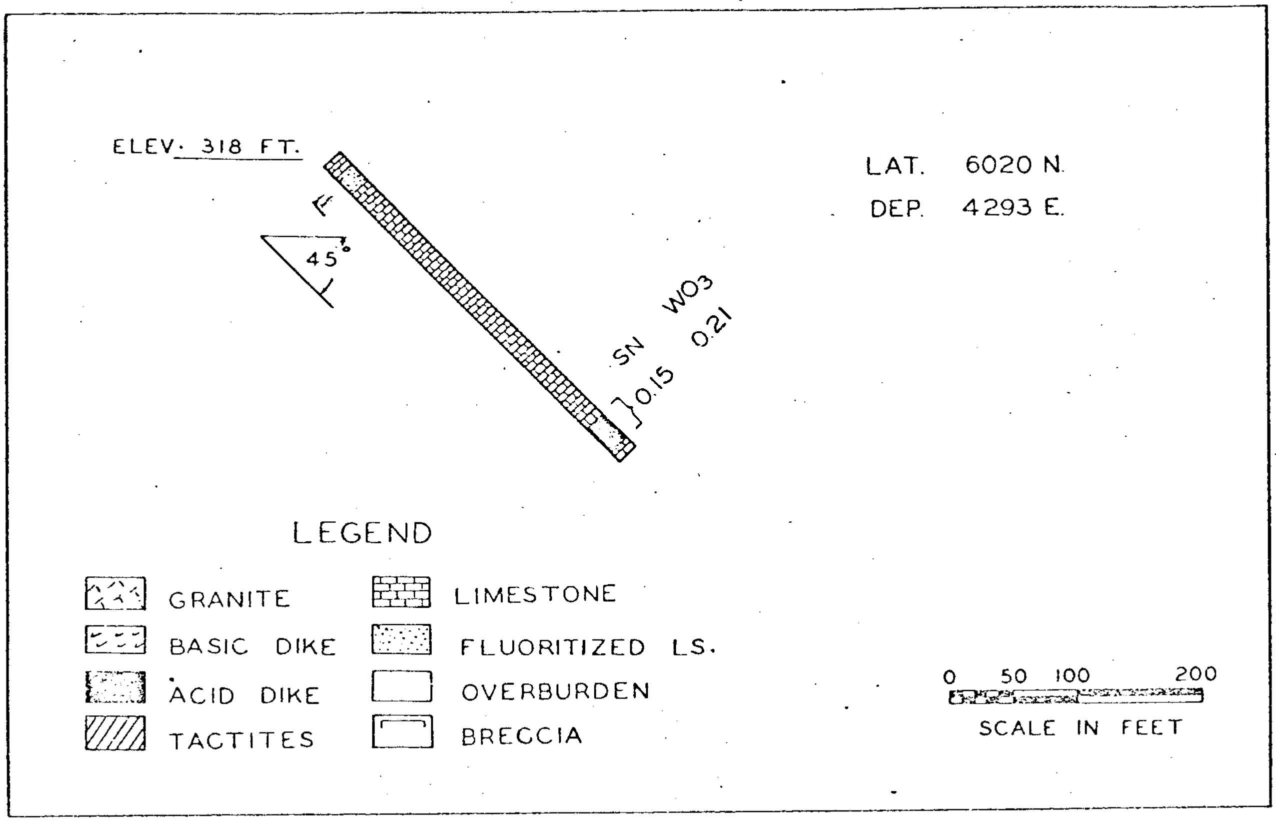


FIG. 11 GEOLOGIC SECTION D.D. HOLE NO. 7

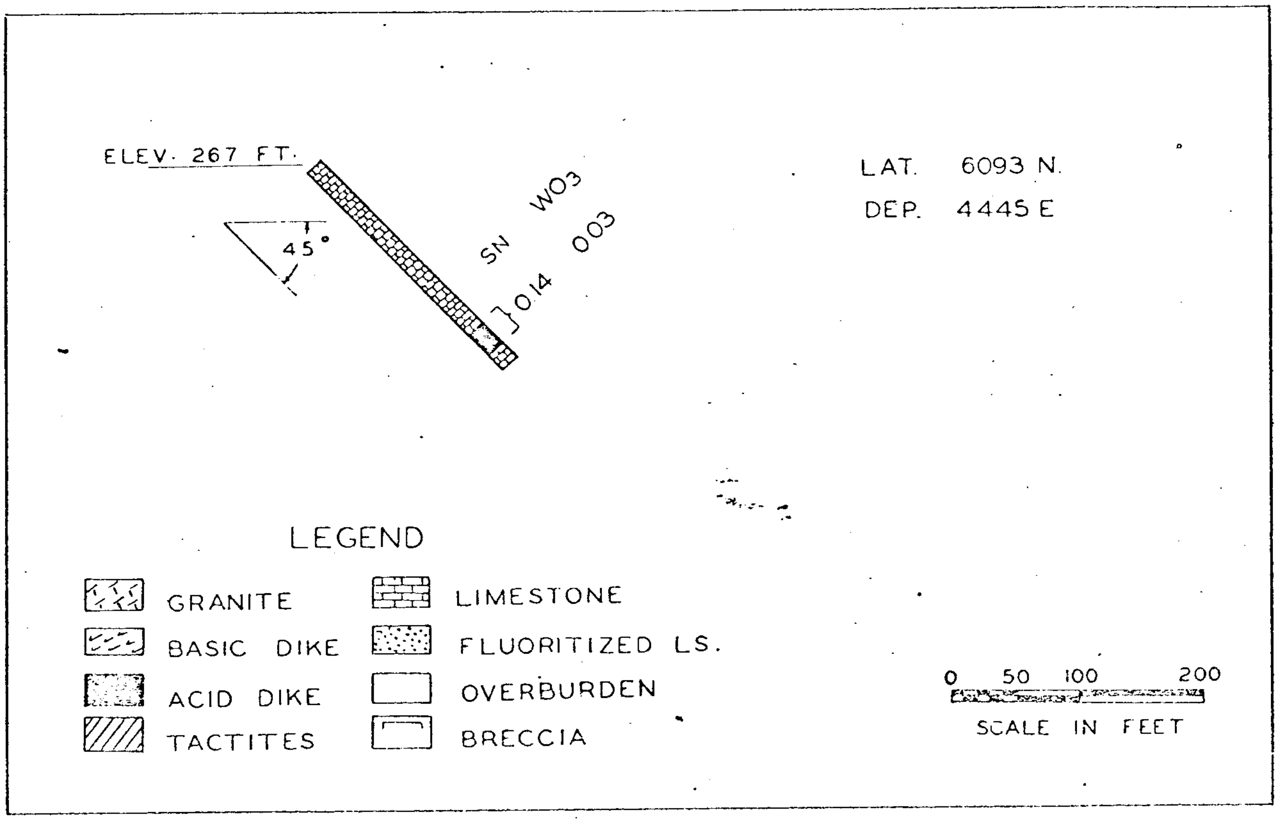


FIG. 12 GEOLOGIC SECTION D.D. HOLE NO. 8

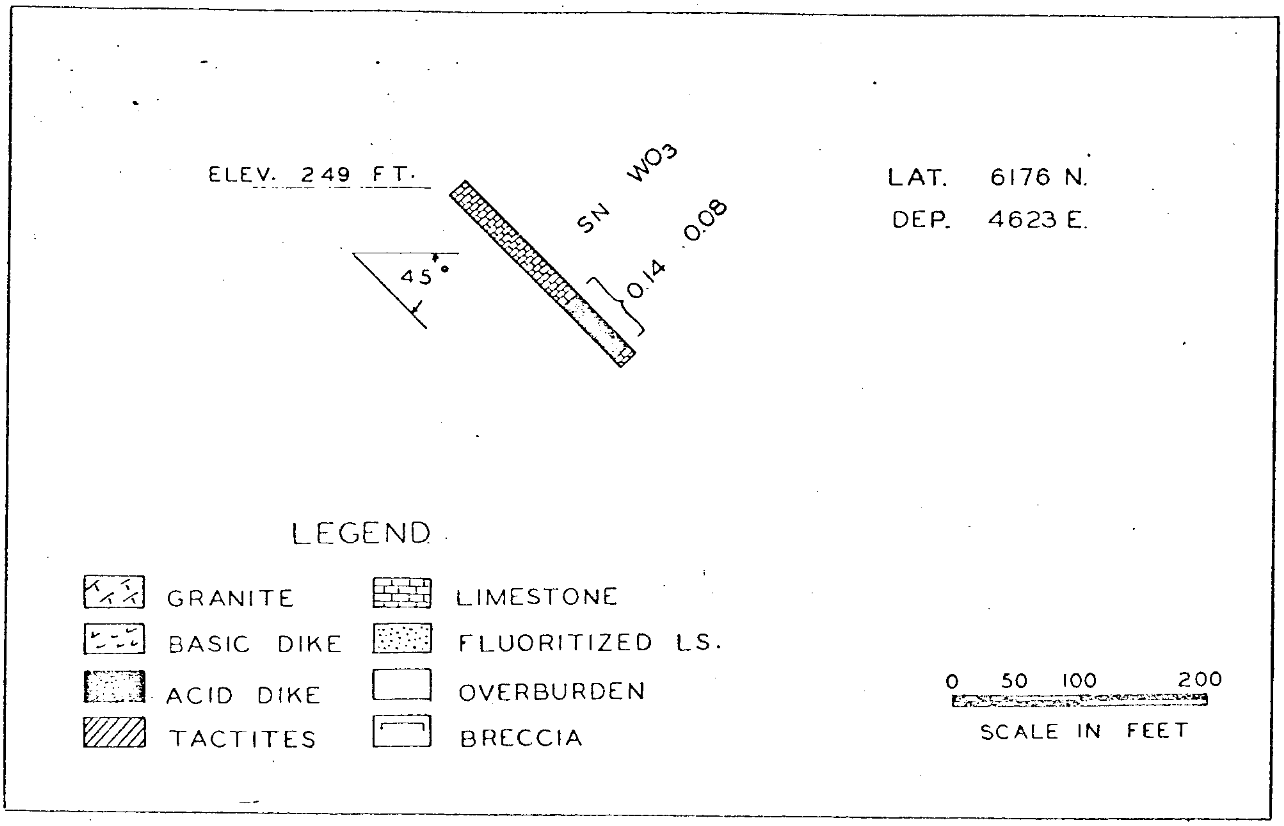


FIG. 13 GEOLOGIC SECTION D.D. HOLE NO. 9

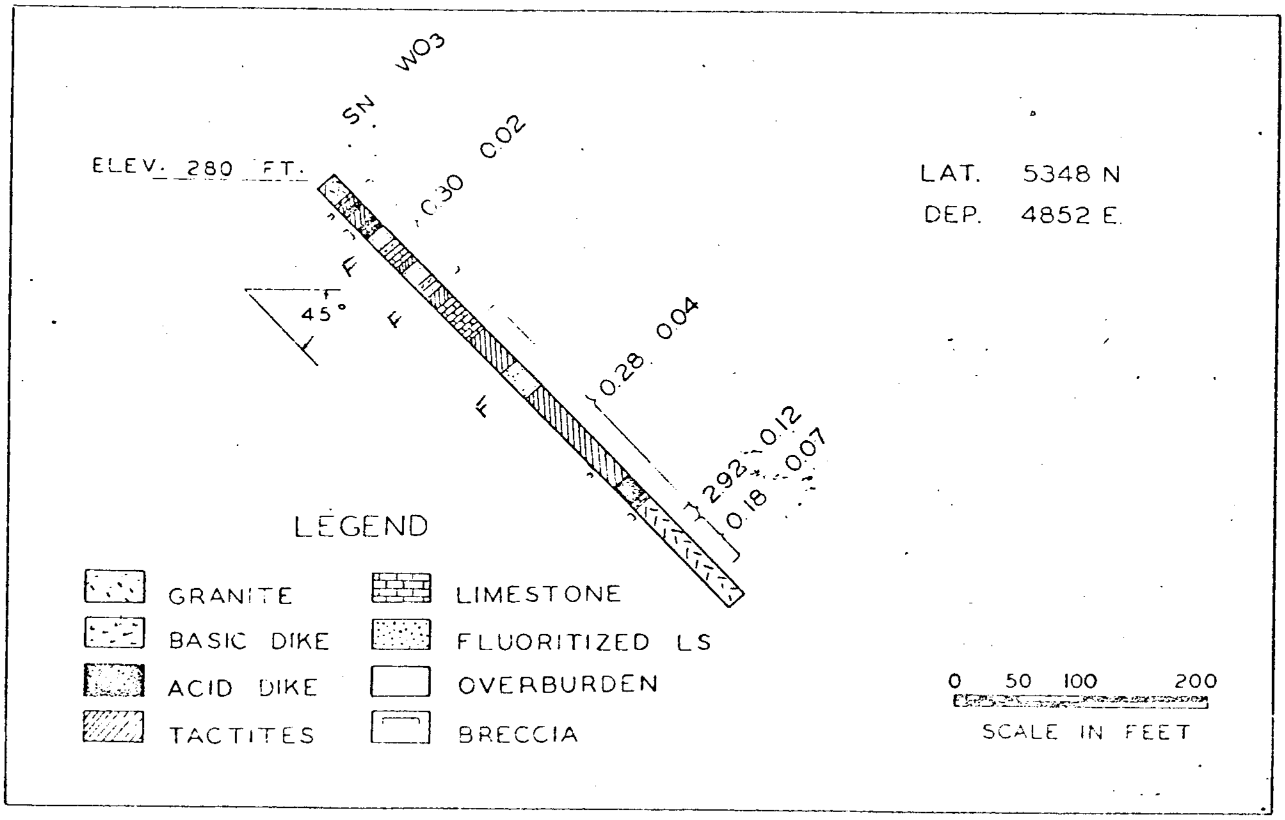


FIG. 14 GEOLOGIC SECTION D. D. HOLE NO. 10

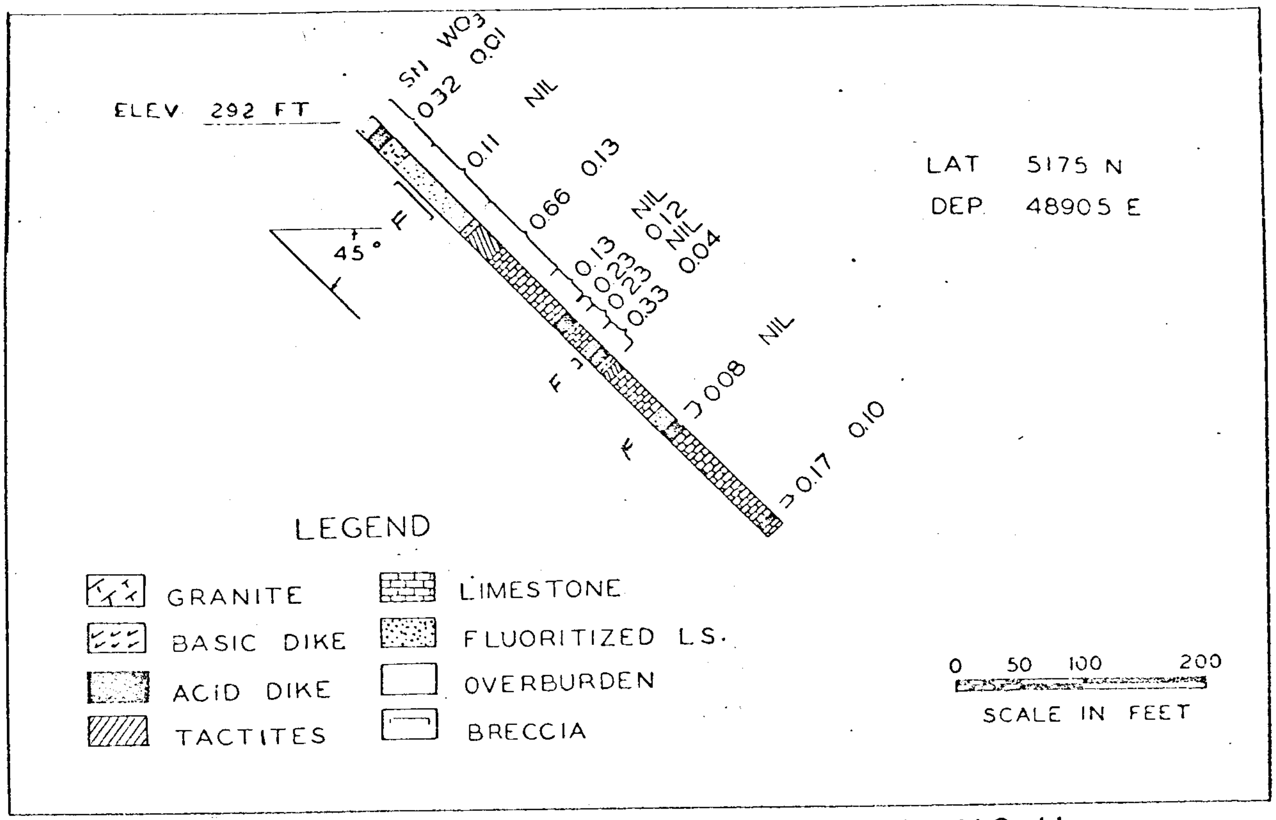


FIG. 15 GEOLOGIC SECTION D.D. HOLE NO. 11

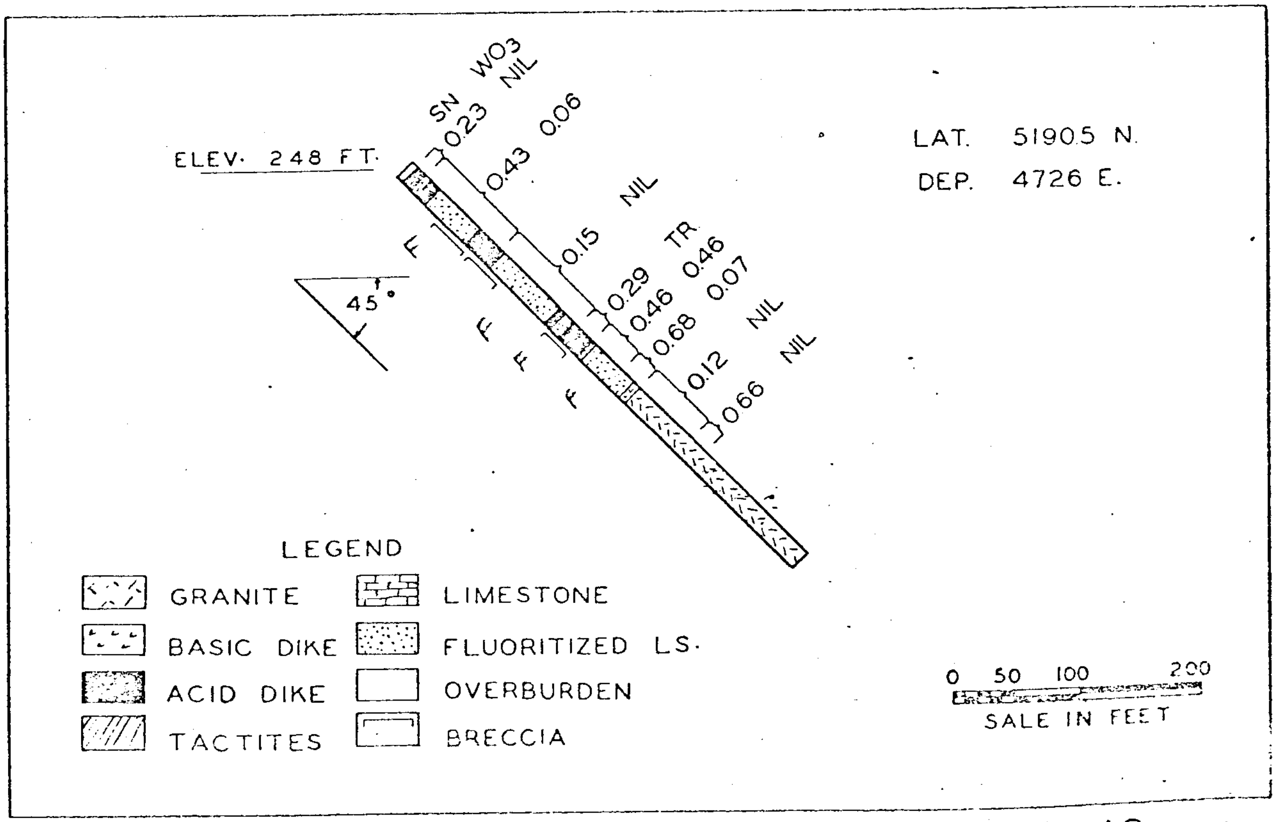


FIG. 16 GEOLOGIC SECTION D.D. HOLE NO. 12

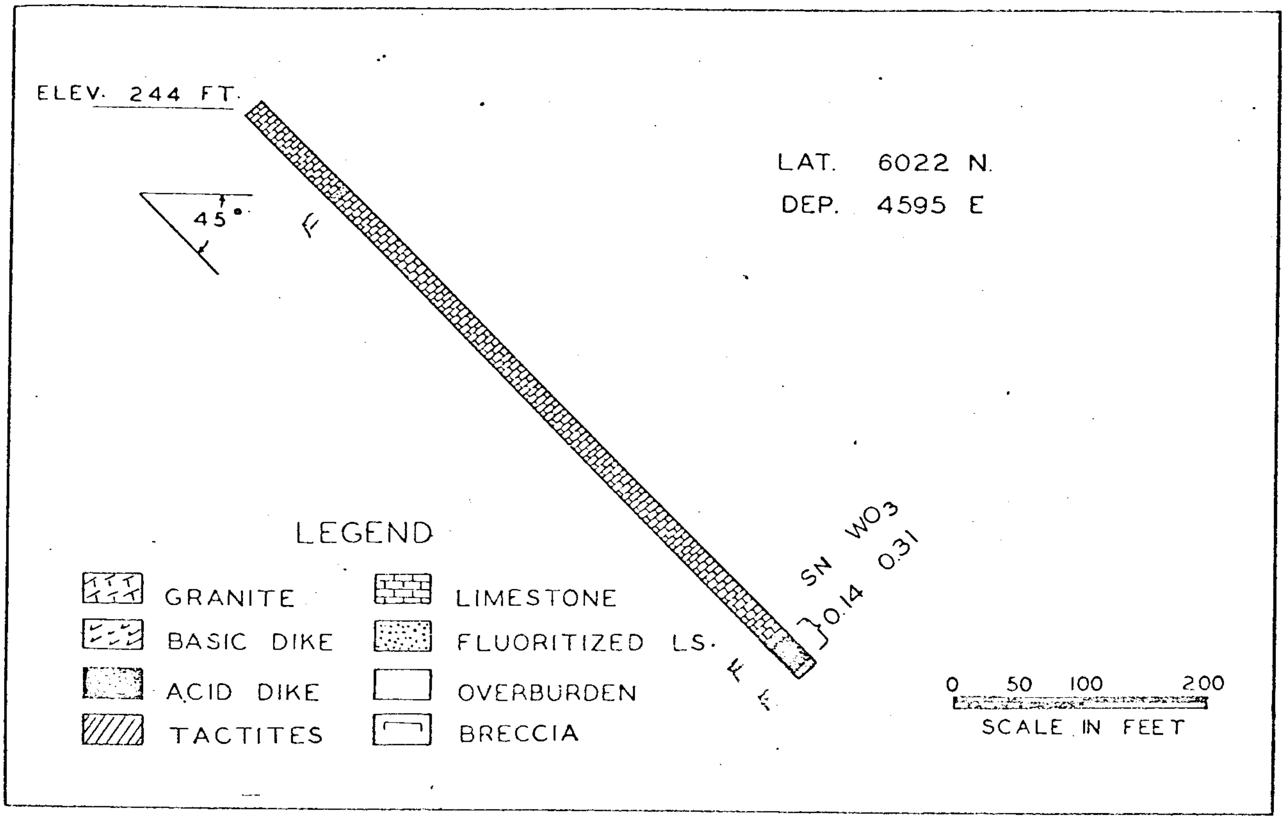


FIG. 17 GEOLOGIC SECTION D.D. HOLE NO. 14

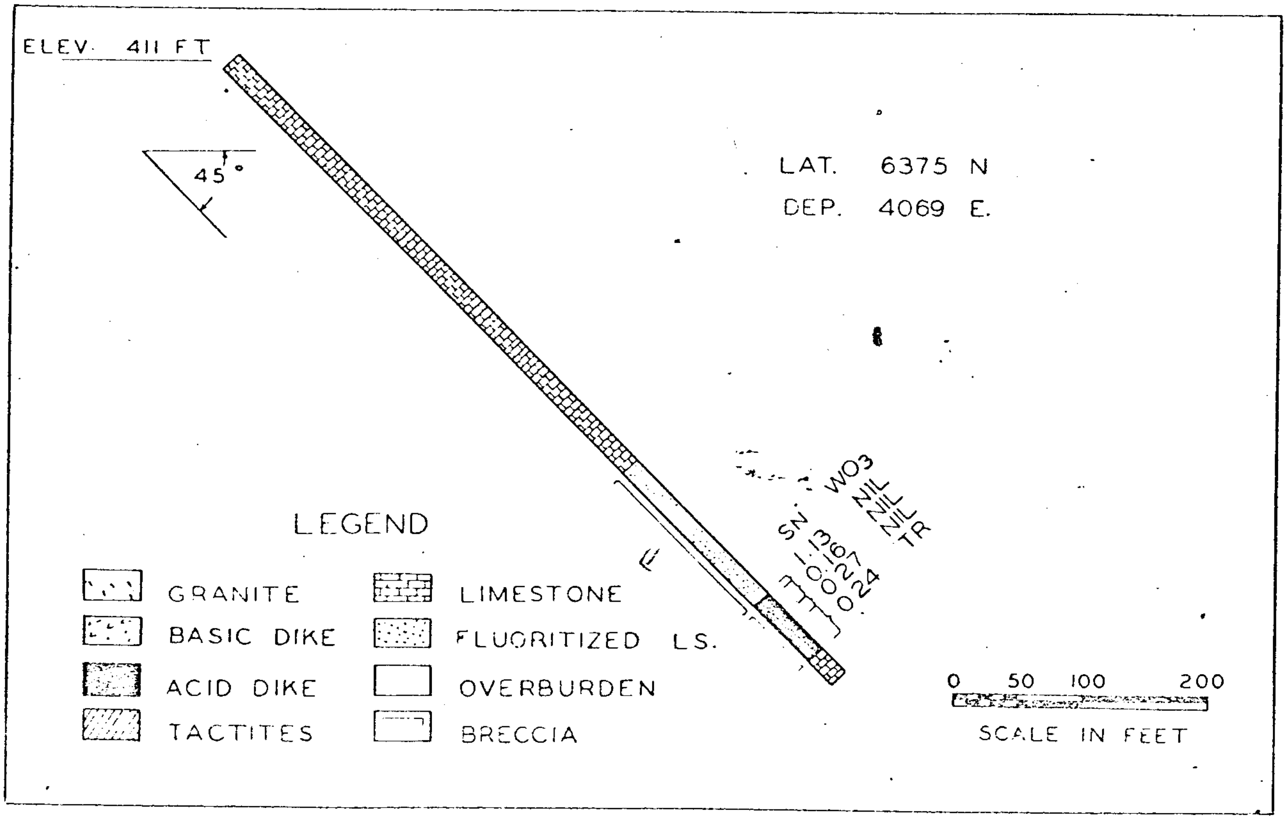


FIG. 18 GEOLOGIC SECTION D.D. HOLE NO. 17

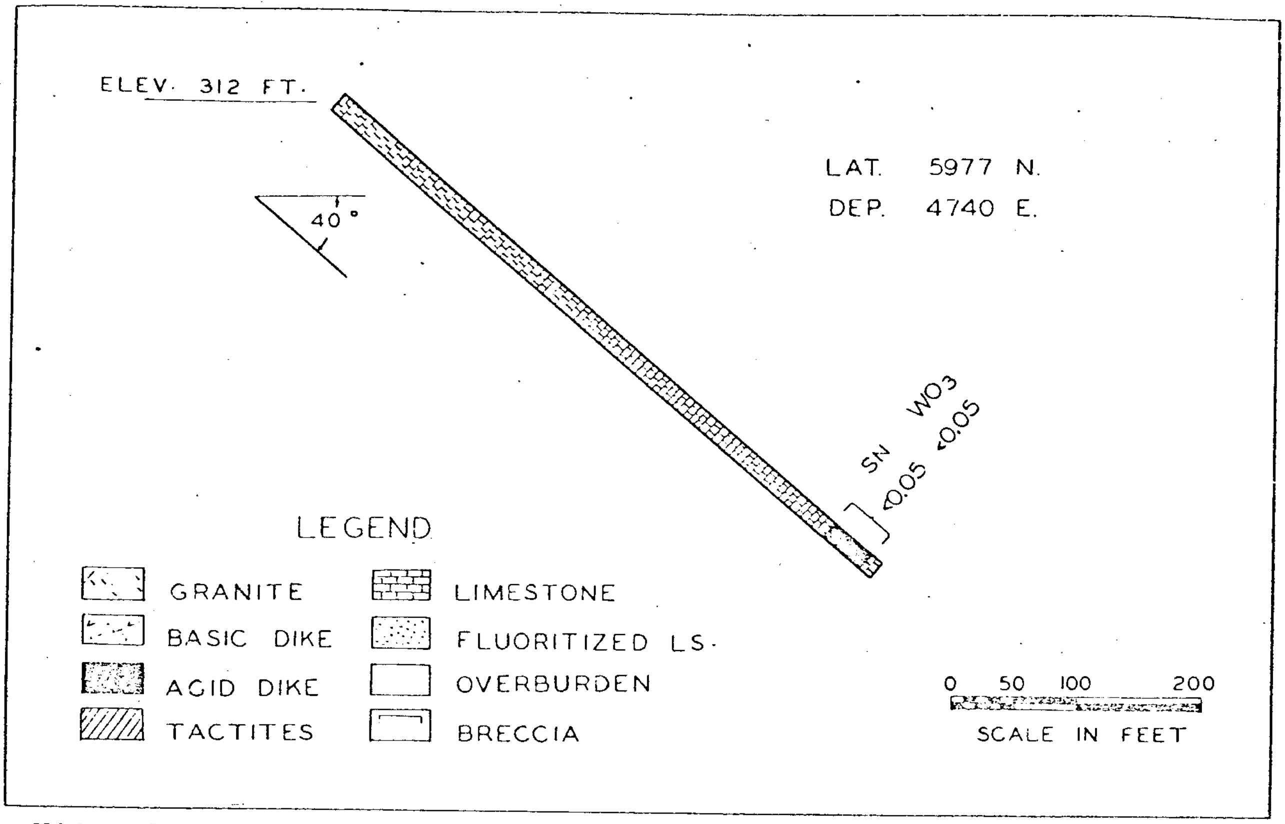


FIG. 19 GEOLOGIC SECTION D. D. HOLE NO. 21

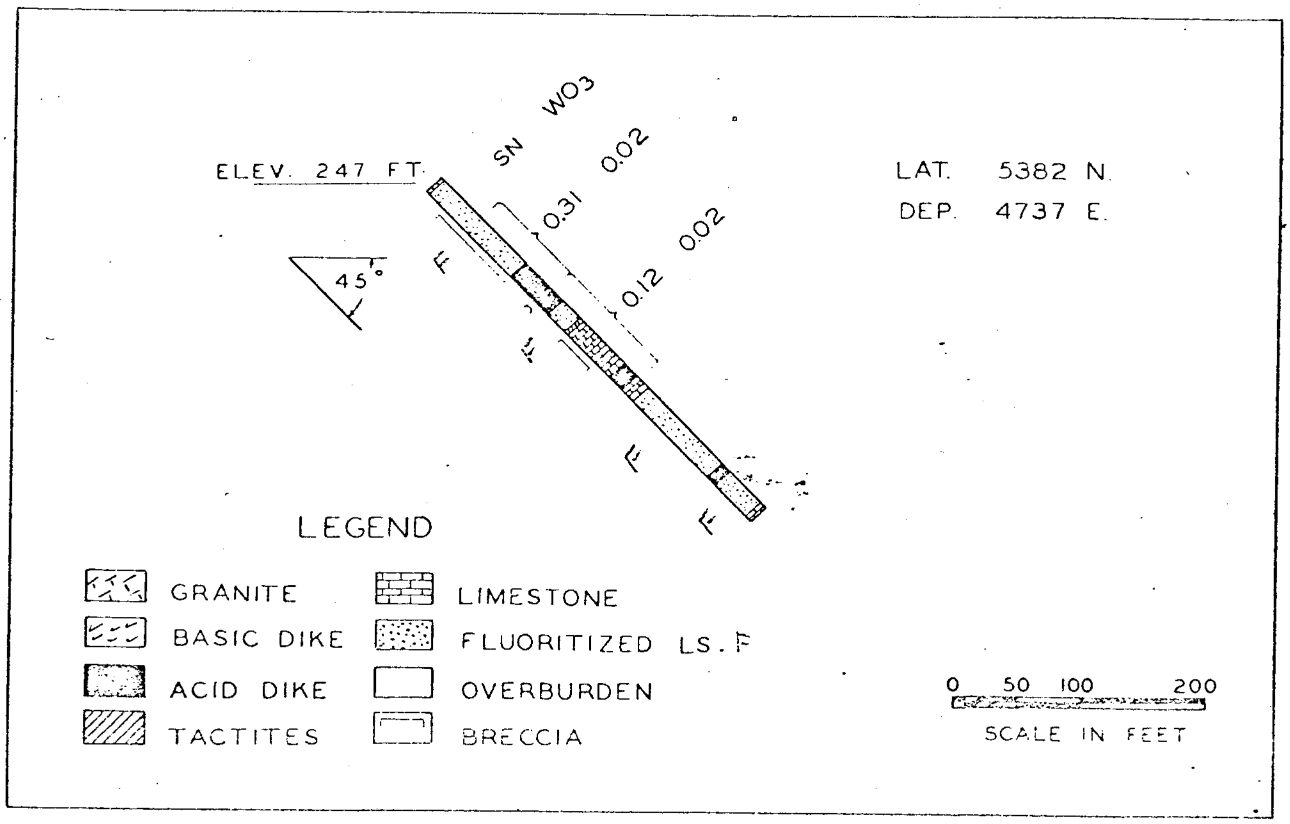


FIG. 20 GEOLOGIC SECTION D.D. HOLE NO. 22

009653

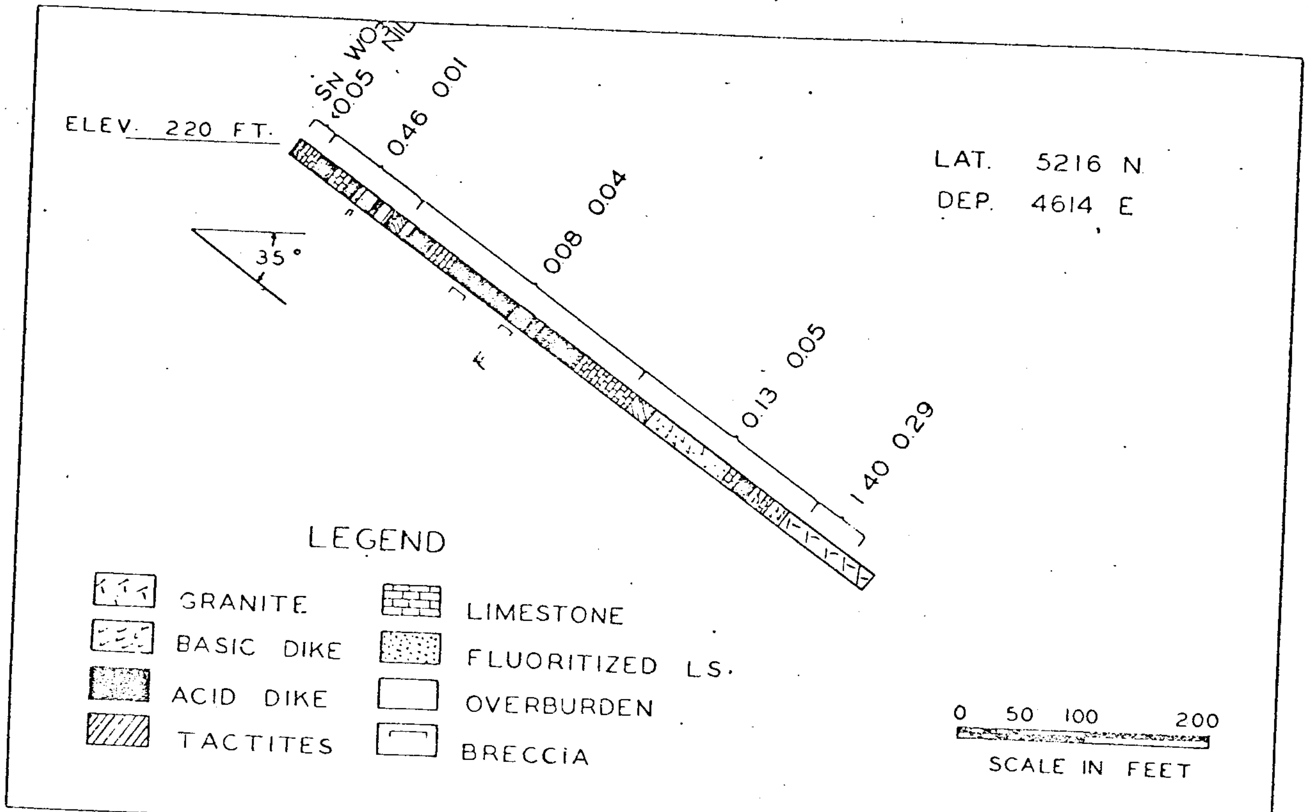


FIG. 21 GEOLOGIC SECTION D.D. HOLE NO. 23

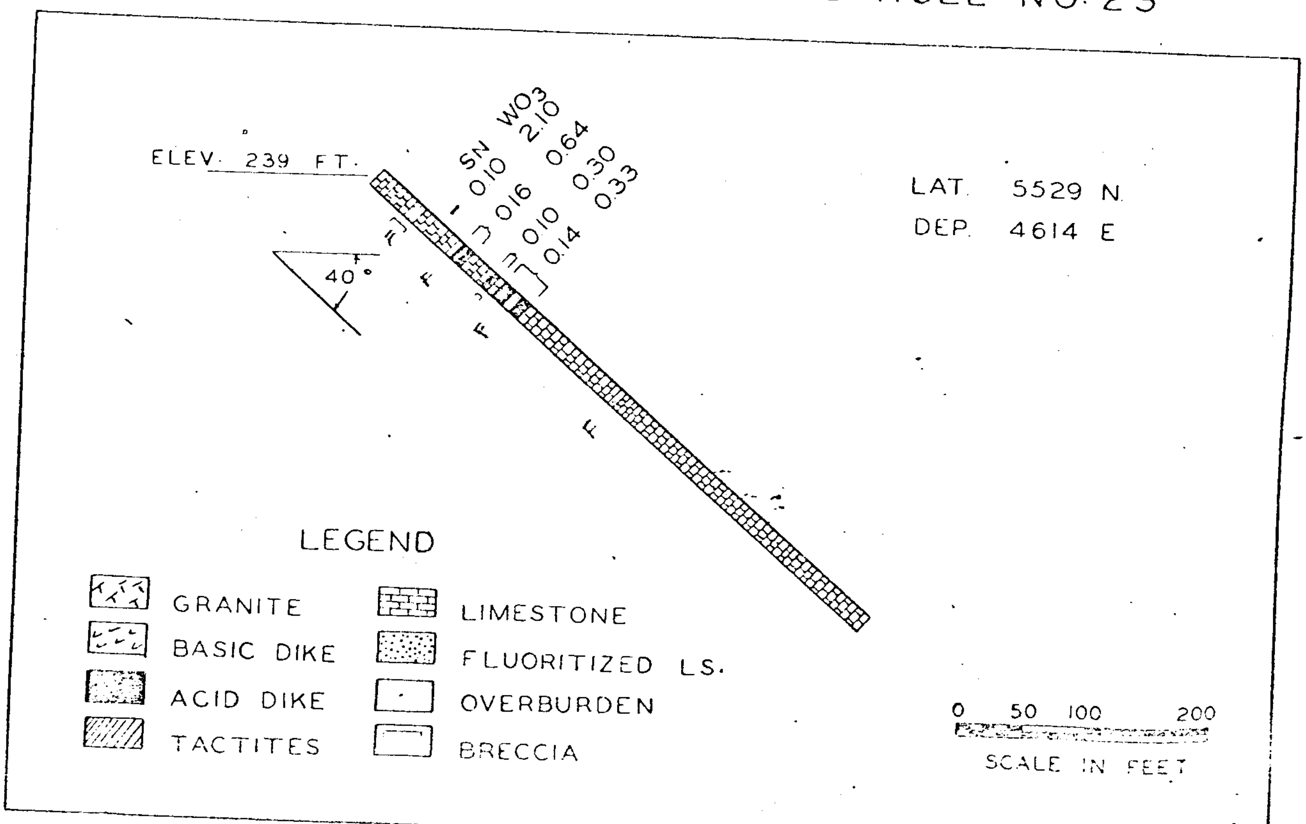


FIG. 22 GEOLOGIC SECTION D.D. HOLE NO. 24

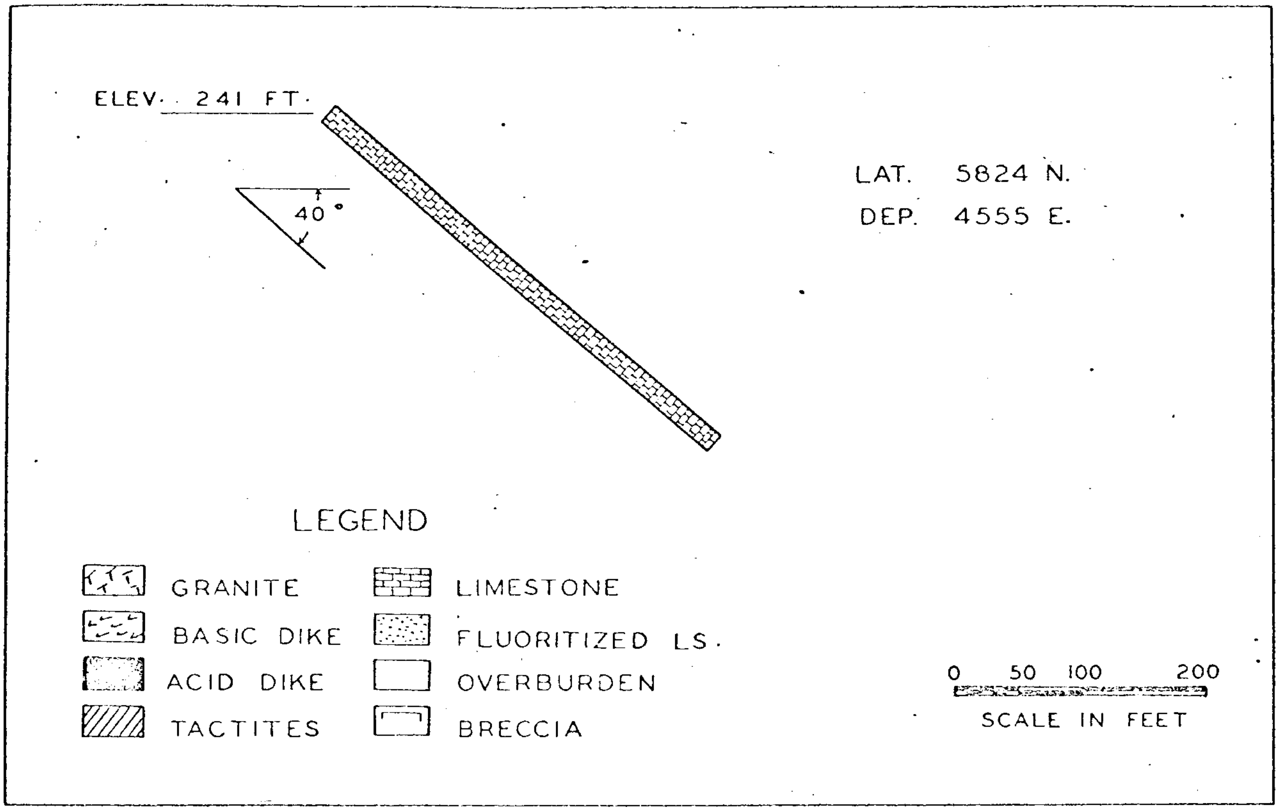


FIG. 23 GEOLOGIC SECTION D. D. HOLE NO. 25

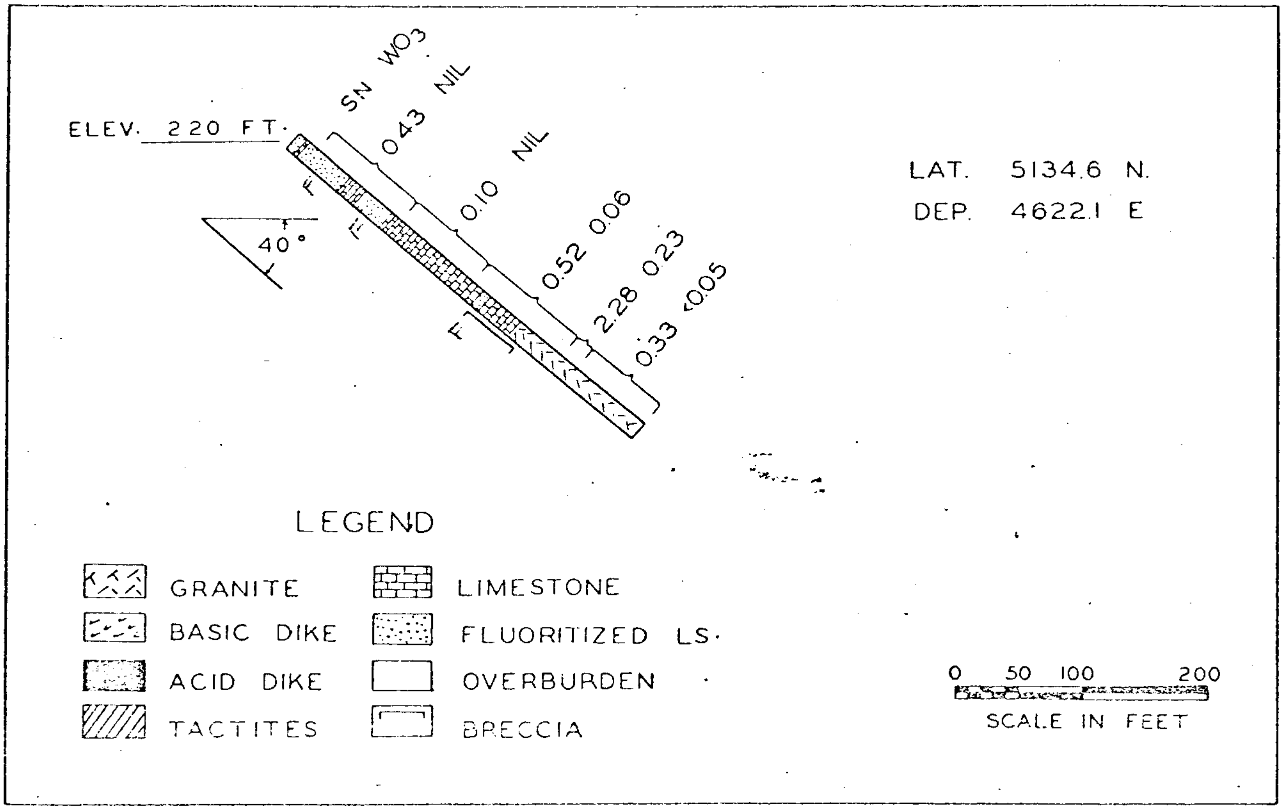


FIG. 24 GEOLOGIC SECTION D. D. HOLE NO. 26

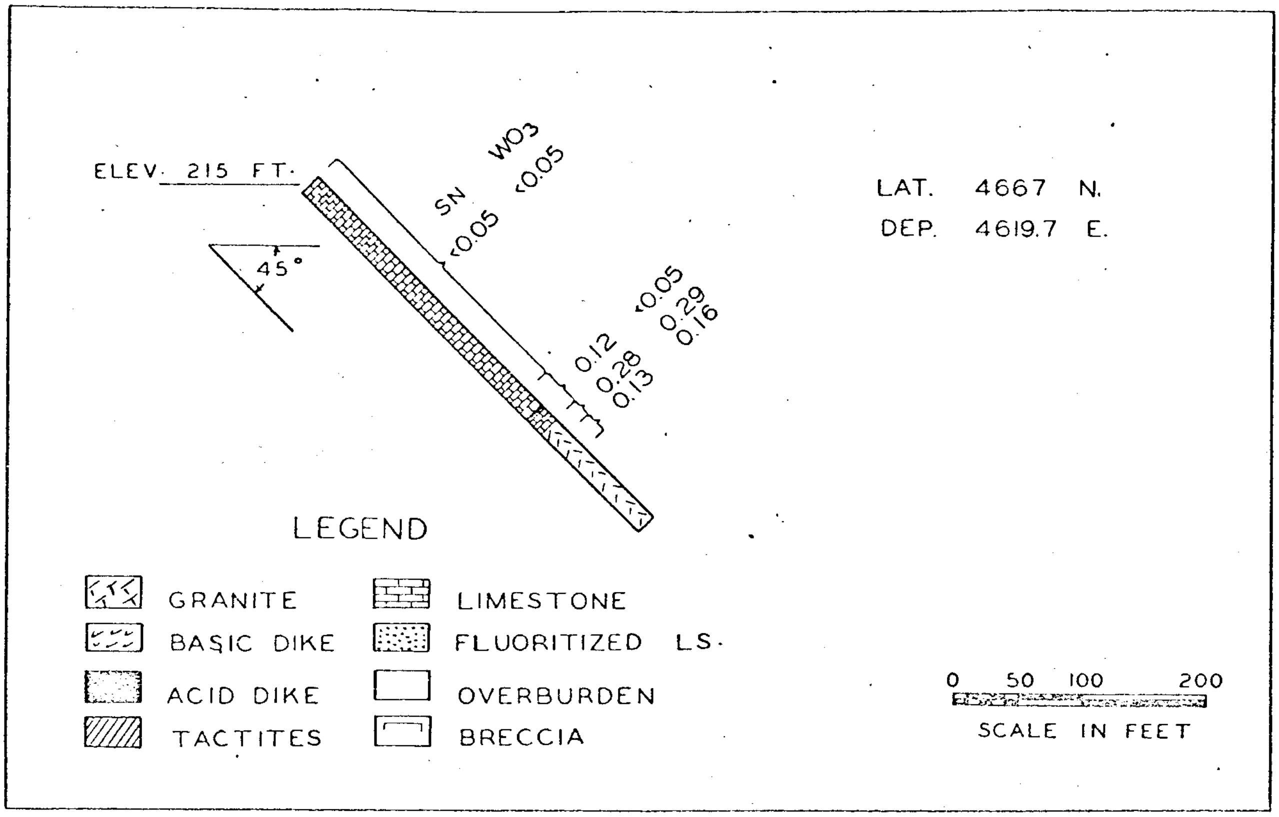


FIG. 25 — GEOLOGIC SECTION D.D. HOLE NO. 27

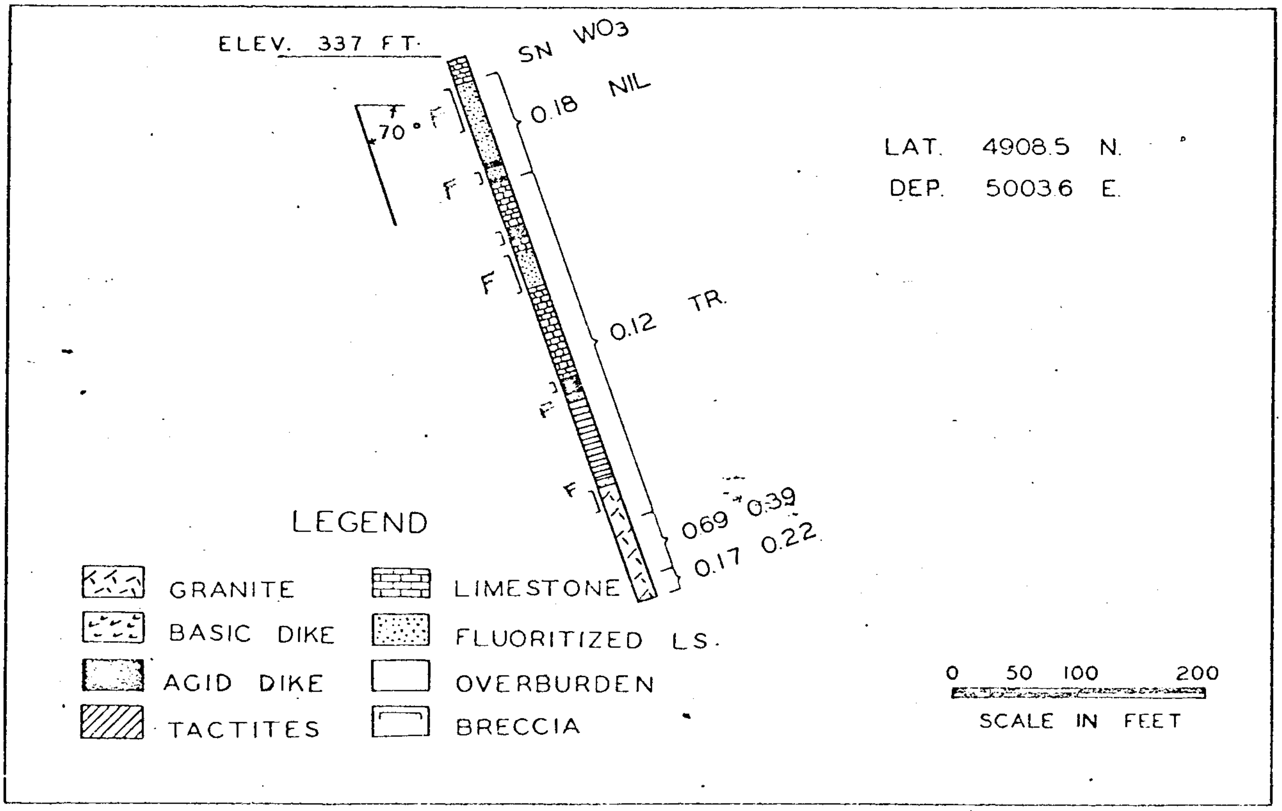


FIG. 26 GEOLOGIC SECTION D.D. HOLE NO. 28

but brecciation appears to be more pronounced in the vicinity of the dike. (See figs. 14, 15, 16, 20, 21, 24, 25, and 26.)

Other dikes that have been cut in drill holes but do not outcrop suggest that the system of flat-dipping fractures and intrusives may be more strongly developed than was at first supposed. Abundant breccia and both acidic and basic dikes have been found along the granite contact in some of the holes. These dikes may be similar flat-lying intrusives. (See figs. 14, 21, 25, and 26.)

The grade of tin and tungsten ores in Cassiterite dike diminishes rapidly so far as explored below the level of No. 3 adit, as revealed in the main winze and in drill holes. This could be due to structural control of the metallization by an east-west, flat-dipping system intersecting the Cassiterite dike. (See fig. 38 and table 5.) There are no criteria to warrant favorable or unfavorable production as to grade of ore in the dike below the lowest hole.

In regard to the chronology of fracturing and intrusion, there is some evidence that a system of fractures tangential to Cassiterite dike exists and that post-mineral faulting may have occurred along the diverging fractures and locally in Cassiterite dike. In No. 3 adit, east of the main winze, the footwall of the dike is composed of gouge and slickenside over widths of 10 to 20 inches. The gouge is studded with large crystals of cassiterite, which show the effects of strain and split into many small pieces along angular fracture planes. This cleavage is not characteristic of cassiterite crystals found elsewhere in the dike. Instead, it may indicate that post mineral stresses have occurred, probably along the plane of fault gouge. A small, high-grade tin vein found on the surface above Randt adit may belong to the same tangential system. The vein diverges from Cassiterite dike at trench No. 36 and runs through pits No. 1 and No. 2 in trench No. 8 (see fig. 33).

Occurrence of the Deposits

Three principal types of deposits occur in acidic dikes (figs. 27 and 28), the contact metamorphic zone (fig. 29 and 30), and in the granite (fig. 31 and 32). Deposits in the acidic dikes have been described in many Geological Survey publications and by engineers in private practice; but previous to exploration by the Bureau of Mines no ore reserves were known to exist in the granite and contact zone (see figs. 5 to 26, inclusive).

Deposits in Acidic Dikes

The only known dikes worthy of economic consideration are the Cassiterite and Ida Bell.

Cassiterite Dike

While the Cassiterite dike can be traced for nearly 2 miles, only a short portion of its length is sufficiently mineralized with tin and tungsten to warrant classification of the dike as a potential source of tin. The mineralized portion lies close to the granite and contact metamorphic zone. It is limited on the west by the approximate intersection with Ida Bell dike and on the east by a point about 100 feet east of No. 0 adit. Approximately 1,900 feet of the dike, therefore, is mineralized (fig. 2). An ore shoot nearly 600 feet long occurs above and below No. 3 adit. It extends to the surface above No. 3 adit, but, although the dike has not been thoroughly probed, the ore shoot apparently declines in grade a short distance below the adit.

F. C. Fearing reported that the grade of tin declined and that of tungsten rose toward the portal of No. 3 adit and as far west as Cassiterite Creek (fig. 37). Tungsten values in the prospect pits were said to range from 1.29 percent to 1.46 percent WO_3. In Bureau of Mines trench No. 17 on Cassiterite dike and east of Cassiterite Creek, both tin and tungsten analyses were low.

In the same report it was stated that tungsten minerals were not found in quantity above No. 3 adit. As the Bureau of Mines did not reopen No. 2 adit, the latter statement could not be confirmed. However, only low tungsten analyses were obtained from No. 1 adit (fig. 33).

Cassiterite dike, in its mineralized portion, strikes approximately N. 70° W. East of Cassiterite Creek it dips from 61° to 84° south. West of the creek dips vary from 78° south to vertical. Table 1 and figures 27 and 28 show apparent dips of Cassiterite dike. The width of the dike ranges from 4.5 feet up to 20 feet, with an average width of 10 feet. Changes in dip or strike apparently have no structural influence over the size or degree of mineralization in ore shoots.

TABLE 1. - Apparent dips of Cassiterite dike

From -	To -	Degree of dip	Direction of dip
Surface	Hole 17*/	78	South
Do.	Hole 6	90	
Do.	Hole 5	90	
Do.	Hole 1	84	South
Hole 1	Hole 14	73	Do.
Surface	Hole 2	79	Do.
Do.	Hole 24	70	Do.
Do.	Hole 3	70	Do.
Do.	Hole 4	61	Do.
Hole 4	Hole 11	80	Do.
No. 1 adit	No. 3 adit	68	Do.

*/ Apparent dip believed due to faulting; vertical dip used in calculating lode width.

Ida Bell Dike

The Ida Bell dike strikes N. 60° E. and is nearly vertical. It is less altered and mineralized than the Cassiterite dike. This dike was explored by surface trenches and diamond-drill holes over a length of 900 feet, commencing a09663

and on
1,900
early
surface
1, the

tung-
e
ge from
Cas-
yses

ound
. 2
ungsten

N.
f the
28
om
p or

ss
by
ing

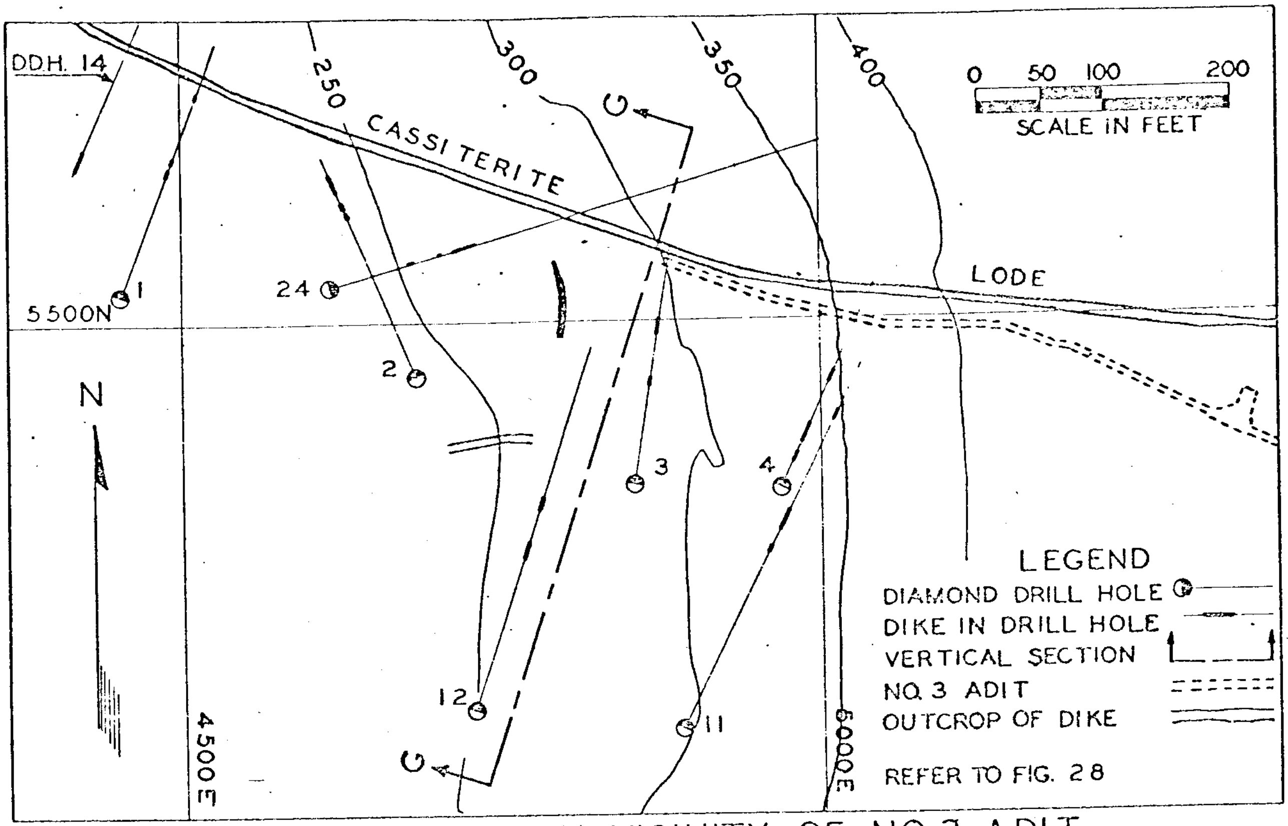


FIG. 27 PLAN OF DIKES IN VICINITY OF NO. 3 ADIT

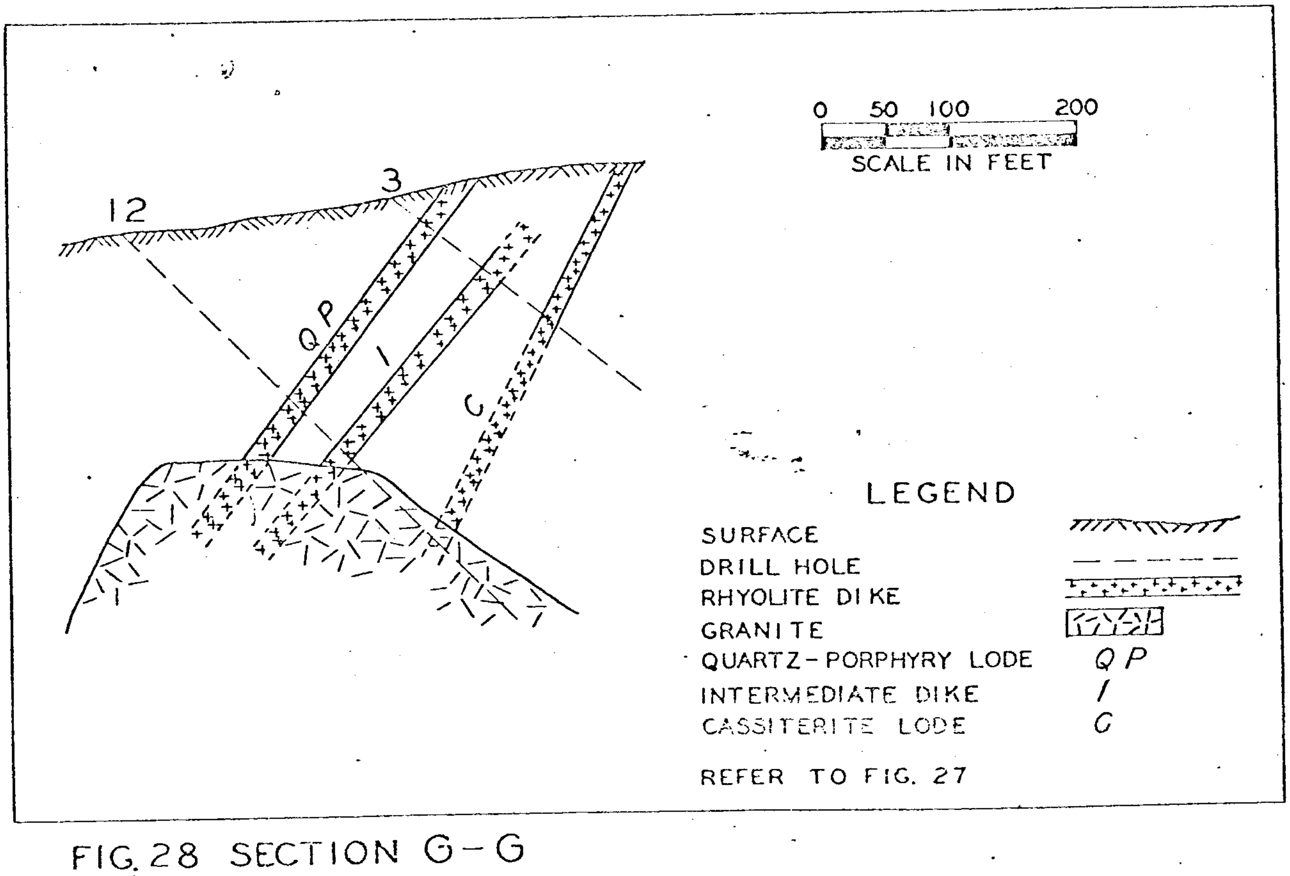


FIG. 28 SECTION G-G

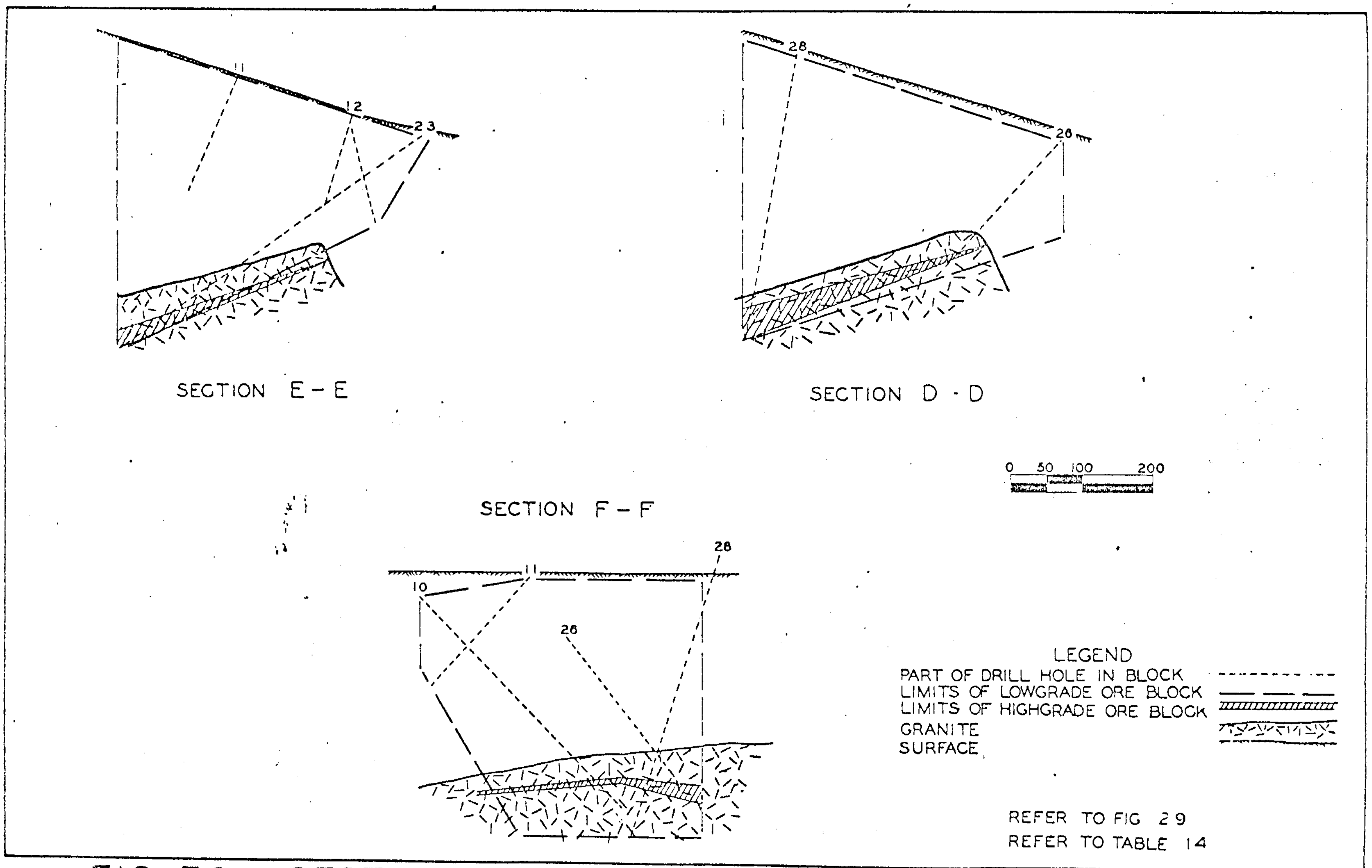


FIG. 30 SECTIONS OF LOWGRADE OREBODY

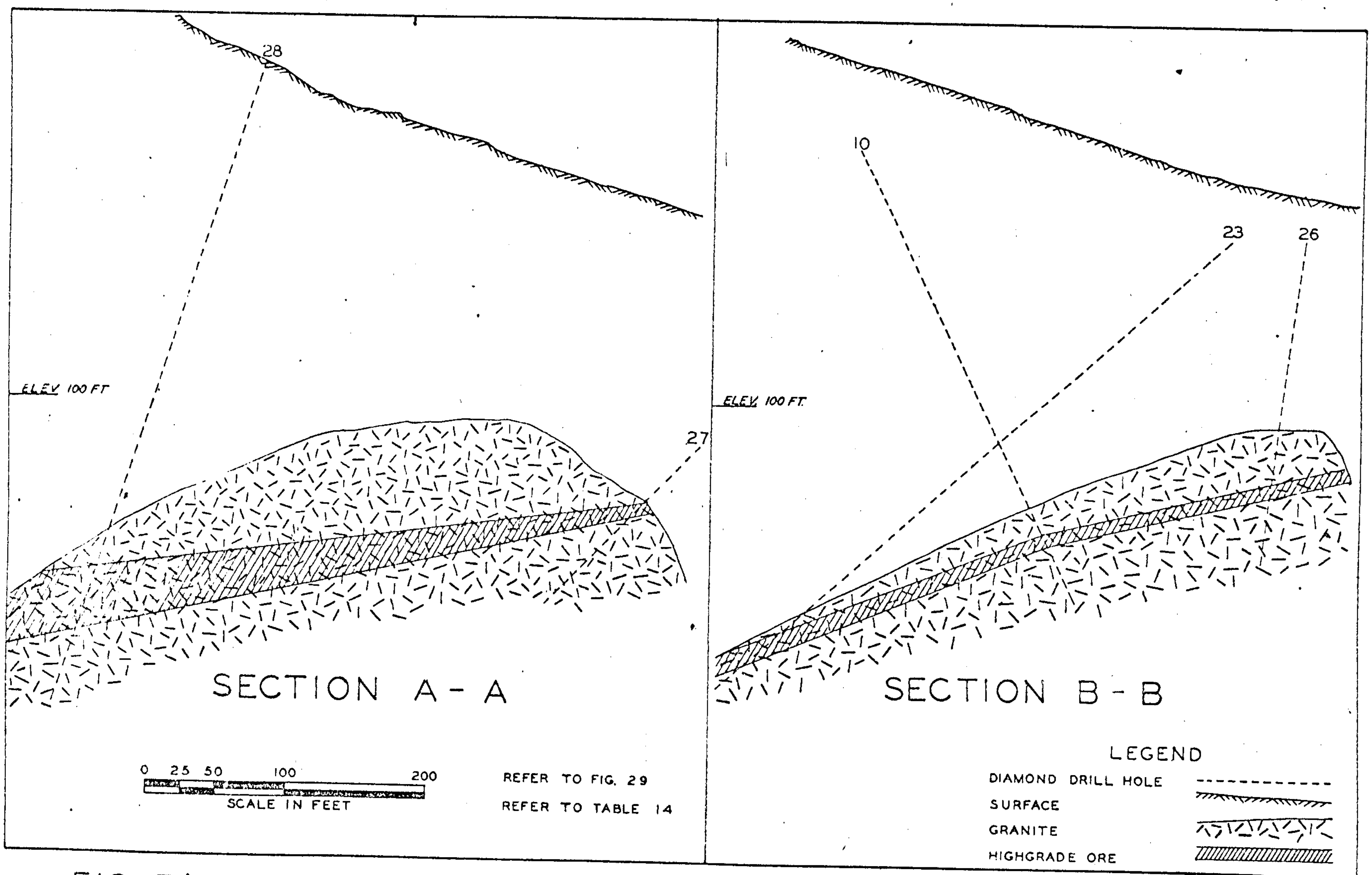


FIG. 31 VERTICAL SECTIONS IN GRANITE

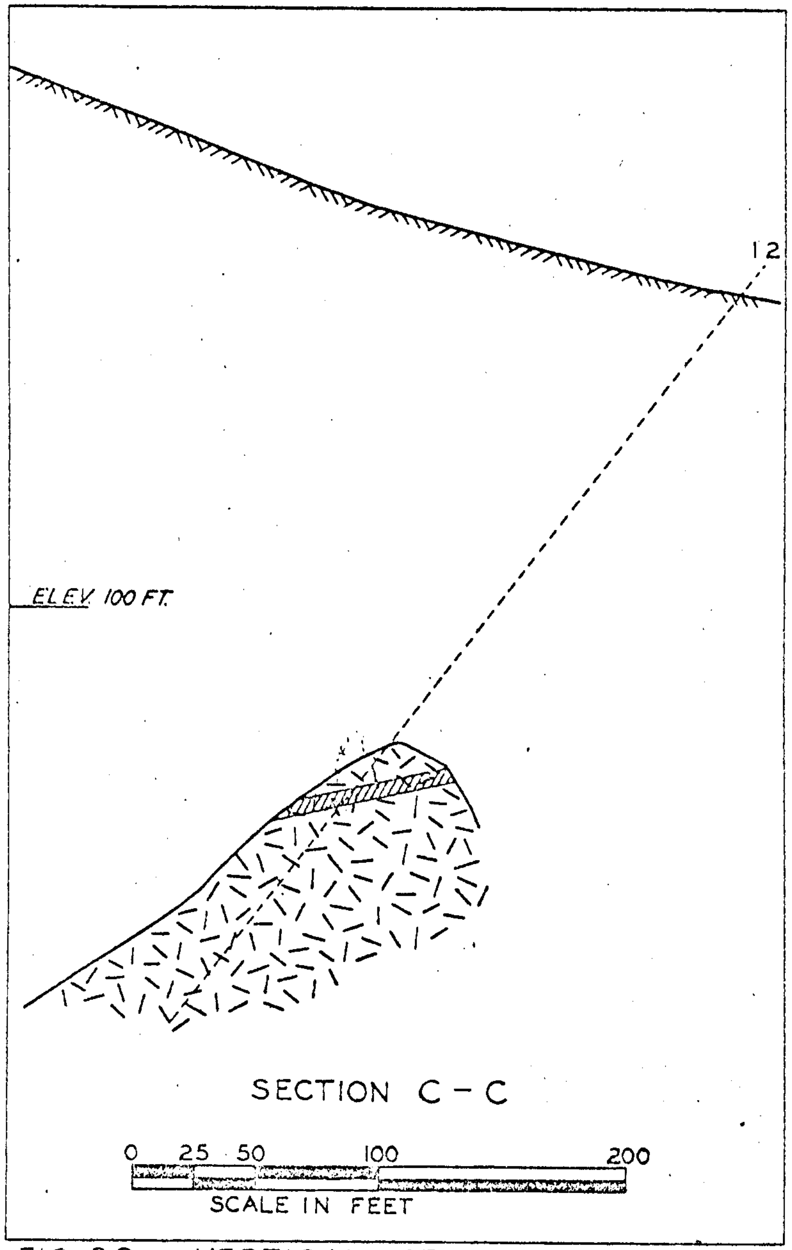
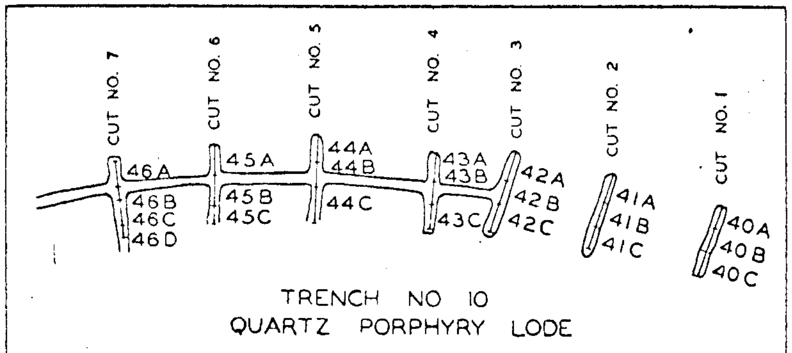
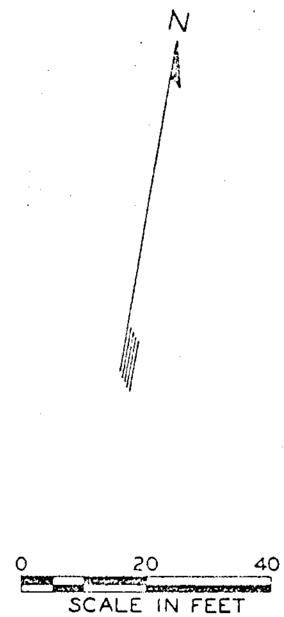


FIG. 32 VERTICAL SECTION IN GRANITE





SAMPLE NO	WIDTH FEET	NORMAL WIDTH	PERCENT SN	PERCENT WO3
40A	3.8		0.83	0.02
40B	3.8		0.59	0.02
40C	4.6		0.20	NIL
41A	4.7		1.54	0.03
41B	4.0		0.77	NIL
41C	4.0		1.01	0.06
42A	5.0		0.74	NIL
42B	4.0		0.74	0.08
42C	5.0		0.32	0.02
43A	4.5		0.38	0.02
43B	4.0		0.35	NIL
43C	4.0		0.83	0.03
44A	4.5		0.12	0.02
44B	4.0		0.65	0.02
44C	6.0		0.23	NIL
45A	5.0		0.06	0.02
45B	5.0		0.26	NIL
45C	2.7		0.23	NIL
46A	5.0		1.07	NIL
46B	2.5		0.53	0.18
46C	4.0		0.50	0.07
46D	2.0		0.65	NIL

FIG. 34 ASSAY MAP - QUARTZ PORPHYRY LODE

at the intersection with Cassiterite dike and extending that distance easterly (fig. 2). A small block of possible ore approximately 150 feet long was found at and near the intersection.

In surface trench No. 8 the Ida Bell dike is 55 feet wide and averages 1.39 percent tin and 0.02 percent WO_3; in trench No. 11, 130 feet easterly from trench No. 8, it is 29.9 feet wide and averages 0.46 percent tin and 0.03 percent WO_3 (fig. 33). The normal width of the dike in D.D. hole No. 17 is 18.2 feet and the average grade is 0.24 percent tin and no WO_3; in D.D. hole No. 7 it is 21 feet wide, and the average grade is 0.15 percent tin and 0.21 percent WO_3 (figs. 11 and 18).

The two trenches and two drill holes outline a potential block of ore with an average width of 31.0 feet and average grade of 0.79 percent tin and 0.05 percent WO_3. The favorable average grade depends upon high values in the two surface trenches, and as very little developing has been done on this dike, geological inferences are not justified, and the deposit is not included in ore reserves.

Contact Metamorphic Zone

The contact metamorphic zone covers a distance of 200 to 600 feet south of the portal of No. 3 adit. It is roughly 400 by 400 feet and overlies the apex of the granite boss (figs. 29 and 30).

Surface exposures and diamond-drill cores show the zone to be composed of a complex group of rocks and minerals. Originally limestone, the country rock has been brecciated, probably both by faulting and explosive forces from the granite magma. Acidic and basic dikes of all sizes have intruded the limestone and many of them form a matrix around limestone fragments. Nearly the entire zone appears to have been invaded by gases and mineralized solutions that have altered or replaced the original minerals. Tin is found in varying amounts throughout the zone. The most active agent appears to have been fluorine, and locally the rock consists wholly of fluorite. Slickenside and abraded breccia fragments were observed in many places in the drill cores and indicate the presence of numerous faults.

Deposits in Granite

Granite does not outcrop at the Lost River mine, but it was encountered in Bureau of Mines drill holes and has been shown to be tin-bearing. Near its contact with the limestone, the granite is highly altered and is essentially kaolin with lesser fluorite, topaz, quartz, galena, sphalerite, and chalcopyrite. This phase carries only small quantities of tin. In general the granite becomes progressively less altered with distance from the contact. Drill holes disclose that the strongest tin mineralization is not at the contact, but instead occurs within the granite at a maximum distance of 40 feet from the contact (see figs. 29, 31, and 32).

In order to delineate any ore body in the granite, it has been necessary to assume that the tin and tungsten found in different holes are connected in some way; and the reasons for making this assumption should be stated fully.

Tin has been found in the granite in six drill holes, all of which pass through the same phases of alteration and mineralization in greater or lesser degree. The alteration and mineralization has been intense and extends for many feet in all drill holes. It is of the same kind, and as drill holes are not spaced over 200 feet from the next nearest hole, with the exception of hole No. 12, it is reasonable to suppose that the mineralizing solutions have emanated from the same source.

The scattered nature of the holes precludes the possibility of the deposit being structurally controlled by any of the known steep-dipping fracture systems or a combination of these systems.

The deposit appears to have been controlled or influenced by the flat-dipping fracture system described before, and the ore body conforms closely to the shape outlined in figures 29, 31, and 32.

The contours of the granite, as plotted in the figures, were determined by interpolation of elevations between holes, and probably do not represent a true configuration of its surface. The breccia and gouge, already mentioned in the contact zone, are derived in part from post intrusive faulting, which has displaced the granite to some extent.

The vertical sections of the granite (figs. 31 and 32), on which the outline of the high-grade ore body is portrayed in straight lines, may suggest that distinct walls or fractures were observed. On the contrary, while the degree of hydrothermal alteration was high in the richer areas, there was no evidence of structure by which these places could be distinguished from other parts of the granite. Tin and tungsten occur in variable quantities down to and even below the limits of the ore body as shown. These limits, as drawn, merely indicate an arbitrarily chosen economic cut-off.

Character of the Ore

Rhyolite porphyry. - The Geological Survey kept a field party at Lost River during the progress of Bureau of Mines exploration. Their field and laboratory work has been correlated, and most of the following description and identification of minerals and rock formations has been abstracted from the unpublished report by Robert R. Coats.

> The alteration of the Cassiterite dike is more widespread than that of the Ida Bell dike, which apparently is most altered near its intersection with the Cassiterite dike.
>
> Much of the Cassiterite dike within the metallized zone is hard and retains the glassy quartz phenocrysts. The chief change in it was the replacement of the groundmass by topaz, lithia mica, and small amounts of fluorite. Locally, secondary quartz has been formed. Subsequent to the development of topaz and fluorite were deposited cassiterite, wolframite, and several sulfide minerals. Final alteration consisted of the development of chloritic products from the mica and of dickite, both from topaz and as a cavity

filling. The formation of chlorite and dickite generally resulted in the softening of the rock and makes artificial support necessary for mine openings in such rocks. Similar alteration is present in the other bodies of rhyolite porphyry, and some, such as the light-colored intrusive exposed in cuts Nos. 5 and 9, are so completely altered that their original composition can be determined only by analogy. The origin of some other masses of rock is exceedingly obscure. In the trenches of cut No. 7 and adjacent bedrock exposures there is considerable aphanitic, brown, or olive-green rock, containing no minerals identifiable in hand specimens except an occasional grain of fluorite and thin, irregular, bluish stringers of tourmaline. Enclosed in this material are angular fragments of a grayish, white-weathering rock, rich in fluorite. This breccia grades, with increasing amounts of fragmental material, into a fluoritic rock with sparse aphanitic veinlets. The aphanitic material appears under the microscope as a very fine-grained aggregate of lithia-mica flakes with minor amounts of fluorite. The angular fragments embedded in it are made up of rounded grains of fluorite separated by a meshwork of an aggregate of minute mica flakes. Little cassiterite is visible in this aphanitic material or in the breccia fragments contained in it. Because of their resemblance to the altered limestone adjacent to the rhyolite porphyry dikes, the fragments in the breccia appear to be the result of the replacement of limestone fragments. The origin of the fine-grained mica-fluorite matrix of the fragments is not known definitely, but the relations suggest that it is the product of the complete replacement of an irregular intrusive mass of rhyolite porphyry. Alteration such as that described above is commonly called greisenization.

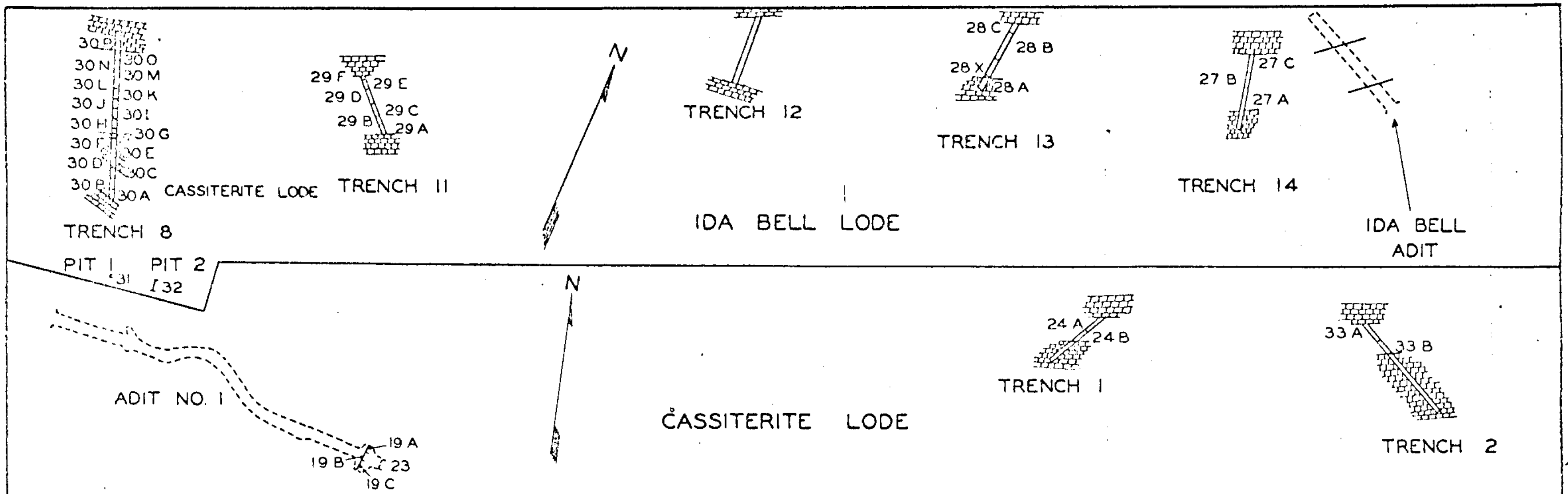
Basalt porphyry. - The basalt porphyry, like the rhyolite porphyry, is extensively greisenized, typically where exposed in cuts 5 and 9. However, the basalt porphyry contains much less topaz, and the lithia mica is generally richer in iron. The higher iron content accounts for the brown or green color of the mica and of the rock. The titanium of the original rock is represented in the alteration product by a variety and abundance of titanium minerals principally rutile, anatase, and sphene. The apatite of the original rock persists through rather intense alteration, though perhaps recrystallized. The solutions that have softened the rhyolite porphyry by the production of dickite have also attacked the basalt porphyry and produced various ill-defined greenish chloritic minerals, principally from the lithia micas. Little cassiterite has been observed in the altered basalt porphyry, but no reason is known why it should not occur there.

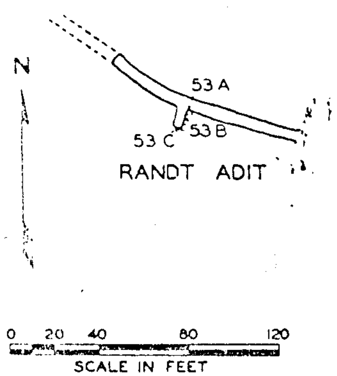
Veinlets. - The altered rocks, both igneous and calcareous, are cut by numerous veinlets ranging in width from a fraction of an inch to a few inches and in length from a few tens of feet to a hundred feet. These veinlets are made up principally of quartz, lithia mica, topaz, cassiterite, wolframite, arsenopyrite, and fluorite. Not all of these minerals are present in any one veinlet. Although some veinlets are rich in cassiterite, or wolframite, or both, they appear too small and widely separated to be minable, either singly or as a composite lode.

Limestone and dolomite. - The calcareous country rock has been altered in several ways. Close to the Cassiterite dike and to many of the small intrusives, the limestone and dolomite have been replaced by an aggregate of fluorite and mica, in which coarse-to-medium grained fluorite is predominant. Such material also makes up the angular fragments embedded in altered rhyolite porphyry. Similar material, but finer-grained and with an irregular banding roughly parallel to the veinlet, replaces the calcareous rock adjacent to many minute veinlets of micaceous material, which is regarded as the product of replacement of rhyolitic material. Cassiterite and other ore minerals are rare in this predominantly fluorite rock. Another common product of the metasomatism of the calcareous rocks is a green fluorite-tactite, which ranges widely in grain and composition. This material occurs both as an irregularly banded replacement of limestone or marble adjacent to fractures or as irregular masses which are not related to any observable fractures. The massive green rock may represent the coalescense of numerous sheets of banded rock, developed either where fractures were closely spaced, or where attack by solutions was especially intense. Idocrase, diopside, hornblende, biotite, chlorite, fluorite, plagioclase, garnet, and magnetite are commonly present. Small amounts of scheelite are disseminated through them in many places as minute grains. Such material is nowhere plentiful enough to constitute a tungsten ore. These rocks contain cassiterite locally, but no specimens especially rich in that mineral have been observed.

At present no Geological Survey report is available on the recent discovery of the granite and the development of the contact metamorphic zone by Bureau drilling.

Granite. - Rock specimens of the granite have been obtained only from drill cores. The material from some sections of the holes was so soft and friable, because of alteration, that the original texture and composition could only be surmised. On the contact, such material chiefly consists of kaolin and calcite with smaller amounts of topaz, tourmaline, zinnwaldite, and sparse amounts of pyrite, sphalerite, galena, arsenopyrite, cassiterite, and wolframite. Locally, variations occur in composition and amount of alteration and mineralization, but in general with increasing depth the granite appears less altered. Harder specimens are essentially quartz, clay, zinnwaldite, and topaz with minor amounts of tourmaline, galena, chalcopyrite, and sphalerite. Feldspars in this zone are still too highly altered to be identifiable.



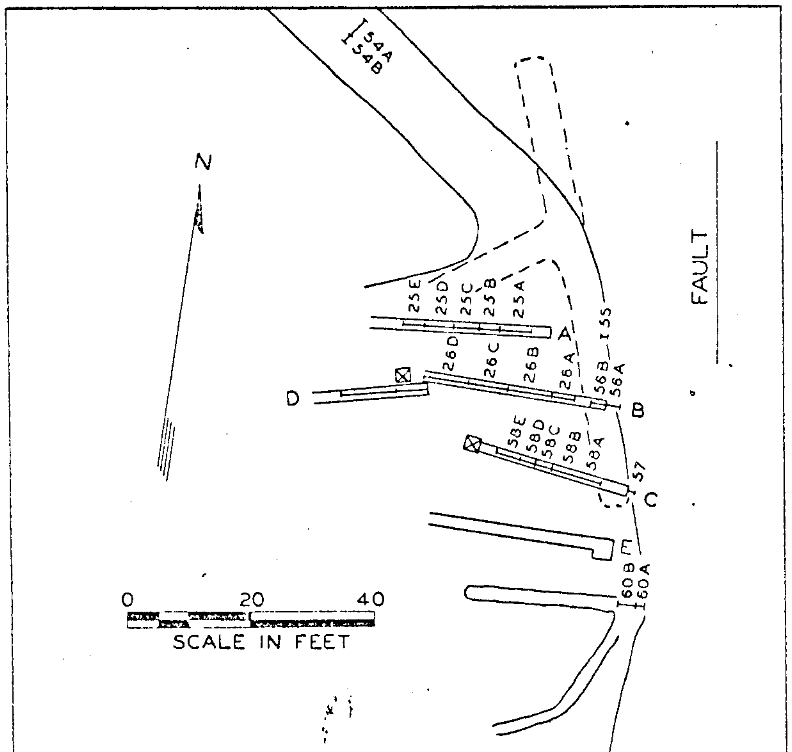


SAMPLE NO	WIDTH FEET	NORMAL WIDTH	PERCENT SN	WO₃
TRENCH NO. 8 – IDA BELL LODE				
30 A	5.0	5.0	0.89	NIL
30 B	5.0	5.0	0.53	NIL
30 C	5.0	5.0	0.74	NIL
30 D	5.0	5.0	1.96	NIL
30 E	5.0	5.0	0.23	0.02
30 F	5.0	5.0	0.74	0.02
30 G	5.0	5.0	0.77	NIL
30 H	5.0	5.0	1.25	0.04
30 I	5.0	5.0	0.59	NIL
30 J	5.0	5.0	2.50	0.02
30 K	5.0	5.0	2.73	0.03
30 L	5.0	5.0	1.96	NIL
30 M	5.0	5.0	2.73	0.02
30 N	5.0	5.0	0.59	NIL
30 O	5.0	5.0	0.44	NIL
30 P	5.0	5.0	0.98	0.07

SAMPLE NO.	WIDTH FEET	NORMAL WIDTH	PERCENT SN	WO_3
TEST PIT NO. 1				
31	1.6	1.6	2.14	0.03
TEST PIT NO. 2				
32	6.0	6.0	2.83	NIL
TRENCH NO. 11 – IDA BELL LODE				
29 A	5.5	5.2	0.26	0.02
29 B	4.5	4.4	0.09	0.02
29 C	7.5	7.1	0.23	0.03
29 D	7.5	7.1	1.10	0.03
29 E	5.0	4.7	0.44	0.02
29 F	1.5	1.4	0.53	0.05
TRENCH NO. 13 – IDA BELL LODE				
28 A	8.0	6.9	0.06	0.02
28 X	9.5	8.2	0.36	0.03
28 B	15.0	13.0	0.29	0.06
28 C	5.0	4.3	0.20	0.05
TRENCH NO 14 – IDA BELL LODE				
27 A	11.5	11.1	0.14	0.04
27 B	8.0	7.7	0.10	0.05
27 C	9.5	9.2	0.11	0.07

SAMPLE NO.	WIDTH FEET	NORMAL WIDTH	PERCENT SN	WO_3
ADIT NO. 1 – CASSITERITE LODE				
19 A	3.9	3.6	1.10	0.05
19 B	4.0	3.8	0.89	0.08
19 C	6.0	5.7	0.50	0.07
23	6.0	5.7	0.98	0.07
TRENCH NO. 1 – CASSITERITE LODE				
24 A	9.5	5.5	TR	NIL
24 B	8.0	4.6	0.17	0.02
TRENCH NO. 2 – CASSITERITE LODE				
33 A	10.5	7.5	0.53	0.07
33 B	7.5	5.4	0.23	NIL
RANDT ADIT – CASSITERITE LODE				
53 A	2.5	1.0	0.77	1.10
53 B	5.8	5.4	0.20	0.28
53 C	3.0	2.8	0.20	0.15

FIG. 33 ASSAY MAP — IDA BELL AND CASSITERITE LODES

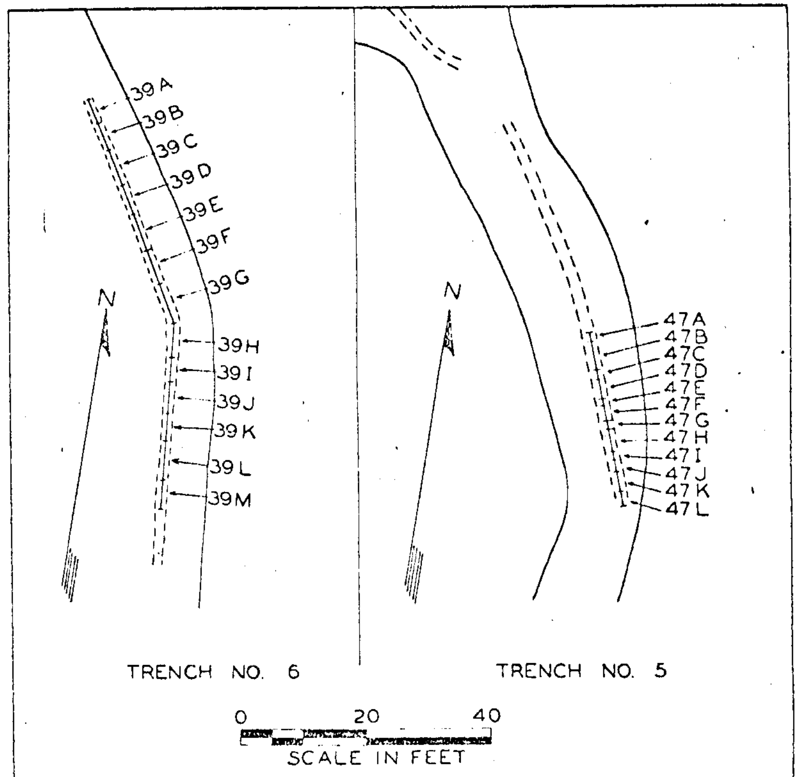


TRENCH NO. 9

SAMPLE NO.	WIDTH FEET	NORMAL WIDTH	PERCENT SN	PERCENT WO_3
54A	3.6		0.15	NIL
54B	1.2		0.23	NIL
25A	6.0		0.17	0.02
25B	3.5		0.17	0.02
25C	4.4		0.05	0.02
25D	4.5		0.26	0.03
25E	3.4		0.26	0.03
26A	3.3		5.00	0.02
26B	7.0		1.72	0.03
26C	6.8		0.21	0.03
26D	8.0		0.08	0.02
56A	2.3		1.07	0.16
56B	2.4		0.30	NIL

SAMPLE NO.	WIDTH FEET	NORMAL WIDTH	PERCENT SN	PERCENT WO_3
59A	5.0		0.14	0.02
59B	8.7		0.11	0.04
57	3.6		0.41	0.03
58A	4.2		2.62	0.02
58B	4.5		3.81	NIL
58C	2.8		0.08	0.02
58D	2.5		0.44	0.03
58E	3.5		0.05	0.03
60A	2.5		0.11	NIL
60B	2.7		0.39	0.03
55	3.3		0.08	0.02

FIG. 35 ASSAY MAP — GREENSTONE LODE



TRENCH NO. 6

SAMPLE NO.	WIDTH FEET	NORMAL WIDTH	PERCENT SN	PERCENT WO_3
39A	4.5		0.12	0.02
39B	5.0		0.20	NIL
39C	6.0		0.50	0.03
39D	5.0		0.09	0.42
39E	6.0		0.41	0.43
39F	5.5		0.38	0.02
39G	7.5		0.53	NIL
39H	5.0		0.06	NIL
39I	4.0		1.28	0.08
39J	5.0		0.53	0.02
39K	4.5		0.18	0.18
39L	6.5		0.59	0.02
39M	5.0		2.02	NIL

TRENCH NO. 5

SAMPLE NO.	WIDTH FEET	NORMAL WIDTH	PERCENT SN	PERCENT WO_3
47A	1.7		0.09	NIL
47B	4.5		0.11	0.02
47C	1.3		0.14	0.02
47D	3.0		0.12	0.02
47E	1.0		0.14	NIL
47F	2.4		0.20	0.03
47G	1.6		0.06	0.02
47H	4.0		0.11	NIL
47I	1.3		0.29	0.03
47J	2.0		TR.	NIL
47K	3.6		0.12	NIL
47L	2.3		0.09	NIL

FIG. 36 ASSAY MAP — GREENSTONE LODE

1899600

In some places sulfides may constitute up to 4 percent by weight of the rock, but there is no evidence of any association of sulfides and tin or constant ratio between the quantities of each.

Contact metamorphic zone. - The rocks of the contact zone are comprised of the metamorphosed limestones and dolomites, rhyolite and basalt porphyries, and veinlets described by Coats. The effects of movement and eruptive forces are pronounced, with the formation of large and small breccia fragments and widespread slickensides. All of the rocks have been attacked by solutions, with complete replacement occurring in many places, so that the original rock constituents are scarcely identifiable, and the entire zone is a complex assembly of rocks and structures. Because of the many intrusives and periods of faulting, much fragmentation and fracturing has occurred, providing innumerable channels for the entrance of metallizing solutions. Although there are richer concentrations of tin locally, tin deposition has occurred in some degree in all formations. Wolframite, molybdenite, pyrite, chalcopyrite, arsenopyrite, galena and sphalerite have also been introduced in various amounts, times, and places.

SAMPLING AND ASSAYING

The ore bodies were sampled by the Bureau of Mines by means of surface trenching, underground sampling in parts of the old workings, and diamond drilling. During the course of the exploration, 53 trenches with a combined length of 5,750 feet were excavated by means of a bulldozer, 22 trenches with a combined length of 1,585 feet were dug by hand, 251 channel samples were taken across a total length of 1,132 feet, 22 core-drill holes with a total depth of 8,693 feet were completed, and 1,434 core and sludge samples were submitted for analysis.

Surface trenches were used to explore for lateral extensions of known ore shoots, for new ore shoots in the known large lodes, and for small high-grade veins. The principal deposits explored were the Cassiterite dike, Ida Bell dike, Quartz-Porphyry dike, Greenstone lode (contact metamorphic zone), and numerous veinlets. (See figs. 2, 33, 34, 35, and 36.)

Trenches were started with bulldozers and carried through frozen overburden to hard rock. As much loose rock as possible was scraped off with the bulldozer, and smaller hand trenches were then dug to solid rock. The line for samples cuts was laid out in the hand trenches, and the rock face was carefully dressed and cleaned. Channel cuts were 1 by 8 inches to 1 by 12 inches. Canvas coverings were spread at all times to prevent loss of chips or contamination of sample.

At the time of Bureau of Mines exploration, many of the old workings were caved and inaccessible. These had been sampled between 1918 and 1920 by F. C. Fearing, an engineer of good reputation, and the Fearing assay maps and report were available to the Bureau.

The Bureau decided that any great increase in ore reserves at the Lost River mine would have to be found in depth and that exploration should be principally in that direction.

Only enough of the old workings were reopened and sampled to draw a conclusion as to the accuracy of the Fearing data. No. 1 adit, No. 3 adit, and the main winze to the water level at 200 feet were opened for the purpose of check-sampling.

In No. 3 adit, much of the old timber was too rotten to make sampling of the roof advisable. Instead, hand trenches were dug across the adit floor as near to 10-foot intervals as was possible. The muck and spill from cars that had accumulated on the floor was frozen solid, and this permitted unusual cleanliness for this type of sampling. Moreover, the rock surfaces had been disintegrated by frost action to a depth of several inches and, by removing this disintegrated material, a smooth, clean surface was obtained for sampling. Channel cuts were 1 by 4 inches in cross section.

As the width of the adit in many places is less than the width of the dike, neither the Fearing samples nor the Bureau of Mines samples represent the full width of the ore body. However, the average width of 18 Bureau samples was 8.3 feet, in contrast with an average width of only 5.1 feet for 31 comparable samples taken by Fearing. Within the length of dike represented by the samples, the average width of the ore body is estimated not to be over 10 feet, and, as some concentration of tin and tungsten minerals was observed on one wall or the other, it is believed that Bureau samples give a conservative, close measure of values for the full width of dike. Figure 37 shows the location and analyses of samples by Fearing and the Bureau of Mines.

Since it was necessary to use sample data from the Fearing report to calculate ore reserves in inaccessible old workings, Bureau sample results were compared with corresponding samples taken by Fearing, as in table 2. The comparison shows Bureau average grades of tin and tungsten to be approximately 80 percent of the Fearing average.

In order to sample the main winze below No. 3 adit, lagging was removed at 25-foot intervals to the 200-foot level, where the water level was encountered. Channel samples were cut across the west end of the winze. Sampling of the full width of the dike was impossible for the same reason that applies to No. 3 adit. Sample 48 is omitted from the calculations because it was impossible to sample the entire width of the dike. However, a crosscut on the 200-foot level enabled sampling a full cross section of the dike on each side of the crosscut. Sample 34 was cut across the dike on the west wall of the crosscut and sample 35 on the east wall. (See fig. 38.)

Two channel samples were cut in No. 1 adit, one across the roof near the face and the second across the face. (See fig. 33.)

Diamond drilling was first undertaken to explore in depth the known lodes, such as Cassiterite dike, Ida Bell dike, Quartz-Porphyry dike, and Greenstone lode. Several prospect holes also were drilled to explore the zone known as the Reef, north of the portal of No. 3 adit, where numerous high-grade veinlets outcropped. Drilling in the Greenstone lode area was done to explore for a granite intrusion that was indicated by geologic evidence. The granite was discovered in 1943, and tin mineralization was so favorable that a large part of subsequent drilling was done in the granite area. (See fig. 2.) The locations of holes and results of drilling are shown in tables 3, 5, 6, 8, 9, and

con-
and
of

g
loor
rs
us-
ad
e-
for

e
nt
sam-
31
ed
over
rved
rva-
s the

cal-
ere
com-
ly

ved

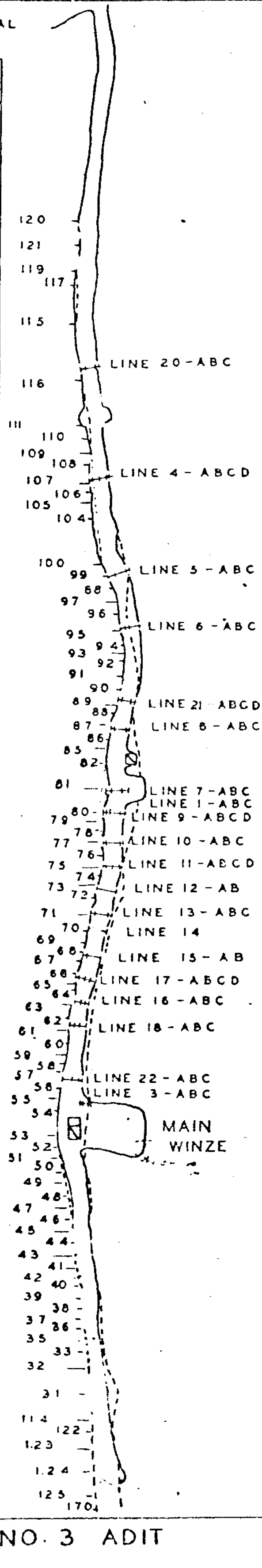
amp-
t
e it
ıt
n

the

lodes,
tone
as
inlets
a
as
part
ca-
and 10

SAMPLE NO.	NORMAL WIDTH	% SN	% WO_3
+120		0.43	0.77
+121	5.5	0.92	1.29
+119	7.5	1.26	1.97
+117		0.50	0.00
+113	5.0	0.52	0.47
20	6.4	0.83	0.73
+116	6.0	1.08	0.00
+111	3.0	0.66	0.47
+110	2.5	0.34	0.39
+109	3.0	0.77	0.60
+108	3.5	0.74	0.33
4	8.6	0.39	0.46
+107	2.0	0.43	0.79
+106	1.5	0.20	0.45
+105	2.0	0.00	0.91
+104	6.5	0.22	1.23
+100	2.0	0.22	3.13
5	8.2	1.11	0.79
+ 99	3.5	0.65	0.80
+ 98	3.5	0.77	0.79
+ 97	4.0	0.58	0.73
+ 96	3.5	0.74	1.39
6	9.2	0.49	1.35
+ 95	3.5	0.94	0.00
+ 94	4.0	0.71	0.40
+ 93	4.0	0.46	0.41
+ 92	4.0	1.91	1.10
+ 91	3.0	0.58	0.54
+ 90	5.0	1.35	0.44
21	9.1	0.48	0.20
+ 89	7.5	2.98	0.25
+ 88	6.0	0.92	0.30
8	7.9	0.30	0.08
+ 87	5.5	0.88	0.00
+ 86	4.5	0.68	0.00
+ 85	5.0	0.74	0.00
+ 82	10.0	1.05	0.75
7	8.5	0.50	0.38
+ 81	12.0	0.37	0.00
9	7.2	0.49	0.37
+ 80	4.0	0.83	0.27
+ 79	4.0	0.80	0.35
+ 78	5.0	1.11	0.75
10	8.1	0.71	0.17
+ 77	7.8	1.60	1.21
+ 76	5.5	1.17	0.44
11	7.7	1.84	0.44
+ 75	4.5	1.23	0.70
+ 74	4.0	0.80	0.60
12	8.3	2.46	0.17
+ 73	6.0	3.44	1.50
+ 72	5.5	4.03	0.93

SAMPLE NO.	NORMAL WIDTH	% SN	% WO_3
13	7.1	2.73	0.23
+ 71	5.5	1.68	0.25
			-
14	7.4	3.33	2.23
+ 70	6.5	2.92	0.87
+ 69	6.5	3.54	0.34
+ 68	6.0	1.97	0.72
13	7.1	2.73	0.23
+ 67	4.5	1.41	0.84
+ 66	4.5	1.23	1.65
17	8.3	0.76	0.07
+ 65	6.5	2.98	0.63
+ 64	4.5	1.69	0.90
16	6.1	1.13	0.31
+ 63	4.5	2.21	0.42
+ 62	5.5	1.23	0.48
18	7.4	0.60	0.44
+ 61	6.0	0.89	1.10
+ 60	8.0	1.41	0.61
+ 59	7.0	3.63	0.25
+ 58	5.0	1.60	0.28
+ 57	7.5	3.57	1.36
22	9.7	1.91	0.37
+ 56	6.5	2.15	0.48
+ 55	4.5	1.23	0.42
+ 54	11.5	2.30	0.50
+ 53	12.0	2.54	0.95
+ 52	5.0	2.37	0.82
+ 51	7.5	3.01	0.63
+ 50	5.0	1.94	0.48
+ 49	5.5	1.23	0.32
+ 48	8.0	3.57	0.54
47	5.0	1.11	1.07
+ 46	5.5	2.46	0.28
+ 45	5.5	1.35	0.29
+ 44	7.5	3.20	0.22
+ 43	7.5	2.40	0.16
+ 42	5.5	1.25	1.07
+ 41	6.0	2.09	0.14
+ 40	6.5	1.91	0.00
+ 39	5.25	2.21	0.00
+ 38	5.5	1.56	0.2[illegible]
+ 37	6.5	2.15	
+ 36	8.0	2.09	
+ 35	5.5	1.63	
+ 33	6.5	3.14	
+ 32	5.0	0.89	
+ 31		2.17	
+114		1.61	
+122		1.01	
+1.23		0.49	
+1.24		2.22	
+125		1.05	
+170		1.32	0.27



SAMPLE NO.		NORMAL WIDTH FEET	% SN	% WO_3
20	A	0.1	0.11	0.09
	B	5.3	0.95	0.83
	C	1.0	0.30	0.48
4	A	0.4	0.71	0.05
	B	6.1	0.32	0.10
	C	0.9	0.47	2.48
	D	1.2	0.59	0.94
5	A	0.3	1.19	0.07
	B	4.4	1.17	0.32
	C	3.5	1.04	1.45
6	A	4.5	0.26	0.07
	B	1.4	0.83	0.54
	C	3.3	0.65	3.46
8	A	0.2	0.26	0.07
	B	7.0	0.32	0.09
	C	0.7	0.80	0.02
21	A	1.0	0.26	0.05
	B	4.3	0.53	0.15
	C	1.3	0.38	0.17
	D	2.5	0.47	0.36
7	A	0.9	0.08	0.07
	B	5.3	0.44	0.54
	C	2.3	0.80	0.15
1	A	2.7	0.01	
	B	1.3	0.09	
	C	3.9	0.07	
			0.23	
9	A	1.8	0.23	0.02
	B	4.7	0.53	0.46
	C	0.9	0.53	0.53
	D	0.8	0.17	0.10
2	A	1.9	0.81	
	B	0.4		
10	A	0.3	0.23	0.06
	B	6.8	0.59	0.19
	C	1.0	1.04	0.13
11	A	2.1	0.08	0.03
	B	2.9	1.66	0.92
	C	2.0	4.28	0.35
	D	0.7	0.95	0.02
12	A	7.5	2.68	0.18
	B	0.8	0.44	0.05
13	A	0.5	0.29	0.03
	B	5.3	3.21	0.31
	C	1.3	0.71	0.03
14		7.4	3.33	2.23
15	A	8.2	1.90	0.87
	B	1.3	0.47	0.02
17	A	2.4	2.80	0.07
	B	2.7	1.72	0.07
	C	2.4	0.44	0.07
	D	0.8	0.77	0.05
16	A	2.0	2.38	0.57
	B	1.6	0.89	0.35
	C	2.5	0.32	0.07
18	A	3.2	0.47	0.83
	C	1.9	1.25	0.12
22	A	2.4	0.64	1.08
	B	2.8	0.68	0.07
	C	4.5	0.05	0.19
3	A	0.5	0.08	
	B	1.7	0.35	
	C	1.1	1.23	

SAMPLES BY U.S.B.M.

* SAMPLES BY FEARING

=== APPROXIMATE OUTLINE OF DIKE

0 50 100

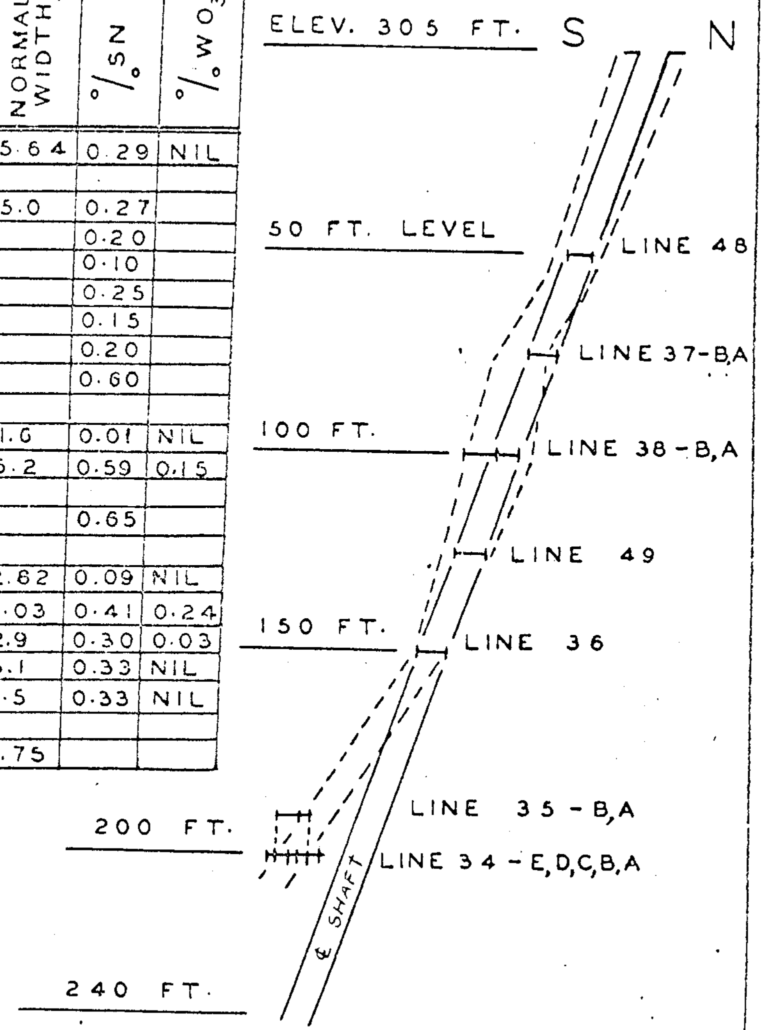
SCALE IN FEET

FIG. 37 ASSAY MAP - NO. 3 ADIT

FEARING FEET	U.S.B.M.	NORMAL WIDTH	%SN	%WO_3
5		7.0	1.25	0.43
10		8.0	1.65	0.18
15		8.0	3.00	0.20
20		8.0	0.50	0.22
25		8.0	1.80	0.31
30		9.0	0.80	0.48
35		8.0	1.00	0.26
40		8.5	1.60	0.30
45		8.0	1.50	0.18
50		9.0	0.75	TR.
	48	5.64	0.06	0.47
55		4.5	0.60	
60		8.0	0.50	
65		6.5	0.30	
70		6.5	0.30	
	37A	2.82	1.72	NIL
75		4.5	0.15	
	37B	2.82	0.36	0.19

FEARING FEET	U.S.B.M.	NORMAL WIDTH	%SN	%WO_3
80		4.0	0.10	
85		5.0	0.08	
90		5.0	0.05	
95		3.5	0.05	
	38 A	7.23	0.23	0.30
100		2.0	0.08	
	38 B	6.4	0.50	0.12
105		2.0	0.25	
115		5.0	0.10	
120			0.20	
	49	5.72	0.47	3.63
125			0.15	
130		5.0	0.30	
135		2.5	0.30	
140		5.0	0.30	
145		5.0	0.30	

FEARING FEET	U.S.B.M.	NORMAL WIDTH	%SN	%WO_3
	36	5.64	0.29	NIL
150		5.0	0.27	
155			0.20	
160			0.10	
165			0.25	
170			0.15	
175			0.20	
180			0.60	
	35A	1.6	0.01	NIL
	35B	5.2	0.59	0.15
200			0.65	
	34 A	2.82	0.09	NIL
	B	1.03	0.41	0.24
	C	2.9	0.30	0.03
	D	3.1	0.33	NIL
	E	1.5	0.33	NIL
240		0.75		



----- DIKE

0 40 80

SCALE IN FEET

FIG. 38 ASSAY . MAP - MAIN WINZE

TABLE 2. - Comparison of samples by F. C. Fearing and by Bureau of Mines

No. 3 adit.

Samples by U. S. Bureau of Mines				Samples by Fearing			
No.	Width, feet	Sn, percent	WO_3 percent	No.	Width, feet	Sn, percent	WO_3 percent
20	6.4	0.83	0.73	116	6.0	1.08	Nil
				108	3.5	0.74	0.33
4	8.6	0.39	0.46	107	2.0	0.43	0.79
				100	2.0	0.22	3.13
5	8.2	1.11	0.79	99	3.5	0.65	0.80
				96	3.5	0.74	1.39
6	4.7	0.70	2.59	95	3.5	0.94	0.00
				90	5.0	1.35	0.44
21	9.1	0.46	0.20	89	7.5	2.98	0.25
				88	6.0	0.92	0.30
8	7.9	0.30	0.08	87	5.5	0.86	0.00
7 & 1	16.4	0.22	0.20	81	12.0	0.37	0.00
9	7.2	0.49	0.37	80	4.0	0.83	0.27
				78	5.0	1.11	0.75
10	8.1	0.71	0.17	77	7.0	1.60	1.21
				76	5.5	1.17	0.44
11	7.7	1.84	0.44	75	4.5	1.23	0.70
				74	4.0	0.80	0.60
12	8.3	2.46	0.17	73	6.0	3.44	1.50
				72	5.5	4.03	0.93
13	7.1	2.73	0.23	71	5.5	1.66	0.25
14	7.4	3.33	2.23	70	6.5	2.92	0.87
				68	6.0	1.97	0.72
15	7.5	1.65	0.72	67	4.5	1.41	0.84
				66	4.5	1.23	1.65
17	8.3	0.76	0.07	65	6.5	2.98	0.63
				64	4.5	1.69	0.90
16	6.1	1.13	0.31	63	4.5	2.21	0.42
				62	5.5	1.23	0.48
18	7.4	0.60	0.44	61	6.0	0.89	1.10
22 & 23	13.0	1.60	0.28	54	11.5	2.30	0.50
	149.4	21.31	10.48		167.1	45.98	22.19
Avg.	8.3	1.16	0.58	Avg.	5.1	1.48	0.716
	18 samples				31 samples		

Factors $\frac{\text{U.S.B.M. - \%Sn}}{\text{Fearing - \%Sn}} = 0.784$ $\frac{\text{U.S.B.M. - \%}WO_3}{\text{Fearing - \%}WO_3} = 0.81$

TABLE 3. - General data of drill holes

Hole	Collar of hole			Bottom of hole			Length hole, feet	Inclin hole
	Elevation feet	Long.E.	Lat.N.	Elevation feet	Long.E.	Lat.N.		
1	236	4452	5523.5	17.5	4530.0	5727.5	309	-45°
2	239	4622	5458	56.0	4613.5	5627.5	259	-45°
3	277	4853.5	5369	121.0	4882.5	5560	248	-39°
4	317.5	4970	5366	116.5	5052.0	5550	284	-45°
5	242	4454	5924	33.5	4314.5	5769	295	-45°
6	318.5	4286	6013	108.0	4197.0	5822.5	298	-45°
7	318.5	4293	6020	86.0	4160	6210.5	329	-45°
8	267	4445	6093	111.5	4387	6237	220	-45°
9	249	4623	6176	113.0	4595	6309	192	-45°
10	280	4852	5348	-47.5	4943.5	5014	463	-45°
11	292	4890.5	5175	-29	5027	5465.5	457	-45°
12	248.5	4726	5190	-60.5	4820.5	5484.5	437	-45°
14	244.5	4595	6022	-194.5	4409.5	5624	621	-45°
17	411	4069	6375	-72.0	3904	5921	683	-45°
21	312	4740	5977	-51.0	4807.5	6403.5	565	-40°
22	247	4737	5382	-11.0	4677	5134	365	-45°
23	220	4614	5216	-105.5	5076	5267	567	-35°
24	239	4614	5529	-94	4995.5	5638.5	518	-40°
25	241	4555	5824	-16.5	4850	5908.5	401	-40°
26	220.5	4622.1	5134.6	-7.5	4832.6	4964	354	-40°
27	215	4619.7	4667	-50	4790	4870	375	-45°
28	337.5	5003.6	4908.5	-88.5	5081.1	5042.7	453	-70°

Before diamond drilling was settled on as a means of exploration, it was recognized that because of the character of the ore drilling results might not be entirely satisfactory.

Mineralization is spotty and generally occurs in small veinlets or seams. Moreover, earlier exploration had shown the enclosing rocks to be greatly altered and softened in the vicinity of concentrated tin deposition. As the cassiterite occurs in hard, relatively coarse grains or crystals, of which probably 90 percent is plus 200-mesh, whereas gangue minerals such as kaolin and dickite occur unconsolidated and easily pulverized, it was expected that an appreciable percentage of the cassiterite might be freed from the gangue in drill sludges. (See sample 57, table 7.) Its high specific gravity would cause the freed cassiterite to lag and become entrapped in the hole.

During drilling in ore formations, extreme care was observed in water pressures not to disintegrate cores more than necessary. After each pulling of the core the hole was washed clean for some time in an effort to recover all cuttings for the sample. Good water recovery was demanded at all times, except in a few extreme cases. Whenever the return water was lost or it was suspected that the hole was sloughing or caving, drilling was stopped and the hole was cemented or cased. In spite of these precautions, drill logs indicate that there was an appreciable lagging of cassiterite in the sludges.

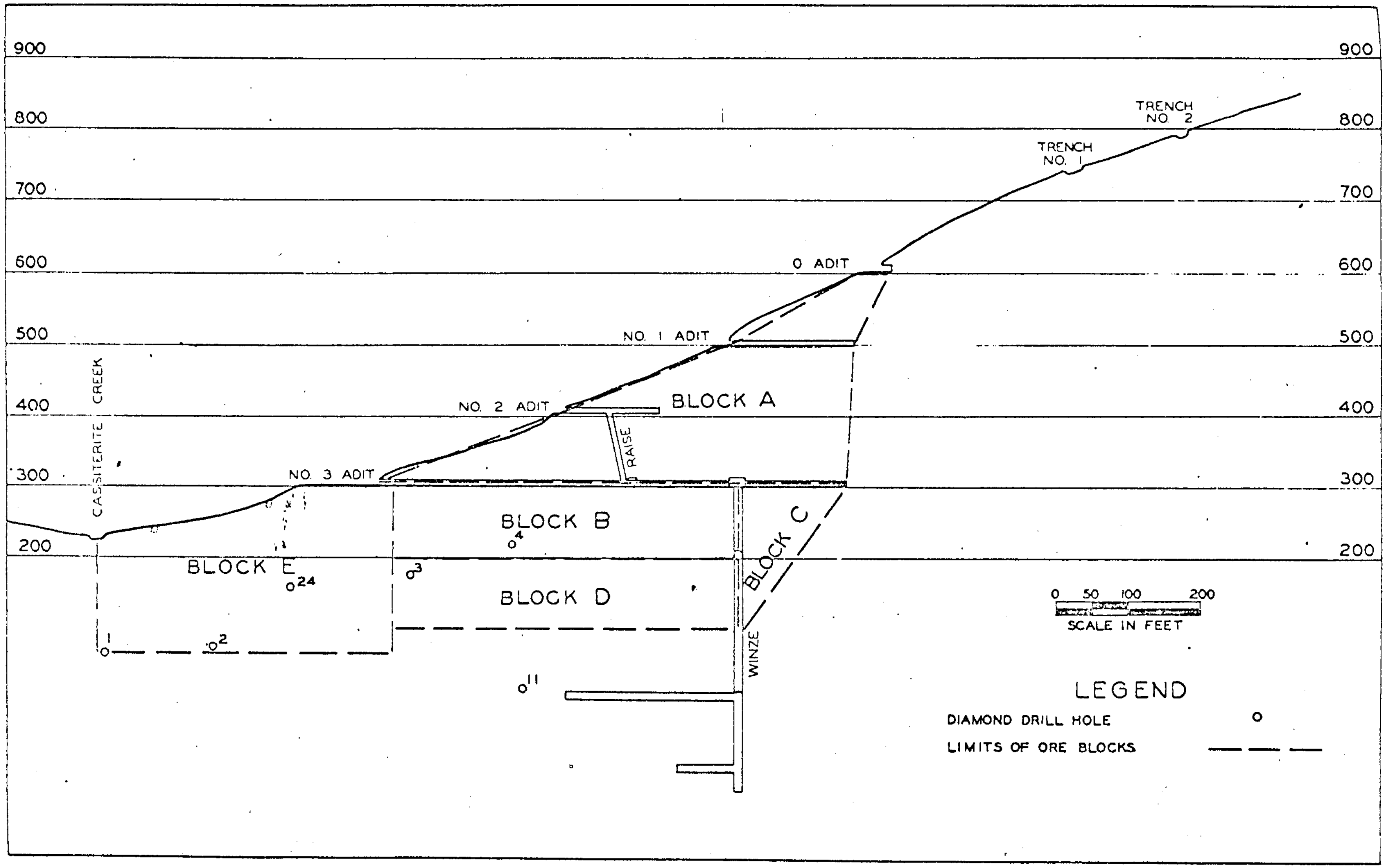


FIG. 39 LONGITUDINAL SECTION CASSITERITE DIKE

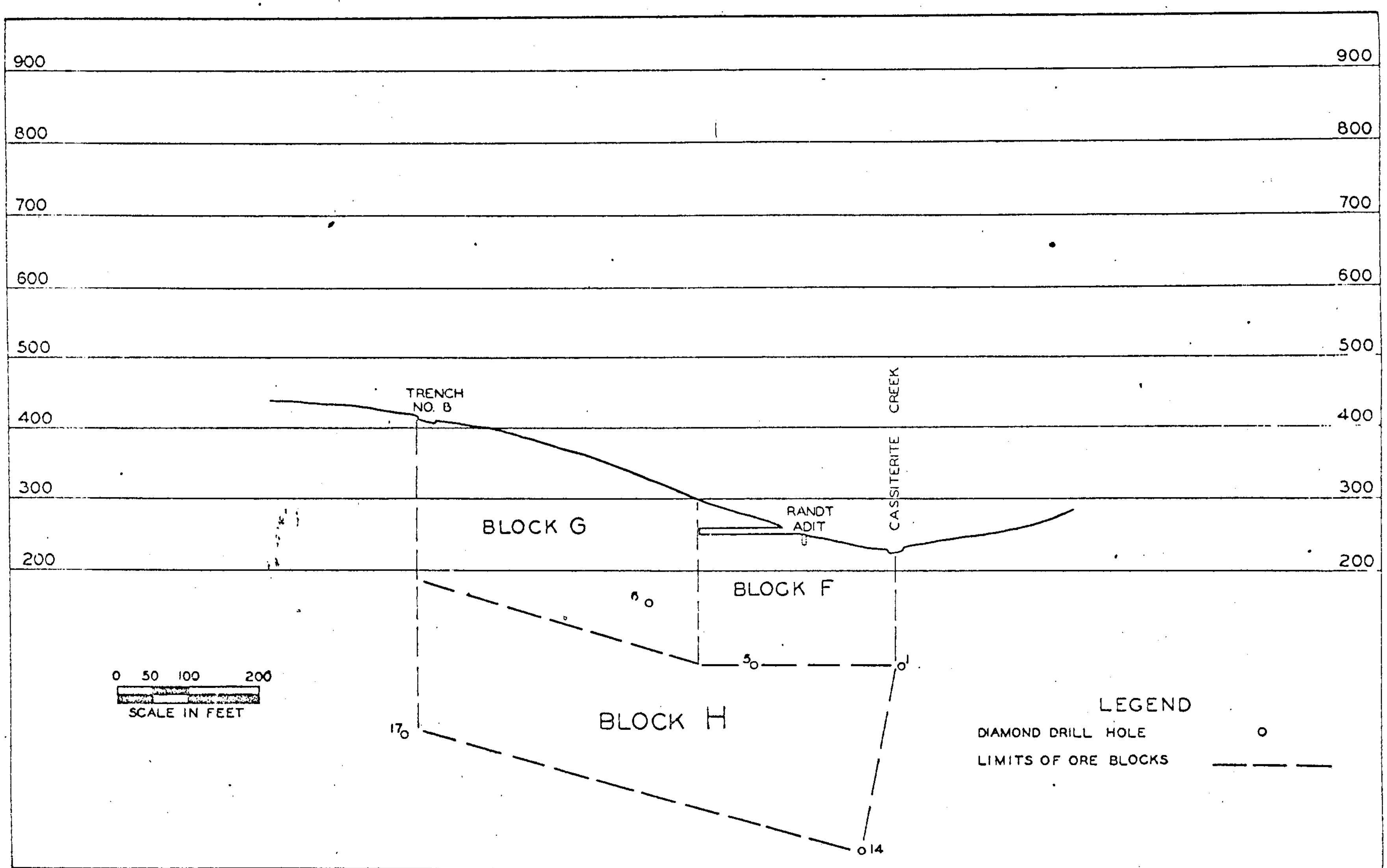


FIG. 40 LONGITUDINAL SECTION CASSITERITE DIKE

Samples from all sources were dried, weighed, and crushed to proper size for splitting on a Jones sampler.

Core samples were analyzed for tin and tungsten by Smith-Emery Co., 920 Santee Street, Los Angeles, Calif. Qualitative spectrographic analyses were made on a few selected samples, as shown in table 7.

Sludge samples were chemically analyzed by the Bureau of Mines at Rolla, Mo., where spectrographic analyses also were made. Table 7 shows a few miscellaneous chemical analyses for fluorite, beryllium, molybdenite, and some base-metal sulfides.

Channel samples were divided between both laboratories.

For the purpose of determining the accuracy of the analyses, duplicate samples were sent to other laboratories. Twenty-one duplicate samples were analyzed by the Bureau at Reno, Nev., and 70 duplicate samples were analyzed by Ledoux & Co., 155 Sixth Ave., New York, N. Y. The results of these check analyses are given in table 4.

Discrepancies in tin analyses are small. Tungsten analyses, on the other hand, are erratic, and some doubt as to the correct results still exists.

A few core analyses were exceedingly higher than corresponding sludge analyses, or vice versa. This is perhaps due to physical factors encountered in the drilling.

Familiarity with the ores and conditions of drilling lead the writer to believe that any possible errors in diamond drill-sample recovery would also be on the conservative side.

A large sample of low-grade material from the granite zone has been obtained and sent to the Rolla laboratory for a beneficiation test.

ORE

Bureau of Mines exploration has shown that tin and tungsten ore reserves exist in the Cassiterite dike (figs. 27, 28, 39 and 40), and table 6 indicates reserves in the granite (see figs. 3, 29, 31, and 32, also tables 8 and 9) and in the contact metamorphic zone (figs. 29 and 30, and table 10). Other potential sources of ore are indicated, notably in Ida Bell dike near its intersection with Cassiterite dike (figs. 33 and 11), in the Quartz-Porphyry dike (figs. 27 and 28), and in the Intermediate dike (figs. 27 and 28). See table 5. Geologic and sampling information on these latter deposits is too scanty to justify estimating any tonnage or grade.

TABLE 4. - Check samples and duplicate assays

Sample	Analyses by U.S.B.M.-Reno		Analyses by Smith-Emery		Analyses by Ledoux & Co.	
	Sn, percent	WO_3, percent	Sn, percent	WO_3, percent	Sn, percent	WO_3, percent
14	3.04	2.11	3.33	2.23		
34A	.08	.01	.09	Nil		
34B	1.30	.21	.41	0.24		
34C	.22	.02	.30	.03		
34D	1.25	-.01	.33	Nil		
34E	.03	-.01	.33	Nil		
35A	1.38	-.01	1.01	Nil		
35B	.17	.07	.59	0.15		
36	.54	.01	.29	Nil		
37A	1.44	.01	1.72	Nil		
37B	.70	.12	.36	0.19		
38A	.28	.27	.23	.30		
38B	.13	.05	.50	.12		
48	.22	.32	.06	.47		
49	.25	3.40	.47	3.63		
50	2.66	1.76	2.53	1.49		
51	3.73	.18	3.86	.57		
52	.42	.48	.47	.22		
53A	.37	.80	.77	1.10		
53B	.22	.31	.20	.28		
53C	.28	.06	.20	.15		
27A			.14	.04	0.29	-0.03
27B			.10	.05	.14	-.03
27C			.11	.07	.11	-.03
28A			.06	.02	.14	-.03
28B			.29	.06	.12	-.03
28C			.20	.05	.15	-.03
28X			.36	.03	.17	-.03
29A			.26	.02	.17	-.03
29B			.09	.02	.21	-.03
29C			.23	.03	.29	-.03
29E			.44	.02	.23	-.03
29F			.53	.05	.30	-.03
30A			.89	Nil	.90	-.03
30B			.53	Nil	1.00	-.03
30C			.74	Nil	1.37	-.03
30D			1.96	Nil	2.09	-.03
30E			.23	0.02	.38	-.03
30F			.74	.02	1.25	-.03
30G			.77		.88	-.03
30H			1.25	.04	1.74	-.03
30I			.59	Nil	1.61	-.03
30J			2.50	0.02	2.68	-.03
30K			2.73	.03	3.01	-.03
30L			1.96	Nil	2.19	-.03
30M			2.73	0.02	3.03	-.03
30N			.59	Nil	.76	-.03
300			.44	Nil	.43	-.03
30P			.98	.07	.20	-.03

TABLE 4. - Check samples and duplicate assays, cont'd.

Sample		Analyses by Smith-Emery, percent		Analyses by Ledoux & Co., percent		Analyses by U.S.B.M.-Rolla percent	
Hole	Feet	Sn	WO_3	Sn	WO_3	Sn	WO_3
Sludges							
23	41- 45			0.37	0.04	0.48	-0.05
26	206-211			1.20	.01	1.05	-.05
26	211-216			2.41	tr.	2.10	-.05
26	216-221			1.36	tr.	1.09	-.05
26	224-229			1.95	tr.	1.78	-.05
26	229-234			1.46	0.01	1.32	-.05
26	310-315			.55	.03		-.05
28	36- 41			.47	.01	.40	-.05
28	93- 98			.61	tr.	.51	-.05
28	98-100			1.03	tr.	.92	-.05
28	100-101			.98	tr.	.82	-.05
28	422-427			.42	0.02	.39	-.05
Cores							
10	381-389	1.09	Nil	1.24	.09		
10	389-394	5.84	0.30	6.06	.22		
12	311-316	.05	Nil	.14	tr.		
12	316-320	Nil	Nil	.21	0.01		
17	590-595	Nil	Nil	.15	.01		
17	595-602	1.13	Nil	1.46	.01		
17	602-608	.10	Nil	.25	.01		
17	608-614	.10	Nil	.37	tr.		
17	614-619	.30	Nil	.43	0.02		
21	514-519?	Nil	Nil	.15	.01		
21	534-539	0.05		.18	.01		
21	549-554	Nil		.10	tr.		
23	19- 21?	1.38	Nil	1.54	0.01		
23	21- 25	.05	0.01	.09	.04		
23	61- 66	1.90	.02	1.91	.02		
23	499-505	5.10	.17	5.34	tr.		
24	85- 88?	.05	1.61	.17	0.90		
24	148-157	.05	.86	.19	.40		
26	270-275	4.48	.38	4.56	.10		
26	310-315	.85	Nil	.93	Nil		
27	267-269	.20	0.21	.20	tr.		
27	269-272	.55	.38	.54	Nil		
27	272-274	.35	.20	.35	tr.		
27	274-277	.05	.32	.15	tr.		
28	87- 93	.25		.34	tr.		
28	93- 98	.35		.68	Nil		
28	266-274	.05		.25	tr.		
28	378-383	.15	0.04	.13	tr.		
28	389-395	.80	.24	.50	tr.		
28	400-405	1.10	.37	2.07	Nil		

\- Less than 0.05

983

TABLE 5. - Locations and analyses of dikes in drill holes

Hole	Elev., feet	Incl.	Incl. dist.	Vert. dist.	Hor. dist.	Elev., feet	Percent Sn	Percent WO_3
Quartz-porphyry lode								
			145	103	103	133		
1	236	-45°	154	109	109	127	0.30	Nil
			192	136	136	103		
2	239	-45°	195	138	138	101	0.20	Nil
3	277	-39°						
			33	23	23	295		
4	318	-45°	47	33	33	285	0.32	0.01
5	242	-45°						
6	318	-45°						
			220	156	156	136		
11	292	-45°	229	163	163	129	0.23	0.12
			180	128	128	120		
12	248	-45°	202	143	143	105	0.25	Tr.
14	244	-45°						
			85	54	65	185		
24	239	-40°	95	61	73	178	0.09	1.12
17	411	-45°						
Trench 17								
Intermediate lode								
			160	114	114	122		
1	236	45°	165	117	117	119	0.18	Nil
			199	141	141	98		
2	239	45°	205	146	146	93	0.28	Tr.
			100	63	78	214		
3	277	-39°	110	69	86	208	0.20	0.03
			57	40	40	278		
4	318	-45°	80	57	57	261	0.18	0.01
5	242	-45°						
6	318	-45°						
			249	177	177	115		
11	292	-45°	275	195	195	97	0.33	0.05
			235	167	167	81		
12	248	-45°	252	179	179	69	0.68	0.07
14	244	-45°						
			117	75	90	164		
24	239	-40°	121	77	93	162	0.10	0.30
17	411	-45°						
Trench 17								

TABLE 5. - Locations and analyses of dikes in drill holes, cont'd.

Cassiterite lode								
Hole	Elev., feet	Incl.	Incl. dist.	Vert. dist.	Hor. dist.	Elev., feet	Percent Sn	Percent WO_3
			236	167	167	69		
1	236	-45°	242	172	172	64	0.44	Nil
			221	157	157	82		
2	239	-45°	239	170	170	69	0.32	0.20
			154	97	120	180		
3	277	-39°	170	107	133	170	0.26	0.16
			128	91	91	227		
4	318	-45°	148	105	105	213	0.62	0.02
			239	170	170	72		
5	242	-45°	251	178	178	64	0.15	0.21
			218	155	155	163		
6	318	-45°	236	168	168	150	0.64	0.09
			340	272	272	20		
11	292	-45°	350	279	279	13	0.08	Nil
12	248	-45°						
			589	418	418	-174		
14	244	-45°	614.5	436	436	-192	0.14	0.31
			131	83	101	156		
24	239	-40°	157	100	121	139	0.14	0.33
			614	436	436	-25		
17	411	-45°	635	451	451	-40	0.27	Nil
Trench 17			16.2				0.35	Nil

TABLE 6. - Summation of cassiterite lode widths and sample analyses

009101

Block	Samples	Normal Width, feet	Percent Sn	Percent WO_3	Remarks	
A	Fearing's	9.51	1.08	0.119	Fearing's calculations.	
B	No. 3 adit 0-100 ft.	9.5	.96	.58	Fearing adj.	
	Winze	11.2	.73	.20	Fearing adj. & Bureau of Mines.	
	DDH. 3	15.1	.26	.16	Angle of intersection=71°V.	945
	DDH. 4	18.8	.62	.02	Angle of intersection = 74°V.80°H.	0.94
C	No. 3 adit 0-200 ft.	11.8	1.47	.15		
	Winze	11.1	.54	.18		
D	100-200 ft.					
	Winze	11.0	.35	.15	Bureau of Mines	
	DDH. 3	15.1	.26	.16	Bureau of Mines	
	DDH. 4	18.8	.62	.02	Bureau of Mines	
	DDH. 11	9.4	.08	Nil	B.M. angle of intersection 74°V.80°H.	0.94
E	Fearing Tr.#2	11.5	.20	1.29	Fearing	
	DDH. 3	15.1	.26	.16	Bureau of Mines	
	DDH. 24	15.6	.14	.33	Bureau of Mines 70°V.40°H.	0.60
	DDH. 2	11.5	.32	.20	Bureau of Mines 56°V.50°H.	0.64
	DDH. 1	4.7	.44	Nil	Bureau of Mines	0.78
	Fearing Tr.#3	10.5	.27	1.46	Fearing	
	Trench #17	16.2	.35	Nil	Bureau of Mines	
F	Randt adit	8.0	.82	.42	Fearing adjusted	
	Trench #17	16.2	.35	Nil	Bureau of Mines	
	DDH. 5	8.3	.15	.21	Angle of intersection = 45°V.75°H.	0.69
	DDH. 1	4.7	.44	Nil	Angle of intersection=51°V.	0.78
G	Trench #8	20.0	1.03	Nil		
	Randt adit	8.0	.82	.42		
	DDH. 6	12.8	.64	.09	Angle of intersection=45°V.	0.71
	DDH. 17	14.7	.27	Nil	Angle of intersection = 45°V.80°H.	0.70
H	DDH. 1	4.7	.44	Nil		0.78
	DDH. 5	8.3	.15	0.21		0.69
	DDH. 6	12.8	.64	.09		0.71
	DDH. 14	12.0	.14	.31	Angle of intersection=28°V.	0.47
	DDH. 17	14.7	.27	Nil		0.71

PETROGRAPHIC AND SPECTROGRAPHIC REPORT ON CORE SPECIMENS

Following is a microscopic laboratory report of the Lost River core samples S-1 and S-53, inclusive, project 607.

The samples submitted for examination were very small in size and in general highly altered. Principally because of this, in most instances the classification of the rocks as to types and origin could not be ascertained definitely by the laboratory study.

A study of the petrographic data showed that cassiterite occurred in appreciable amount only in one sample, S-11, which was estimated at 5 percent by mineral count. Small amounts or traces of cassiterite were found in 10 of the other core samples.

In relative terms only, considerable or appreciable amounts of fluorite were found in 19 of the core samples, and small amounts or traces of fluorite were present in 13 of the samples.

A small amount of beryl was found in sample S-1.

Phenacite (2 $BeO.SiO_2$) was identified in three samples, S-4, S-28, and S-48. Chemical analyses showed 0.22 percent BeO in sample S-28 and 1.49 percent BeO in sample S-48. The BeO analysis of the latter sample when converted to terms of the mineral phenacite amounted to 3.5 percent mineral in the sample. Sample S-4 was not analyzed chemically but the microscopic data indicated that this sample contained a lesser amount of phenacite than sample S-28. Sample S-27 contained 0.08 percent BeO by chemical analysis but the source of the beryllium was not located.

The presence of the beryllium in the five samples was substantiated by qualitative spectrographic analysis.

No scheelite was observed in any of the samples. Wolframite, chalcopyrite, arsenopyrite, and sphalerite were present in some of the cores but in all instances the amounts were low. The lithium-bearing mica zinnwaldite was present in many of the samples but usually occurring in small amounts. Topaz was likewise present in many of the samples.

The table below gives the amounts of cassiterite and fluorite in the core samples in relative terms only.

009705

TABLE 7. - Relative amounts of cassiterite and fluorite in core samples submitted for examination

Samples with considerable or appreciable amount		Samples with small amount or traces		Samples in which none was observed	
Cassiterite	Fluorite	Cassiterite	Fluorite	Cassiterite	Fluorite
S-11	S-1	S-2	S-5	S-1	S-2
	S-3	S-12	S-10	S-3	S-13
	S-4	S-13	S-12	S-4	S-15
	S-6	S-15	S-17	to	S-16
	S-7	S-16	S-18	S-10	S-23
	S-8	S-23	S-26	inc.	S-29
	S-9	S-43	S-30	S-14	S-24
	S-11	S-44	S-32	S-17	S-25
	S-14	S-45	S-39	to	S-31
	S-19	S-52	S-42	S-22	S-33
	S-20		S-46	inc.	S-34
	S-21		S-50	S-24	S-35
	S-22		S-51	to	S-36
	S-27			S-42	S-38
	S-28			inc.	S-40
	S-37			S-46	S-41
	S-48			to	S-43
	S-49			S-51	S-44
	S-52			inc.	S-45
				S-53	S-47
					S-53

S-1 DDH. No. 1, 145-151 feet. Fluorite

The sample was composed essentially of kaolin with considerable fluorite, a small amount of beryl, and a trace of garnet. The kaolin was extremely fine-grained, being 3 to 10 microns in size.

S-2 DDH. No. 1, 236-242 feet.

A soft mixture of kaoline with topaz and chlorite. The sample was slightly stained with iron oxide and contained a trace of cassiterite.

S-3 DDH. No. 2, 149-202 feet. Fluorite

Mostly kaolinite, with considerable fluorite and a trace of quartz. No evidence as to origin was present.

S-4 DDH. No. 3, 100-110 feet.

Essentially iron-stained kaolinite with considerable calcite and fluorite and a trace of phenacite ($2\ BeO.SiO_2$).

S-5 DDH. No. 3, 159-165 feet.

Mostly kaolinite with iron oxide staining, a little fluorite, topaz, calcite, chlorite, and zinnwaldite.

S-6 DDH. No. 3, 165-169 feet.

Crystalline and massive fluorite with considerable kaolin, small amounts of chlorite, zinnwaldite, and a few grains of goethite. Iron oxide staining was abundant.

S-7 DDH. No. 4, 18-23 feet.

Mostly kaolin with considerable fluorite, small amounts of iron oxide staining, chlorite and calcite, and a few grains of wolframite, chalcopyrite, and arsenopyrite.

S-8 DDH. No. 4, 40-47 feet.

Mostly kaolin with fluorite scattered throughout, considerable iron oxide staining, and some manganese coatings. The rock appeared to be a gouge clay; slickensides were present.

S-9 DDH. No. 4, 57-70 feet.

The sample was essentially fluorite in a kaolin matrix. One side of the sample showed slickensided areas. The fluorite showed the slickensides as well as the kaolin, indicating that the fluorite was present before the slickensides were formed.

S-10 DDH. No. 4, 57-70 feet.

The sample was composed essentially of kaolin with minor amounts of fluorite and iron oxide staining. The sample showed slickensides, indicating movement.

S-11 DDH. No. 4, 133-138 feet.

The sample was composed of quartz, kaolin, considerable chlorite, cassiterite, fluorite, topaz, and a sparing amount of wolframite altered to a brown, waddy, manganese oxide. The association of minerals present indicated a highly altered cassiterite vein or dike material.

S-12 DDH. No. 4, 138-148 feet.

The sample was composed essentially of zinnwaldite with quartz, kaolin, and partly kaolinized orthoclase feldspar in considerable amounts. Sparing amounts of cassiterite, fluorite, arsenopyrite, and sphalérite were present also. The rock appeared to be similar in type and origin to sample S-11.

S-13 DDH. No. 5, 239-246 feet.

The rock was made up essentially of calcite, topaz, and kaolinite intimately associated with each other. Very sparing amounts of molybednite, cassiterite, and arsenopyrite were present. One end of the sample showed a slickensided area.

S-14 DDH. No. 6, 220-225 feet.

Essentially fluorite and topaz with lesser kaolin, sparing iron oxide staining, and very sparing amounts of wolframite. The rock was quite crystalline.

S-15 DDH. No. 6, 225-231 feet.

The samples was composed of an altered groundmass of greenish kaolin and quartz with phenocrysts of white kaolinite. Small amounts of calcite, topaz, pyrite, and cassiterite were present.

S-16 DDH. No. 9, 134-144 feet.

Traces of chalcopyrite and cassiterite in a quartz-feldspar gangue. A thin section showed quartz phenocrysts in a crystalline mass of quartz and orthoclase. Sericite was present to a considerable extent.

S-17 DDH. No. 10, 28-32 feet.

Topaz and kaolinized feldspar (orthoclase) with a little fluorite. No opaque minerals were present. The material was rather soft and nondescript.

S-18 DDH. No. 10, 41-49 feet.

Essentially kaolin with small amounts of fluorite, topaz, muscovite, and zinnwaldite. A trace of sphalerite was noted.

S-19 DDH. No. 10, 70-75 feet.

Essentially an iron oxide-stained mass of hydromuscovite, kaolin, and fluorite with small amounts of topaz, muscovite, and zinnwaldite. A few slickensides were noted. At one place, a veinlet of pyrite and chalcopyrite cut through the sample. A little manganese oxide was present as coatings.

S-20 DDH. No. 10, 110-123 feet.

Essentially muscovite, zinnwaldite, and hydromuscovite with considerable fluorite. The material was very fine-grained.

S-21 DDH. No. 10, 180-193 feet.

The rock was made up essentially of fluorite with a considerable amount of chlorite, minor amounts of garnet, and sparing amounts of calcite, topaz, and vesuvianite. A little hydromuscovite also was present. The material appeared to be a contact metamorphic rock.

S-22 DDH. No. 10, 274-295 feet.

The sample was a contact metamorphic rock consisting of fluorite, chlorite, vesuvianite, garnet, calcite, and hydromica.

S-23 DDH. No. 10, 330-342 feet.

A light-greenish mass of soft clayey material, essentially kaolin with lesser chlorite, a little pyrite, and a trace of cassiterite.

S-24 DDH. No. 10, 437 feet.

Coarse-grained quartz with fine-grained kaolin and muscovite. A small amount of topaz and sparing amounts of sphalerite and chalcopyrite also were present. The rock could possibly have been originally a granite, or acid dike material.

S-25 Greenstone Lode, surface.

Essentially topaz in a kaolinized matrix with considerable muscovite and chlorite and sparing amounts of arsenopyrite, molybdenite, pyrite, and sphalerite.

S-26 DDH. No. 11, 33.5-42.5 feet.

The sample was a chloritic schist with a little fluorite, muscovite, and hydromuscovite. It was dark and fine-grained.

S-27 DDH. No. 11, 114-121 feet.

Considerable fluorite was present in an iron oxide-stained mass of kaolin, hydromuscovite, and chlorite. The material was nondescript and resembled a gouge clay. It contained 0.08 percent BeO, but the source was not located.

S-28 DDH. No. 11, 139-148 feet - 5 feet.

Mostly fluorite with lesser chlorite and iron oxide staining. Chemical analysis showed 0.22 percent BeO. A sparing amount of phenacite was observed.

S-29 DDH. No. 11, 219.5-229 feet.

An iron oxide-stained claylike mass of kaolin, muscovite, zinnwaldite, and chlorite. A small amount of topaz was present.

S-30 DDH. No. 11, 258-263 feet.

Iron oxide-stained kaolin, muscovite, zinnwaldite, chlorite, topaz, and a small amount of fluorite and a sparing amount of wolframite.

S-31 DDH. No. 11, 263-268 feet.

A soft mass of muscovite with kaolin, zinnwaldite, and a small amount of iron oxide staining.

S-32 DDH. No. 11, 268-275 feet.

A dark, rather fine-grained, somewhat altered rock composed of muscovite, zinnwaldite, and chlorite with a small amount of fluorite and a trace of molybdenite.

S-33 DDH. No. 11, 339.5-345 feet.

The sample was composed of quartz and topaz with kaolin, chlorite, and soft waddy manganese. Iron oxide staining was present.

S-34 DDH. No. 11, 345-350 feet.

The sample was composed essentially of quartz with a little topaz, muscovite, chlorite, and kaolin. A small amount of iron oxide staining was present.

S-35 DDH. No. 11, 445-451 feet.

Light-colored rock composed essentially of quartz and orthoclase feldspar with a little muscovite. Some of the feldspar was slightly altered to sericite and kaolin. The rock had an aplitic texture, probably of dike origin.

S-36 DDH. No. 12, 60-65 feet.

A soft greenish mass of chlorite, muscovite, and a clay mineral of the montmorillonite group. The material was extremely fine-grained.

S-37 DDH. No. 12, 80-100 feet.

The sample was a nondescript mass of iron oxide-stained kaolin, muscovite, zinnwaldite, chlorite, and hydromuscovite. Considerable fluorite and a small amount of topaz were present.

S-38 DDH. No. 12, 80-85 feet.

Lightly iron oxide-stained zinnwaldite, muxcovite, quartz and kaolin. A sparing amount of wolframite was present. The rock could have been of pegmatitic origin.

S-39 DDH. No. 12, 186-192 feet.

A greenish mixture of kaolin and muscovite with a veinlet of fluorite penetrating it. Iron oxide staining was abundant. The material appeared to be a gouge clay and showed a slickensided area.

S-40 DDH. No. 12, 192-202 feet.

Fine-grained, even granular rock composed essentially of quartz and muscovite, with a small amount of topaz.

S-41 DDH. No. 12, 244.5-252 feet.

Fine to medium-grained mass of muscovite and zinnwaldite. It was highly stained with iron oxides. Manganese oxide also was present as coatings. The association of minerals indicated that the rock may have been of pegmatitic origin.

S-42 DDH. No. 12, 252-260 feet.

Light-green, fine-grained, altered mass of iron-stained kaolin with minor amounts of muscovite, chlorite, fluorite, and topaz.

S-43 DDH. No. 12, 260-269.5 feet.

Massive quartz and topaz with pyrite and minor amounts of cassiterite, chalcopyrite, and sphalerite.

S-44 DDH. No. 12, 269.5-284 feet.

Light-colored, even, fine, granular quartz with lesser topax, kaolin, and chlorite. A small amount of iron oxide staining and sparing amounts of pyrite and cassiterite were present. The data indicated that the rock was cassiterite vein or dike material.

S-45 DDH. No. 12, 381-391 feet.

The sample was a light-colored, somewhat kaolinzed rock of orthoclase, quartz, and zinnwaldite. Small amounts of pyrite and cassiterite were present. The rock resembled an aplite.

S-46 DDH. No. 14, 589-595 feet.

The sample consisted of intergrown, matted, very fine-grained muscovite and zinnwaldite with a small amount of fluorite and a trace of apatite. Iron oxide staining was abundant.

S-47 DDH. No. 14, 595-614 feet.

The sample was composed of quartz and orthoclase with a small amount of zinnwaldite and sparing amounts of wolframite and pyrite. The rock was thought to be granite pegmatite.

S-48 DDH. No. 22, 59-70 feet.

Mostly fluorite and kaolin with minor amounts of topaz, chlorite, zinnwaldite, and phenacite ($2BeO.SiO_2$). Chemical analysis of this sample showed 1.49 percent BeO; this would be approximately 3.5 percent phenacite. The data indicated pegmatitic origin.

S-49 DDH. No. 22, 92-98 feet.

The sample was essentially massive fluorite with kaolin, small amounts of manganese oxide and iron oxide staining, and a sparing amount of topaz. A slickensided area was present on one surface.

S-50 DDH. No. 22, 110-116 feet.

The sample contained considerable pyrite and a trace of sphalerite in a gangue of massive topaz with small amounts of kaolin, chlorite, and fluorite.

S-51 DDH. No. 22, 126-132 feet.

The sample was composed essentially of quartz, topaz, and muscovite, with a small amount of fluorite and a trace of pyrite.

S-52 DDH. No. 22, 210-218 feet.

The sample consisted of a sparing amount of cassiterite in a gangue of soft, white to slightly iron oxide-stained kaolin, with a lesser amount of fluorite. The fluorite was present in small veinlets in the kaolin.

S-53 DDH. No. 22, 316-320 feet.

The sample was composed essentially of kaolin and kaolinzed feldspar, zinnwaldite, and muscovite with a minor amount of topaz. The rock may have been a pegmatite, but it was now a soft mass of kaolin and kaolinzed feldspars with the mica flakes dispersed throughout.

S-56 DDH. 17 - 421.5 feet; mostly calcite with lesser kaolin.

DDH. 17 - 453.5 feet; almost entirely crystalline calcite with included limestone fragment.

DDH. 17 - 580 feet; essentially kaolin with a trace of pyrite, cassiterite, and calcite. Spectrographic analysis shows low Sn, no W.

DDH. 17 - 582 feet; mostly calcite and kaolin with a trace of pyrite and cassiterite. Spectrograph shows low Sn, no W.

DDH. 17 - 597 feet; essentially kaolin with a sparing amount of pyrite, iron oxide staining, and cassiterite. A trace of galena is present. Spectrographic analysis shows high Sn, no W.

DDH. 17 - 622.5 feet; essentially kaolin and calcite with a sparing amount of pyrite, topaz, cassiterite, sphalerite, and arsenopyrite. Spectrograph shows Sn present, no W.

DDH. 21 - 516 feet; mostly sericite, sericitized feldspar, orthoclase, quartz, and topaz. A sparing amount of pyrite is present. The rock is apparently an altered rhyolite. No Sn or W are present.

DDH. 21 - 531 feet; potash feldspar and quartz with lesser pyrite, plagioclase feldspar, and a sparing amount of biotite and topaz. The rock is a rhyolite composed of a microcrystalline granular groundmass with phenocrysts of quartz and feldspar. No Sn or W is present.

DDH. 21 - 553 feet; dense white rhyolite with scattered phenocrysts of quartz. The groundmass is composed of orthoclase, quartz, and sericitizes orthoclase.

DDH 23. - 104 feet; fluorite, orthoclase, chlorite, and garnet with considerable magnetite, cassiterite, galena, sphalerite, and pyrite. The spectrograph shows high Sn, no W, trace of Cd and low In.

DDH 23 - 120 feet; mostly kaolin and zinnwaldite with considerable fluorite and topaz. Some iron oxide staining is present. Spectrographic analysis shows low Sn and no W.

DDH. 23 - 156 feet; essentially kaolin and topaz with lesser muscovite and a sparing amount of sphalerite and pyrite. Spectrographic analysis show low Sn and no W.

DDH. 23 - 183 feet; essentially zinnwaldite and muscovite with a smaller amount of kaolin. Spectrographic analysis shows low Sn and no W.

DDH. 23 - 184.8 feet; essentially kaolin, topaz, and zinnwaldite with a sparing amount of sphalerite. The spectrograph shows Sn present, no W.

DDH. 23 - 187 feet; fluorite, topaz, tourmaline, and kaolin with considerable galena, chalcopyrite, arsenopyrite, and cassiterite. The spectrograph shows Ag and Sn as high plus; no W is present.

DDH. 23 - 236.5 feet; essentially mica (muscovite and zinnwaldite) with considerable fluorite and lesser kaolin. A trace of galena was noted. Spectrographic analysis shows Sn present, but no W.

DDH. 23 - 300.5 feet; fine-grained intermixture of fluorite, chlorite, zinnwaldite, pyrite, and sphalerite. Galena and cassiterite are present. The spectrograph shows high Sn, trace of In, and no W.

DDH. 23 - 330 feet; zinnwaldite, muscovite, fluorite, and lesser topaz with accessory cassiterite and sparing amounts of pyrite, galena, molybnite, and arsenopyrite. A sparing amount of phenacite was observed. Spectrographic analysis shows high Sn, Bi, and Be. No W is shown.

DDH. 23 - 397 feet; essentially fluorite and chlorite with pyrite, cassiterite, and sphalerite. Spectrographic analysis shows high plus plus Sn and W present.

DDH. 23 - 427.5 feet; this rock is essentially calcite, sphalerite, and chlorite with lesser cassiterite, kaolin, and fluorite. A trace of phenacite was observed. Spectrographic analysis shows high Sn and Be but no W.

DDH. 23 - 449 feet; essentially white muscovite with lesser darker zinnwaldite and a trace of topaz and molybdenite. Sn is present; no W.

DDH. 23 - 478 feet; essentially biotite and altered feldspar with calcite widely dispersed throughout the rock. The feldspar is too highly altered to be identified. Sn is low and W is absent by spectrographic analysis.

DDH. 23 - 550 feet; essentially quartz with lesser fluorite, kaolin, and tourmaline and a minor amount of galena, chalcopyrite, cassiterite, and sphalerite. Spectrographic analysis show Sn high, but no W.

DDH. 23 - 502.5 feet; essentially kaolin with lesser fluorite, topaz, and quartz. A sparing amount of cassiterite, chalcopyrite, and sphalerite is present. Sn is high plus and W absent, according to the spectrographic analysis.

DDH. 23 - 508 feet; essentially topaz with lesser fluorite, quartz, galena, sphalerite, chalcopyrite, and a trace of cassiterite. Spectrographic analysis shows Sn present, W absent.

DDH. 23 - 528 feet; this sample is essentially kaolin and topaz with considerable fluorite, calcite, and sphalerite and a minor amount of galena, pyrite, and arsenopyrite. The sphalerite is very dark brown in color. Sn is present; W is absent, according to spectrographic data.

DDH. 23 - 550 feet; essentially quartz and altered feldspar with lesser zinnwaldite. The feldspar is too highly kaolinized to be identified. Spectrographic analysis shows both Sn and W as being low.

DDH. 24 - 32.5 feet; essentially calcite, clay, and fluorite with a sparing amount of pehnacite. The clay is kaolin, highly stained by iron and manganese oxides. Spectrographic analysis shows Be, high plus; Sn, high, and W, absent.

DDH. 24 - 34.5 feet; essentially calcite with considerable manganese oxide stainings and coatings. Some iron oxide staining and sparing amount of clay is present. Spectrographic analysis shows a trace of Sn and no W.

DDH. 24 - 48.5 feet; the rock is a contact rock composed essentially of bands of vesuvianite, garnet, fluorite, and magnetite. A considerable amount of scheelite, calcite, and tourmaline and a minor amount of chlorite is present. Spectrographic analysis shows both Sn and W as being high.

DDH. 24 - 134 feet; essentially calcite and zinnwaldite with manganese oxide staining and coatings. A sparing amount of iron oxide and molybdenite is present. Spectrographic analysis shows both Sn and W as being very low.

DDH. 24 - 152 feet; essentially calcite, topaz, tourmaline, mica, and clay minerals stained by iron and manganese oxide. The clay mineral is a montmorillonite-saponite clay, the mica is zinnwaldite. Sn is shown as a trace and W is very low, according to spectrographic analysis.

DDH. 24 - 257 feet; the sample is essentially zinnwaldite with clay, iron oxide staining, malachite coatings, and a minor amount of quartz and calcite. The clay mineral is montmorillonite-saponite. Spectrographic analysis shows Sn present, W trace.

DDH. 24 - 452 feet; essentially orthoclase with lesser quartz, muscovite, zinnwaldite, fluorite, and tourmaline. Spectrographic analysis shows a trace of Sn and no W.

DDH. 25 - 23 feet; essentially fluorite, topaz, and sericite with a few specks of scheelite. Some chlorite is present. Thin-section study shows the fluorite and sericite intimately intermixed and cut by veinlets of topaz. Some zinnwaldite is present. Spectrographic data shows Sn and W as being present.

DDH. 25 - 319-320 feet; the sample is essentially zinnwaldite with lesser montmorillonite-saponite clay, an appreciable amount of cassiterite, minor amounts of galena and sphalerite, and a trace of fluorite. The cassiterite is a light, pale-yellow color. Spectrographic analysis shows Sn as being high with two pluses and W as being absent.

DDH. 25 - 347 feet; essentially muscovite and zinnwaldite with considerable wolframite and lesser fluorite and potash feldspar. Spectrographic analysis shows Sn low, W high double plus.

DDH. 25 - 352 feet; this rock is a banded fluorite and zinnwaldite rock with considerable galena, sphalerite, and chalcocite. The chalcocite is highly tarnished. Spectrographic analysis shows Sn present, W absent.

DDH. 25 - 385 feet; fluorite, zinnwaldite, and sericitized orthoclase feldspar with a veinlet of sphalerite, galena, and cassiterite. Spectrographic analysis shows Sn with a high plus and W as being absent.

DDH. 26 - 44 feet; fluorite being replaced by biotite, hornblende, and sericite. A veinlet of feldspar and a sparing amount of quartz and magnetite are present. Sn is present and W very low, according to spectrographic analysis.

DDH. 26 - 48 feet; fluorite replaced by feldspar and hornblende, with much sphalerite and lesser magnetite, pyrite, calcite, and garnet. Spectrographic analysis shows Sn low and W very low.

DDH. 26 - 71 feet; essentially fluorite with tourmaline, chlorite, and calcite. A minor amount of sphalerite and cassiterite is present. The tourmaline, chlorite, and calcite are replacing the fluorite. Spectrographic analysis shows Sn high plus and W very low.

DDH. 26 - 121.5 feet; essentially fluorite with calcite and veinlets of chlorite and tourmaline. A trace of galena is present. Spectrographic analysis shows Sn as being low and W as being absent.

DDH. 26 - 207 feet; essentially an intimate mixture of calcite and kaolin cut by stringers of calcite with a sparing amount of topaz and fluorite. Iron oxide staining is present. Spectrographic analysis shows Sn low and W absent.

DDH. 26 - 214 feet; essentially kaolin and chlorite with lesser pyrite, calcite, and a minor amount of cassiterite and arsenopyrite. Considerable iron oxide staining is present. Spectrographic analysis shows Sn as being high, W is absent.

DDH. 26- 216.5 feet; essentially calcite cut by veinlet of zinnwaldite and tourmaline. A sparing amoung of cassiterite and pyrite is present. Spectrographic data show Sn low and no W.

DDH. 26 - 234 feet; on border of soft kaolinite section, contains fine needles of silvery metallic more plentiful in kaolinite section. Essentially kaolin with lesser topaz. A minor amount of pyrite and a trace of cassiterite and galena is present. The silvery metallic needles are galena. Spectrographic analysis shows low Sn and no W.

DDH. 26 - 235 feet; essentially kaolin with fine silvery needles of galena and needles of topaz. Spectrographic analysis of the sample shows very low Sn and no W. Spectrographic analysis of the fine silvery mineral shows high Pb and silver present.

DDH. 26 - 236 feet; the greenish inclusions in the kaolin are made up of fine crystals of dravite (magnesium tourmaline). Some arsenopyrite is present in this sample. Spectrographic analysis shows low Sn and no W.

DDH. 26 - 279.5 feet; essentially quartz with much topaz, minor amounts of pyrite, galena, and sphalerite and a trace of cassiterite. A sparing amount of kaolin is present. Spectrographic analysis shows Sn present, no W.

DDH. 26 - 298.5 feet; essentially quartz, zinnwaldite, and kaolin with a minor amount of topaz, pyrite, and galena. The kaolin is probably altered feldspar. Spectrographic analysis shows Sn present, W absent.

DDH. 26 - 322 feet; essentially kaolin and saponite, zinnwaldite, and chlorite, with lesser fluorite and topaz. A sparing amount of sphalerite, molybdenite, pyrite, arsenopyrite, and quartz is present. Spectrographic analysis shows low Sn, no W.

DDH. 26 - 330 feet; essentially quartz with lesser zinnwaldite and topaz and a sparing amount of kaolin, pyrite, arsenopyrite, galena, molybdenite, and sphalerite. Spectrographic analysis shows low Sn, no W.

DDH. 26 - 331.5 feet; this sample is composed essentially of quartz and zinnwaldite, with lesser topaz and a minor amount of pyrite and sphalerite. The sphalerite is a very dark-brown variety. Spectrographic analysis shows Sn present and W absent.

DDH. 26 - 345 feet; essentially quartz with considerable kaolin, saponite, and zinnwaldite, a minor amount of topaz, and a trace of sphalerite. Spectrographic analysis shows low Sn, no W.

DDH. 26 - 353 feet; essentially quartz, kaolin fluorite, and zinnwaldite, with a minor amount of topaz, tourmaline, galena, chalcopyrite, and cassiterite. Spectrographic analysis shows Sn as high plus, W absent.

DDH. 27 - 182 feet; essentially fluorite with considerable tourmaline and goethite. Much iron-oxide staining is present. Spectrographic analysis shows a trace of Sn, no W.

DDH. 27 - 252 feet; essentially a montmorillonite-saponite clay intersected by a veinlet of zinnwaldite with considerable cassiterite, wolframite, and arsenopyrite. Iron oxide and manganese oxide stainings are present. Spectrographic analysis shows Sn high with two plus signs, W high.

DDH. 27 - 257.5 feet; essentially calcite with considerable montmorillonite-saponite clay, a small amount of tourmaline, and a sparing amount of cassiterite. Spectrographic analysis shows low Sn and no W.

DDH. 27 - 266 feet; essentially calcite with included grains of a clay-like mineral, probably saponite. Some zinnwaldite is present. Spectrographic analysis shows low Sn, no W.

DDH. 27 - 267 feet; essentially saponite and calcite with iron and manganese oxide staining. A sparing amount of cassiterite is present. Spectrographic analysis shows Sn present, W absent.

DDH. 27 - 268.5 feet; this sample is essentially saponite with zinnwaldite. A minor amount of topaz and calcite is present, but no quartz was observed. Iron and manganese oxide staining is present. Spectrographic analysis shows low Sn, no W.

DDH. 27 - 271 feet; essentially saponite with considerable calcite and topaz and a sparing amount of tourmaline. Ore minerals include pyrite, cassiterite, sphalerite, wolframite, galena, and arsenopyrite. No quartz is present. Spectrographic analysis shows Sn as high plus, W high.

DDH. 27 - 276.5 feet; essentially quartz and zinnwaldite, with minor amounts of saponite and topaz and sparing amounts of galena, sphalerite, cassiterite, and pyrite. Spectrographic analysis shows Sn low, W absent.

DDH. 27 - 281.5 feet; essentially quartz and topaz with a little chlorite and tourmaline and minor amounts of pyrite, galena, sphalerite, and cassiterite. Spectrographic analysis shows Sn present and W absent.

DDH. 27 - 301.5 feet; essentially quartz, kaolin, zinnwaldite, and topaz, with sparing galena, sphalerite, and cassiterite. Spectrographic analysis shows low Sn, no W.

DDH. 27 - 313 feet; essentially quartz, kaolin, and zinnwaldite, with lesser topaz and sparing fluorite, pyrite, galena, sphalerite, and cassiterite. Spectrographic analysis shows low Sn, no W.

DDH. 27 - 325 feet; essentially quartz, kaolin, and zinnwaldite, with lesser topaz and fluorite. A sparing amount of sphalerite and galena is present. Spectrographic analysis shows low Sn, no W.

DDH. 27 - 347 feet; essentially quartz, kaolin, zinnwaldite, topaz, and fluorite, with lesser saponite and a sparing amount of arsenopyrite, sphalerite, and galena. Spectrographic analysis shows low Sn, no W.

DDH. 27 - 359 feet; mostly quartz, clay, and zinnwaldite with lesser topaz and fluorite. The clay mineral is identified as being saponite. Spectrographic analysis shows Sn present, W absent.

DDH. 27 - 369 feet; quartz, clay, and zinnwaldite, with lesser topaz and a sparing amount of sphalerite. The clay is a mixture of kaolin and saponite. Spectrographic analysis shows low Sn, no W.

DDH. 27 - 375 feet; essentially quartz, clay, zinnwaldite, and topaz. A small amount of calcite and a trace of sphalerite is present. The clay is a mixture of kaolin and saponite. Spectrographic analysis shows low Sn, no W.

S-57 DDH., No. 10, 387-394 feet sludge.

This sample is essentially zinnwaldite, chlorite, and topaz, with a little calcite and sparing fluorite, garnet, vesuvianite, and tourmaline. Ore minerals include cassiterite, molybdenite, sphalerite, with traces of arsenopyrite, chalcopyrite, and wolframite.

Chemical analysis shows 0.49 percent Sn.

This material is very fine-grained, and much of it is too fine to be definitely identified by microscopic means. A screen analysis made on a sample of the material is shown below.

Mesh	Weight, percent
Plus 100.......	0.3
Minus 100-plus 150.......	1.8
Minus 150-plus 200.......	2.0
Minus 200-plus 400.......	6.6
Minus 400.......	89.3
	100.0

TABLE 8. - Drill-hole location of ore in high-grade portion of granite

Hole			Top of granite				High-grade, top & bottom				Feet		Analyses, percent	
Number	Elevation	Incl.	Incl. dist.	Vert. dist.	Hor. dist.	Elev.	Incl. dist.	Vert. dist.	Hor. dist.	Elev.	From-	To-	Sn	WO_3
10	280	45°	354	251	251	-29	381 394	270 279	270 279	+10 +1	381	394	2.917	0.115
12	248	45°	252	178	178	-62	311 320	220 227	220 227	+28 +21	311	320	0.64	Nil
23	220	35°	484	272	397	-52	499 543	283 309	409 445	-63 -89	499	545	1.401	0.287
26	221	40°	210	134	162	+87	270 C 285 C	172 182	207 219	+49 +39	270	285	2.28	0.227
27	215	45°	267	189	189	+26	267 282	189 200	189 200	+26 +15	267	282	0.28	0.29
28	338	70°	360	338	122	0.00	389 437	365 410	132 148	-27 -72	389	437	0.680	0.388

Pages 40 to 57 Not Copped

983

1091731

009733

TABLE 9. - Summation of drill-hole sample analyses in granite area

			Percent				
Hole	Feet		Sn	WO_3	Ft.xSn	Ft.xWO_3	
10	*0- 22	22	0.15	0.02	3.30	0.44	
	22-123	101	0.30	0.02	30.82	1.87	
	123-161	38	0.08	Nil	3.04	-	
	161-381	220	0.30	0.04	65.14	8.74	
	394-438	44	0.18	0.07	7.83	3.00	
		425	0.24	0.03	110.13	14.05	
10	381-394	13	2.92	0.11	37.92	1.50	
11	0- 50	50	0.32	0.01	16.07	0.40	
	50-114	64	0.11	Tr.	7.21	Tr.	
	114-191	77	0.66	0.13	50.78	10.55	
	191-220	29	0.13	Tr.	3.87	Tr.	Qtz-Porph. at 220
		220	0.35	0.05	77.93	10.95	
12	0- 15	15	0.23	Nil	3.45	-	
	15-100	85	0.46	0.06	38.90	5.60	
	100-186	86	0.15	Tr.	13.18	-	Qtz-Porph. at 186
		186	0.30	0.03	55.81	5.60	
22	*0- 47	47	0.11	0.04	5.20	1.98	
	47-126	79	0.31	0.02	24.83	1.67	
	126-223	97	0.12	0.01	11.95	1.45	
		223	0.19	0.02	41.98	5.10	
23	*0- 19	19	Tr.	Nil	Nil	Nil	
	19-107	88	0.46	0.01	40.82	1.24	
	107-326	219	0.08	0.03	17.44	7.77	
	326-499	173	0.13	0.05	23.10	8.68	
		499	0.16	0.04	81.36	17.69	
23	499-543	44	1.40	0.29	61.65	12.64	
26	*0- 20	20	0.10	Nil	2.40	-	
	20-106	86	0.43	Nil	37.22	-	Sludges
	106-180	74	0.10	Nil	7.75	-	Sludges
	180-270	90	0.52	0.06	46.34	5.71	Cores
	285-325	40	0.23	0.01	9.30	0.55	
		310	0.33	0.02	103.01	6.26	
26	270-285	15	2.28	0.23	34.20	3.41	
27	*0-237	237	Tr.	Tr.	-	-	
	237-267	30	0.12	Tr.	3.50	-	
	282-300	18	0.13	0.16	2.40	2.91	
		285	0.02	0.01	5.90	2.91	
27	267-282	15	0.28	0.29	4.15	4.37	
28	*0- 16	16	?	?			
	16- 21	5	0.24	Tr.	1.20	-	
	21-104	83	0.18	Tr.	15.10	-	
	104-389	285	0.12	Tr.	33.11	0.70	
	*437-453	16	0.17	0.21	2.66	3.40	
		405	0.13	0.01	52.07	4.10	
28	389-437	48	0.69	0.39	33.07	18.62	

*Not used in ore block.

TABLE 10. - Drill-hole location of ore in contact zone and granite

Hole	Elev.	Incl.	Combined top and bottom I. dist.	V. dist.	H. dist.	Elev.	Highgrade top and bottom I. dist.	V. dist.	H. dist.	Elev.
			22	16	16	+264	381	270	270	+10
10	280	-45°	438	311	311	-31	394	279	279	+1
			0.0	0.0	0.0	+292				
11	292	-45°	220	156	156	+136				
			0.0	0.0	0.0	+248	311	220	220	+28
12	248	-45°	186	132	132	+116	320	227	227	-21
			47.0	33	33	+214				
22	247	-45°	223	158	158	+89				
			19.0	11	16	+209	499	283	409	-63
23	220	-35°	543	309	445	-89	543	309	445	-89
			20.0	13	15	+208	270	172	207	+49
26	221	-40°	325	208	250	+13	285	182	219	+39
							267	189	189	+26
27	215	-45°	Nil				282	200	200	+15
			16.0	15	5	+323	389	365	132	-27
28	338	-70°	437	410	148	-72	437	410	148	-72

DEVELOPMENT

Intermittent development was carried on from 1903 until 1930. Since the latter date the property has been idle, and some of the old workings are now inaccessible. The principal mine entrances are reported to have been No. 0 adit, No. 1 adit, No. 2 adit, No. 3 adit, and Randt adit, all on Cassiterite dike; Ida Bell adit on Ida Bell dike; Deveraux adit on Quartz-Porphyry dike; and a small adit and a shaft on the Greenstone lode.

The locations of some of these old workings are shown in figures 2 and 3. At the start of Bureau of Mines explorations only No. 3 adit and Randt adit were partly open. No. 1 adit was reopened a short time after work was in progress.

No. 3 adit was caved at about 550 feet from the portal. In other places rotten timber was reinforced or replaced. However, the timber as a whole has so deteriorated that it is doubtful if the adit will remain open much longer. Timbering is particularly bad over the station and around the collar of the main winze. The adit is about 6.5 feet wide by 7.5 feet high outside the timber.

At a distance of 480 feet from the portal of No. 3 adit, an inclined 2-compartment winze had been sunk to a reported depth of 425 feet below the adit. In July 1942 the water level was found to be at the third level at a depth of nearly 300 feet in the winze. By August of the same year the water had risen to the 200-foot level and remained there for the remainder of the year.

Crosscuts of drifts are reported to have been turned off the winze at 100-foot intervals. In 1942 a 12-foot hanging-wall crosscut was found at the 100-foot level, a 16-foot hanging-wall crosscut at the 200-foot level, and a drift to the west at the 300-foot level. There is reported to be a hanging-wall crosscut, footwall crosscut, drift west on Cassiterite dike, and drift east on Quartz-Porphyry dike at the 400-foot level. These are stated to be, respectively, 80, 80, 75, and 60 feet in length.

No systematic sampling was done below the 200-foot level in the winze, and no assay maps are available for these lower workings. According to one of the owners, only grab samples were taken. A group of such samples was found by the Bureau of Mines engineer in the assay office. The samples were evidently many years old, but some tags were still decipherable. Those that could be identified were examined and analyzed by the Bureau in an endeavor to find some clue regarding development results on the lower levels. The information gained is contained in table 11.

TABLE 11. - Old grab samples from 300- and 400-foot level in winze

Sample	Percent tin	Location		Description
		Level	Distance from winze	
1	0.13	400	West 75 feet	Limestone.
2	.15	300	West 70 feet	Limestone.
3	.60	400	South Drift No. 27	Alt.ls.& min. dike.
4	.23	400	South Drift No. 26	Alt. dike, limestone.
5	.08	300	West Drift at 70 feet	Alt. dike, limestone.
6	.11	300	West Drift at 150 feet	Limestone
7	.25	400	South Drift, No. 25	Min. dike & some ls.
8	.10	300		Limestone.
9	.18	400	South Drift, No. 24	Limestone and dike.
10	2.64	400	South Drift, No. 28	Min. dike & alt. ls.
11	1.07	400	North 60 feet	Min. dike & alt. ls.
12	.60	300	West 40 feet	Limestone.
13	.43	300	West 100 feet	Limestone with sulfides.

Physical characteristics of Cassiterite dike are variable. In some places it is hard and tight to the walls; in others it is comparatively soft and may have gouge along one wall. However, it has been stated by one of the owners that the adits stood open for 20 years without timber. No heavy ground was observed by the Bureau of Mines engineer. In some places decayed timber was falling of its own weight. In the worst-appearing ground, in No. 3 adit, east of the winze, where the dike was highly altered and softened and gouge occurred along the footwall, neither caps nor posts showed the common effects of pressure. Apparently the dike has undergone some slacking and swelling as a result of exposure for many years, but no extensive movement or pressure has taken place.

Inside No. 1 adit, where air circulation had been circumvented by closure of the portal, there was even less evidence of rock pressure. Narrow openings in the dike on the 100- and 200-foot levels in the winze have stood unsupported for years.

009739

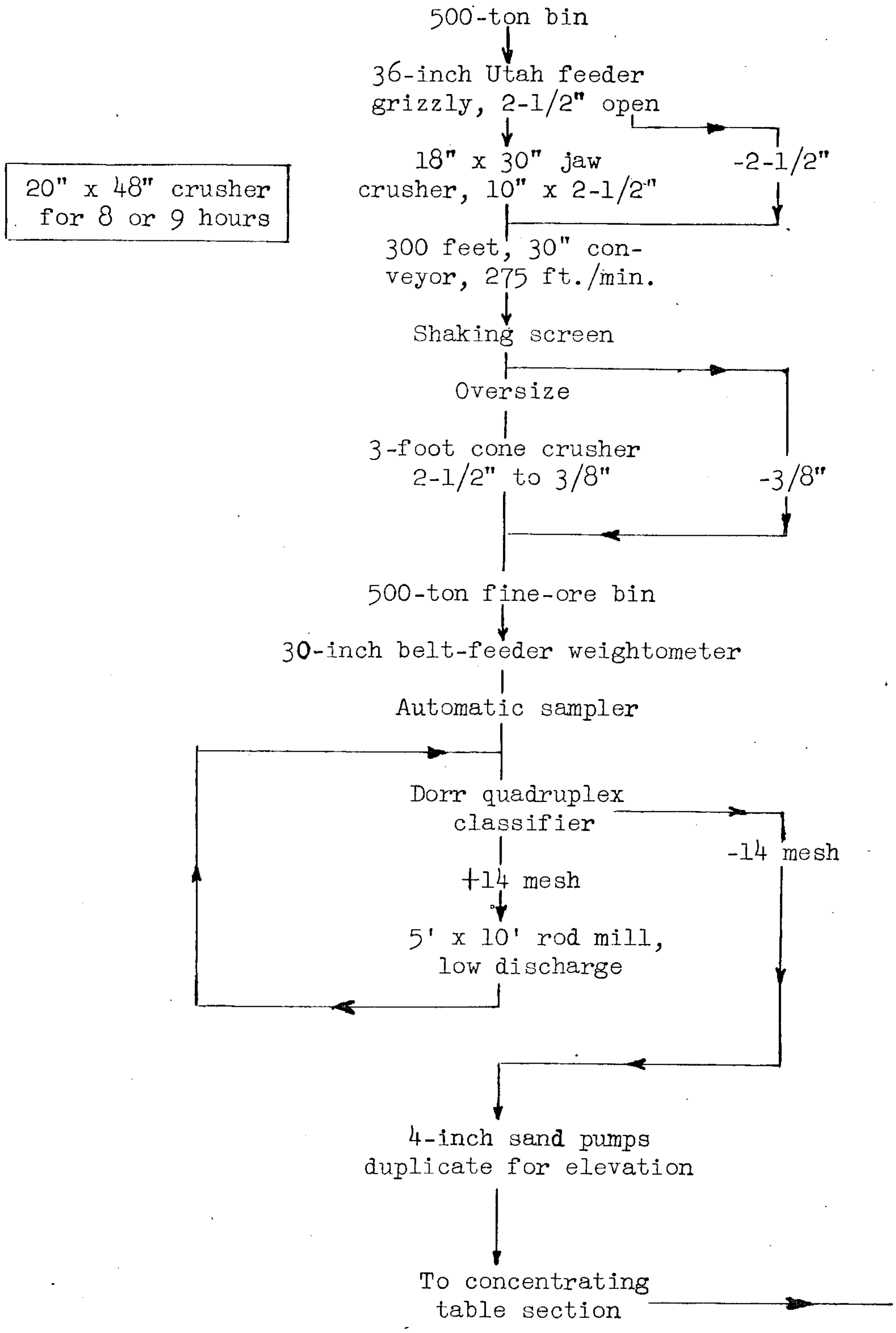


Figure 41. - Crushing section.

009741

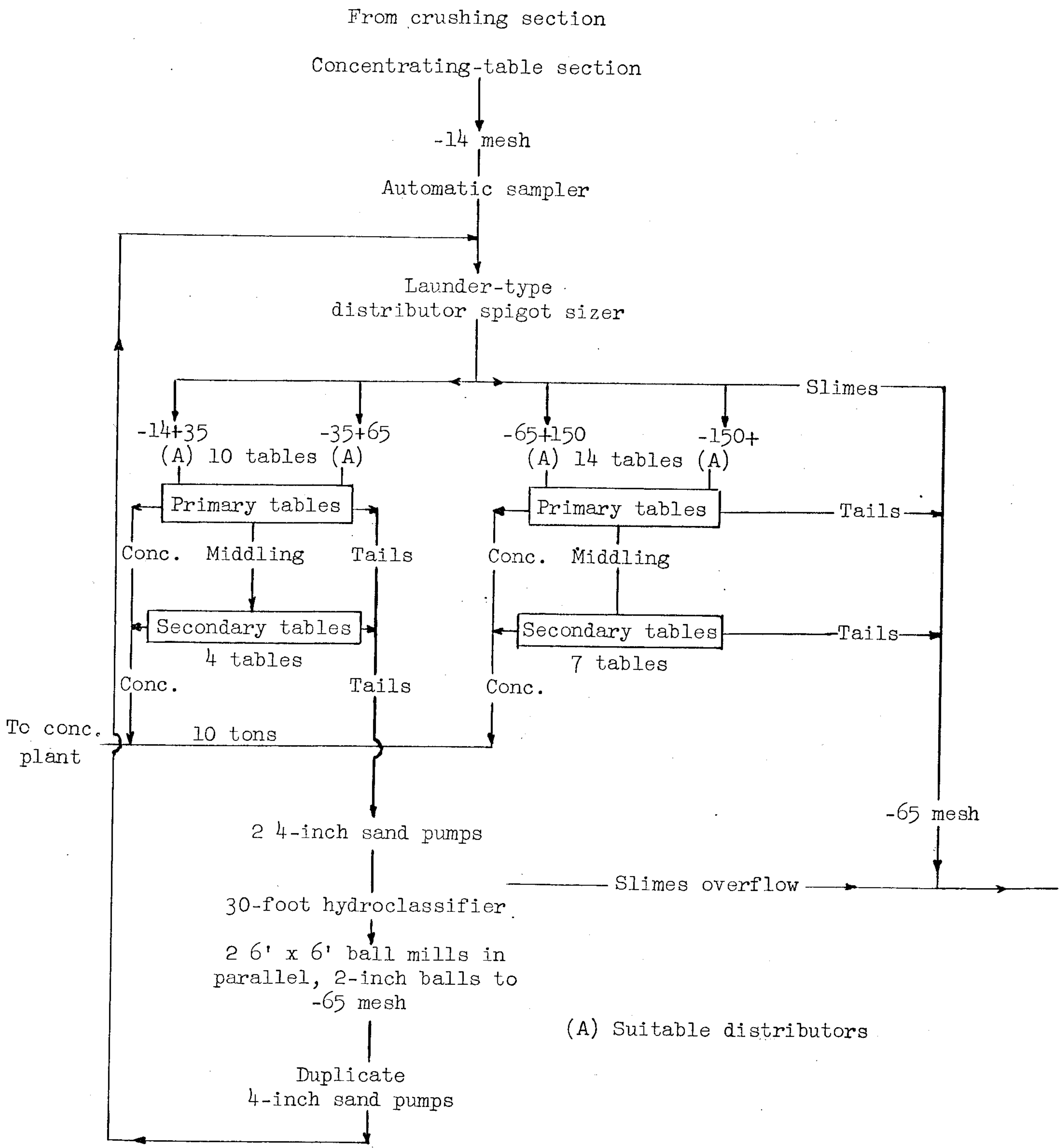
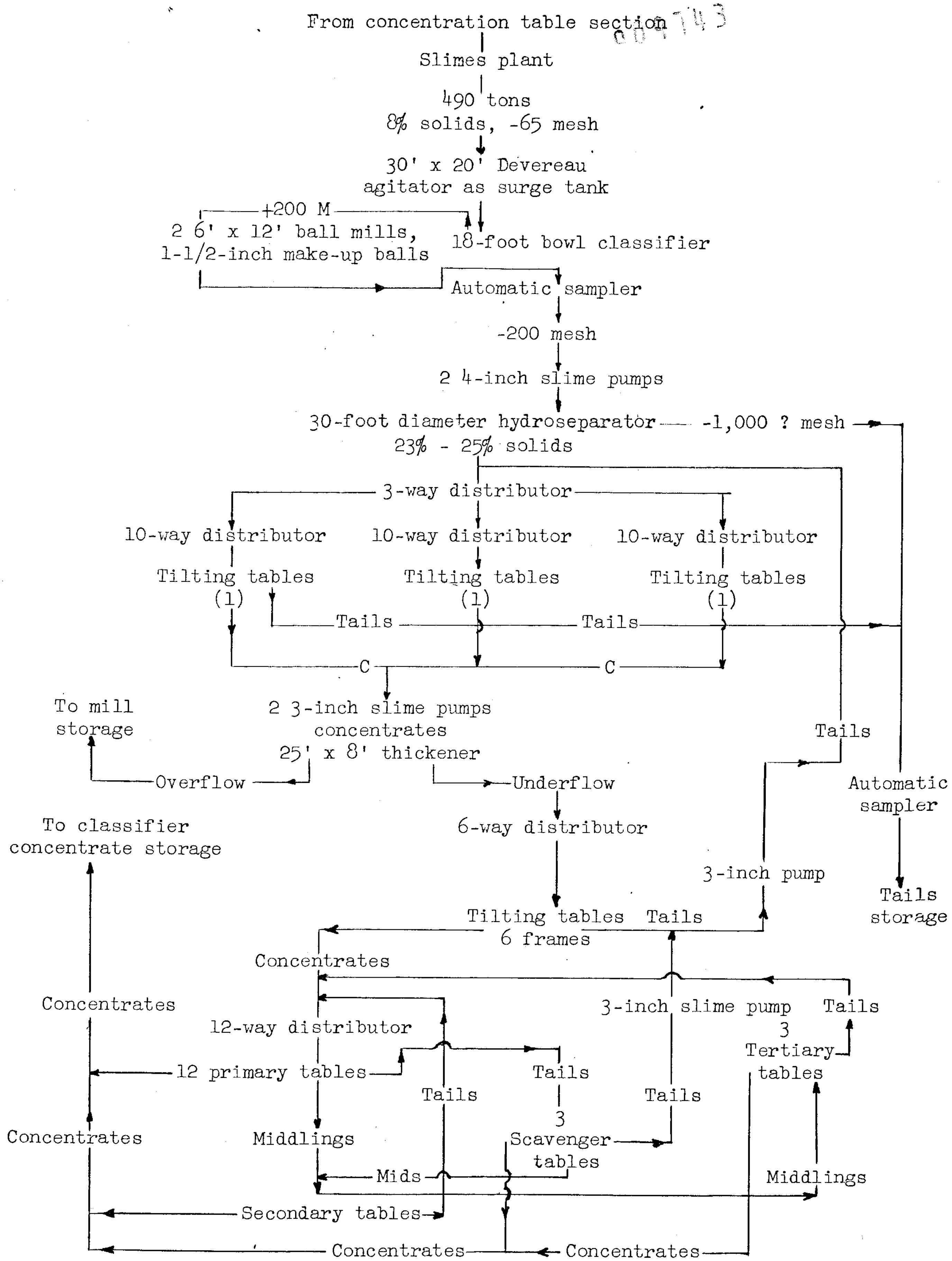


Figure 42



(1) 10 frames of 5 decks to each frame, with 1,800 square feet of blanket frames are synchronized with distributor.

Figure 43

009745

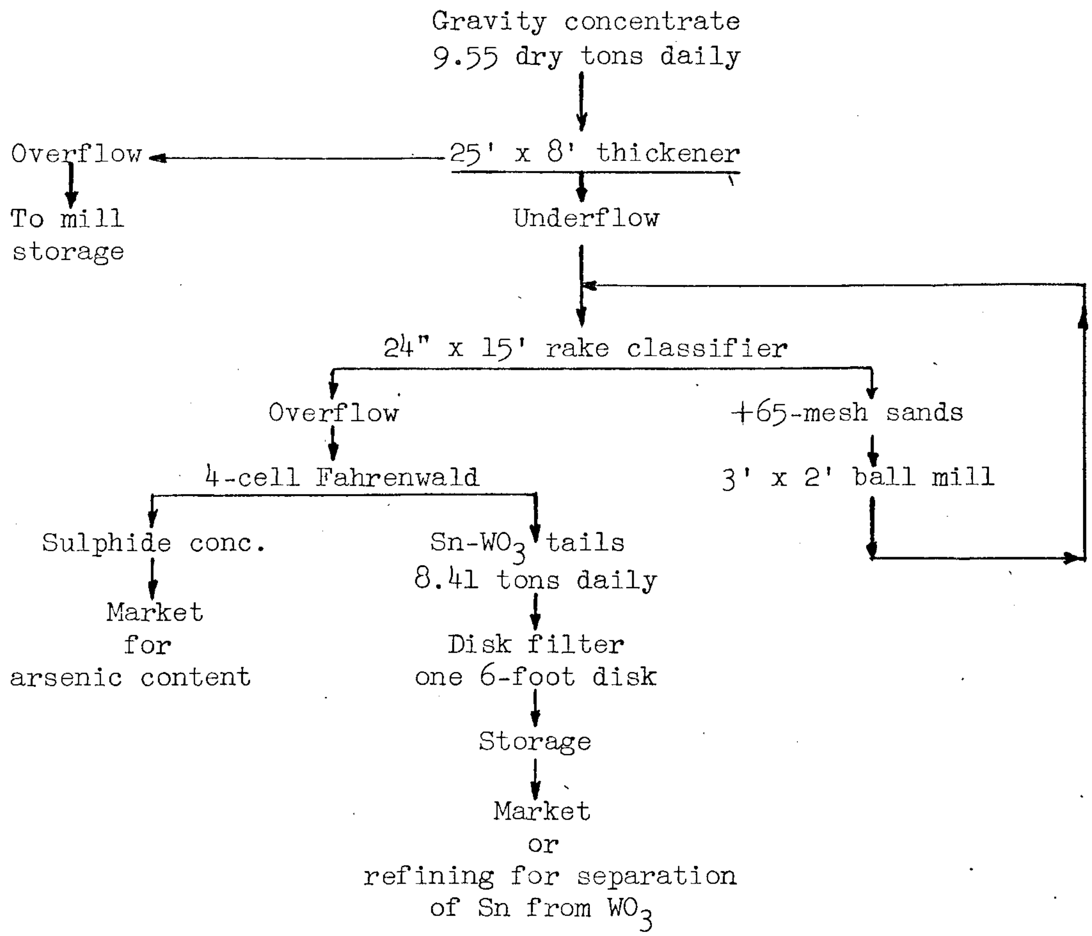


Figure 44. - Flow sheet for flotation of gravity concentrates

Wall rocks are metamorphosed limestone and are free of slips and solution fractures. Thin slabs occur along the hanging wall in places, and these would need to be barred down, but the limestone walls of open stopes would stand indefinitely.

Conditions in the granite ore body are entirely different. The walls or limits of stopes would be determined by economic considerations and not by physical character of the rock. The ore is part of a general zone of intense alteration in which the original granite has been changed in a large degree to a soft, coherent mass of clay and other decomposition products. Excavations in this material would require close, artificial support.

The contact metamorphic zone is composed of a variety of rocks and minerals. It is characterized by numerous intrusives, widespread brecciation, many faults, and almost complete alteration. Metamorphism of the intrusives has destroyed their original physical qualities, and the rock mass as a whole can have very little tensile or compressive strength.

BENEFICIATION TESTS

Beneficiation tests were made on a representative sample of ore from the rhyolite-porphyry dikes. The sample was tested in the Bureau of Mines laboratories at Rolla, Mo. Satisfactory recoveries were obtained, and a flow sheet designed for a tentative 500-dry ton-daily mine production is shown in figures 41, 42, 43, and 44. The flow sheet includes a slimes plant similar to that in operation at the Sullivan tin concentrator of the Consolidated Mining & Smelting Co. of Canada. The Sullivan concentrator is treating ore containing 0.065 percent tin in the tin concentrator and is making a recovery of 45 percent. The percentage of recovery should be higher in the Lost River plant. Within the last few years several South American tin producers have successfully operated similar slime plants.

The following is a report on the beneficiating tests made at the Rolla laboratory.

Ore Dressing Report on Tin-Tungsten Ore from Project 607, Lost River District, Seward Peninsula, Alaska

A description of the samples composited for the metallurgical test follows:

Description of samples composited for testing

Sample description	Weight, pounds	Lode	Source of sample
No. 1 and 2.......	121	Cassiterite.....	No. 3 adit, No. 1 adit, Randt adit, surface trench No. 1 and surface trench No. 8.
No. 3.............	25	Cassiterite.....	Main winze to 200-ft. depth.
No. 4.............	62	Ida Bell........	Surface trench No. 8.
		Greenstone......	Surface trench No. 9
		Quartz-Porphyry.	Surface trench No. 10.

Physical Character:

A microscopic examination of the composite sample showed that the ore was composed of numerous minerals. The minerals of economic value, listed in the order of their abundance, were cassiterite, wolframite, scheelite, arsenopyrite, sphalerite, galena, and molybdenite. The gangue was composed chiefly of topaz and zinnwaldite, with lesser amounts of calcite, fluorite, quartz, muscovite, chlorite, beryl, pyrite, oxides of iron, and clay minerals.

The cassiterite occurred mainly in amber to dark transparent and translucent crystal fragments. Most of it was freed by minus 14-mesh grinding, but part of the cassiterite remained intimately associated with the wolframite and iron oxides.

The wolframite occurred largely as dark-brown to black platy grains. Some was associated with arsenopyrite, pyrite, scheelite, and calcite. Grinding to minus 14-mesh was sufficient to free most of the wolframite.

Chemical Character:

The chemical analysis of the ore for testing was:

Analysis, percent

Sn	WO_3	Fe	Zn	As	Pb	Mo
1.02	0.29	2.79	0.18	0.10	0.10	0.03

The tin and tungsten analyses of the individual samples comprising the composite are:

Sample	Analysis, percent Sn	WO_3
Cassiterite Lode 1...........	1.52	0.44
Cassiterite Lode 2...........	0.80	0.27
Cassiterite Lode 3...........	0.60	0.46
Ida Bell.....................		
Quartz-Porphyry..............	0.60	0.03
Greenstone...................		
Weighted composite...........	1.02	0.29

Treatment Procedure:

1. Table concentration.
2. Flotation of table concentrate.
3. Magnetic separation of tin and tungsten.
4. Chemical separation of tin and tungsten.

Table Concentration

Treatment:

An 85-pound sample of ore crushed through 14-mesh was hydraulically classified into a slime product and four sand products. The approximate screen sizes of the classified products, ranging from coarse to fine, were 14- to 35-mesh, 35- to 65-mesh, 65- to 150-mesh, and 150- to 200-mesh. The two coarsest sand products were tabled to produce finished concentrates and tailings with a high tin content. These tailings were combined and crushed to pass 65-mesh and then hydraulically classified into the next two sizes of sand products and the slime. These sand products were tabled to produce finished concentrates, middlings, and tailings. The middlings were combined, ground through 200-mesh, and mixed with the slime. The slime and middling mixture was tabled to produce a finished table concentrate, a final middling, and a tailing. In normal plant operations this middling would be circulated, the circuit would eventually be equalizing by it passing into either the concentrate or the tailing. The tabling results follow:

Metallurgical data

Product	Weight, percent	Analysis, percent		Percent of total	
		Sn	WO_3	Sn	WO_3
1st concentrate	0.89	32.16	14.60		
1st tailing	(34.50)	(1.13)	(0.16)		
2d concentrate	0.11	40.68	14.80		
2d tailing	(16.30)	(0.36)	(0.01)		
3d concentrate	0.50	45.36	16.40		
3d tailing	38.51	0.23	0.02		
4th concentrate	0.12	40.80	16.40		
4th tailing	20.56	0.15	0.01		
Slime concentrate	0.29	35.40	12.00		
Slime middling	3.04	2.16	0.59		
Slime tailing	35.98	0.12	0.06		
Combined concentrate	1.91	37.14	14.80	75.73	85.90
Middling	3.04	2.16	0.59	7.01	5.44
Combined tailing	95.05	0.17	0.03	17.26	8.66
Calculated head	100.0	0.94	0.33	100.0	100.0

Figures within parentheses do not enter into the calculations, as this material was reground for further treatment, as previously stated. They do serve, however, to indicate the grade of products at various stages of t treatment.

The gangue mineral in the concentrate consisted largely of free particles of topaz, but much of the sulfide content of the ore was also concentrated, for the concentrate contained 6.2 percent Fe. 3.1 percent As, 1.1 percent Pb, 0.3 percent Zn, and 0.10 percent Mo.

Flotation of Table Concentrate

To avoid the tin-smelter and tungsten-refinery penalties on base metals in the table concentrate, flotation of the sulfides preliminary to separation of the tin and tungsten is necessary.

Treatment:

A representative portion of the minus 14-mesh table concentrate was screened on 65-mesh, and the plus 65-mesh material was stage ground in a pebble mill to pass 65-mesh. The primary minus 65-mesh material and the ground fraction were combined to make a pulp of 25 percent solids. After conditioning in a mechanical-type cell, the sulfides were floated and cleaned once. The rougher tailing and cleaner tailing were combined, dried, and reserved for a subsequent tin and tungsten separation. The metallurgical and operating data are:

Metallurgical data

Product	Weight, percent	Analysis, percent							Percent of total	
		Sn	WO_3	As	Fe	Pb	Sn	Mo	Sn	WO_3
Sulfide conc....	11.9	0.42	0.95	30.7	30.4	4.1	1.5	0.1	0.13	0.76
Tin-tungsten tailing......	88.1	42.15	16.70	0.05	2.85	0.7	0.18	0.05	99.87	99.24
Calculated feed.	100.0	37.2	14.8	3.7	6.1	1.1	0.34	0.06	100.0	100.0

Operating data

Reagents	Pounds per ton			
	Conditioner		Rougher	Cleaner
Copper sulfate.....	0.5			
Amyl xanthate......		0.16		
Pine oil No. 5[1]...		0.18		
Time, minutes......	2	2	5	4

1/ General Naval Stores Co.

Some flaky molybdenite remained on the screen after the final grinding stage, and it was discarded. The limited amount of sample did not permit a separation of all the economic minerals of the ore.

Magnetic Separation of Tin and Tungsten

A series of magnetic-separation tests was made on the table concentrate, both with the sulfides remaining and with the sulfides removed by flotation. Separations were poor, and little difference in the results on either product was noted.

A series of oxidizing roasts was made on the flotation tailing to prepare the product for magnetic separation. The samples were heated for 30 minutes at 100° intervals of temperatures from 400° to 1,000° C. The roasted material was treated magnetically to remove the wolframite. Results were poorer than those obtained on unroasted material.

A Frantz high-intensity ferro-filter was used for all tests, as it proved superior to the Stearns type-K separator. The objection to the Frantz ferro-filter is that it has no mechanical motion to release entrapped nonmagnetics and it is not designed for continuous operation.

The results of a typical test on the minus 14-mesh table concentrate without previous sulfide removal or roasting are given below:

Magnetic separation data

Product	Weight, percent	Analysis, percent		Percent of total	
		Sn	WO_3	Sn	WO_3
Magnetic....	15.0	8.8	43.8	3.5	44.1
Nonmagnetic.	85.0	42.7	9.8	96.5	55.9
Composite...	100.0	37.6	14.9	100.0	100.0

Chemical Separation of Tin and Tungsten

Two methods of treatment were used in these tests. In the first series of tests the enriched minus-65-mesh tin-tungsten flotation tailing was mixed with sodium carbonate and sintered at various temperatures. The sintered masses were then crushed and leached with boiling water to dissolve the sodium tungstate; the cassiterite remained as an insoluble residue. In the second method sodium tungstate was formed by digesting the tin-tungsten flotation tailing in a solution of sodium hydroxide. After a sufficient time for digestion the solution was diluted, and the insoluble cassiterite was removed by filtration.

The tin-bearing residues from either type of treatment are salable to tin smelters, and the sodium tungstate solutions are suitable for the usual type of acid treatment used at tungsten refineries to precipitate tungstic acid.

Either the sodium carbonate or sodium hydroxide treatment, if properly controlled, would give a good separation of the tungsten from the tin mineral. The costs would be high, as an excess of sodium carbonate or sodium hydroxide is necessary, and this is lost upon acidification. Table concentration of the flotation tailing would be advisable to reject most of the topaz and to reduce the bulk of material to be chemically treated. Such a procedure was not possible with the limited amount of concentrate produced, but examination of the material showed that much of the topaz could be rejected.

Sodium Carbonate Treatment

Preliminary tests showed that a treatment time of 30 minutes at 850° C. was sufficient to make the wolframite in the flotation tailing-sodium carbonate mixture water-soluble. Higher temperatures tended to fuse the mass and form more sodium stannate, whereas lower temperatures required longer heating periods to complete the conversion of wolframite to sodium tungstate.

A series of sinters was made in an electrically heated furnace using a mix of 10 parts, of flotation tailing and varying the amount of sodium carbonate from 1 part to 20 parts by weight. The larger amounts of sodium carbonate increased the soluble-tin content, whereas smaller amounts failed to react with all of the tungsten. The soluble materials were removed from the crushed sinter by leaching with boiling water for 15 minutes.

The sintering results are summarized below:

Sintering with soda ash

Test	Time	Temperature °C.	Parts by weight		Percent rendered soluble	
			Soda Ash	Flot. Tail.	Sn	WO_3
1	1/2 hr.	850	1	10	0.05	65.2
2	1/2 hr.	850	2	10	0.10	All*
3	1/2 hr.	850	4	10	0.11	All*
4	1/2 hr.	850	6	10	6.13	All*
5	1/2 hr.	850	8	10	6.23	All*
6	1/2 hr.	850	10	10	16.0	All*
7	1/2 hr.	1,000**	20	10	68.5	All*

*The residues contained less than 0.05 percent tungstic oxide.
**The charge melted at this temperature.

The best results were obtained in test No. 2, in which two parts of sodium carbonate were used. This is nearly 300 percent of the theoretical amount required for 100 percent conversion of the wolframite to sodium tungstate; however, 23.5 percent of the sodium carbonate was unconsumed.

Sodium Hydroxide Treatment.

Two tests were made to dissolve the tungsten content of the minus 65-mesh flotation tailing with sodium hydroxide after preliminary tests indicated the conditions to be maintained. A solution of less than 25 percent sodium hydroxide does not give complete dissolution of the tungsten. In tests wherein the tungsten was completely dissolved, solutions were allowed to concentrate by boiling, so an exact figure for strength of solution necessary at a constant volume cannot be given. Evidently the wolframite is attacked rather rapidly by hot sodium hydroxide solutions of the proper strength. The time of digestion was 2 hours. A minimum of 300 percent of the theoretical amount of sodium hydroxide was used, of which 38.6 percent was not consumed. Less initial sodium hydroxide probably would suffice, and equally good results should be obtained by allowing digestion to proceed through evaporation to the pasty stage without allowing the temperature to rise enough to form soluble stannates.

In the first test 12 parts of sodium hydroxide and 40 parts of water (600 percent of the theoretical amount required) were added to 40 parts of flotation tailing, and the resulting pulp was boiled for 2 hours. Water was added occasionally to prevent the final volume from being less than 50 percent of the original volume. The pulp was diluted and filtered, and the filtrate and residue analyzed.

The same procedure was followed in the second test, but only one-half as much sodium hydroxide solution was used. The residue contained an appreciable quantity of tungstic oxide, which probably would have been dissolved had the solution been allowed to concentrate further through evaporation. The results of the two tests are:

Digestion with sodium hydroxide

	Parts by weight					Residue analysis		Filtrate analysis, grams/liter		
Test	Ore	H_2O	NaOH	Residue	Filtrate	Sn	WO_3	Sn	WO_3	NaOH
1	40	40	12	34.0	200	52.7	0.01	0.048	29.8	32.1
2	40	20	6	34.8	200	51.4	.28	.036	26.1	11.6

Percent of total

Residue		Filtrate		
Sn	WO_3	Sn	WO_3	NaOH
99.95	0.05	0.05	99.95	53.5
99.96	1.82	.04	98.18	38.6

Both sodium carbonate and sodium hydroxide probably react with the flotation tailing gangue, hence its removal by gravity tabling would not only lessen the bulk to be treated but would reduce the reagent consumption.

On the basis of the results obtained in test 1, the recoveries of tin and tungstic oxide in the original ore were 73.6 and 85.2 percent, respectively.

Discussion

Several tests were made for the purpose of recovering the tin and tungsten minerals from the crude ore by flotation. None of them were successful.

Attempts to make a separation of the tin and tungsten minerals in the table concentrate by flotation likewise gave unsatisfactory results.

The results obtained by magnetic separation may not represent the best separation possible by this method of concentration. Possibly, by using a wet, high-intensity or a dry, cross-belt type of separator results comparable to those obtained in similar operations in England and South America would be obtained.

Two additional table tests were made, one as a preliminary test and the other to learn whether or not a simplified gravity treatment would give results comparable to those reported herein.

The preliminary test involved crushing through 14 mesh, desliming, and hydraulically classifying the sands into four sand products, which were tabled separately to produce a concentrate, middling, and tailing. The tailings were discarded and the middlings were crushed, deslimed, and tabled with the next smaller fraction. The slime was not treated. The combined concentrate contained 61.8 percent of the tin and 76.2 percent of the tungstic oxide in the feed and analyzed 33.5 percent tin and 16.1 percent tungstic oxide.

In the table test designed to simplify the tabling procedure, a sample of prospect rejects from a commercial analytical laboratory was utilized in lieu of the ore originally received. This material is believed to be more representative of the Lost River deposits, as it was taken by numerous cuttings across the width of several veins. The characteristics and nature of the ore, however, varied but little from those of the sample originally received. The tin and tungstic oxide analyses for the ore were 0.90 and 0.28 percent, respectively. The ore was dry-crushed by rolls through 48-mesh and hydraulically classified into a slime and two sand products. The sands were tabled separately to produce concentrates, middlings, and tailings. The tailings were discarded. The middlings were combined, ground through 150-mesh, and mixed with the slime, and the resultant pulp was tabled to produce a concentrate, final middling, and tailing. The combined concentrate contained 32.6 percent tin and 12.0 percent tungstic oxide; the recoveries were 70.6 and 85.5 percent, respectively. The slime contained considerable tin, which indicated the necessity of stage grinding with tabling to obtain the best recovery.

In the proposed generalized flow sheet, which follows, all the mill feed is ultimately ground through 200-mesh. Better recoveries than any herein reported should be obtained by this treatment along with a slight increase in recovery brought about by circulation of the middling.

The feasibility of omitting the acid treatment of the sodium tungstate-sodium hydroxide solution and substituting an evaporation treatment to selectively crystallize out crude sodium tungstate should be investigated. Sodium tungstate is not as soluble in water as sodium hydroxide. Possibly much of the sodium hydroxide could be reclaimed.

Table Concentration

A preliminary tabling test was made on this ore to determine its characteristics and the general procedure for obtaining optimum recovery. A sample of the ore was crushed to pass a 14-mesh screen and hydraulically classified into a slime and four sand products. The two coarser sand fractions were tabled separately to produce finished table concentrates, middlings, and tailings, which were discarded. The middlings were crushed through 65-mesh and hydraulically classified into two sand products and a slime, all of which were mixed with the corresponding product obtained in the classification of the minus 14-mesh ore. The sand fractions were tabled separately to produce concentrates, middlings, and tailings. The slime was not treated. The grade of the combined concentrates was 33.5 percent Sn and 16.1 percent WO_3; the recoveries were 61.2 percent and 76.2 percent, respectively. The metallurgical data in the preliminary War Minerals Report on this ore were based upon this test.

A second tabling test was performed with the improvements and modifications indicated by the preliminary test. The second tabling test resulted in minor improvement of grade of concentrate, but recovery was increased considerably. The combined concentrate from the second tabling test analyzed 37.1 percent Sn and 14.8 percent WO_3, with recoveries of 75.7 percent and 85.9 percent, respectively.

An 85-pound sample of the ore was crushed through 14-mesh in a jaw crusher and rolls. The crushed ore was then hydraulically classified into four sand products and slime. Proximate screen ranges of these sized fractions were: -14 + 35, -35 + 65, -65 + 150, -150 + 200, and -200. The first and second fractions were tabled to produce finished concentrates and high tin tailings. These tailings were combined and crushed to pass a 65-mesh screen and then hydraulically classified into the third and fourth sized fractions and slime. The third and fourth fractions were then tabled to produce finished concentrates, middlings and final tailings. The middlings were combined, ground through 200-mesh, and then mixed with the slime. The slime was tabled to produce a finished concentrate, a final middling and a tailing. Tabulated tabling results are given below.

Table concentration of Lost River ore

Product	Daily tonnages*	Weight, percent	Analysis, percent						
			Sn	WO_3	Pb	Zn	Mo	As	Fe
1st conc	4.45	0.89	32.16	14.60					
1st tailing	(172.50)	(34.5)	1.13	.16					
2d conc	.55	.11	40.68	14.80					
2d tailing	(81.50)	(16.3)	.36	.04					
3d conc	2.50	.50	45.36	16.40					
3d tailing	192.5	38.51	.23	.02					
4th conc	.60	.12	40.80	16.40					
4th tailing	102.80	20.56	.15	.01					
Middling	15.20	3.04	2.16	.59					
Slime conc	1.45	.29	35.40	12.00					
Slime tailing	179.95	35.98	.12	.06					
Combined gravity conc.	9.55	1.91	37.14	14.80	1.10	0.34	0.10	3.07	6.17
Middling	15.20	3.04	2.16	0.59					
Combined tailing	475.25	95.05	.17	.03					
Head, calc	500.00	100.00	0.94	0.33					
Head, anal			1.02	0.29	(Ratio of concentration 52.4 : 1)				

	Percent of total	
	Sn	WO_3
Com. gravity conc	75.73	85.90
Middling	7.01	5.44
Com. tailing	17.26	8.66
Head, calc	100.00	100.00

*Based upon a daily mill tonnage of 500.

Flotation of Sulfides from Lost River Tin-Tungsten Table Concentrate

To avoid the smelter penalties on base metals present in the table concentrate, a sulfide float preliminary to separation of the tin and tungsten is necessary.

A representative sample of the minus 14-mesh table concentrate was screened on 65-mesh, and the plus 65 was stage-ground in a pebble mill to pass 65-mesh. The ground fraction and the primary minus 65 were combined and pulped to 25 percent solids in a Minerals Separation-type cell and conditioned 2 minutes with 0.5 pound of $CuSO_4$ per ton. At the end of 2 minutes, 0.16 pounds of amyl xanthate and 0.18 pound per ton of methyl amyl alcohol were added, and conditioning continued for 2 minutes more. Roughing took 5 minutes, and the rougher concentrate was cleaned once in a cell of one-half the rougher-cell capacity. The cleaner tailing was added to the rougher tailing and reserved for subsequent tin and tungsten separation testing. The sulfide concentrate weighed 11.9 percent of the feed and analyzed 0.42 percent Sn, 0.95 percent WO_3, 4.1 percent Pb, 1.5 percent Zn, 0.1 percent Mo, 25.5 percent As, and 30.4 percent Fe.

The enriched tin-tungsten tailing was sufficiently low in all constituents to meet maximum tin specifications, but the Pb was over the maximum allowable for tungsten concentrates. Tabulated analyses follow:

Product	Daily ton-nages	Weight percent	Percent							Percent, total	
			Sn	WO_3	Pb	Zn	Mo	As	Fe	Sn	WO_3
Sulfide conc....	1.14	11.9	0.42	0.95	4.1	1.5	0.1	25.5	30.40	0.14	0.74
Sn-WO_3 tailing.	8.41	88.1	42.15	16.70	.7	.18	.05	.05	2.85	99.86	99.26
Composite	9.55	100.0	37.14	14.80	1.1	.34	.05*	3.07	6.17	100.00	100.00

*Some flaky MoS_2 remained on the 65-mesh screen after the final grinding stage and was discarded. Analysis of the flotation feed was 0.10 percent Mo. The high flotability of molybdenite makes it very probably that 90 percent of it would be recovered in the sulfide concentrate.

The gravity section of the concentrator will produce 9.55 tons of tin-tungsten concentrate daily when treating 500 tons of feed per 24 hours.

This concentrate will be enriched by partial removal of the contaminating sulfides in a flotation circuit. This operation will result in a final metallurgical balance, as tabulated below.

Summarized metallurgical data

Product	Dry tons per day	Content, ton		Analysis, percent		Percent of total	
		Sn	WO_3	Sn	WO_3	Sn	WO_3
Heads, calculated......	500.00	4.700	1.650	0.94	0.33	100.00	100.00
Gravity tailing........	475.25	.808	.143	.17	.03	17.26	8.66
Gravity middling.......	15.20	.328	.090	2.16	.59	7.01	5.44
Sulfide concentrate....	1.14	.005	.011	.42	.95	.11	.64
Sn-WO_3 concentrate.....	8.41	3.545	1.404	42.15	16.70	75.62	85.26

Discussion

The flow sheet for the proposed 500-ton tin-tungsten concentrator for the Lost River deposits involves stage grinding, with intermediate tabling, of the entire feed to minus 200-mesh. This should enable the operators to obtain a higher recovery than was possible in the second table test as 65-mesh was the cut-off.

Tin and tungsten loss in the flotation concentrate was slight. The overall recovery in the flotation tailing was 75.6 percent of the Sn and 85.3 percent of the WO_3.

Magnetic separation of the wolframite from the cassiterite in the flotation tailing gave merchantable products, but much of the tungsten remained in the tin concentrate. Tin smelters do not pay for tungsten.

To effect better separation of the tin and tungsten in the flotation tailing, a series of tests is in progress in which leaching is employed. To date it is apparent that much better results will be obtained by this method than by magnetic separation, though the cost will be substantially higher.

In April 1943, the plant of the U. S. Vanadium Corporation, 568 W. 8 S., Salt Lake City, would accept Lost River bulk concentrate on the following basis:

Process for treating all tungsten ores or concentrates is patented. Would be necessary to grind Lost River concentrates to 250-mesh.

Treatment Charges

Start at \$30 per ton for 3 percent WO_3 and increase \$1 per ton for each 1 percent WO_3 up to 15 percent WO_3. Fifteen percent WO_3 at \$42 per ton is the maximum charge.

Payments

Pay at \$30 per unit for 90 percent of WO_3 up to and including 15 percent WO_3; for all WO_3 above 15 percent will pay for 100 percent of WO_3. As example, a 20 percent WO_3 concentrate will pay \$508 Net.

The plant is not equipped to treat cassiterite residue, which would have to be shipped to Texas City.

Tin residues, after extraction of the tungsten, would be acceptable to the Metals Reserve smelter at Texas City, Tex. The concentrate would be free of penalized impurities and would be paid for at the Metals Reserve price for domestic tin.

Beneficiation tests are being made on ores in the granite or contact zone. However, the ores are similar in grain sizes and mineral composition to those of the rhyolite porphyry dikes. Possibly, a larger amount of slimes would result from treatment of granite ores. This would affect the reclaiming of mill water, but over-all recovery of metals should be comparable to recovery from dike ores.

Tests on a composite of reject sludge samples were unsatisfactory, as the material was exceedingly fine; only 1 percent of it was plus 100-mesh, and 78 percent was minus 400-mesh. Additional tests will be made on a composite of diamond drill-core samples from the granite zone.

Mill Water

During the months of November to April, inclusive, subzero temperatures prevail and all surface water freezes. Cassiterite Creek has an ample supply of water for all purposes during the summer months, but has not flow of water in the winter.

Possible sources of water for winter operations are from the mine, from a warm spring below Camp Creek, and from Lost River.

The quantity of water that the mine produces is not known. A rough estimate of the amount encountered in the winze would be 200 g.p.m., but it is questionable if this would be a continuous volume. No measurements have been made of the quantity of flow from the warm spring mentioned. Unless there is a subsurface flow, the spring does not make more than 100 g.p.m. These two sources of water, if dependable, would supply sufficient water, in conjunction with recovery of mill water, to operate a moderately large plant.

Inasmuch as all thickeners would need to be housed against the weather, the cost of buildings and heat would be high. Heating costs in similar climates run from 5 to 15 percent of total mill operating costs.

A dependable supply of winter water is known to exist at the junction of Rapid River and Lost River. No knowledge could be obtained as to the amount of water in Lost River above the junction. Utilization of water from near Rapid River would entail pumping a distance of about 4.5 miles against a head of over 100 feet.

Water has been pumped successfully under similar temperatures in wooden pipes. To accomplish this, when freezing weather commences, the outside of the pipe is sprinkled with water, which freezes and forms an ice jacket. The process is repeated until a coat of ice several inches thick is obtained. Heat from the pipe melts a thin film of inner ice, in which an insulating air pocket is formed.

It is suggested that for any large operation the winter water resources of Lost River should be explored above Rapid River, and that operating plans should include the proposal to pump from the nearest source on Lost River.

GENERAL

A large volume of fluorite rock is contained in the tin ores. Flotation tests to recover calcium fluorite as a byproduct of the tin concentrator were inconclusive, but it is roughly estimated that over 2 million tons of the tin ores, which will average 18 to 20 percent CaF_2, probably could be treated successfully in an auxiliary plant.

The following letter describes results of a flotation test made at the Rolla laboratory for the recovery of fluorite.

> We have completed a flotation test on a composite of the Lost River samples to show the possibility of recovering fluorite from the ore. The sample tested, which contained 28.6 percent calcium fluoride, was a composite of the assay pulps of sludge samples from holes 1, 3, 4, 10, and 12 that were analyzed for calcium fluoride and reported in my letter of March 4 to Heide. These pulps had all been ground through 200-mesh in a pulverizer, and material after this treatment usually is not amenable to flotation.
>
> In the flotation test, 63.3 percent of the calcium fluoride was recovered as a concentrate containing 82.3 percent calcium fluoride, 8.5 percent calcium carbonate, and 0.8 percent silica. A considerable amount of the various minerals in the ore also segregated in the concentrate. Some of them probably could be rejected by gravity concentration and some by selective flotation.
>
> Both the character of the sample and the limited quantity available prevent further testing, but it looks as though something might be done with the ore.

Current prices of fluorspar do not warrant production from the Lost River mine. Freight and lighterage costs are greater than the value of the product. Moreover, seasonal shipping would necessitate storing and rehandling shipments. No docking facilities are available, vessels are not equipped for bulk loading, and concentrates would need to be sacked.

Flotation treatment of the tin-tungsten concentrate would produce a sulfide concentrate containing approximately 25 percent arsenic and smaller percentages of other metals. Further tests may show the possibility of profitably recovering one or more of these metals.

Developmett of waterpower in the Lost River area appears impractical owing to climatic and topographic conditions. No natural fuel resources are known to exist nearer than Candle, about 130 miles from Lost River, where a small amount of coal has been produced. It is doubtful that utilization of this coal for steam power would be cheaper than Diesel power.

As the principal products of the Lost River mine would be tin and tungsten, the value of the mine depends on market prices of these metals.

It is likely that the present price for tin will be maintained several years after the war, as supplies will depend on the rehabilitation of foreign mines. On the other hand, high prices for tungsten have stimulated domestic production and development, and it is possible that the postwar market for tungsten will be oversupplied. Previous to the war, tungsten prices fluctuated widely. Average price per unit WO_3 was \$14.83 in 1936. By 1940 it had risen to \$20.61 per unit.

BIBLIOGRAPHY

COATS, ROBERT R., Supplementary report on the lode tin deposits of the Lost River mine, York District, Seward Peninsula, Alaska: U. S. Geol. Survey Rept. (Unpublished).

COLLIER, ARTHUR J., The tin deposits of the York region, Alaska: Geol. Survey Bull. 229, 1904.

COLLIER, ARTHUR J., Recent developments of Alaskan tin deposits: Geol Survey Bull. 259, 1905, pp. 120-127.

EAKIN, HENRY, Tin mining Alaska: Geol. Survey Bull. 622, 1915, pp. 81-94.

FEARING, FREDERICK C., Alaska tin deposits. Eng. and Min. Jour., vol. 110, 1920, pp. 154-158.

FEARING, FREDERICK C., Report covering the property of the National Tin Mining Corporation. Unpublished.

HARRINGTON, GEORGE L., Tin mining in Seward Peninsula, Alaska: Geol. Survey Bull. 692, 1919, pp. 353-361.

HESS, FRANK L., The York tin region, Alaska: Geol. Survey Bull. 284, 1905, pp. 145-157.

KNOPF, ADOLPH, The mineral deposits of the Lost River and Brooks Mountain region, Alaska: Geol. Survey Bull. 345, 1908, pp. 268-271.

__________, Geology of the Seward Peninsula tin deposits, Alaska: Geol. Survey Bull. 358, 1908.

MERTIE, J. B., Jr. and COATS, ROBERT R., Lode tin deposits of the Lost River area, Alaska: Geol. Survey Rept. (unpublished).

STEIDTMANN, EDWARD, and CATHCART, S. H., Geology of the York tin deposits, Alaska: Geol. Survey Bull. 733, 1922.