LODE-TIN MINING AT LOST RIVER, SEWARD PENINSULA, ALASKA

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by

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R. L. Thorne,[6] and J. A. Herdlick[7]

SUMMARY

Tin mining on Seward Peninsula, Alaska, has been intermittently active since the early 1900's. Most of the recorded production (over 2,200 short tons of metallic tin) has been obtained from placer deposits, but recent production also has resulted from development of lode deposits associated with intrusive rocks occurring in the western part of the peninsula. Generally, the lode deposits containing tin minerals (principally cassiterite) are found in dikes, in granite masses, and in silicate zones in limestone overlying the granite. Both the placer and lode deposits have been of considerable strategic interest to the Government because of the almost total absence of a domestic supply of tin.

The most important tin-lode deposit discovered to date is in the Cassiterite dike at the Lost River mine, on Cassiterite Creek about 85 miles northwest of Nome. This highly altered, rhyolite porphyry dike averages 12 feet in thickness, strikes N. 75° to 85° E., and dips steeply to the south; it can be traced on the surface for about 8,000 feet. Throughout its explored length of about 2,200 feet the dike everywhere contains some tin and tungsten in veinlets and disseminations; economic concentration of the ore minerals, however, is confined to definite shoots. The ore minerals include cassiterite, stannite, hulsite, paigeite, wolframite, scheelite, and base-metal sulfides; gangue minerals include fluorite, boron silicates, and lithium-bearing minerals. Some ore minerals also occur in the numerous other dikes and in the granite and intruded limestone in and adjacent to the mine.

Although the Lost River mine was discovered in 1903 and was extensively developed during several periods of activity, production did not result until Government funds were made available by the Defense Production Act of 1950. Under this act and the stimulus of the Korean War, Government capital was used for purchasing and operating the mine and plant. Rehabilitation was begun May 15, 1951; all operations ceased in October 1955, when Government support was withdrawn. A Defense Minerals Exploration Administration (DMEA) exploration project (90-percent Government participation) was conducted during the operating period and resulted in several important

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contributions toward clearer understanding of the nature and extent of the Lost River deposit; some additional reserves were indicated.

During the operating period company records indicate that the gravity-type mill treated more than 51,000 short tons of ore averaging 1.13 percent tin, from which was produced 687 tons of concentrate averaging 52 percent tin. Tin recovery was 60.8 percent. No tungsten concentrate was produced as the mill was not equipped to make such a recovery. Discrepancies between mill records and smelter returns indicate that the weight figures of the ore and concentrate probably are high. Excessive losses were due chiefly to mechanical limitations of the grinding circuit, which prevented disintegration of ore without excessive sliming of cassiterite.

All the ore treated was obtained from the Cassiterite dike. An advancing system of shrinkage stoping was used, which employed several different methods of stope development, preparation, drilling, and ore drawing; various sizes of stope blocks also were tried.

Cost records maintained by the United States Tin Corp., although not well adapted to a breakdown into basic-cost units, indicate that the approximate cost of mining and processing ore during the 6 months of comparatively stabilized operation (July through December 1954) was $17.56 per ton of ore milled, excluding exploration costs and capital charges but including smelting charges and Seattle office expense. The cost per pound of tin recovered during the same period was $1.33, but the average tin recovery was only 13.25 pounds per ton of ore milled or approximately 60 percent of the tin contained in the heads.

INTRODUCTION

The western part of the Seward Peninsula has been the principal source of primary tin produced on the North American Continent; the area also contains the largest known reserves of lode and placer tin on the continent. Because nearly all tin for United States industry has been obtained from highly vulnerable foreign sources, the Seward Peninsula area has been of particular interest to Government mineral and defense agencies, especially when overseas supplies were threatened. As a result, the Federal Bureau of Mines and the Federal Geological Survey have conducted numerous investigations to indicate the strategic and commercial-production potential of the tin deposits in the area, and other Government agencies have participated in the wartime phases of the program.

In 1941, when United States participation in World War II began to appear imminent, a bill (H. R. 96, January 3, 1941) to appropriate $2 million for exploration of Alaskan tin deposits was introduced into the Congress but was not enacted into law.

In late 1942 and early 1943, after two-thirds of the tin resources and nearly three-quarters of the tin-smelter capacity of the world had been denied to Allied Nations by the Japanese conquest of southeast Asia, the Operating Committee of the War Production Board approved Government construction of a 500-ton mill at the Lost River mine to recover tin from the most promising lode deposit so far discovered on Seward Peninsula. Construction was contingent on noninterference with military shipping in that area. Because of the critical shipping situation brought on by the Japanese invasion of the Aleutian Islands, the Army decided that the production potential of the Lost River mine did not have enough comparative importance to the war effort to justify diversion of badly needed cargo space; consequently, the construction project was not begun.

Under authority conferred by the Stockpiling Act of June 7, 1939, the Bureau of Mines, in cooperation with the Geological Survey, began diamond-drilling and trenching investigations of the Lost River-mine area in August 1942. These operations were continued during the summer seasons until October 1944. During this period the Bureau of Mines completed 7,335 linear feet of surface trenches, an 8,899-linear foot bed of diamond-drill holes, and took 3,393 samples from the mine, trenches, and diamond-drill cores and sludges. Beneficiation studies of ore from the Cassiterite dike were conducted at Bureau laboratories, and a flowsheet for treating the ore was developed. Factual data resulting from the investigation have been published.8/ The Bureau project was reactivated in 1951, then suspended in 1952, when the exploration program was taken over by DMEA.

Under the stimulus of the Korean War and the authorities created by the Defense Production Act of 1950, the Government advanced funds to the United States Tin Corp. for constructing plant facilities, for mine equipment and development and for mine and mill operation from May 15, 1951, to October 1955. Under the same authority, through DMEA, the Government on August 25, 1952, also entered into an exploration contract, at 90-percent Government participation, with the United States Tin Corp. This contract and later amendments resulted in 2,792 linear feet of drifting and crosscutting, 74.5 feet of raising, 1,984 feet of diamond drilling, and 1,795 feet of longhole percussion drilling.

By arrangement with the United States Tin Corp. and the various Government agencies involved, Bureau of Mines engineers made special studies of mining and milling methods and costs at the Lost River mine during the operating period. Through participation in the various exploration programs conducted on Seward Peninsula since 1942, these engineers also were thoroughly familiar with conditions peculiar to mining operations in the area. These general conditions, as well as factual data resulting from the special studies, are presented in this report as aids to possible future mining in the Lost River area.

Because the entire tin-bearing area of western Seward Peninsula is a geologic and economic unit, the discussion of history and production includes information pertaining to the entire area. The data on labor, wages, transportation, weather, and housing requirements would be almost equally applicable to other parts of the area. Much of the milling data would be applicable to other Seward Peninsula tin ores, but those on mining methods and costs would be applicable only to such other deposits as might be closely similar to the ore bodies that have been mined at Lost River.

ACKNOWLEDGMENTS

The cooperation of officials of the United States Tin Corp. in making available the cost and operational information in this report is gratefully acknowledged; special mention is due J. J. Gilmour, secretary-treasurer, and Everett Hougland, manager, for the many courtesies extended to Bureau of Mines personnel by the Seattle and Lost River offices of the company.

Acknowledgment also is made to officials of the General Services Administration (GSA), the Defense Minerals Exploration Administration (DMEA), and the Defense Minerals Production Administration (DMPA) for making information available regarding their particular phases of the Lost River operation.

8/ Heide, H. E., Investigation of the Lost River Tin Deposit, Seward Peninsula, Alaska: Bureau of Mines Rept. of Investigations 3902, 1946, 57 pp.

HISTORY AND PRODUCTION

Tin Mining on Seward Peninsula

Tin was discovered on Seward Peninsula during the first rush of gold miners, following discovery of the rich gold deposits at Nome. It was first recognized by A. H. Brooks, of the Federal Geological Survey, in gold-placer concentrate mined from Buhner Creek (tributary of the Anikovik River) in the summer of 1900.

During the summer of 1901 placer tin was discovered on Buck Creek, across a low divide from the Anikovik River. Production of placer tin from Buck Creek and other creeks draining the slopes of Potato Mountain was begun during the summer of 1902 and continued each summer thereafter for 19 years. Placer-mining operations on these creeks were revived in 1948 and were continued during six seasons until they were discontinued because of uncertainty regarding continuance of the Texas smelter. Although tin-bearing veins have been discovered on Potato Mountain, no serious attempt has ever been made to explore them.

Almost simultaneously with discovery of placer tin on Buck Creek, W. J. C. Bartels discovered lode tin on Cape Mountain. For a number of years thereafter, Bartels and his associates tried to develop a lode mine on Cape Mountain but succeeded in producing only a few tons of concentrate in 1906. Placer tin was discovered, however, in several creeks that drained the slopes of Cape Mountain; consequently, placer-mining operations on two of these creeks were begun in 1924. Between 1924 and 1942 the Cape Creek-Goodwin Gulch placer produced several hundred tons of concentrates. Recent DMEA exploration on Cape Creek have disclosed commercially valuable deposits of placer tin. Other creeks that have not been explored may contain additional reserves.

By 1903 tin had been discovered on Cassiterite Creek (a tributary of Lost River), on Brooks and Ear Mountains, and in streams draining Ear Mountain. Since then, tin or traces of tin have been found in rock specimens or in placer concentrate from almost every part of western Seward Peninsula. The positions of the above-mentioned geographical features are shown in figures 1 and 2.

To the end of 1955 enough tin production to be reported officially had been made from the vicinity of Potato and Cape Mountains and Lost River. The total output from these areas from 1902 through 1955 has been approximately 2,200 short tons of metallic tin.

Most concentrate produced before World War II was shipped to Singapore, but some was shipped to Perth Amboy (N. J.) and some to England. Most, if not all, concentrate produced since 1941 has been shipped to the Government-owned smelter at Texas City.

Tin Mining on Lost River

During the summer of 1903, Charles Randt, Leslie Crim, and W. J. O'Brien found tin-bearing minerals in float or in panning concentrate on Lost River. Assisted by Arthur J. Collier and Frank L. Hess, Federal Geological Survey, they traced the tin to its source in a rhyolite dike on the banks of Cassiterite Creek, a tributary of Lost River. This dike is known as the "Cassiterite dike" or the "Cassiterite lode"; it is the source of nearly all lode-tin production from Seward Peninsula to the date of this report. Most of the placer tin that has been recovered from Cassiterite Creek was probably also derived from the Cassiterite dike.

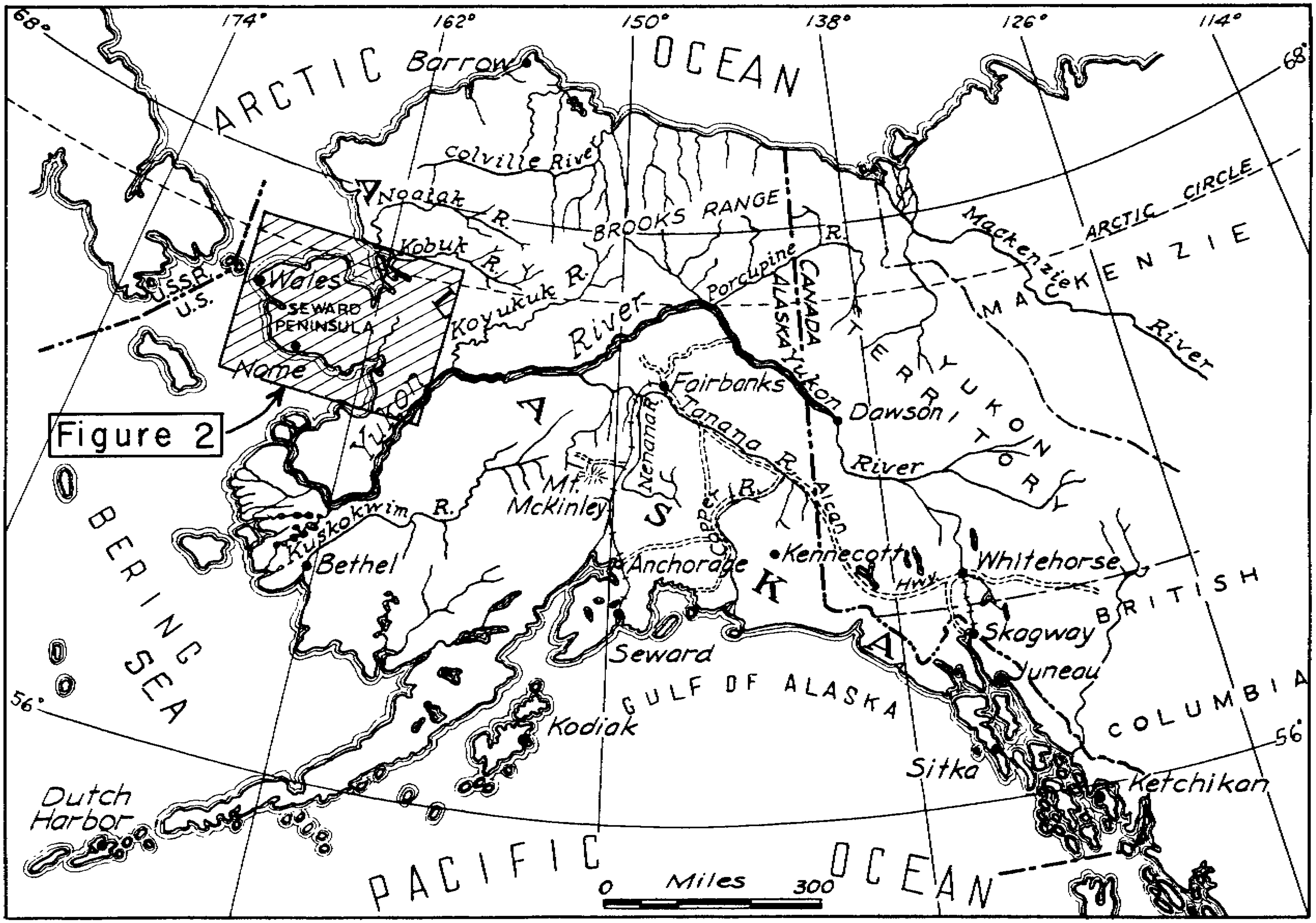

FIGURE 1. - Index Map of Alaska.

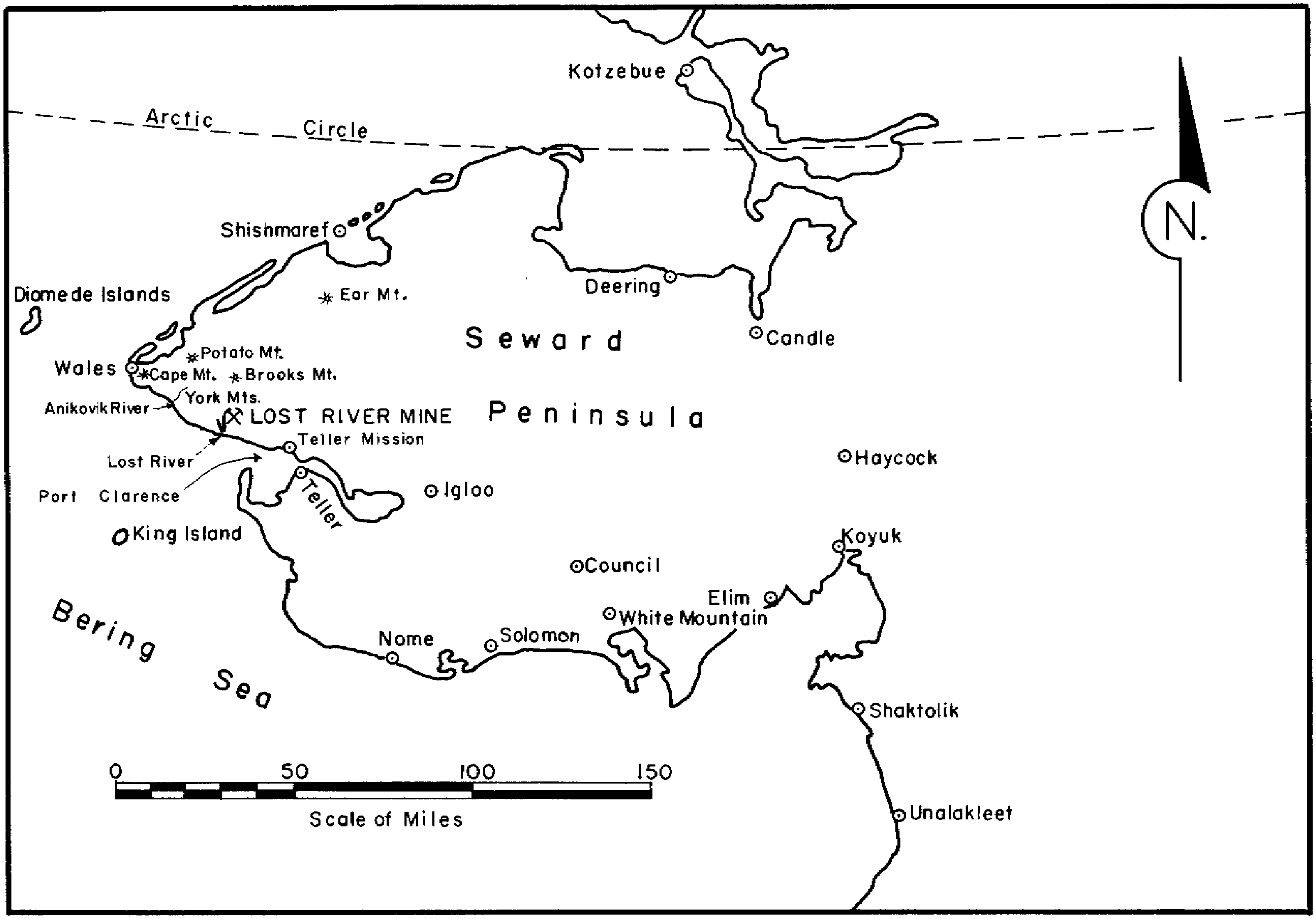

FIGURE 2. - Seward Peninsula.

By the summer of 1904, a 60-foot adit drift along the dike at the point of discovery and a short adit drift on the dike about 200 feet higher than the discovery adit had been completed. During the next 9 years these workings were extended, several other adits were started, and numerous test pits were dug near the Cassiterite dike and at various other prospects along Lost River, Tin Creek, and Cassiterite Creek. This work showed that the Cassiterite dike is at least 8,000 feet in length but that tin mineralization at surface is limited to a strike of about 2,500 feet; it demonstrated also the existence of a highly altered and mineralized area of approximately 100 acres immediately adjacent to the original discovery. In addition, the work showed that lead, copper, zinc, tungsten, and tin mineralization occurs at a number of other places along the upper 6 miles of the Lost River Valley. Meanwhile (July 1907) Randt, Crim, and O'Brien had incorporated the Lost River Tin Mining Co.

In 1913 operations were taken over by the Jamme Syndicate, which constructed a gasoline-driven, 10-stamp mill with Card tables, from which about 5.6 tons of concentrate, containing about 3.5 tons of tin and 0.6 tons of tungsten, was produced during the seasons of 1913 and 1914. During the same period, about 2,000 tons of ore was mined and stockpiled, and probably about 150 tons of ore was milled. By this time about 1,094 feet of drifts on 5 levels and 108 feet of raises had been completed.

In 1916 the property was optioned by W. W. Johnson and his associates, who purchases the materials and equipment for a 50-ton mill. These were delivered to Teller but never moved to the mine. Apparently this option was released almost at once, because no work, except assessment, was done during 1917.

James F. Halpin and his associates optioned the property in 1918 and reactivated exploration during 1919 and the winter of 1919-20. A compressor plant was built and a large warehouse constructed on the beach near the mouth of Lost River, the underground workings were extended, and a shaft was sunk to a depth of 250 feet. Operations were discontinued in the summer of 1920. During the early months of Halpin's option, a detailed examination and report were made by Frederick C. Fearing, consulting mining engineer, of New York City. Fearing's report included detailed sampling data, ore-reserve estimates, and a geological appraisal that inferred the existence of the apex of a tin-bearing granitic intrusive at comparatively shallow depth.

The mine was idle from 1920 to 1928, when an option was awarded the National Tin Mining Co., a Nevada corporation. This company deepened the shaft to about 425 feet below the adit level and completed about 245 feet of crosscut at 294 feet below the adit level; it also completed 100 feet of crosscut at 398 feet below the adit level. Operations ceased in 1930, and title reverted to the Lost River Tin Mining Co.

The Lost River Tin Mining Co. remained inactive until its charter was revoked on January 2, 1942, when title to the claims reverted to W. J. O'Brien of Seattle, Wash., and to the heirs of the Crim and Randt estates - the latter represented by Charles F. Hutson, attorney, of Seattle.

In 1948 the property was optioned by the United States Tin Corp., which initiated placer-mining operations on Cassiterite Creek in the summer of 1949. These placer-mining operations were continued during the summers of 1950 and 1951 and resulted in a production of concentrate that contained approximately 93.4 tons of tin. Meanwhile (1951) the company obtained financing from DMA (later Defense Materials Procurement Administration) and began constructing a mill, which from early 1952 to October 1955 produced concentrate containing 309 short tons of tin. Because income at current metal prices did not meet costs, the operation was discontinued September 26, 1955.

LOCATION AND ACCESSIBILITY

The Lost River mine is 70 miles south of the Arctic Circle at latitude 65°31' N., longitude 167°9' W., near the westernmost extremity of Seward Peninsula - the westernmost extremity of the North American continent (fig. 1). It is on Cassiterite Creek, approximately 1 mile above the junction of Cassiterite Creek with Lost River, 400 feet above sea level, and 6-1/2 miles due north from the Bering Sea (fig. 2).

Lost River and its tributary, Cassiterite Creek, drain the southern flank of the York Mountains. From its juncture with Cassiterite Creek the valley of the Lost River forms a natural route from the mine to the sea (see figs. 1 and 2); a gravel-surfaced road has been built down this valley.

Nome, Alaska, is the nearest important trading center. The airline distance southeasterly from the flying field at Lost River mine to Nome is 85 miles; the distance by sea from the mouth of Lost River to Nome is approximately 95 land miles. Overland communication between Lost River and Nome is not practicable, except by winter sled road; however, the Territorial Department of Roads has begun constructing an 85-mile road from Nome to Teller. From Teller Mission, on the north side of Port Clarence, an 18-mile road would connect with the existing road from the mouth of Lost River to the mine; this road could be built, for summer use only, along the sandbars that parallel the north shore of Port Clarence. The cost of this road, including one 40-foot bridge and one short ferry, was estimated by the Alaska Road Commission in 1951 to be $85,000.

Port Clarence is the only sheltered anchorage along the western coast of Seward Peninsula, but it is unimproved. If traffic becomes heavy enough to justify the expense, this harbor or possibly its inner harbor, known as Grantley Harbor, could be dredged and otherwise improved to provide docking facilities for western Seward Peninsula. Meanwhile all freight must be lightered ashore from ships anchored along the open coast; obviously this restricts the lightering operations to periods of calm weather.

This part of the Seward Peninsula coast is free of ice from some time in June to late October or November only (see table 1); consequently all heavy freight shipments must be landed during this period. Normally, three regular steamship sailings a season are made from Seattle to the Bering Sea; the first is in June or early July, depending on ice conditions, the second in midseason, and the third in September for deliveries along the Bering Sea during October. These ships deliver general freight to Nome and to other places along the Bering Sea where the volume of freight is large enough to justify lightering; therefore, mining equipment and bulk supplies, except fuel oil, may be delivered directly from Seattle to the beach at the mouth of Lost River (2,960 land miles). All fuel oil for the area is distributed through local dealers at Nome or Kotzebue, who deliver the oil by barge to the various consumers along the coast; ocean-going tankers could, however, deliver directly to any place on the coast where consumption is large enough to justify competitive shipments from Seattle.

Personnel, mail, light freight (up to about 1 ton, depending on dimensions), and perishable goods are transported from Nome to Lost River by air. When the mine was being operated, daily flights by light, single-engine planes were made except during exceptionally bad weather; during the winter the flights may be impracticable for 1 to 2 weeks. Nome is served by daily flights of large planes from Anchorage and Fairbanks, where overnight connections with stateside flights can be made.

The normal plane service between Lost River and Nome is by single-engine planes, which usually land at a small airfield near the junction of Cassiterite Creek with Lost River, about 1 mile from the mining camp. This field, about 1,000 feet long, could be enlarged, but heavy planes could not maneuver safely in the Lost River Valley. Another landing field about 5,000 feet long has been built near the beach west of the mouth of Lost River; this field is suitable for DC-3's (or larger planes with equivalent landing and takeoff requirements), which may be chartered at Nome for transporting larger shipments or heavier pieces. The principal limitations on single pieces of freight are the shape, size, and concentrated weight that will permit loading and unloading without damage to the plane.

PROPERTY AND OWNERSHIP

As constituted in 1955, the realty holdings of the United States Tin Corp. included 15 patented lode-mining claims, 3 patented placer claims, and 33 unpatented lode claims; in addition, the company held purchase options on the 6 patented lode claims comprising the Bessie-Maple group.

The various claim groups held by the company are listed as follows:

Patented lode claims

(U. S. Mineral Survey 1234, Patent S73035)

Surveyor	3 prospectors	Rob Roy.
Shon Rue	Collier	Jenny Lynn.
Klondike	Mars	Triangle.
Bald Eagle	Jupiter	Lincoln.
Carry Gow	Green	Engineer.

Patented placer claims

(U. S. Mineral Survey 1235, Patent S13326)

Thrassa	Margaret	Gertrude.

Patented lode claims (optioned)

(U. S. Mineral Survey 1243, Patent S73037)

Maple	"J"	Poorman.
"I"	Bessie	Tiger.

Unpatented lode claims

Name	Date	Volume	Page	Name	Date	Volume	Page
Sandra K	8/31/51	228	152	No. 6	2/16/53	228	164-5
Penny S	8/31/51		154	No. 1	2/16/53		164-5
Patsy K	8/31/51		153	No. 2	2/16/53		164-5
U.S.T.S.	9/4/51		155	No. 3	2/16/53		164-5
U.S.T.N.	9/4/51		154	No. 4	2/16/53		164-5
U.S.T.X.	9/12/51		166	No. 7	7/21/51		131
U.S.T.K.	8/31/51		150	No. 8	7/21/51		131
U.S.T.L.	8/31/51		150	Dalcouth No. 1.	10/16/52		246-7
U.S.T.M.	8/31/51		151	Dalcouth No. 2.	10/16/52		240-2
U.S.T.W.	9/4/51		156	Dalcouth No. 3.	10/16/52		242
Fog	9/12/51		165	Cogruk	10/16/52		245
Blizzard	9/12/51		164-5	Beluga	10/16/52		248
Kenny	10/16/52		164-5	Scotty	2/16/53		252
Dale K	10/16/52		164-5	Harry	2/16/53		252
Cy. S	10/16/52		164-5	Bench No. 1	2/16/53		254
Butch	2/16/53		164-5	Bench No. 2	2/16/53		256
No. 5	2/16/53		164-5				

All claims are recorded in the office of the United States Commissioner, Cape Nome recording precinct, Nome, Alaska.

The real and personal property of the United States Tin Corp. was offered to the highest bidder at the United States Marshal's sale held at the mine office October 30, 1957, to satisfy a judgment in favor of the United States of America. No acceptable bid was submitted; therefore, the Government took possession of the property.

CLIMATE

The climate is subarctic, but temperature extremes are prevented by the moderating influence of the Bering Sea, only a few miles from the mine. Within the 5-year period 1950-54, the lowest recorded temperature has been minus 40° and the highest 72°. Temperatures may be expected to remain at 0° or below for about 100 days in January, February, and March; on the other hand, about 100 days of such freezing weather may also be expected during June, July, August, and September; during the remaining months of spring or fall the temperature ranges between 0° and freezing. The coast is free of ice from some time in the latter half of June to about the middle of November. Although the average annual precipitation is only about 11 inches and the average annual snowfall less than 3 feet, strong winds from the Bering Sea and the Arctic Ocean cause very unpleasant, driving, cold rains during the summer and severe drifting during the winter. The nearly continual, strong, cold winds combined with rain or dry snow, render outside work nearly always uncomfortable and, in winter, almost impossible. Fortunately, the Eskimos are accustomed to it.

The drifting snows create difficulties with maintenance of roads or paths in winter but eliminate any problem with regard to snow on buildings, because the snow is blown off the roofs.

Although the weather at Lost River has not been recorded officially, close approximation should result from an average of the United States Weather Bureau records at Teller and at Cape Wales, about 50 miles southeasterly and 20 miles northwesterly, respectively, from Lost River; the average at these 2 stations for the 5-year period is given in table 1.

TOPOGRAPHY AND VEGETATION

The Lost River mine is in the east central part of the York Mountains, which occupy an area about 16 miles in length, west to east, and about 10 miles in width, north to south. These mountains are steep-sided but, except on the highest peaks, are well rounded. The rounded ridges and most of the slopes are covered with finely disintegrated detritus; consequently, bare outcrops of bedrock are scarce. The summits, almost uniformly, are about 2,000 feet above sea level, although Brooks Mountain, the highest peak in the range, rises to 2,918 feet. The several streams and small rivers that drain the mountains rise to altitudes of less than 1,000 feet and flow through relatively broad, straight valleys on gentle gradients; the lower parts of these streams are braided.

No vegetation but moss and lichens grows in those parts of the area that are underlain by limestone detritus; a few small willows grow on the gentler slopes and in sheltered valleys, which are underlain by slate or by slate detritus. Most of the York Mountains, except the western flanks of the range and a small area on the north slope of Brooks Mountain, are underlain by limestone; consequently, they are completely barren.

TABLE 1. - United States Weather Bureau records
(average of Teller and Wales)

Average monthly temperatures 1950-54, inclusive

Year	Jan.	Feb.	Mar.	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.
	Degrees Fahrenheit											
1950	15.5	-1.7	3.9	16.2	26.2	38.8	50.6	50.8	42.6	34.1	24.2	10.3
1951	-10.4	-4.0	-3.4	16.3	32.7	43.5	44.9	45.5	40.7	28.9	22.8	6.4
1952	-3.2	-3.9	1.9	9.1	26.8	40.7	46.1	48.5	40.1	31.3	21.6	6.5
1953	-4.3	-7.8	-4.9	14.9	31.8	42.9	48.3	46.8	41.8	28.6	18.5	-1.5
1954	-.4	-11.1	6.9	18.3	35.9	42.1	46.1	48.1	41.5	30.2	16.5	-6.8
Average..	-.6	-5.7	.9	15.0	30.7	41.6	47.2	47.9	41.3	30.6	20.7	3.0

Average annual temperature, 22.7° F.

Average monthly precipitation 1950-54, inclusive

Year	Jan.	Feb.	Mar.	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.
1950	0.46	0.40	1.02	0.44	0.27	0.97	1.26	2.72	2.33	1.71	0.95	0.58
1951	.20	.39	.04	.34	.24	.63	.87	5.19	.70	.37	.83	.96
1952	.78	.17	.43	.20	.54	.28	1.64	2.31	.86	.99	.79	.52
1953	.19	.26	.14	.37	.05	.65	.70	2.39	1.88	.44	.12	.28
1954	.20	.05	.70	.00	.18	.78	2.28	5.24	1.98	.37	1.49	.61
Average..	.37	.25	.47	.27	.25	.66	1.35	3.57	1.55	.78	.84	.59

Average annual precipitation, 11.15 inches

Year	Temperature, ° F.		Number of days at temperature			Freezing temperature		Ice on coast 1/		Total annual snowfall, inches
	Maximum	Minimum	70° or -	32° or -	0° or -	First in autumn	Last in spring	Closed in autumn	Open in spring	
1950	72	-22	3	243	70	Sept.20	June 25	Nov. 20	June 15	43
1951	61	-40	0	249	108	Sept.1	June 10	?	June 15	25
1952	68	-40	0	255	110	Aug. 10	June 18	Dec. 11	June 19	42
1953	73	-31	2	246	110	Sept. 1	June 12	Oct. 29	June 29	23
1954	67	-36	0	247	105	Sept.17	June 18	Nov. 13	June 3	?
Average..	68	-35	1	248	101	Sept.4	June 17	Nov. 18	June 14	33

Average annual snowfall, 33 inches

1/ Data for Wales only, because sheltered harbor at Teller provides different conditions.

Summer travel throughout the area is relatively easy because of lack of vegetation, rounded topography, and the general absence of unfordable streams. Trails suitable for jeeps or four-wheel-drive trucks can be made with very little effort where grades are suitable; many parts of the area can be traveled by four-wheel-drive vehicles without any preliminary roadwork.

LABOR

Skilled labor for lode mining and milling on Seward Peninsula must be drawn from the continental United States or Canada. The Coeur d'Alene district of Idaho, and Butte, Mont., are the most available sources in the United States. Many miners in these two districts have had experience in Alaska or are inured to northern climates enough to be adaptable to Alaskan conditions. Because miners, like construction workers, are inclined to roam, no great difficulty is experienced in obtaining a nucleus of skilled workers.

Much of the labor at lode-mining operations on Seward Peninsula may be recruited from the native Eskimo population. Although Eskimos generally have not been trained in mining or mechanical trades, recent operations at Lost River demonstrated that they have innate mechanical aptitude to a high degree. They were quickly trained to operate all kinds of mechanical equipment and gradually were becoming competent underground miners. To retain them on the job, however, it was necessary to provide family quarters and schools for their children and to permit them to return to their village homes at somewhat irregular intervals. This difficulty is offset, to a large degree, by the fact that the Eskimos are remaining permanently in the area and depend almost entirely on mining for any earnings above their normal subsistence.

The population 1950 census of the various settlements on Seward Peninsula and of settlements closely adjacent to Seward Peninsula is listed below:

	Population		Population
Candle	105	Koyuk	134
Council	41	Nome	1,930
Deering	174	Noorvik	248
Diomede	103	Selawik	273
Elephant Point	108	Shaktolik	127
Elim	154	Shishmaref	194
Golovin	94	Shungnak	141
Haycock	24	Solomon	93
Igloo	64	Teller	160
Kiana	181	Teller Mission	109
Kobuk	38	Unalakleet	469
Kotzebue	623	Wales	141
		White Mountain	129

Total population	5,857	
Total native (Eskimo)	4,207	(estimated)
Total nonnative	1,650	(estimated)

Note. - King Island Eskimos probably were counted with the population of Nome, where they migrate each summer.

The geographical distribution of the settlements is shown in figure 3. The population of the smaller settlements is almost entirely Eskimo. Assuming that the ratio of able-bodies Eskimo males to the total Eskimo population is about 1:4, it is estimated that the potential Eskimo labor supply in the area is about 1,000. Although most of the Eskimos who worked at the Lost River mine were from Teller, Wales, or Shishmaref, a larger and more sustained demand undoubtedly would draw workers from the more remote villages.

WAGES

The wage and salary rates paid at the Lost River mine during 1954 - the most stabilized period of continuous operation - are listed below. Supervisory personnel, office and clerical workers, mechanics, and most of the contract miners were white; surface and underground laborers, mill operators, hoistmen, trammers, mucking-machine operators, and a few contract miners were Eskimo. For comparison, the rates for a number of similar jobs in the Coeur d'Alene district of Idaho are given; these rates, about average for the district, are from the October 1, 1955, agreement between the Sunshine Mining Co. and local No. 5089, United Steelworkers of America:

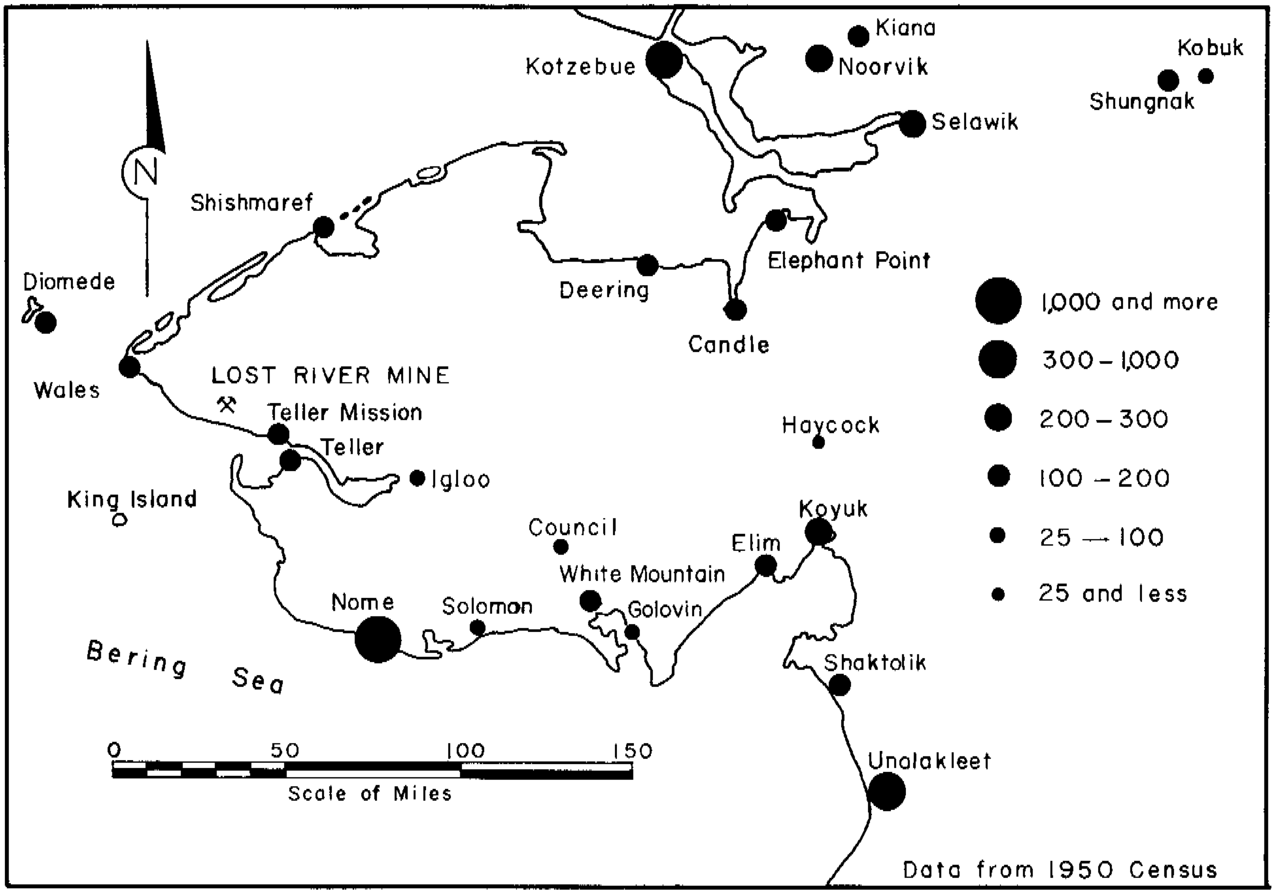

FIGURE 3. - Population Distribution in Seward-Peninsula Area.

	Lost River rate per month	Lost River rate per hour	Coeur d'Alene rate per hour
General superintendent	$750		
Mill superintendent	700		
Mine engineer	660		
Mine shift foreman	600-900		
Assayer	500		
Bookkeeper	500		
Warehouseman	450		
Cook	595		
Power-plant operator	590		
Electrician		$2.50	
Mechanics		2.00	
Bulldozer operator		2.00	
Surface laborer		1.50	$1.82
Mill operator		2.00	2.14-2.20
Miner		Contract	2.01-2.27
Timberman		do.	2.14
Mucker-trammer		1.75	2.14
Mucking-machine operator ...		1.75	2.14
Hoistman (single drum)		1.75	2.14
Motorman		1.75	2.01

Virtually all mining and all timbering were on a "labor contract" basis. The company provided all equipment and supplies; the miners were paid for labor at unit prices, as follows:

Drifts and crosscuts	cost per foot	
(depending on conditions)		\$12.00-\$17.00
Raises (timbered)	do.	18.00
Raises (untimbered)	do.	10.00
Stoping	cost per ton	.65

Earnings of the contract miners ranged from the guaranteed minimum of \$1.75 per hour to more than \$5.00 per hour, depending on conditions and the ability and energy of the miner. The hourly rates quoted in the above table are for day shiftwork; for other shifts a differential was paid as follows:

Starting time (inclusive)	Differential per 8-hour shift, cents
Afternoon (11:31 a.m. to 3:14 p.m.)	40
Night (3:15 p.m. to 7:30 p.m.)	60
Graveyard (7:31 p.m. to 4:59 a.m.)	80

Note. - Contract rates were negotiated for each contract.

All hourly rates were based on a 40-hour workweek; overtime was paid at 1-1/2 times the base rate. The Lost River mine operated 7 days a week.

Board at the Lost River mine was charged at the rate of \$3 per day. No charge was made for room rent in the bunkhouse or for family housing.

Transportation from the point of hire and return was paid after 1 year of service.

Details of labor costs are presented in the section on costs (p. 56).

SPECIAL OPERATING PROBLEMS

Lode-mining operations at Lost River or in other parts of western Seward Peninsula are subject to few, if any, difficulties that are uncommon to lode-mining operations in isolated parts of other northern countries. Nevertheless, four problems deserve some special discussion; these are: (1) Transportation and communication, (2) housing and recreation, (3) water supply, and (4) permafrost.

Transportation and Communication

When compared with transportation problems or many mines in the mountains of Western United States and in interior Alaska and northern Canada, those problems at Lost River are relatively simple, because no long overland freight haul is involved. The chief problem is created by the short season during which heavy freight may be shipped or received. Because of this brief transportation season, large inventories of supplies and concentrates must be maintained, requiring large warehousing facilities and considerable interest charges on unproductive capital. These charges or the freight charges for the long ocean haul are offset, at least in part, by lower requirements for the equipment and labor that would be involved in overland haulage to inland locations.

Interruptions to mail communications and, during frequent, short periods, to radio communications constitute a relatively minor annoyance, for which adequate provisions should be made but which should not be serious for a well-organized operation.

Housing and Recreation

The very rigorous climate, combined with extreme isolation, requires special provision for well-insulated, well-heated buildings, cheerful quarters for families and single-status employees, and facilities for indoor recreation. All these are needed, especially for the married employees, if a competent staff is to be maintained with minimum turnover. Work at Lost River was started with minimum housing for single employees and almost no family quarters. It soon became evident that a good staff could not be retained without better living quarters for single men and without at least a few houses for staff families. It also became evident that only the younger and less responsible Eskimos could be retained unless provision was made for family accommodations. Consequently, it was necessary to build a new bunkhouse, minimum quarters for Eskimo families, and church, school, and store facilities for the Eskimos. The camp buildings and other equipment are described in detail in the cost section of this report (p. 49); the camp layout is shown in figure 16.

The housing would be adequate, on a minimum basis, for resumption of limited-duration operations on the existing (100-ton-per-day) basis or for the exploration program that would be an advisable prelude to any plans for permanent operations. Any long-range production plans, however, should include provisions for a single unit or for a closely knit group of units of well-insulated, fireproof construction, which would provide all housing for single employees, mess hall, change rooms, recreation and reading rooms, offices, and warehouse; these should be connected to the mill and mine by covered walkways. Small but modern houses for at least 3 or 4 key-management personnel should be built, and the other family accommodations should be improved gradually.

Water Supply

One of the chief problems during the recent operations at Lost River mine was obtaining a steady, dependable water supply for milling. Although the total annual runoff should be adequate for an establishment many times larger than the existing plant, the surface flow is not dependable, because the total lack of vegetation results in rapid surface runoff; furthermore, surface flow stops during the long, cold winters.

Subsurface flow in the gravels just above bedrock on Lost River provided mill water during the one winter of continuous operation. This flow had been known to exist and was considered in the original plans; it was developed for use by means of a 10- by 12- 15-foot wood-lined sump in the gravels of Lost River a short distance below its confluence with Cassiterite Creek. To increase the catchment for this sump, a bedrock drainage drift was driven 30 feet across the river channel. This development provided a nearly steady flow of 300 gallons per minute, which was pumped to the mill through 9,000 feet of heavily insulated transite pipe, heated with a thermostatically controlled electrical heating system; the system is described in detail in the section on costs. From February to May, however, the flow decreased to the extent that mill operations had to be suspended a number of times while the water in the sump was being replenished. The aggregate shutdown time per month (1954) because of water failure follows:

Month	Hours	Shifts
February	86.0	10.75
March	44.0	5.50
April	285.9	35.74
May	76.9	9.60
Total	492.8	61.59

Note. - Equivalent to 20.5 days of operation.

This test indicated an actual recovery of approximately 150 million gallons, or 620,000 tons of water per year. Undoubtedly recovery could be increased by additional development of the Lost River channel. The width of the channel at this point is estimated at 300 feet. Inasmuch as only 40 feet, or about one-seventh of the total width, was developed, it may be assumed that by extending drainage drifts across the entire channel substantially more water could be recovered. Whether the additional recovery would be proportional to the additional width of the developed channel could be determined only by trial. Because of continuous silting, the sump and drainage channels require periodic maintenance. The gravel in the valley of Lost River is not frozen permanently, but seasonal frost extends to a considerable depth and prevents rapid recharge of underground channels during the initial spring runoff.

The combined drainage area of Cassiterite Creek and Lost River above the confluence of the 2 streams is approximately 8 square miles. The average yearly precipitation (11 inches) in this area should yield a runoff of about 6.5 million tons, which would equal the requirements of a 1,000-ton-per-day operation (at 18 tons of water per ton of ore) if no water is reclaimed. Unfortunately, the very unequally distributed runoff would require costly and probably impracticable storage facilities.

An unlimited supply of salt water is available from the Bering Sea (6-1/2 miles from the mine), but the costs involved in utilizing this source of supply might well be prohibitive.

Results of recent diamond drilling from the bottom level of the mine indicated that an additional supply of water may be obtained from the underground flow through the highly fractured limestone and granite that underlie the mine area. This flow may be derived from inland sources considerably beyond the limited drainage basin of Cassiterite Creek. Old reports to the effect that a strong flow of water had been encountered on the bottom level off the old shaft never were investigated fully, because the bottom level off the new shaft was driven about 30 feet above the old bottom level, which never was unwatered. At that time it was believed that the new bottom level (365) was below sea level, but information since developed indicates that it is nearly 60 feet above sea level and therefore probably above the regional water table. During the last phases of the DMEA exploration program, diamond-drill holes 5, 6, and 7, which penetrated relatively unaltered but highly fractured granite and limestone outside the zones of intense kaolinization and, partly, considerably below the 365 level, each tapped a heavy flow of water which had to be sealed off to prevent flooding the pumps. The drill-pressure gages indicated 50 pounds per square inch on each hole. It was impossible, in the time available, to make dependable measurements of the amount or continuity of flow that could be obtained.

Summarizing, it can be said that a little additional development of the bedrock flow on Lost River will provide adequate water for a 100-ton-per-day operation and probably for a somewhat larger one. Moreover, a substantial or possibly a large flow probably could be obtained by developing underground, water-bearing fractures at slightly greater depths in the Lost River mine. It appears likely that the size of future operations, aside from the question of ore reserves, will be limited by the amount of underground water that can be developed. Continuous measurements of flow from existing drillholes would provide some indication of potential supply, but adequate testing would require that another level be driven at least 100 feet below the 365 level.

Permafrost

Permafrost has been encountered at Lost River in mine workings and in diamond-drill holes as much as 200 feet below the surface. This condition affects exploration and mining in several ways and requires special operational procedures and precautions. Openings driven in frozen material require little or no timbering if protected from seasonal thawing. If air is allowed to circulate freely through such openings during the summer, excessive caving is likely to occur because of the intensified frost action caused by repeated thawing and freezing. Diamond drilling in permafrost must be conducted on a continuous basis or water circulated continuously to prevent formation of ice in the holes. Broken ore left too long in stopes that are in the frozen zone tends to freeze solidly and must be loosened by blasting to facilitate drawing. Air and water lines must be protected. Most of the problems due to permafrost can be solved by a well-planned mining schedule, control of air circulation, and judicious use of insulating materials.

NATURE OF DEPOSITS

General Geology

The tin deposits of the Seward Peninsula have been the subject of numerous investigations by the Federal Geological Survey since their discovery in 1900, and much information has been published. The various investigations and general geology of the tin-bearing areas are described by Steidtmann and Cathcart.9/ Detailed geologic studies of the Lost River deposits were made by the Federal Geological Survey during the recent period of mining activity.10/

Observations of the various geologists and engineers may be summarized briefly as follows: The tin-bearing areas are underlain by slates and limestones that have been intruded by granite and associated dikes. Granite stocks are exposed at Cape and Brooks Mountains, Tin Creek, and Black and Ear Mountains. Similar stocks were inferred from surface indications at Potato Mountain and Cassiterite Creek; existence at the latter was confirmed by diamond drilling and underground workings at the Lost River mine. Tin mineralization has been found associated with all known and inferred granitic intrusives except that at Black Mountain. Because all the granites have similar chemical composition and almost identical association with quartz-bearing rhyolite and dacite dikes, they are believed to have a common deep-seated origin.

At Lost River numerous metallic minerals occur in widely varying amounts in the acid and basic dikes, in the contact-metamorphic zone of altered limestone around and overlying the granite stock, and, to some extent along zones of fracture in comparatively unaltered granite or limestone. Cassiterite and wolframite are the chief ore minerals; other metallic minerals present in lesser amounts include scheelite, galena, marmatite, chalcopyrite, stannite, arsenopyrite, pyrite, molybdenite, stibnite, and bismuthinite. Traces of cadmium, indium, and other rare elements also have

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9/ Steidtmann, Edward, and Cathcart, S. H., Geology of the York Tin Deposits, Alaska: Geol. Survey Bull. 733, 1922, 125 pp.

10/ A bulletin on these studies is being prepared by C. L. Sainsbury, Geological Survey.

been identified. Nonmetallic gangue minerals include quartz, fluorite (often in concentrations), topaz, tourmaline, zinnwaldite, clay minerals, and lime-magnesium silicates. The approximate location of the principal prospects, as well as the generalized geology of the Lost River area, is shown in figure 4.

For mining purposes, the deposits at Lost River may be classified as vein-type or "dike" deposits and "disseminated" deposits; the latter includes the contact-metamorphic zone and the zones of fracture in granite or limestone.

Disseminated Deposits

Cassiterite and other metallic minerals occur in numerous discontinuous lenses and veinlets and as disseminations throughout a wide, irregular zone of altered granite and metamorphosed limestone adjacent to and overlying the buried granite stock. Although many high assays for tin and tungsten may be obtained from samples taken within this zone, the general average is comparatively low, and distribution of the metal content appears to be erratic. Along the contact, particularly where strong mineralization has developed, both the granite and the limestone are intensely fractured and altered. In many places the granite is almost completely kaolinized, and the clay constituents swell rapidly when exposed to air and water. In these areas openings that are not heavily and tightly timbered will close completely in a few months.

Except for comparatively minor exploration, no development of the disseminated deposits has been attempted. On the basis of limited information now available, the recovery of tin from these deposits could only be accomplished by using large-scale mining methods that require no ground support, such as block caving or opencutting.

Dike Deposits

All ores mined to date and all reserves indicated by mine workings are in the Cassiterite dike or at the intersection of the Cassiterite dike with the Ida Belle dike (fig. 4).

The surface of the Cassiterite dike has been traced throughout a strike length of about 8,000 feet; it has been proved by exploratory workings to be tin bearing throughout a strike length of about 2,200 feet and throughout a vertical range of about 1,000 feet. The width of the Cassiterite dike ranges from about 5 to about 20 feet; the dip ranges from 60° to 80°, and the strike from S. 60° E. to N. 85° E. Generally, the width and dip are nearly uniform at about 12 feet and 70°, respectively, but locally the dike may roll and pinch or swell. Usually the tin content is lower where the dike narrows.

Where it is unaltered, the dike appears to be a normal rhyolite porphyry dike, but within the ore-bearing sections it has been shattered by irregular fractures and differentially altered by hydrothermal solutions. The degree of shattering as well as the nature and intensity of alteration differs from place to place in the dike; generally, the numerous fractures and irregular seams and masses of claylike material create an unstable condition in any mine workings that may be driven within the dike.

The limestone walls of the dike are only slightly fractured and altered; consequently, they generally are firm, except in places where a breccia zone, which may be as much as 6 feet in width and usually is tin-bearing, is on the hanging wall of the dike. From the above description it is apparent that mining operations on the Cassiterite dike involved only the relatively simple problem of mining a soft vein between firm walls; the principal difficulties are caused by hanging-wall breccia

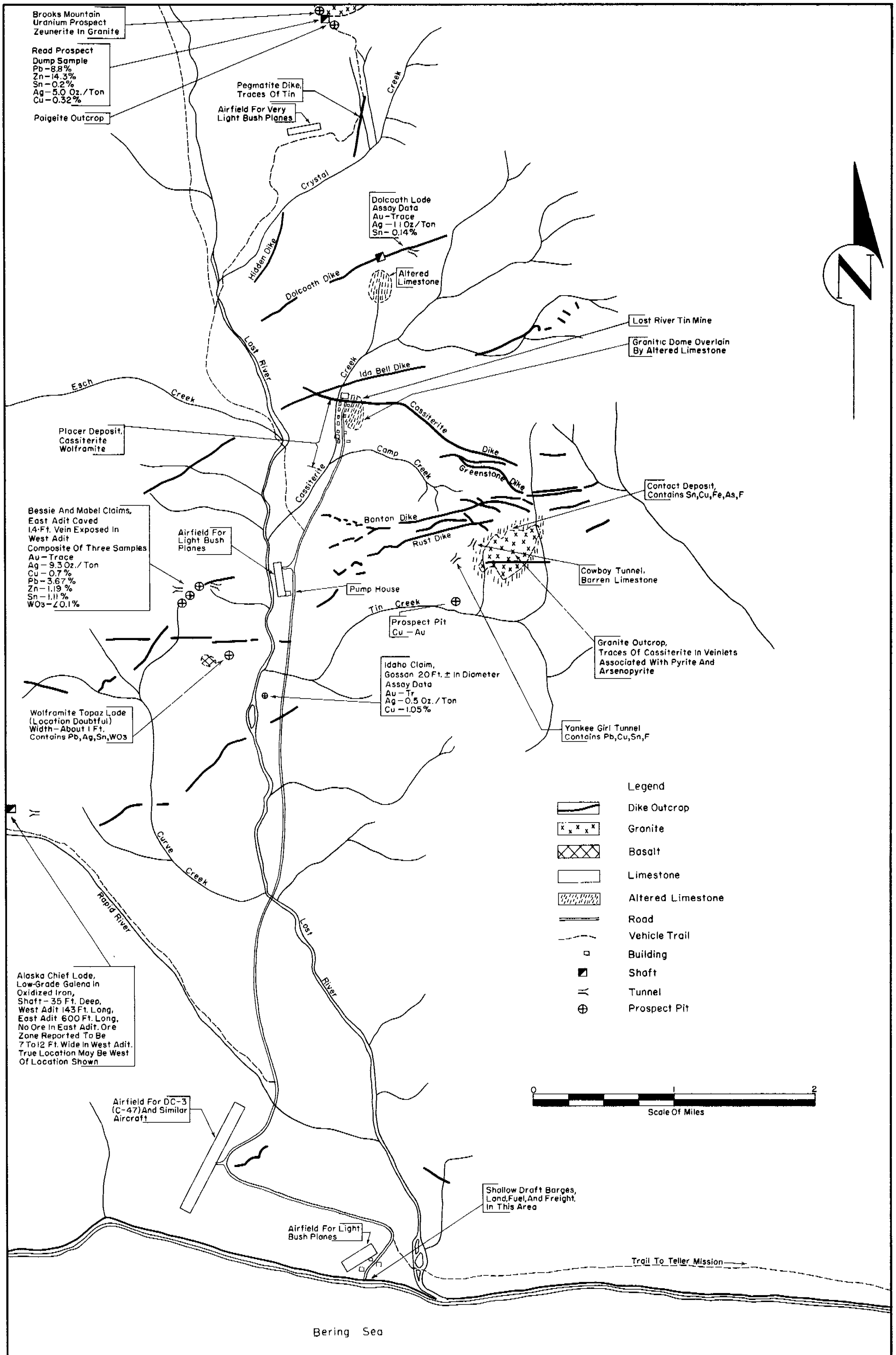

FIGURE 4. - Prospects and General Geology, Lost River Area. Note.—Data from reports and maps by U. S. Tin Co., Federal Bureau of Mines, and Federal Geological Survey. Compiled by J. J. Mulligan, Bureau of Mines, 1956.

and by the fact that the dike material usually is so unstable as to require close timbering of its permanent openings. Fortunately, most openings in the limestone will stand indefinitely without support.

The Ida Belle dike is 15 to 55 feet in width; this dike or dike system has been traced by intermittent surface indications throughout a strike length of about 3 miles but is known to be tin bearing only within the first 750 feet easterly from its intersection with the Cassiterite dike. The composition and general character of the Ida Belle dike and its walls are similar to comparably mineralized parts of the Cassiterite dike; therefore it may be assumed that, if ore shoots of minable grade are discovered in it, the mining conditions will be similar to those in ore shoots in the Cassiterite dike. Mining costs in the Ida Belle however, should, be appreciably lower because of its greater width.

Numerous other dikes in the immediate vicinity of the Lost River mine are known to be tin bearing but have not been explored (fig. 4). If ore shoots are discovered in them, presumably they would present essentially the same mining problems as the Cassiterite dike.

EXPLORATION AND DEVELOPMENT

Summary of Work Completed

Although several extensive tin-bearing structures occur in the Lost River area, the major exploratory effort and all development have been confined to part of the Cassiterite dike and the associated contact metamorphic zone. The exploration programs completed at the Lost River tin mine are summarized as follows:

Method	Operator, period and units			Total All Operators 1903-55 feet
	Various owners and lessees 1903-20 1928-30 feet	Federal Bureau of Mines 1942-45 1951-52 feet	U.S. Tin Corp. DMEA - DMPA 1951-55 feet	
Trenching:				
Hand	500	1,585	-	2,085
Dozer	-	5,750	150	5,900
Drilling:				
Core, surface	-	8,899	-	8,899
Core, underground	-	-	1,984	1,984
Percussion, longhole	-	-	1,795	1,795
Exploratory openings:				
Drifts and crosscuts	1,553	-	2,795	4,384
Raises	200	-	1/75	275
Shaft or winze	425	2/200	2/185	820

1/ Exploratory rise from 365 level; other rises driven for stope preparation are not listed.

2/ Driven to replace caved, inclined winze, completed during first exploration period, 1903-20.

The principal workings are shown in plan in figure 6 and in section in figure 7.

FIGURE 5. - DMEA Sampling Data, 195 Level and 365 Level, Lost River Mine. Note.--Values are percent Sn unless otherwise designated, Nil = 0.05 percent Sn. Channel samples assayed at Lost River by U. S. Tin Co. Longhole and diamond-drill samples assayed at Juneau by Federal Bureau of Mines. Map traced from prints of U. S. Tin Co. maps. Drill holes, sample locations and assay data compiled from reports by U. S. Tin Co. and DMEA field team. Compiled by J. J. Mulligan, Bureau of Mines, 1956. (In pocket at end of report.)

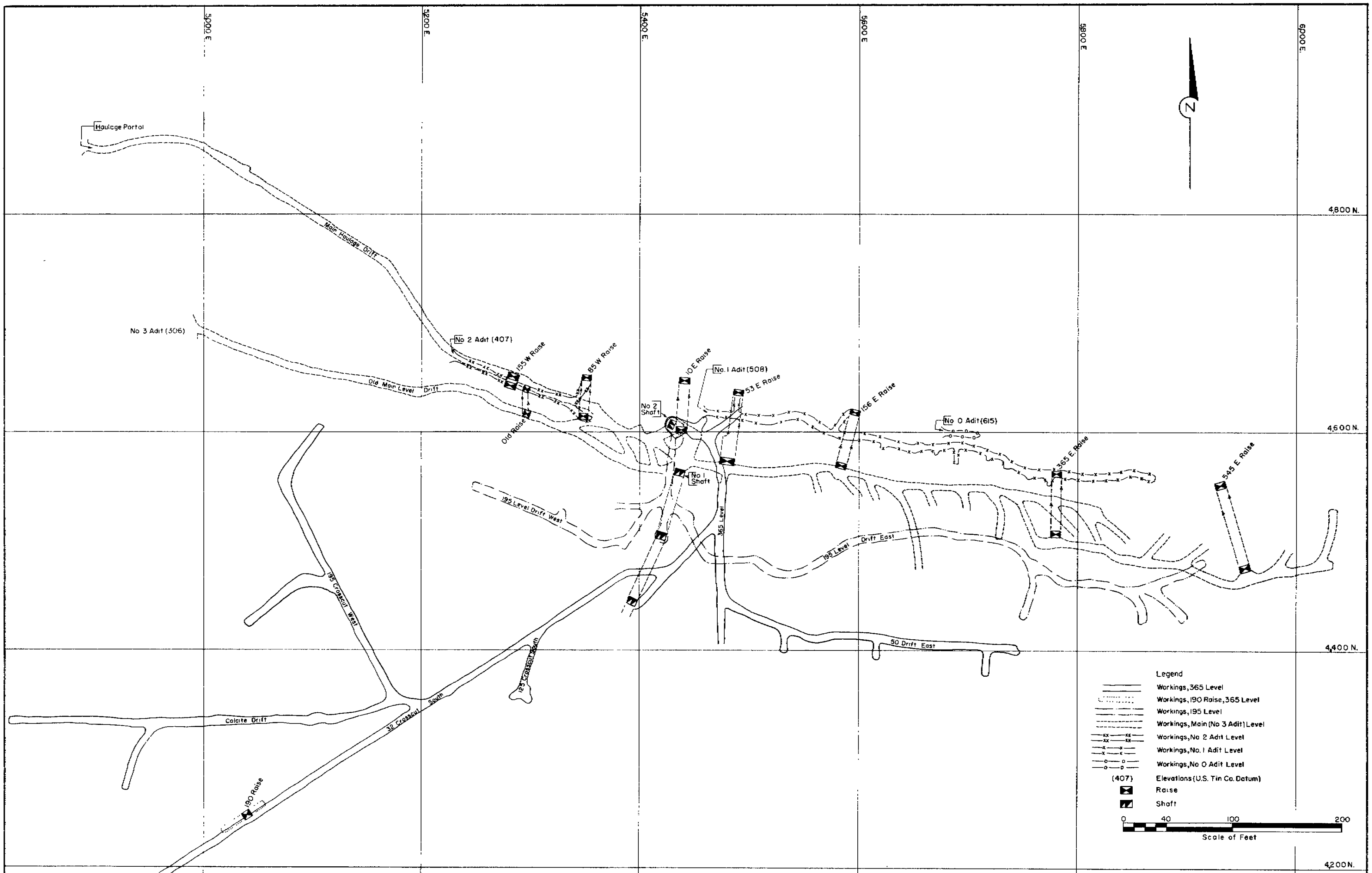

FIGURE 6. - Composite-Level Map, Lost River Mine. Note.—Map compiled from blueprints and reports by U. S. Tin Co. and DMEA field team. Compiled by J. J. Mulligan, Bureau of Mines, 1956.

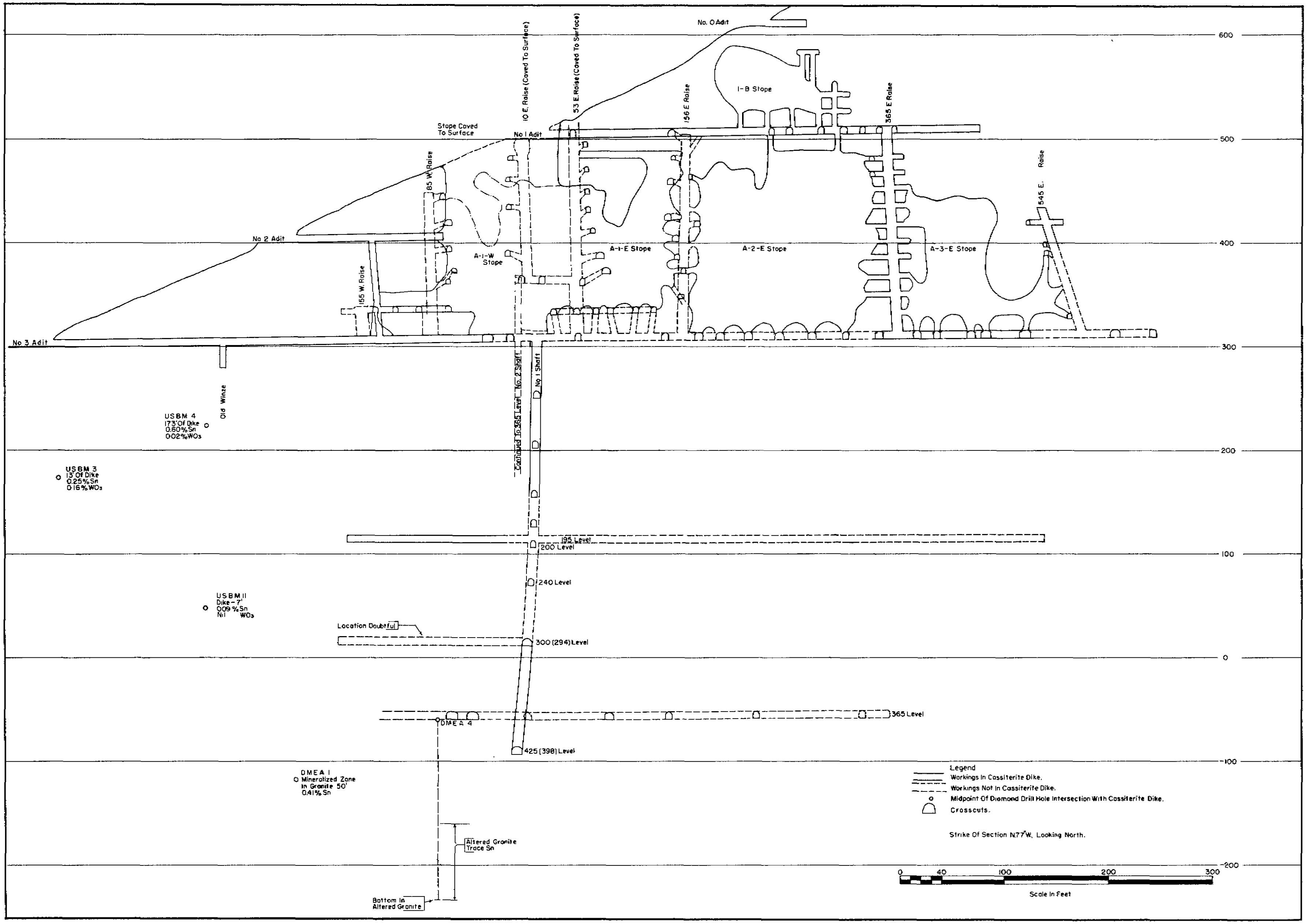

FIGURE 7. - Vertical Longitudinal Section, Lost River Mine. Note.–Data compiled from reports and maps by U. S. Tin Co. and DMEA field team. Compiled by J. J. Mulligan, Bureau of Mines, 1956.

Drilling

Diamond Core Drilling, Surface

Diamond drilling from the surface was undertaken by the Federal Bureau of Mines in 1942 and 1943 to explore in depth the Cassiterite and Ida Belle dikes, as well as to attempt to prove the existence of a postulated granite cupola. The surface diamond drilling was completed in 2 field seasons, using 2 drills 2 shifts per day during the greater part of the program. Overall drilling progress (reaming, casing, and cementing) was 23.8 feet per drill-shift. Of the total 8,899 feet of hole drilled, 45 percent was cased, 17.7 percent cemented, and 8 percent reamed.

Most of the holes were inclined 45°, but 1 was drilled at 70°, and a few at 30°. Holes were collared through frozen overburden into firm rock with a BX-size bit, advanced with AX size until loss of return drill water or caving necessitated casing, and completed with EX size. Holes ranged in depth from 192 to 683 feet but averaged 404 feet.

The diamond drills used were the skid-mounted, gas-powered, screw-feed type, chucking EX rods, and rated at 1,000 feet of EX hole. Cast-set diamond bits and 5-foot, rigid, double-tube core barrels were used to recover core. Core recovery was nearly complete in the limestone walls of the dikes. In the soft, altered dikes core recovery averaged 60 percent.

Continuous circulation of water in the holes was necessary to eliminate the formation of hard ice, which completely filled the holes if circulating water was stopped for periods exceeding 4 hours. The ice thus formed had to be core-drilled before the hole could be advanced in rock. The drilling rate in ice averaged 50 feet per hour. The results of the surface drilling program were presented in detail by Heide.11/

Diamond Core Drilling, Underground

Diamond drilling from underground stations on the 365 level was authorized by DMEA contract to explore the metallization and contour of the limestone-granite contact, as well as to test for the existence of the Cassiterite dike structure in the underlying granite. A total of 1,984.5 feet of drilling in 8 holes was completed. Overall drilling progress, including casing and cementing, was 14.8 feet per drill-shift. The average depth of all holes was 247.6 feet; the average depth drilled at each bit size was 16.7 feet BX, 111.1 AX, and 119.8 EX. The holes were drilled at various inclinations between horizontal and vertical. Approximately 20 percent of the total drilling was in limestone and the remainder in granite.

The drill used was a column-mounted screw-feed type (Chicago-Pneumatic 55A), powered by a rotary air motor and rated at 500 feet of EX hole. Feed gears were 200, 300, 400, and 500 revolutions per inch of advance. Cast-set diamond bits and 5-foot rigid-type core barrel were used to recover core.

Overall core recovery averaged the same in limestone or granite - 84 percent. Difficult drilling conditions were encountered in the soft, highly altered contact zone, which ranged in character from clay to a friable, decomposed granite. Although clay sections of the contact zone were recovered as core, this core was sometimes exceedingly difficult to remove from the inner tube because of the swelling properties of the clay. On exposure to air, the decomposed granite air-slacked and disintegrated to a mass of quartz grains, mica flakes, and clay.

11/ Work cited in footnote 8, p. 3.

Experiments with a swivel-tube core barrel in drilling the contact zone indicated that neither superior core recovery nor faster penetration rates could be obtained than those with a rigid-tube core barrel. In clay, a swivel-tube core barrel penetrated only to the depth of the bit and shell before it blocked at the junction of the inner tube and shell.

Experience in drilling the contact zone proved the necessity for drilling with more than the normal amount of water. Various types of drill bits were tried in the contact zone; the most successful was a diamond bit with greatly enlarged waterways. The additional volume of water obtained at the bit crown by enlarged waterways kept the bit face free of the sticky clay. A stellite-tipped sawtooth bit was used successfully in drilling the highly kaolinized parts of the contact. In the granitic phase of the contact zone the stellite-tipped teeth of the bit were rapidly destroyed by abrasion from the coarse quartz in the granite. In the contact zone the holes were cased to the greatest possible depth, and the casing was kept as close to the face as possible.

In the unaltered granite several holes tapped watercourses that maintained a static water pressure of 50 to 60 p.s.i. at the hole collar when capped. The volume of water from the holes was not measured. Locations of the DMEA diamond-drill holes, sampling results, and graphic logs are shown in figure 5.

Percussion Drilling, Longhole

A total of 1,795 feet of longhole percussion drilling was completed from the 195 and 365 levels as part of the DMEA program. The chief purpose of longhole was to sample the cassiterite dike at intervals of 20 to 25 feet along the levels. One longhole was drilled to sample the contact zone and 6 to sample metallization of the adjoining granite. The average depth of hole was 51 feet; several holes were 60 feet deep. Drilling progress was from 35 to 50 feet per drill-shift; the depth depended largely on the skill of the driller.

All holes were inclined upward 5° to assure maximum recovery of drill cuttings; most of the holes were drilled nearly normal to the strike of the Cassiterite dike. Generally, longholes were collared and drilled 6 to 10 inches with a 3-inch Timken steel bit. The collars were cased with a piece of pipe (2-1/2 inches i.d. and 10 to 16 inches in length), which deflected drill cuttings into boxes. Holes were completed with tungsten carbide bits, sized from 2-1/4-inch diameter. Drill rods were 1-inch hexagonal, in 2-, 3-, or 5-foot lengths. Case-hardened couplings were 4-1/2 inches in length with an outside diameter of 1-3/4 inches. The drill was a 4-inch Sullivan hand-cranked drifter, mounted on a 4-foot aluminum shell with a sliding cone. Hole locations and sampling results are shown in figure 5.

Drifting and Crosscutting

Exploration and development drifts and crosscuts were predominant in limestone, although 730 feet of drifting on the 365 level was in granite. Drifts normally were 5 by 7 feet in cross section; they required timbering only when driven in some parts of the Cassiterite dike or in the altered limestone and granite of the contact zone on the 365 level.

A normal drift round consisted of 16 holes, including a 4-hole burn cut. The depth of round drilled generally was 7-1/2 feet, with a corresponding advance per round blasted. The rounds were blasted by 40- or 45-percent gelatin dynamite in "Redi-slit" cartridges, detonated by fuse and No. 6 caps or electric delays. Stemming holes or the use of spacers was seldom practiced, although it was tried in an effort to alleviate coarse fragmentation experienced in some parts of the granite.

Several types of drills were used in drifting. A drill jumbo, mounting two DB-30 automatic feed drifters; hand-cranked, 4-inch drifters, mounted on columns; and jackhammers, mounted on pneumatic feed legs. Drill steel was seven-eights inch hexagonal 2-foot changes. Detachable tungsten carbide-insert bits, as well as detachable reground steel bits, were used for drilling the round. A 2-inch compressed-air line and a 1-inch water line along the side and bottom of the drifts supplied the drills. Air and water lines were all victualic coupled.

A rocker shovel was used for loading end-dump cars of 21-cubic foot capacity. On the lower levels cars generally were hand-trammed and dumped directly into the skip at the shaft. Empty cars were supplied to the rocker shovel from the nearest crosscut. A storage-battery locomotive was used to haul cars where the distance from the drift heading to the shaft exceeded 500 feet. A compressed-air locomotive, intended for haulage on exploratory drifts on the lower levels, could not be used efficiently for this service, because track curves were too sharp for the wheelbase of the air locomotive. The track gage was 18 inches; the rail weighed 30 pounds per yard in the main haulage and headings, but smaller rails were used in some exploration drifts. Load-favoring grades in the drifts were between 0.5 and 2.0 percent.

Drift headings in the lower levels were ventilated by air drawn from the No. 3 adit by a blower at the collar of the shaft; air was transmitted down the shaft to the heading by a 12-inch vent tube. Compressed air was used in clearing smoke from the face; muckpiles were sprinkled with water to alleviate obnoxious fumes and dust.

Where immediate support was required, drifts were timbered with 8- by 8-inch-square timbers for caps and posts, 4- by 4-inch girts or collar braces, and 2- by 12-inch lagging on the back and sides. The cap was blocked at the ends and bulkheaded to the back. Timbers were not framed. The frame for members of a set was nailed on 2-inch planks of appropriate length and width. Drift sets had no standard spacing in the exploratory drifts; the spacing ranged from 4 to 7 feet, center-to-center. Inside dimensions of drift timbers were at least 5 by 6 feet. As a rule, the drifts were timbered only where immediate support was absolutely necessary; almost no attempt was made to support exploration drifts for permanent or semipermanent maintenance.

A drifting crew of 2 men normally completed in 1 shift a cycle of removing a blast round and drilling and blasting another; the same crew also extended the track, vent tube, and air and water lines.

All new drifts, crosscuts, and raises below the haulage level (No. 3 adit) were driven under the terms of the DMEA contract. This work totaled 2,867 linear feet, including the 75-foot 190 raise. Channel samples were cut from all altered or mineralized zones encountered in the new openings, and the geology was mapped in detail. The location of the DMEA exploratory openings and the sampling results are shown in figure 5.

Raising

Most stope-development raises (figs. 8, 9, and 10) were driven in the limestone on the footwall side of the Cassiterite dike; however, the 365 raise was driven for its entire length in the dike, and the 545 raise was begun in the hanging wall of the dike. The 190 raise was driven vertically from the 365 level in granite to test the metallization of the granite-limestone contact.

The principal raises completed during the operating period, the average inclination, and the approximate length of each are listed as follows:

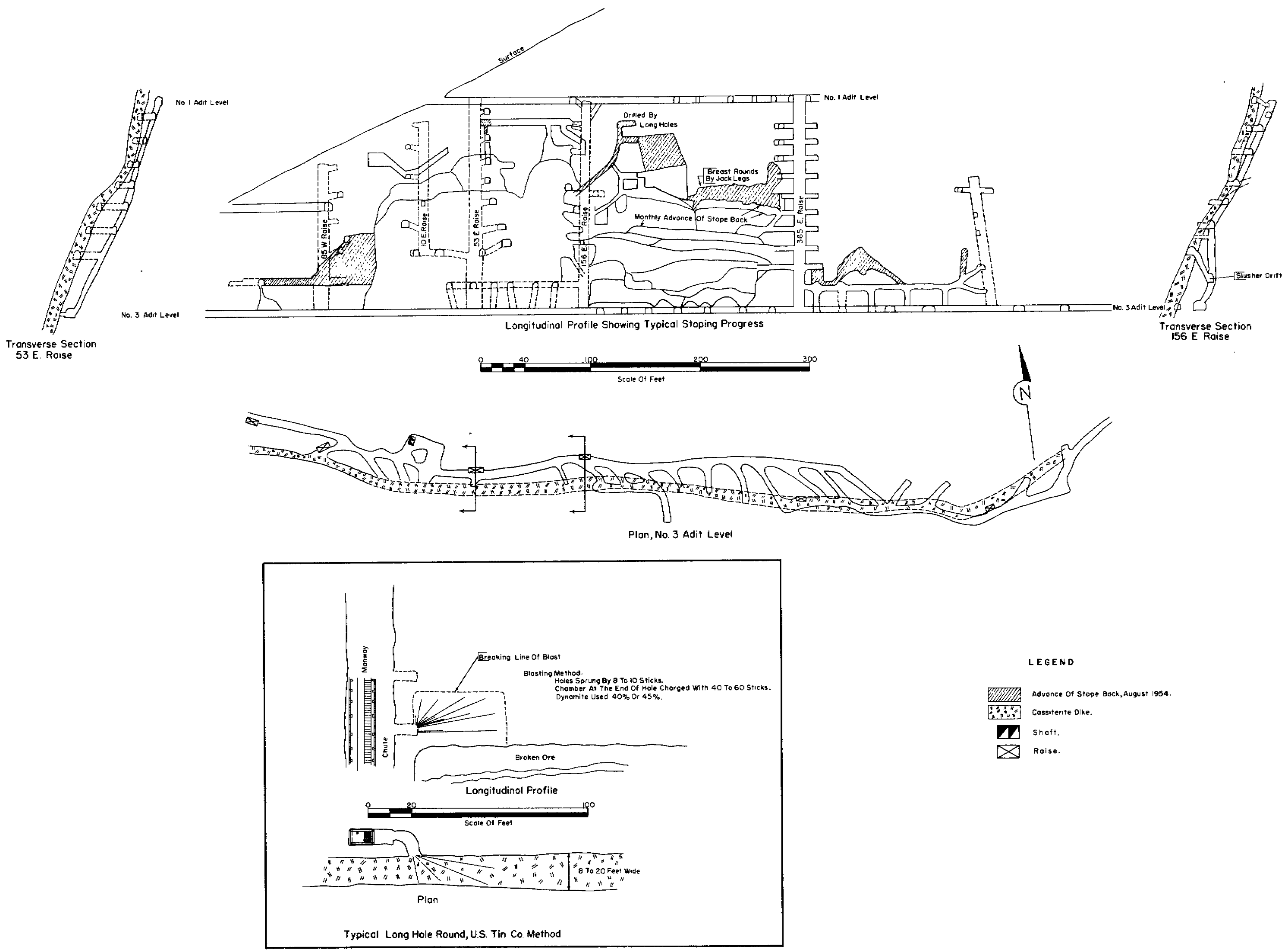

FIGURE 8. - Mining Method Used by U. S. Tin Co., Lost River Mine. Note.–Traced From Map by U. S. Tin Co. by Miro Mihelich, Bureau of Mines, 1956.

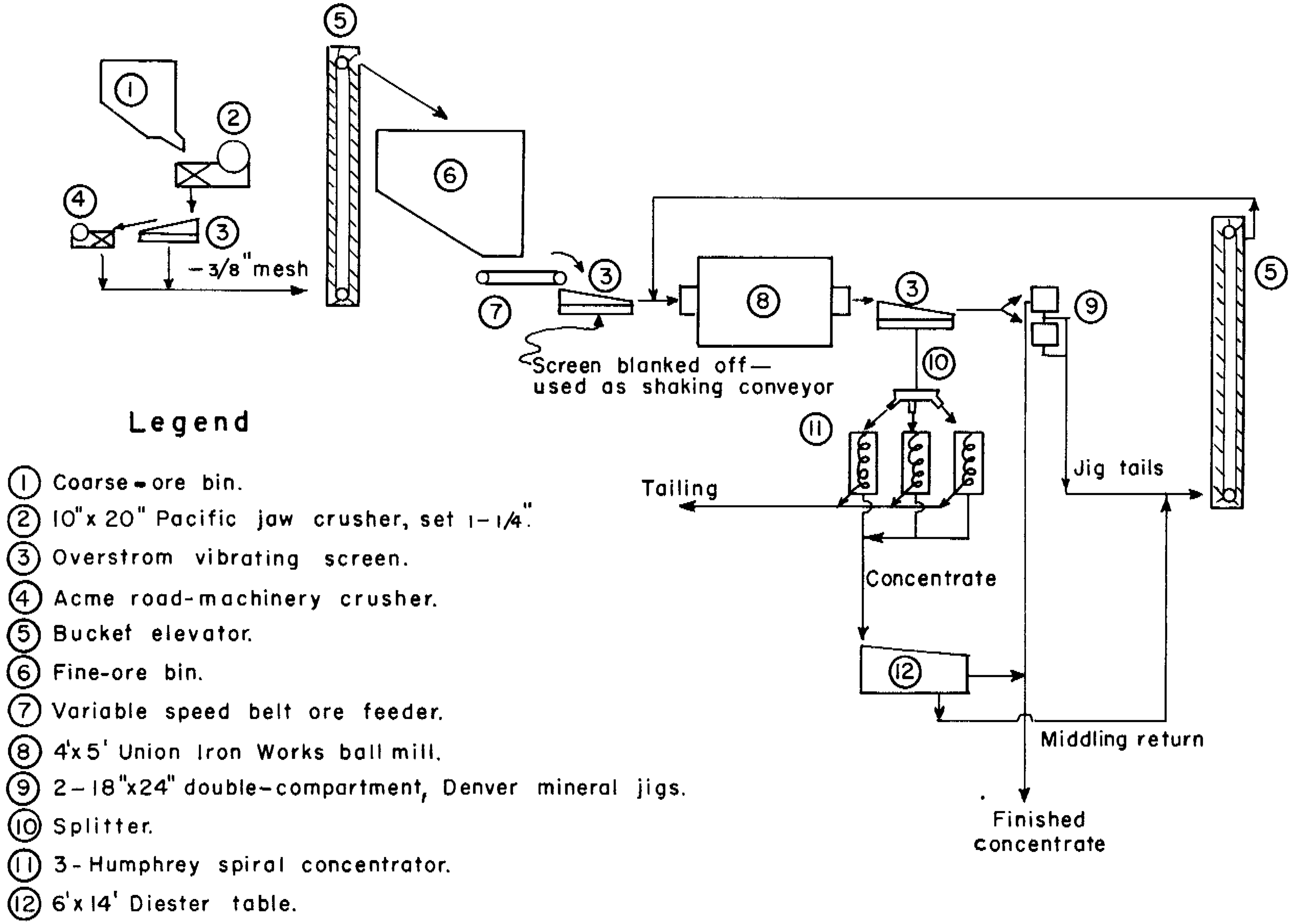

FIGURE 9. - Flowsheet of Initial Mill, Lost River, June 1952.

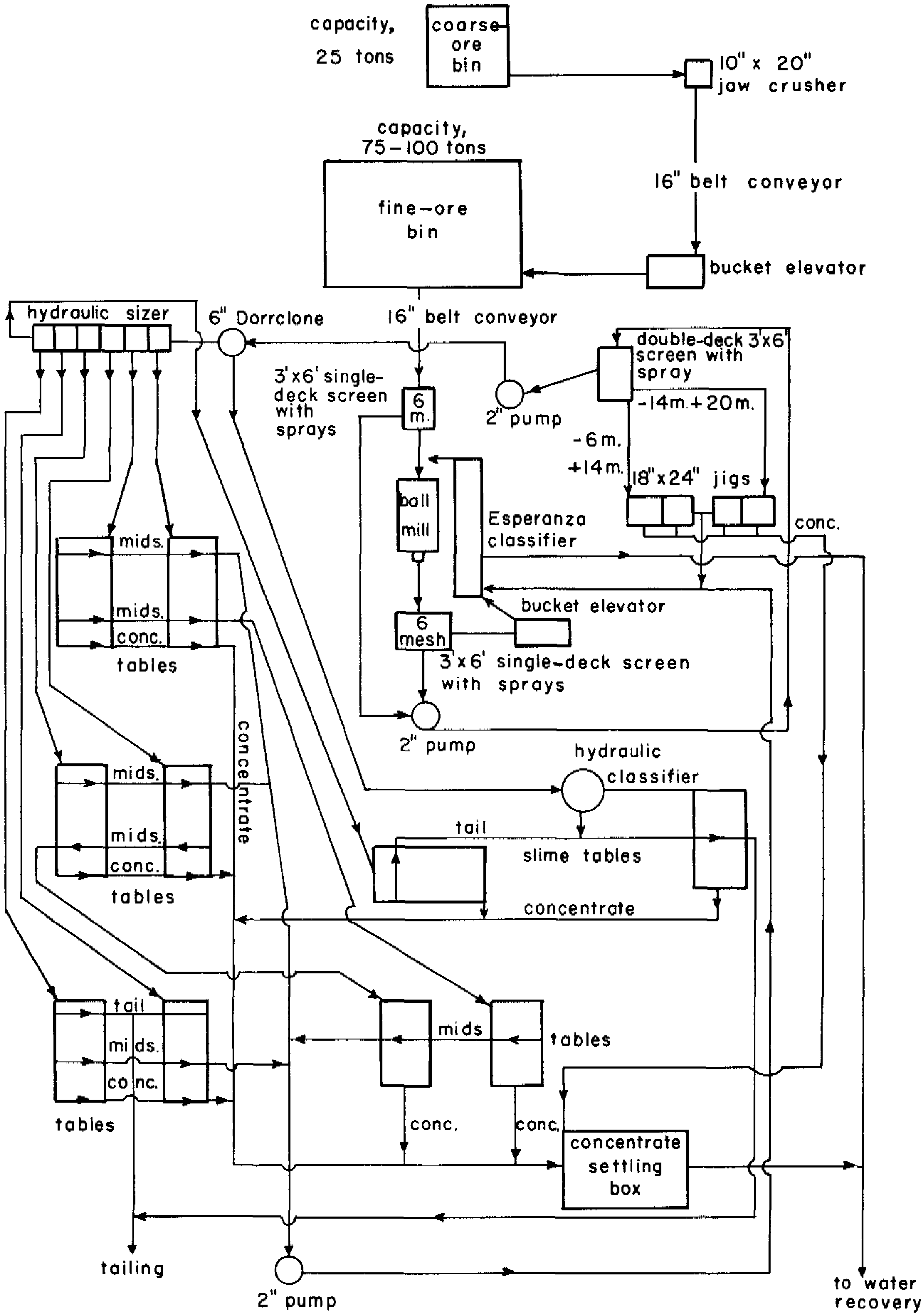

FIGURE 10. - Flowsheet, Lost River Mill, May 1953.

Raise No.	Length, feet	Average inclination	Location
545 E.	154	55°45'	In hanging wall of dike.
365 E.	204	67°30'	In Cassiterite dike.
156 E.	204	75°15'	In footwall of dike.
53 E.	204	72°15'	Do.
10 E.	125	69°00'	Do.
85 E.	142	79°15'	Do.
190	75	Vertical	In granite 365 level.

The stope-development raises in the footwall of the Cassiterite dike were generally inclined to conform as nearly as possible to the dip of the dike and, accordingly, were inclined from 56° to vertical. From the footwall of the dike at sublevel intervals of 20 to 25 feet up, the raises ranged from zero to 30 feet.

Stope-development raises consisted of a manway-timberslide, timbered compartment, and an untimbered chute compartment. The average raise section excavated was 6 by 10 feet. The manway-timberslide compartment, which occupied half the raise section, was timbered by 2 sets on 5-foot centers. Caps and posts were 8- by 8-inch; the posts were of different lengths but generally not longer than 6 feet. Caps, normal to inclination of raise, were blocked at both ends by 2-inch headboards. The walls of the manway compartment were lagged by 2- by 12-inch boards behind the posts. The timber was unframed; 2-inch spreaders (boards) of appropriate width were nailed in place. The timberslide was formed of 4 boards (1 by 10 inches each), 2 on the bottom and 1 on each side. The chute compartment was formed by lining the adjacent set of the manway-slide compartment with 2- by 12-inch boards; other sides of the chute were formed by the excavation. The average total timber used per foot of raise was 53.4 board-feet.

Raise rounds were drilled by self-rotating stopers, chucking 1-inch hexagonal drill steel in changes of 2 feet. Detachable, reground steel bits were used in raises. A drill round comprised 16 to 24 holes; a burn- or V-cut was drilled over the chute compartment of the raise. Normally a round was 6.5 feet in depth and advance as per round blasted; 40- or 45-percent gelatin dynamite, detonated by fuse and No. 6 caps, was used. Powder consumption was 9.45 pounds per foot of raise.

Air tugger hoists with 5/8-inch cable serviced the raises. A victualic-coupled, 2-inch airline and a 1-inch waterline, installed between the ladder and timberslide, served the raise and subsequent drilling in the stopes.

Shaft Sinking

A vertical shaft was completed to a depth of 385 feet below the No. 3 adit level to provide access for exploration of the Cassiterite dike and the granite cupola.

The shaft collar is in the limestone footwall of the Cassiterite dike off the No. 3 adit, 600 feet east of the portal. The shaft extends above the adit level to a skip dumping point 40 feet above the collar set. The capacity of the skip pocket at the adit level is 26 tons.

The shaft contains a 4- by 5-foot hoisting compartment and a 2- by 5-foot manway, pipe, and power-cable compartment. The average shaft section excavated was 7 by 10 feet.

The sinking hoist was a Ledgerwood steam hoist converted to compressed air. The drum of the hoist was 3 feet in width and 4 feet in diameter; the drum was grooved for 7/8-inch cable; the rope speed was 300 feet per minute.

Sinking was on a 2-shift-per-day basis, with a 4-hour interval between shifts for clearing smoke. The sinking crew was composed of 2 men per shift in the shaft bottom; these men, with the services of 1 hoistman, performed all phases of the sinking cycle. One timber framer provided both sinking crews with framed timber as needed. Each crew mucked, drilled, and blasted for three consecutive shifts; a set of timber and required pipe, ladders, etc. were installed on the fourth shift.

Jackhammers were used for drilling; both men of the crew normally operated one drill. Tungsten carbide bits and seven-eighths-inch hexagonal steel in changes of 2 feet completed the drilling equipment. Approximately half the shaft section, 4 feet in depth, was drilled per round of 9 holes. A bench cut, which alternated from side to side, was used; thus, 1 end of the shaft bottom always was 2 feet below the other. This type of round not only facilitated drainage and mucking but reduced damage to timbers, as it tended to throw the broken rock toward one end of the shaft instead of up. Drill rounds were charged with 40-percent gelatin dynamite and detonated by electric delays (Nos. 0 to 5), wired to a blasting switch at the collar of the shaft. Holes were spaced widely and blasted lightly to minimize production of fines. Consumption of powder was 10 pounds per foot of shaft advance.

Blasted rounds were mucked into a bail-type sinking bucket of 10-cubic-foot capacity, which rode temporary guides to the dumping point. The dumping mechanism consisted of a weighted door actuated by cable and an air tugger operated by the hoistman. In the closed or dumping position the door would guide the bucket as it was lowered, so that projecting ears at the bottom of the bucket would engage in notches and form a pivot for tilting the bucket. Approximately 20 buckets were mucked per round blasted.

Timbering followed advance of the shaft bottom by 20 feet, with sets installed on 5-foot centers and end-plate bearing sets placed at intervals of 100 feet in depth. All shaft sets were made of sawed timbers, framed and trial-assembled on the surface. Members of the set were 8 by 8 inches, except for the divider, which was 8 by 6 inches. Side lagging was 2 by 12 inches. Sets were hung by 3/4-inch bolts. A blasting set of 10-inch-diameter round timbers, suspended by cables, served as a stage to install the sets and also protected the timber from blasting. Timber used per foot of shaft was 178 board feet.

The shaft was ventilated during sinking operations by 12-inch collapsible tubing connected to a blower at the collar of the shaft.

Water presented no problem to shaft-sinking operations; the shaft was nearly dry to a depth of 385 feet below the adit level. The water accumulating in the bottom of the shaft resulted mostly from drilling operations and was easily bailed out. The flow of water encountered by diamond drilling below the 365 level and water encountered in the old winze at 40 feet below the 365 level indicate that pumping may be a serious problem at increased depths.

A Kimberly-type skip of 30-cubic-foot capacity and an electric, 40-hp. Coeur d'Alene Jr. hoist were installed in the shaft after sinking operations were completed, but no skip pockets were excavated on the lower levels; cars were dumped directly into the skip. The shaft is in hard, siliceous limestone, relatively free of fractures and faults; consequently, the shaft should remain in usable condition until the timber decays.

STOPING PRACTICE

Almost all tin ores obtained from the Cassiterite dike were mined by an advancing system of shrinkage stoping; insignificant production was obtained from stope development and preparation headings. All stopes were above the No. 3 adit level. Nearly all development and much preparation of the stope were done in the limestone foot wall of the dike; only one stope (the A-I-E) was developed in or on the hanging wall of the dike (see figs. 7 and 8). Several methods of stope development, preparation, drilling, and ore drawing were tried; various sizes of stope blocks also were tried.

Development

The greater part of the development drift under the area stoped is in limestone at distances ranging from 5 to 30 feet from the footwall of the dike. The reaminder of the drift is in or near the hanging-wall side of the dike. Stoping blocks were developed by 2-compartment raises, spaced at intervals of 50 to as much as 200 feet apart. Raises in the limestone under and at variable distances from the foot wall of the dike were inclined to roughly parallel the dip of the dike; however, the 365 raise was in the dike, and the 545 raise in the hanging wall. At vertical intervals of 15 to 30 feet, small crosscuts or inclined raises were driven from the main raise to the footwall of the dike to provide access to the stope for subsequent mining. In the A-I-E stope these access sublevels were driven in the dike from the 365 raise in the dike. Most of the raises were driven from the No. 3 adit level approximately 200 feet to the No. 1 adit level.

Preparation

The first stopes mined were prepared for drawing ore by means of sublevel-slusher drifts, in which the ore was transferred to raises and then to chutes on the haulage level. Later the ore was drawn from the stopes through shovel-loader drawpoints at the main haulage level.

For the slusher-drift-chute method of drawing ore, a sublevel slusher-grizzly drift, 20 feet above the back of the haulage level, was driven in the footwall between the main raises. Short crosscuts or inclined raises from the slusher sublevel were driven into the dike at intervals of 20 feet to serve as ore passes for the broken ore to the sublevel; from the sublevel the ore was moved into the chute compartment of the main stope raise by a 34-inch scraper.

Shovel-loader drawpoints were prepared by driving crosscuts from the haulage drift to the footwall of the dike and by driving short raises, which were enlarged upward to connect with similar raises from adjacent drawpoints. A small pillar between adjacent drawpoints remained in the dike. An alternative method was to drive a sublevel in the dike 20 feet above the back of the haulage level between main stope raises and then drawpoint crosscuts and belled raises to the sublevel. Drawpoint crosscuts were spaced at intervals of 20 to 30 feet along the haulage drift, but intervals in the dike varied more because the drawpoint crosscuts were not driven parallel to each other. These crosscuts were driven at various angles with the main haulage drift, some as slight as 30°; others as much as 90°.

Drilling and Blasting

Two methods of drilling and blasting were used in the stopes: (1) The longhole from sublevel stations either in the footwall of the dike near main raises or in the dike itself some distance in the stope from the main raises; or (2) the conventional breast round, drilled by jacklegs from the top of the broken ore. Selecting the drilling method to be employed in the stopes was governed largely by the availability of competent miners and by the hardness of the dike; the longhole system was used more often in the relatively softer parts of the dike.

Longhole rounds usually consisted of 8 to 12 holes up to 35 feet in length, drilled by a column-mounted, 4-inch drifter with sectionalized steel and tungsten carbide bits. The holes were drilled diagonally across the dike; the average burden was 8 feet. The longer holes of a round generally were sprung with 8 to 10 sticks of powder; the chamber thus formed at the end of the hole was loaded with 40 to 60 sticks of 40- or 45-percent gelatin dynamite, detonated by electric delays. Figure 8 illustrates a typical longhole round.

A breast round by jacklegs usually consisted of 12 holes drilled to a depth of 7 feet; the average burden on the holes was about 4 feet. The round was blasted with 40- or 45-percent bulk-strength gelatin dynamite, detonated by No. 6 caps and fuse. Figure 8 illustrates stoping progress.

Both methods of drilling were used in all stopes. Essentially, excavation was in horizontal slices, which were started from raises at each end of the stope block. Because the end of the stope block mined by longholes was allowed to advance vertically at a greater rate than the end mined by jacklegs, the back of the stope soon was sloped steeply instead of remaining horizontal.

Drilling and blasting of ore were contracted to the miners at 65 to 75 cents per ton broken - the price depended on the working place. The small sublevel crosscuts and raises inside the stope, incidental to preparing the drilling stations, were driven by the miners at a contract price of $10 per linear foot. The company furnished all tools, supplies, and equipment provided extra help for piping, etc., and occasionally allowed pay for extra shifts to compensate for lost time caused by stuck steel, machine breakdowns, or other delays not the fault of the miners. Payment to the miners on contract work was based on the measured volume of stope excavated or on the linear feet of sublevels or raises driven.

Drawing Ore

Shovel loading from drawpoints was superior to the chute and slusher-grizzly-sublevel method of drawing ore from stopes. The only difficulty experienced by the drawpoint method resulted from drawpoint crosscuts being too short, too sharply angled to the main haulage drift, or too widely spaced. Some coarse waste was sorted at the drawpoints. Ore was hauled to the mill in trains of 5 end-dump cars (21-cubic-foot capacity each) by a 1-1/2-ton storage-battery locomotive. The two-man haulage crew also operated the shovel loader.

Support

Small pillars of ore were left in the dike in the vicinity of the main raises; otherwise, the stope openings were unsupported.

Discussion of Methods and Results

Direct comparison of the longhole and conventional methods of shrinkage stoping as practiced at Lost River is not possible, because both methods often were employed in the same stopes and performance records were not maintained separately.

Productivity at the stoping face ranged from 12 to 70 tons of ore broken per (driller) man-shift. Powder consumption in individual stopes and by both drilling methods ranged from 0.6 to 1.3 pounds per ton of ore broken. Average powder consumption for the total ore reported broken was 0.76 pound per ton. Slightly more than 4.5 tons of ore was broken per hole drilled, regardless of the drilling method used. Records of bit wear and steel consumption per foot drilled were not maintained.

Accurate figures on recovery and dilution are difficult if not impossible to obtain, because ore faces were not sampled or measured systematically. Because many longholes were drilled diagonally across the dike, they penetrated the walls and broke waste, which caused high dilution. For the same reason and also because stopes could not be inspected or sampled properly, ore was left in the walls where it soon was covered by broken rock. The average grade of ore drawn from the stopes was only 1.1 percent tin. The average tin content of all dike samples in or adjacent to the stoped blocks was approximately 1.4 percent. Maximum fluctuation in the grade of mill heads was 0.7 percent. A progressive decrease of 0.1 percent tin in the average grade of ore drawn during successive 6-month periods apparently was largely attributable to progressively increasing amounts of dilution as stoping progressed. (See fig. 12.)

The length of most of the stope blocks, the level interval of 200 feet, and the unbalanced stoping methods resulted in driving numerous branch raises and sublevels to provide working faces; about 1 foot of interstope-preparation headings, raises, and sublevels was driven per 130 tons of ore broken.

A systematic method of stoping, based upon observations of the operation, is outlined in the Appendix.

From September 1951 to April 1955, 23 compensable injuries occurred in the mine. Falls of rock caused 1 fatality and 7 minor injuries, nearby all of which occurred in 53 E. raise during the first year of operations, when many of the crew were inexperienced miners. There were no injuries from rock falls in stopes. One fatality occurred in drifting when a miner delayed too long at the face after spitting a round.

MILLING

Summary and Introduction

The United States Tin Corp. mill at Lost River, Alaska, was opened in June 1952 and operated semicontinuously until September 1955.

According to company records, over 51,000 short tons of ore (with an average 1.13 percent Sn) was treated by gravity-concentration methods to produce 687.1 tons of concentrate (with an average 52.0 percent Sn). The average recovery of tin was 62.0 percent. The gravity concentrate was treated by flotation to remove sulfides; the beneficiated concentrate was barreled and shipped to the tin smelter at Texas City, Tex.

Mill recovery was well below that indicated by laboratory mineral-dressing tests, due chiefly to mechanical limitations of the grinding circuit that prevented disintegration of ore without excessive sliming of cassiterite. Information in this report has been obtained from personal observation, laboratory investigations, unpublished memoranda, and correspondence. The mill data are from monthly reports of the plant superintendent to the board of directors of the United States tin Corp.

Development of Mill

Original Plant, 1951-52

Because of financial complications and a tight time limitation imposed by shipping conditions, the original mill at Lost River was designed to use most of the heavy machinery already at the mine, supplemented by such new and used equipment as was available immediately. The mill building was built and equipment installed during the fall and winter of 1951, but, owing to water-supply difficulties, the plant was not operated until June 1952.

The plant was poorly equipped and not easily adaptable to changes in flowsheet; furthermore, an urgent demand for maximum production caused an all-out effort to pass as much tonnage through the mill as possible without much attention either to grinding methods or efficient concentration. The ore that was first treated could be ground easily; consequently the mill operators were able to greatly overload the concentration equipment.

The flowsheet of the original plant is shown in figure 9. Crushing was effected by a 10- by 20-inch jaw crusher, set for 1-1/4-inch discharge and fed by chute from a small, coarse-ore bin, followed by a 3- by 6-foot shaking screen, with 3/8-inch square mesh, and an Acme road-machinery jaw crusher of indeterminable size to reduce the screen oversize to minus-3/8 inch. The screen and secondary crusher were bypassed after 1 or 2 days of operation, because it was found that the bulk of the discharge from the primary crushers was minus-3/4 inch, and the ball mill apparently could handle such feed without a secondary crusher stage.

The crushed ore was charged into the fine-ore bin by means of a bucket elevator, which maintained elevation without excessive blasting for erecting the lower part of the mill.

Ore was removed from the fine-ore bin by a belt conveyor to a 3- by 6-foot shaking screen. The screen was originally installed to remove ore fines before milling but was employed only as a vibrating feeder for the 4- by 5-foot Union Iron Works ball mill. Pulp from the ball mill was discharged to a 14-mesh screen, with the oversize passing to two Denver duplex jigs; the jig tailing was returned to the ball mill. The screen undersize was split and fed to three Humphreys spirals, which produced tailing and a rough concentrate. The latter was cleaned on a Wilfrey table to yield a final concentrate and a middling, which was returned to the grinding circuit.

Operations for the first few days consisted of sporadic 4- to 6-hour runs at a feed rate of 60 tons per 24 hours. The results were extremely unsatisfactory, and the screen and jigs were eliminated after four short runs.

The spirals, which had been installed without the bottom 1-1/2 tunrs, produced no low-tin reject and were discarded after only a brief trial. Much of the spiral feed was below the optimum-size range for spiral concentration; for fine feed the spirals apparently were overloaded. Subsequent laboratory-testing results indicated that tabling is superior to spiral concentration for treating Lost River ore.

With the spirals and jigs removed from the circuit, the only remaining piece of concentration equipment was the single Wilfley table; consequently, the effective capacity of the mill was about 10 to 12 tons per 24 hours. Because of the demand for marketable concentrate, however, the company felt that it was important to treat as much ore as possible. Up to 40 tons of ore per day was passed over the single table. Milling loss was excessively high; tin recovery ranged from 35 to 45 percent.

An attempt was made during this period to prevent overgrinding during a three-shift run with half the balls removed from the mill. No attempt was made to control the density of the pulp. The test was discontinued because the treatment effected no noticeable improvement in recovery from the badly overloaded table.

During the summer, six additional shaking tables and a hydraulic classifier were installed. With these and controlled grinding, the plant should have made fairly effective recovery of tin from a moderate tonnage. Treatment of ore, however, was increased to 100 tons per operating day during August and September, and no further attempt was made to prevent overgrinding. Results were inconsistent and erratic. The recovery of tin from ore averaging nearly 2 percent Sn ranged from 45 to 65 percent.

The mill was closed early in October because of a shortage of water.

Additions and Changes, 1953

During the winter of 1952-53 the arrangement of mill equipment (see flowsheet, fig. 10) eliminated many disadvantages of the original treatment plan. Fines were removed before grinding, coarse sands were jigged to remove ore minerals as soon as possible (adding an Esperanza classifier allowed better control of water to the ball mill) and sized feed was supplied to a concentration section of adequate capacity.

Operation and maintenance problems, however, were intensified by the expanded concentration section. Equipment was crowded together within the limits of the mill building, and the remaining passageways were narrow. Lack of space hampered routine repairs on many machines. Operating the tables was difficult, because one set of tables was often obscured from another set by intervening equipment. Limitations on both cost and space precluded mechanical transportation of jig concentrate to the concentrate settling box, so this job was accomplished by hand-carrying the concentrate in "powder box" lots.

Furthermore, it soon became evident that the concentration section had been expanded beyond the capacity of the crushing and grinding section. A crushing and grinding procedure was planned, suggested by laboratory tests conducted on Lost River ore by the Bureau of Mines. These tests showed that by crushing the ore to one half-inch and wet-screening it on a 6-mesh screen, at least 50 percent (containing about 75 percent of the total tin) could be sent directly to the concentrating circuits without being ground. For a 100-ton-per-day operation, the ball mill was expected to grind not to exceed 50 tons of minus 1/2-inch feed per day and to regrind the middling returned from the tables and jigs. Only about 25 percent of the total tin was expected to be in the ball-mill feed, and thus sliming of a major portion of the cassiterite would be prevented. Sliming was expected to be minimized further by employing a light ball load, low pulp density, and a high circulating load in the ball-mill circuit.

When milling was resumed in June, a series of tests was made over a period of 13 days to determine the effect of varying ball loads and pulp densities in the ball mill. These tests showed that attrition grinding with small ball loads minimized sliming, and recovery of tin was 76 to 81 percent; also that low pulp densities had the same general effect. With both treatments, the ball-mill discharge contained a large proportion of plus-6-mesh material for recirculation; consequently, the tonnage treated was limited by the capacity of the return elevator. Conversely, it was found that increased ball loads and pulp densities reduced the oversize in the discharge, allowed increased tonnage, and resulted in lower recoveries.

The secondary jaw crusher proved to be incapable of crushing below three fourths of an inch; it failed mechanically several times. During the breakdown periods, it was necessary to operate the primary crusher with a minimum-discharge opening, which seriously limited the capacity of that unit and resulted in an ore product of approximately minus-1 inch.

The grinding problem was complicated further by the fact that part of the mill feed proved to be much harder than the friable ore that had been treated the previous year and harder than any that had been tested in the laboratories. The relative hardness of the ore milled during the summer of 1953, compared with ore milled during other periods, is illustrated by typical screen analyses of ball-mill feed (see tables 2 and 3).

TABLE 2. - Screen analyses, ball-mill-feed composite, July 18-22, 1953

Product, mesh	Weight, percent	Assay, percent Sn	Sn distribution, percent
Plus 3	71.1	1.10	64.6
3 to 6	9.3	1.36	10.4
6 to 8	3.7	1.56	4.8
8 to 10	3.0	1.68	4.1
10 to 14	2.5	1.78	3.6
14 to 20	2.2	1.64	3.0
20 to 32	1.6	1.56	2.1
32 to 48	1.1	1.56	1.4
48 to 65	1.0	1.82	1.5
65 to 100	.9	1.72	1.3
100 to 150	.75	1.40	.9
150 to 200	.75	1.20	.7
200 to 325	1.3	1.10	1.2
Minus 325	.5	.90	.4
Calculated head	100.0	1.21	100.0

TABLE 3. - Screen analyses, ball-mill-feed composite, February 1955

Product, mesh	Weight, percent	Assay, percent Sn	Sn distribution, percent
Plus 3	14.5	0.71	10.4
3 to 6	29.1	.68	19.9
6 to 8	8.3	1.04	8.7
8 to 10	6.6	.92	6.1
10 to 14	6.2	.96	5.9
14 to 20	6.1	1.14	7.0
20 to 32	5.4	1.40	7.6
32 to 48	4.1	1.45	5.9
48 to 65	2.7	1.60	4.3
65 to 100	2.5	1.75	4.4
100 to 150	2.2	2.07	4.6
150 to 200	2.7	2.12	5.7
200 to 325	1.9	1.65	3.4
Minus-325 sand	.7	4.50	3.2
Minus-325 slime	7.0	.41	2.9
Calculated head	100.0	1.00	100.0

Combining harder, coarser ore resulted in removing (as minus-6-mesh material) only 20 percent of the original weight before grinding. Mechanical limitation of the return elevator precluded attrition-grinding treatment of ore at a rate exceeding 0.5 ton per hour; consequently, attrition grinding was abandoned. Grinding with a full ball load was resumed, and the grinding capacity was measured at approximately 1.5 tons per hour. For one 24-hour run at this rate, 16 percent of the ball-mill discharge was coarser than 6-mesh and contained 11 percent of the tin; 44 percent of the discharged material, however, was finer than 200-mesh and contained 37 percent of the total tin. Concentration of 1-percent tin ore ground in this manner yielded a recovery of about 60 percent of the total tin.

Because the ball mill was incapable, with the harder ore, of supplying enough feed to operate the expanded concentration section efficiently and because a major ball-mill trunnion broke, it was decided that the crushing and grinding section of the mill should be enlarged. The old secondary jaw crusher was replaced by a 22-inch cone crusher capable of reducing ore from 1-1/4 inches to 1/4 inch. A 4- by 10-foot Marcy rod mill was installed as the primary grinder. The rod mill also was used for regrinding table middling for a time, but later the ball mill was reconditioned and used for this purpose. With the new grinding section, tonnage was maintained at about 100 tons per day, and mill results gradually were improved.

Final Flowsheet, 1954-55

During the winter of 1953-54 the company installed several additional shaking tables, a flotation unit for removing sulfides, and a thickening tank to facilitate the treatment of fines and the reclamation of water. This equipment was housed in a new building adjacent to the existing mill. Figure 11 shows the schematic flowsheet of the mill when this program was completed.

Installing the thickening tank and adding tables increased the recovery of tin but did not alleviate the crowded conditions in other sections of the mill.

The plant layout was not altered significantly after February 1954 and remained as indicated in figure 11, but the flow of material was changed. The most important change was occasioned by mechanical failure of the ball mill and the necessary rerouting of coarse table middling to the rod-mill circuit. Numerous minor changes also were effected on an experimental basis in an effort to improve plant efficiency; thus distribution of feed, middling return ratio, and treatment of slime were varied from time to time.

The small ball mill installed for grinding coarse concentrate before flotation treatment and the filter installed for dewatering the flotation cassiterite product never were used.

Gravity Concentration

Gravity-concentration data, compiled from monthly reports of the superintendent, are summarized in table 4.

Evaluating these data requires a knowledge of sampling and calculating methods used in the plant. No weightometer or other device was used to determine the weight of material entering the mill. No automatic sampling equipment was employed. Samples of ball-mill feed and plant tailing were taken at fairly regular intervals by diverting the flow of material for a short time. Concentrate was weighed and sampled daily or on a shift basis by a "grab sample" technique.

Mill recovery was calculated from the tin analyses of the products. The weights of feed and tailing were computed from the calculated recovery and weight of the concentrate.

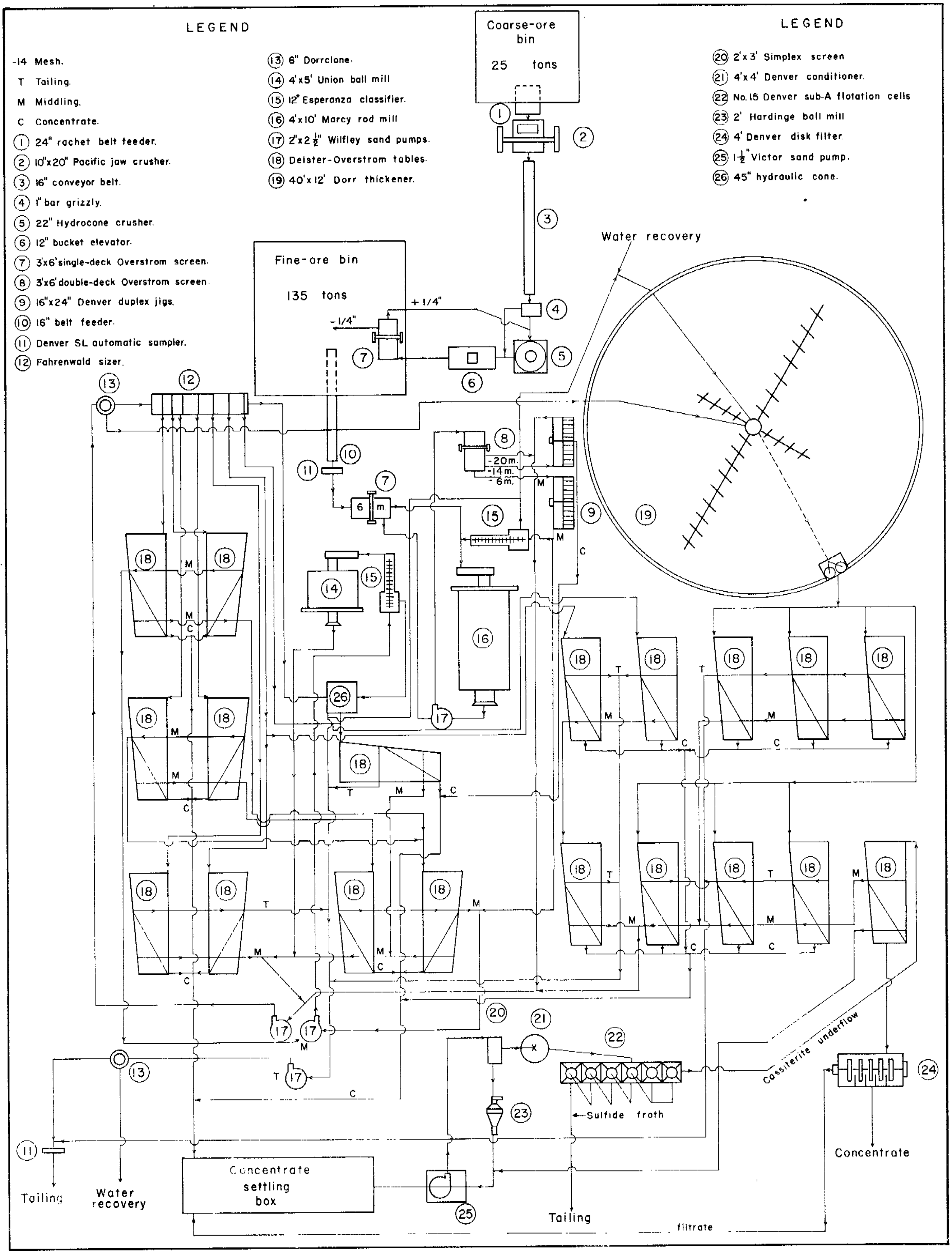

FIGURE 11. - Flowsheet, Lost River Mill, 1954.

TABLE 4. - Summary of gravity milling operations

Period	Tons milled (dry basis)	Hours operated	Rate, tons/24 hours	Concentrate, tons (dry)	Average assay, percent Sn: Head	Average assay, percent Sn: Concentrate	Average assay, percent Sn: Tail	Tin content, pounds: Head	Tin content, pounds: Concentrate	Calculated tin recovery, percent
1952:										
July	561.	279.	48.2	10.90	2.38	70.37	1.03	26,738	15,338	57.36
August	719.	164.	105.1	12.576	1.93	72.20	.70	27,730	18,158	65.48
September	1,900.	456.	100.0	27.262	1.74	54.50	.98	66,228	29,718	44.87
October	257.	97.	63.6	4.434	1.93	56.13	.91	9,920	4,980	50.20
November-December1/	-	-	-	-	-	-	-	-	-	-
Total	3,437.	996.	82.8	55.172	1.90	61.80	.92	130,616	68,194	52.21
1953:										
January-May2/	-	-	-	-	-	-	-	-	-	-
June	58.0	113.5	12.3	.982	1.75	53.44	.86	2,029	1,050	51.74
July	669.4	307.2	52.3	10.169	1.26	45.98	.57	16,851	9,352	55.50
August	1,223.4	519.4	56.5	18.320	1.17	44.57	.51	28,532	16,329	57.23
September	348.0	148.3	56.3	4.432	1.16	50.99	.51	8,052	4,520	56.13
October3/	-	-	-	-	-	-	-	-	-	-
November	821.5	187.0	105.4	11.106	.88	36.13	.39	14,374	8,026	52.84
December	2,448.5	495.3	118.6	31.226	1.02	41.83	.50	50,081	26,125	52.16
Total	5,568.8	1,770.7	75.5	76.235	1.08	42.89	.50	119,919	65,402	54.54
1954:										
January	2,323.	605.3	92.1	41.419	1.20	40.84	.48	55,815	33,831	60.61
February	1,784.	498.8	85.8	30.904	1.10	40.86	.40	39,261	25,252	64.32
March	1,341.	471.0	68.3	19.097	.93	43.81	.31	24,911	16,731	67.16
April	922.	283.9	78.0	12.457	1.04	49.24	.39	19,195	12,266	63.90
May	2,486.	582.5	102.4	31.229	1.31	61.07	.55	65,254	38,145	58.96
June	2,617.	585.5	107.3	29.690	1.25	64.54	.53	65,614	38,324	58.41
July	2,541.	695.4	87.8	32.789	1.28	54.69	.58	65,100	35,864	55.09
August	2,893	684.7	101.4	33.330	.98	53.36	.37	56,486	35,568	62.97
September	2,247.	498.9	109.7	26.788	.98	55.23	.33	44,152	29,591	67.02
October	3,182.	685.8	111.3	39.309	1.01	54.02	.35	64,323	42,468	66.02
November	3,318.	672.6	118.4	40.749	1.04	55.31	.37	69,249	45,074	65.09
December	3,039.	666.1	109.4	35.003	1.02	57.12	.37	62,080	39,988	64.41
Total	28,693.	6,930.5	99.6	372.764	1.10	52.73	.42	631,440	393,102	62.25
1955:										
January	3,006.	701.0	102.9	35.142	.95	55.28	.30	56,888	38,854	68.30
February	2,231.	561.9	95.3	31.226	1.06	50.00	.37	47,436	31,224	65.82
March	427.	87.7	116.9	4.990	1.03	57.15	.40	8,676	5,704	65.68
April4/	-	-	-	-	-	-	-	-	-	-
May	593.	153.5	92.7	7.382	1.20	49.41	.43	14,203	9,233	65.01
June	988.	287.5	82.5	14.058	1.08	53.06	.33	28,116	14,918	70.17
July	1,928.	503.2	92.0	25.674	1.01	50.61	.34	38,905	25,989	66.80
August	2,560	687.1	89.4	31.574	.97	50.73	.35	49,719	32,037	64.44
September	2,509.	599.0	100.5	32.916	.99	45.60	.40	49,618	30,018	60.50
Mill closed September 26	-	-	-	-	-	-	-	-	-	-
Total	14,242.	3,580.9	95.5	182.962	1.03	51.37	.38	293,561	187,977	64.03
Grand total	51,940.8	13,278.1	93.9	687.133	1.13	52.00	.45	1,175,536	714,675	60.80

1/ Mill closed - lack of water.
2/ Remodeling reported almost completed.
3/ Rod mill and cone crusher installed.
4/ Mill closed March 5-May 23; lack of fuel oil.

When the flotation section was begun, a discrepancy existed between the weight of products from flotation and that of concnetrate presumably produced in the gravity section (flotation feed); this was labeled "unaccounted-for loss". Metallurgical balance was maintained by altering the flotation-tailing analysis. Investigation showed that the method of sampling the gravity concentrate did not reflect the true moisture content; therefore, the record gravity-concentrate weights (and consequently the calculated mill-feed weights) were high. This condition was corrected in April 1955.

If all discrepancy in weight, reported as "unaccounted-for loss" in the monthly reports of the superintendent, is assumed to be due to errors in weighing the gravity concentrate, the total fine tin in the gravity concentrate should be reduced by 24,655 pounds from the figure reported in table 4. Using the corrected concentrate figure, the calculated tonnage of ore fed to the mill would be reduced to 50,230 short tons. These calculations cannot be accepted as correct, however, because of the absence of flotation data for February 1955.

Using the revised tonnage and the amount of tin received at the Texas City smelter, the indicated overall recovery of tin at the Lost River plant was about 54.5 percent. This figure agrees with the recovery figure obtained, based on the total receipts of tin at the smelter and calculated by the same methods used by the United States Tin Corp. Thus, it is reasonable to assume that a corrected figure of 50,230 tons treated is approximately correct. It should be noted, however, that an average correction cannot be applied directly to any one period, because the largest discrepancies probably occurred during the early stages of operation and were negligible during the last few months.

Flotation

A flotation section was added to the circuit during the spring of 1954 to remove sulfides from the gravity concentrate. The object of this treatment was twofold: First, to avoid heavy smelter penalties on excess sulfur and base metals and, second, to raise the grade of concentrates to lessen the cost of freight and treatment per ton of tin contained.

The flotation section was operated whenever warranted by the supply of gravity concentrate. In spite of untrained flotation operators and the usual problems of sporadic operation, the treatment was satisfactory. Generally, the flotation of sulfides increased the tin grade of the concentrate 5 to 7 percent with little loss of tin. Reagents used per ton of flotation feed included: Sulfuric acid, 2.5 pounds; pine oil, 0.4 pound; and amyl xanthate, 1.0 pound.

Results of flotation treatment are shown in table 5; these data were compiled from the monthly reports of the superintendent. For calculation, it was assumed that the moisture content (hence the calculated dry weight) of the semidry, final tin concentrate was more accurate than that of the gravity concentrate (flotation feed). Recoveries were calculated from analyses of flotation feed, concentrate, and tailing. Weights of the feed and sulfide concentrate were computed, using the calculated recovery and the corrected dry weight of the tin product.

Data pertaining to shipments of final concentrate are summarized in table 6.

TABLE 5. - Summary of flotation results

Year	Feed			Tailing (sulfide)			Concentrates (tin)			
	Weight, pounds (dry)	Assay, percent Sn	Tin content, pounds	Weight, pounds (dry)	Assay, percent Sn	Tin content, pounds	Weight, pounds (dry)	Assay, percent Sn	Tin content, pounds	Recovery, percent Sn
1954:										
July	265,736	48.206	113,751	73,374	1.398	1,024	192,358	58.603	112,727	99.10
August	154,095	46.666	71,910	34,132	1.200	410	119,963	59.602	71,500	99.43
September	49,633	54.701	27,150	4,963	1.007	50	44,670	60.667	27,100	99.82
October	71,275	53.932	38,440	8,414	.998	84	62,861	61.017	38,356	99.78
November	64,705	55.098	35,652	5,284	.618	33	59,421	59.943	35,619	99.91
December	69,268	56.947	39,446	5,305	.432	24	63,963	61.633	39,422	99.94
1955:										
January	66,659	54.875	36,579	3,752	.446	18	62,907	58,119	36,561	99.95
February1/	-	-	-	-	-	-	-	-	-	-
March	28,905	49.383	14,274	3,274	.641	21	25,631	55.608	14,253	99.85
April2/	-	-	-	-	-	-	-	-	-	-
May	17,067	49.405	8,432	2,303	.738	17	14,764	56.997	8,415	99.80
June	21,389	52.976	11,331	1,197	.835	10	20,192	56.067	11,321	99.91
July	55,787	49.899	27,837	4,656	.322	15	51,131	54.413	27,822	99.95
August	58,835	50.001	29,418	3,713	.539	20	55,122	53.333	29,398	99.93
September	72,981	45.283	33,048	10,943	1.151	122	62,038	53.074	32,926	99.63

1/ Data not available.
2/ Mill shutdown.

TABLE 6. - Concentrate received at Texas City, Tex.

Lot No.	Drums	Net pound (wet)	Moisture, percent	Net pound (dry)	Assay, percent								
					Sn	Soluble Sn	S	Pb	Sb	As	Bi	Cu	Zn
29810	27	42,533	8.25	39,020	57.31	0.07	2.29	0.55	0.02	2.61	0.07	0.16	0.25
29811	27	42,907	7.85	39,535	59.08	.09	2.23	.38	.02	2.51	.06	.17	.21
29812	27	42,715	7.78	39,388	59.65	.09	2.05	.36	.02	2.80	.06	.15	.19
29813	27	43,057	7.73	39,725	63.00	.09	1.64	.32	.02	1.99	.06	.27	.19
29814	27	38,993	7.86	35,928	58.38	.07	2.24	.37	.02	2.50	.07	.17	.22
29815	27	42,348	7.46	39,185	59.52	.07	2.15	.50	.02	2.41	.07	.15	.19
29816	27	42,092	7.78	38,817	61.22	.09	2.07	.26	.02	2.25	.07	.16	.21
29817	27	42,682	7.82	39,341	59.40	.08	2.00	.51	.02	2.36	.07	.15	.20
Shipment 1 ...	215	337,327	7.82	310,939	59.71	.08	2.07	.40	.02	2.42	.07	.17	.21
30233	54	85,557	8.06	78,661	60.04	.09	1.81	.71	.02	2.25	.08	.09	.14
30234	54	85,717	8.21	78,680	59.66	.06	1.58	.37	.02	1.55	.07	.07	.16
30235	54	85,645	8.46	78,399	60.24	.02	1.07	.24	.02	1.49	.06	.09	.17
Shipment 2 ...	162	265,919	8.24	235,740	59.98	.06	1.49	.44	.02	1.76	.07	.08	.16
Shipment 3 1/..	236			336,231	2/57.00								
31439	50	71,223	8.13	65,433	52.69	.07	.83	.18	.02	1.49	.03	.09	.27
31440	50	71,484	8.09	65,701	51.73	.08	.93	.16	.02	1.70	.02	.09	.29
31441	50	68,338	8.85	62,290	50.89	.08	.80	.23	.02	1.47	.02	.13	.32
Shipment 4 ...	150	211,045	8.35	193,424	51.79	.08	.86	.19	.02	1.55	.02	.10	.29
Total				1,076,334	2/57.448	(618,340 pound of tin)							

1/ Data for shipments 1, 2, and 4 summarized from smelter settlement sheets; complete data for shipment 3 not made available to this office.
2/ Average assay after deduction of soluble tin.

Operating Efficiency

The operating efficiency of the Lost River mill, based upon the ratio of time operated to time idle, was poor. Records are incomplete concerning the reasons for all lost time. Such records were detailed from January to December 1954, however, and are summarized in table 7.

TABLE 7. - Operating efficiency, 1954

Remarks	Hours	Percent of total lost time	Percent of total time
Time operated	6,930.5	-	78.2
Time lost	1,832.6	100.0	21.8
Cause:			
Mechanical failures and repairs	376.6	20.6	-
Operational difficulties	139.5	7.6	-
Lack of power	398.8	21.8	-
Lack of water (February to May)	493.7	26.9	-
Lack of oil (September)	178.7	9.7	-
Sundays and holidays	144.0	7.9	-
Experimental tests on individual units	101.3	5.5	-

Analytical Laboratory

Analytical requirements for operating the mine and mill include the determination of tin, and to a smaller extent, tungsten, lead, zinc, arsenic, and sulfur. For accurate determination of tin and tungsten in Lost River ores and mill production, it is necessary to use fusion methods to dissolve the samples and to pay meticulous attention to certain conditions and procedures. As a result, only a limited number of determinations can be made by an analyst in a day.

The analytical laboratory at Lost River was small. The facilities were adequate, for one chemist to perform the analyses necessary for routine mill operations but they were inadequate for a chemist to perform all the additional analyses necessary during experimentation with various mill circuits or during periods of mine exploration and development.

Concentration Problems

Prevention of Overgrinding

The Lost River ore essentially contains kaolinite, altered feldspar, quartz, white topaz, cassiterite, a magnesia-bearing calcite, fluorite, sericite, and zinnwaldite; some muscovite, limonite, pyroxene, amphibole, and chlorite; varying amounts of galena and sphalerite; small amounts of arsenopyrite, wolframite, scheelite, cerussite, and apatite, trace amounts of altered pyrite, chalcopyrite, smithsonite, malachite, bismuthinite, and stibiconite.

Petrographic study indicates that most cassiterite is liberated in the minus-48-, plus-100-mesh size range, although some cassiterite is liberated as coarse as 10-mesh, and a small amount remains locked in the minus-200-mesh fraction.

Laboratory testing and mill operations show that grinding the ore with light ball load and low pulp density liberates the cassiterite in the coarser sand fractions and increases recovery. Mechanical limitations of the mill grinding circuit, however, did not allow this practice; consequently, a large quantity of the tin mineral was ground finer than 200-mesh. Much of the fine cassiterite thus formed was held mechanically by the kaolinite slime and thus rejected into the mill tailing.

The data in tables 8, 9, and 10 show that most of the tin in the tailing was present in the minus-200-, plus-400-mesh sand.

TABLE 8. - Size analysis, sand tailing
(September 16-23, 1955)

Product, mesh	Weight, percent	Assay, percent Sn	Sn distribution, percent
Plus 20	1.14	0.22	1.0
20 to 28	4.52	.24	3.5
28 to 35	13.32	.28	11.7
35 to 48	31.18	.24	23.8
48 to 65	21.58	.20	13.6
65 to 100	9.17	.30	8.9
100 to 200	11.31	.41	14.6
200 to 400[1/]	5.10	1.30	21.0
Minus 400[1/]	2.68	.21	1.9
Calculated sand tailing	100.00	.32	100.0
Assay-sand tailing	-	.32	-

1/ Separated by decantation.

TABLE 9. - Size analysis, slime tailing
(September 16-23, 1955)

Product, mesh	Weight, percent	Assay, percent Sn	Sn distribution, percent
Plus 65	4.95	0.28	3.9
65 to 100	3.60	.25	2.5
100 to 200	10.97	.33	10.0
200 to 325[1/]	11.54	1.15	36.8
325 to 400[1/]	6.14	.50	8.6
400 to 800[1/]	11.47	.31	10.0
800 to 1,600[1/]	31.19	.23	19.9
Minus 1,600[1/]	20.14	.15	8.3
Calculated slime tailing	100.00	.36	100.0
Assay slime tailing	-	.35	-
Combined 200 to 400	17.68	.93	45.4

1/ Minus-200-mesh sand divided into 5 sized products by elutriation; average particle size (theoretical mesh) of each product determined with Andreasen pipette.

Variations in Ore

In addition to variations in hardness of the ore (mentioned previously) considerable variation in grade of ore was noted from day to day. This, of course, necessitated constant adjustment of table operations (transverse tilt, wash water, position of product splitters), unnecessary with constant feed grade.

Figure 12 shows variations in the grades of ore treated in the mill during 1954.

TABLE 10. - Size analysis, total plant tailing
(September 16-23, 1955)

Product, mesh	Weight, percent	Assay, percent Sn	Sn distribution, percent
Plus 20	0.55	0.22	0.3
20 to 28	2.20	.24	1.5
28 to 35	6.48	.28	5.4
35 to 48	15.16	.24	10.7
48 to 65	13.04	.21	8.0
65 to 100	6.31	.29	5.4
100 to 200	11.14	.38	12.5
200 to 325 1/	7.42	1.17	25.9
325 to 400 1/	3.67	.60	6.5
400 to 800 1/	6.59	.36	7.1
800 to 1,600 1/	16.32	.23	11.3
Minus 1,600 1/	11.12	.16	5.4
Calculated tailing	100.00	.34	100.0
Assay tailing	-	-	-
Combined 200/400	11.09	1.16	32.4

1/ Minus-200-mesh sand divided into 5 sized products by elutriation; average particle size (theoretical mesh) of each product determined with Andreasen pipette.

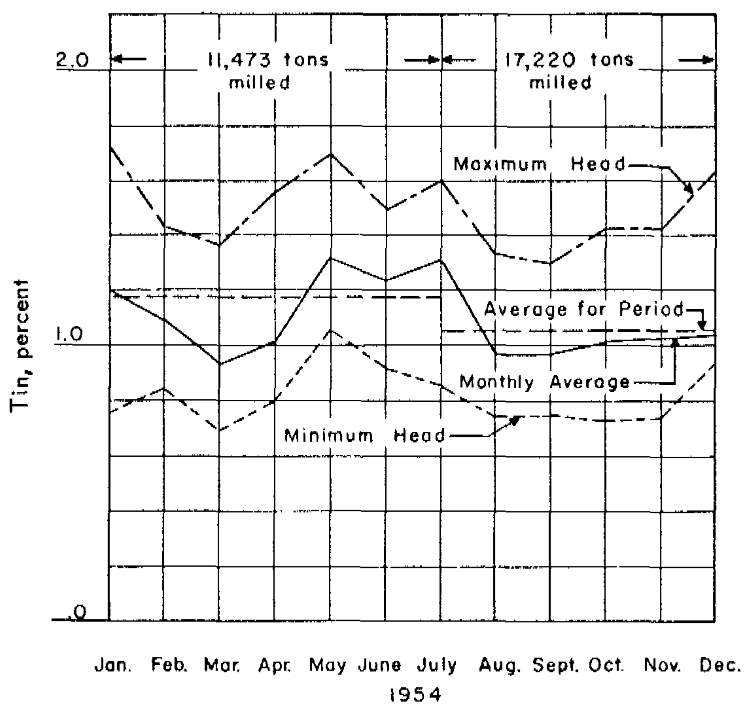

FIGURE 12. - Variation in Grade of Ore Supplied to Mill.

Water Requirements

The preponderance of fine clay in Lost River ore requires using extremely low pulp density during classification to prevent mechanical inclusion of fine cassiterite in clay flocs and to allow rejecting a slime of low tin content. In addition, water requirements of the table-concentration section are high because of fineness of table feed.

Thus, the total water consumed in the mill was higher than normal. Approximately 300 gallons of water per minute was supplied to the mill. This amount equals 27 tons of water per ton of ore, based upon treatment of 100 tons of ore per day.

During certain periods in 1954 and 1955, the fresh water was supplemented by an unknown quantity of "reclaimed" water. Water requirements, therefore, probably are 30 to 40 tons of water per ton of ore treated.

Reclamation of water from the clay-slime fraction was not very satisfactory because of the time required for the extremely fine clay particles to settle. Table 9 shows that over 50 percent of the total slime tailing is finer than 800-mesh (18.5 microns). Settling possibly could be accelerated by adding flocculating agents.

Such treatment, however, would necessitate supplemental treatment of the reclaimed water with dispersing agents to prevent formation of flocs in the grinding and classification circuits.

Plant Deficiencies

The Lost River mill was not planned and constructed as a unit; instead it was built piecemeal to remedy initial deficiencies and to meet the needs of increased production. The result was a plant not only inconvenient to operate but extremely difficult to service.

Most obvious among the operating inconveniences were the following:

1. The coarse-ore bin was entirely too small, and its placement did not provide adequate space for handling and dumping ore trains.

2. Transporting ore from the coarse-ore bin to the primary crusher was difficult. Originally only a chute led from the bin to the crusher; later the chute was replaced by a stationary grizzly. Although the latter was an improvement, with either device the full time of one man was required to drag the ore into the crusher hopper.

3. The kaolinitic material in the ore tended to pack and to plug the secondary cone crusher; this made constant adjustment to the machine necessary, which in turn resulted in lack of uniform size of the ball-mill feed.

4. Equipment was crowded; passageways between most machines barely allowed access to some for operational adjustment.

5. Crowded equipment and low ceilings obscured the view. All equipment in a section or unit could not be watched from a single position; therefore, additional operators were required.

6. Lack of headroom limited the slope of some of the launders and resulted in work stoppage and overflow problems.

7. The mill had no concrete floor and sump, so that spilled material could not be cleaned up easily or returned to the circuits.

8. There was no mechanical method for handling concentrate. The jig concentrate was carried to the settling box in powder boxes, the combined gravity concentrate was hand-fed to the flotation circuit, and the final concentrate was stacked and charged into shipping barrels by hand.

Repairs and servicing mainly were hampered for three reasons:

1. Crowded conditions allowed no space for making repairs or for laying tools and spare parts.

2. Lack of an overhead crane made movement of heavy equipment slow, laborious, and costly.

3. Because the equipment added from time to time was largely whatever was available for the lowest price, the mill contained a variety of types and models of machinery. Consequently, an unusually large supply of spare parts was stocked, and the mechanics had to familiarize themselves with a number of different machines.

COSTS AND RELATED OPERATIONAL DATA

Basis for Cost Analysis

The costs presented in this section were obtained by analyzing company records, consulting the mine management, and directly studying the operations. Because company records were maintained primarily for financial purposes rather than as a guide to operations, it was necessary to reassemble operational-cost data from basic documents, such as freight bills, invoices, time reports, and warehouse receipts. By this method and a first-hand study of operating procedures, it was possible to reach a fair approximation of actual costs for exploration, development, stoping, and milling, although many inconsistent items could not be reconciled. It was impossible to analyze all parts of these operations, because the records were not detailed. For the same reason, it was impossible to segregate construction costs by specific jobs or functions or to segregate power costs in terms of kilowatt-hour consumption.

The operating costs for mining and milling operations have been based upon those for the 6 months between July 1, 1954, and January 1, 1955, because this was the longest period of uninterrupted operation and provides the best measure of potential accomplishment with the existing plant and the operating methods. Descriptions and cost analyses of the several components of the operation follow.

Transportation of Supplies

The annual gross tonnage of supplies to the mine during the period of relatively stabilized operations was slightly less than 1,000 tons per year. The principal freight item was 705 tons of fuel oil, which was barged from Nome and pumped to 2 tanks (combined 50,000-gallon capacity) on the beach. From the beach the fuel was hauled to the mine in a 1,500-gallon tank truck or in a 3,000-gallon tank mounted on a tractor-drawn sled; at the mine it was stored in three 25,000-gallon tanks. Lubricants and gasoline in drums and other supplies in bulk packages were transported by truck or tractor-drawn sled. Perishable goods and emergency supply items were transported by jeep, truck, or sled from airfields.

Ocean-freight rates, including stevedoring, terminal and tax charges from Seattle to Lost River, and lighterage charges at Lost River were as follows:

Commodity	Average costs			Percent of purchase cost
	Ocean	Lighterage	Total	
Mine and mill machinery .... per ton	$47.88	$8.80	$56.68	-
Explosives .... do...	97.50	20.00	117.50	29.5
Timber .... per M ft. b.m.	45.78	12.00	57.78	71.5
Cement .... per ton	30.50	8.80	39.30	-
Oil (bulk, crude, diesel)1/ ... per gallon	.074	.015	.089	-
Groceries .... per ton	40.00	20.00	60.00	20.0
Meat .... do...	125.60	14.40	140.60	14.7
Household goods2/ .... do...	23.60	7.20	30.80	-
Building materials2/ .... do...	32.20	9.60	41.80	-

1/ Purchased on basis of delivered cost to tanks on beach at Lost River, but for this table the cost is itemized to show lighterage charges.

2/ Average for items that involve many different rates; stevedoring and terminal charges (Seattle) and ocean-freight tax excluded.

Air-transportation rates from Nome and Seattle to either airfield at Lost River were as follows:

	Freight per pound		
	Less than 100 lb.	More than 100 lb.	Passenger fare, one way
Nome, Alaska	$0.10	$0.10	$35
Seattle, Wash.	.59	.36	165

The additional cost of transporting freight from the beach or airfield to the mine, including the cost of servicing and maintaining the airfields, is based proportionately upon use during 1954, as follows:

Operation	Total supply cost, including freight	Ratio, percent	Distributed cost of local transport
Mine	$23,018.79	15.47	$2,517.10
Mill	17,381.28	11.68	1,900.43
Power and compressors	61,605.03	41.40	6,736.12
Camp	8,624.78	5.80	943.71
Boardinghouse	30,191.90	20.29	3,301.35
Shop	2,345.07	1.58	257.08
Mobile equipment	4,283.43	2.88	468.60
Technical	1,310.64	.88	143.10
Clerical	17.28	.02	3.25
Total	148,788.20	100.00	1/16,270.74

1/ 10.94 percent of cost of supplies.

Mill-Water Supply

After attempts to obtain an adequate supply of mill water from underground sources had failed, the ground water in the valley of Lost River was developed, and a pumping system was constructed during the summer and fall of 1953. The piping system, with its heating elements, was designed by L. N. Roberson Co., Seattle, Wash. The specifications for a pumping system having a capacity of 300 gallons of water per minute at 32° F. follow:

1. 9,000 feet of 9-inch transite pipe enclosed and centered in a 20- by 22-inch wooden duct constructed (aboveground) of 2-inch lumber, the inside of the duct to be filled with black mineral wool and the pipe to be heated by six "R14 Heatsum" cables spaced equally around the outside of the pipe. This protection was designed to maintain waterflow at minimum temperatures of minus -40° F.

2. Power for the heating cables and pumps to be supplied at 440 volts from the mill powerplant. Power to heat the north half (4,500 feet) of the pipeline to be obtained directly from the generators; power to heat the south half of the pipeline and to drive the pumps to be transmitted at 4,160 volts and reconverted to 440 volts at the pumphouse. Each of the six cables on the pipes was to be controlled by a separate thermostat set to cut in at a half degree below the temperature for the preceding thermostat. The current to each half of the pipeline was distributed through two circuits.

Complete details of the costs of construction and operation of the system could not be found in company records, but approximate principal construction costs are listed below:

Transite pipe:		
9,000 ft. at $1.48	$13,320	
Freight	2,100	$15,420
Insulation:		
376 bags (?)	7,983	
Freight	5,665	13,648
Heating cables, 54,000 ft. at $48/M		2,592
Wire, 2,700 ft. No. 6 WP, 3,300 pound at $55.70 cwt.		1,838
Transformers, switches, circuit breakers, etc.		6,100
Miscellaneous electrical fittings		6,116
Lumber for boxes and trestles:		
208 M bd.-ft.	15,409	
Freight	7,000	22,409
Pumps, 2 6-inch		2,244
Grading for pipeline		2,700
Labor for construction		28,714
Freight not listed above, engineering services, and other unlisted items		22,277
Total		124,058

Power-consumption data were not recorded; consequently, power costs can be stated generally only.

Pumping required a 30-hp. motor to drive one 6-inch centrifugal pump, which delivered approximately 400,000 gallons in 24 hours against a static head of 200 feet. The pump operated intermittently, determined by the water level in the pump sump. A standby pump and motor were provided.

The pumphouse required 2 heating elements, which drew 6 to 10 amperes each during freezing weather.

The maximum connected load on the heating cables was 44 kw. for each half of the pipeline; however, maximum power seldom was used. During relatively mild winter weather, the heating circuits were cut out.

The probable maximum power requirement for the entire system was about 118 kw. (148 hp.); the average load probably did not exceed 45 kw., which at 440 volts would require approximately 100 amperes. The company electrician states that rough measurements made by him indicated a normal consumption of about 40 amperes.

After its installation late in 1953 and until operations were discontinued in late 1955, the water-supply system functioned satisfactorily. No difficulty with freezing was experienced, very little maintenance was required, and power consumption was much less than had been expected. Part of the pipeline is shown in figure 13.

Buildings and Equipment

Camp and mill construction was begun in July 1951 and continued intermittently until May 1954. Because the work necessarily was done irregularly without unified advance planning, the unit construction costs were integrated with other costs for the same period; furthermore, no attempt was made to segregate them until after April 1953, when over half of all construction work had been completed. Therefore, no attempt was made to analyze building and equipment costs for this report; the total costs, as nearly as could be determined from available records, were as follows:

"Book" value of buildings on site at beginning of operations	$30,000	
Mill building (including cost of installing mill and powerplant equipment)	159,000	
Mine surface plant and camp	156,000	$345,000
Water-supply plant (as detailed in preceding section)		124,000
Oil-storage tanks		33,000
Trucks and tractors		69,000
Power- and compressor-plant equipment		63,000
Mining equipment		98,000
Milling equipment		111,200
Accessory equipment:		
Shop equipment	4,000	
Small tools	4,000	
Sawmill equipment	2,000	
Laboratory equipment	2,600	
Kitchen equipment	8,000	
Camp equipment, furniture, fixtures	12,000	
Miscellaneous (otherwise unlisted)	8,700	41,300
Total		884,500

FIGURE 13. - Pipeline From Pumphouse on Lost River to Mill.

The approximate locations of the buildings are shown in figure 14; the purpose and size of the various units are listed below:

Map reference No.	Number	Description
1	1	Mill building (125 by 151 feet):
1A	1	Recovery plant.
1B	1	Powerplant (5 diesel generators).
1C	1	Assay laboratory.
1D	1	Change room (incomplete).
2	1	Shop building (46 by 155 feet):
2A	1	Carpenter shop (23 by 46 feet).
2B	1	Vehicle and heavy equipment repair shop (20 by 60 feet).
2C	1	Warehouse and supply room (23 by 60 feet).
2D	1	Machine shop, lamp-charging station, compressor room, and oil room (46 by 33 feet).
2E	1	Electrical shop (10 by 14 feet).
2F	1	Foreman's office (14 by 16 feet).
2G	1	Blacksmith shop (12 by 20 feet).
3	1	Boardinghouse, 2-story (23 by 60 feet).
3A	1	Camp water-storage tank room (8 by 15 feet).
3B	1	Change room (23 by 20 feet).
3C	1	Mess hall (23 by 20 feet).
3D	1	Kitchen 23 by 25 feet).
3E	10	Sleeping rooms (2-man), above 3B, 3C, and 3D.
3F	1	Cold-storage room (6 by 8 feet).
4	1	Grocery warehouse, lean-to (16 by 40 feet).
5	1	Grocery warehouse, lean-to (18 by 60 feet).
6	1	Grocery warehouse, Quonset (16 by 40 feet).
7	1	House, insulated prefab (16 by 25 feet).
8	1	Cabin, 2 sleeping rooms.
9		Snowsheds. Above units either adjoin or are connected by snowsheds to form 1 continuous structure.
10	1	Cabin, mill superintendent (10 by 15 feet).
11	1	House, mine superintendent (20 by 20 feet).
12	1	Mine office (20 by 30 feet).
13	1	House, insulated prefab (16 by 25 feet).
14	1	Building, Army barracks, contains recreation hall (20 by 60 feet) and dwelling (20 by 20 feet).
15	7	Diesel-oil storage tanks; (five 25,000-gallon tanks at mine) (one 25,000- and one 35,000-gallon tanks near mouth of Lost River).
16	1	Cabin, post office (13 by 23 feet).
17	1	House, insulated prefab (16 by 25 feet).
18	1	Cabin, dwelling (10 by 14 feet).
19	1	Quonset hut, 2-family dwelling (16 by 40 feet).
20	1	Building, Army barracks, 3-family dwelling (20 by 60 feet).
21	1	Quonset hut, 2-family dwelling (16 by 35 feet).
22	1	Building, Army barracks, 2-family dwelling (20 by 50 feet).
23	2	Buildings, Army barracks (20 by 50 feet).
24	1	Sawmill.
25	1	House, minister's residence (25 by 25 feet).
26	1	Church (20 by 50 feet).
27	1	Store (20 by 60 feet).
28	1	Schoolhouse (20 by 50 feet).
29	1	House, insulated prefab, teacher's residence (16 by 25 feet).

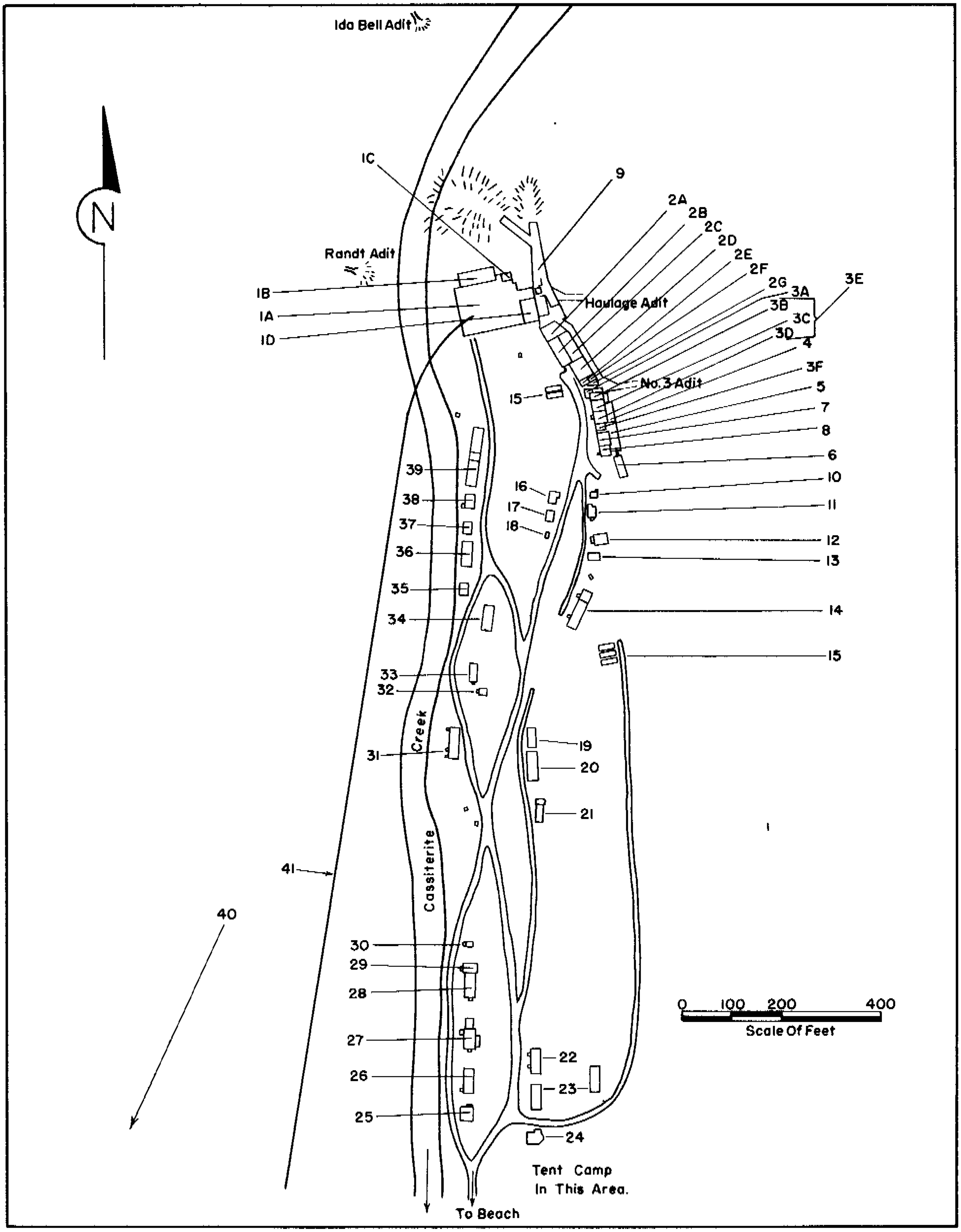

FIGURE 14. - Plan of Surface Plant, Lost River Mine.

Map reference No.	Number	Description
30	1	Cabin, dwelling (10 by 12 feet).
31	1	Building, Army barracks, dwelling (20 by 60 feet).
32	1	Cabin (18 by 24 feet).
33	1	Log cabin, 2-family dwelling (18 by 57 feet).
34	1	Warehouse building, Army barracks (20 by 50 feet).
35	1	House, insulated prefab (16 by 25 feet).
36	1	Warehouse, Army barracks (20 by 50 feet).
37	1	House, insulated prefab (16 by 25 feet).
38	1	Cabin, dwelling (20 by 30 feet).
39	1	Bunkhouse, 20 2-man rooms and washroom (20 by 120 feet).
40	1	Powderhouse, 2,500 feet S. 25° W. of camp area.
41	1	Building, dwelling near mouth of Lost River (20 by 50 feet).
42	2	Warehouses, near mouth of Lost River (20 by 50 feet).
43	1	Pipeline, to pumphouse 9,000 feet.

All construction except Quonset huts was of wood. The mill building is worthy of special note; it was built of shiplap on 2- by 2-inch framing, supported by 6- by 6-inch posts, resting on concrete piers; the roof beams are laminated, 2- by 10-inch lumber. The outside was covered with heavy building paper, secured with wooden strips; the inside was lined with 1-inch Celotex board. The diesel powerplant was housed, without partitions, on the north side of the mill; the heat produced by the engines warmed the mill comfortably in all weather and, with the compressors, also heated the shop. Other buildings were heated by oilstoves. The camp is shown in figures 15 and 16.

Power

The connected powerload for mine, mill, and camp was approximately as tabulated below:

Operation	Horsepower	Remarks
Mine:		
Battery chargers	20.0	8 hours.
Blower	5.0	Time variable.
Hoist	40.0	
Pump	75.0	24 hours, intermittently.
Compressors (2 at 60, 2 at 75)	270.0	
Total	410.0	
Mill:		
Crushing and screening	65.5	
Fine crushing: Ball mill, 40 hp.; rod mill, 75 hp.	115.0	
Jigs: 1 coarse, 1.5 hp.; 1 fine, 2 hp. ...	3.5	
Tables: 9 sand, 20 hp.; 8 slime, 24 hp....	44.0	
Thickeners, Dorr with pumps	4.0	
Drag (Esperanza) classifier	6.0	
Flotation	11.0	
Miscellaneous pumps	65.0	
Total	314.0	24 hours.
Shops	17.5	8 hours, intermittently.
Laboratory	2.0	Do.
Water supply	148.0	Estimated averages: Winter 60 hp., summer 30 hp.
Camp	30.0	
Village (Eskimo)	45.0	15 hp. estimated average.
Total	966.5	

FIGURE 15. - Lost River Mill.

FIGURE 16. - Lost River Camp. Waste dump, snowshed, mill building, shops, and boardinghouse in left-center foreground; pipeline (mill-water supply) and bunkhouse in right foreground. Snowshed covers mine track from No. 3 adit.

Power was generated by the nine diesel units listed below:

Items	Horsepower
Compressors (near mine portal):	
2 LeRoi, 315 cubic feet	120
1 I. R., 365 cubic feet	75
1 Worthington, 365 cubic feet	75
Subtotal, 1,360 cubic feet	270
Generators (in mill building):	
1 Caterpillar, D-364	247
1 Caterpillar, D-375	300
3 Superior, converted, model GDB-8, 100 kw. each	400
Subtotal	947
Total	1,217

Although the total generating capacity was slightly more than the total connected load and considerably more than the operating load, some part of the mine or mill plant usually was short of power because of mechanical difficulties with the diesel equipment.

The hours operated and the fuel consumption by units from July through December 1954 are listed below:

Unit	Hours operated	Fuel oil, gallons	Average, gal./hr.	Fuel cost
Compressors	No record	6,497	-	$1,463.46
D-364 Caterpillar	2,349	25,384	10.8	7,017.77
D-375 Caterpillar	3,398	35,326	10.4	9,757.48
Superior No. 1	2,066	14,153	6.8	3,895.40
Superior No. 2	321	1,839	5.7	500.21
Superior No. 3	1,867	13,005	6.9	3,580.85
Total		96,204		26,215.17

Caterpillar-fuel consumption per rated horsepower per hour = 0.038 gallon.
Superior fuel cost per rated horsepower per hour = .051 gallon.
Caterpillar fuel cost per rated horsepower per hour = $0.0105.
Superior fuel cost per rated horsepower per hour = .0122.

The direct cost of power from July through December 1954 was as tabulated below:

	Labor	Supplies	Shop	Mobile service	Total
July	$1,232.57	$4,717.68	$283.75	$426.38	$6,660.38
August	1,330.70	5,349.12	334.44	257.33	7,271.59
September	1,442.01	5,013.70	642.50	270.57	7,368.78
October	1,470.69	5,388.20	88.91	1,064.71	8,012.51
November	1/614.88	5,069.49	441.17	106.05	6,231.59
December	1/590.00	5,727.83	160.94	141.53	6,620.30
Total	$6,680.85	31,266.02	1,951.71	2,266.57	42,165.15

$\frac{\$42,165.15 \text{ (cost)}}{17,220 \text{ (tons milled)}} = 2.45$ per ton milled.

1/ Used operator 1 shift only.

Fuel Consumption

Fuel consumption by prime consumer, July through December 1954, was as follows:

Consumer	Gallons		Cost		Total cost	Total, percent
	Diesel	Gas	Diesel	Gas		
Powerplant	96,204		$26,215.17		$26,215.17	84
Mobile equipment	2,125	1,448	585.17	$538.95	1,124.12	4
Buildings and offices .	9,030		2,494.12		2,494.12	9
Kitchen	3,859		1,060.21		1,060.21	3
Total	111,218	1,448	30,354.67	538.95	30,893.62	100

$\frac{112,666 \text{ gal.}}{17,222 \text{ tons}}$ = 6.5 gallons of fuel oil per ton milled.

$\frac{\$30,893.62}{17,222 \text{ tons}}$ = 1.79, cost of fuel oil per ton milled.

Boardinghouse

The boardinghouse served meals to employees at $3 a day; one man and his wife cooked, served, and washed dishes. Operating income and costs from July through December 1954 were as follows:

Month	Board-days	Income	Cost			Cost board-day	Loss
			Labor	Supplies	Total		
July	952.83	$2,858.48	$1,003.50	$2,032.96	$3,036.46	$3.186	$177.98
August	1,045.31	3,135.95	1,000.00	2,746.48	3,746.48	3.580	610.53
September ..	905.00	2,715.20	1,050.00	2,842.59	3,892.59	4.300	1,177.39
October	1,025.74	3,005.22	1,112.00	1,980.11	3,092.11	3.010	86.89
November ...	742.74	2,228.21	1,100.00	1,416.00	2,516.00	3.390	287.79
December ...	759.75	2,313.18	1,136.50	1,418.38	2,554.88	3.520	241.70
	5,431.37	16,256.24	6,402.00	12,436.52	18,838.52	3.490	2,582.28

$\frac{\$2,582.28 \text{ (cost to company)}}{17,222 \text{ tons}}$ = $0.15, cost per ton milled.

Labor

Records of labor distribution were not detailed enough to permit cost analyses of subactivities in terms of man-hours or man-shifts; however, the records do permit analyses of labor costs by major activities from July through December 1954, as tabulated below:

	Number of men	Man-shifts	Tons per man-shift		Average wage per shift	Cost per ton milled
			Broken[1]	Milled[2]		
Mine:						
Labor	12	2,119	8.70	8.13	$24.51	$3.016
Foreman	2	323	57.09	53.31	20.62	.387
Engineer	1	183	100.76	94.10	22.67	.241
Average or total	15	2,625	7.02	6.56	23.90	3.644
Mill:						
Labor	9	1,702	-	10.12	15.07	1.489
Superintendent .... Operators	3	576	-	29.90	22.54	.754
Assayers	1	183	-	94.10	19.64	.209
Average or total	13	2,461	-	7.00	17.15	2,452
Average or total, mine and mill	28	5,086	-	3.39	20.64	6.096

See footnotes at end of table.

	Number of men	Man-shifts	Tons per man-shift Broken[1/]	Milled[2/]	Average wage per shift	Cost per ton milled
Service:						
Powerplant	1	306	-	56.27	$21.83	$0.388
Shop	4	602	-	28.61	19.43	.679
Camp	4	660	-	26.09	14.35	.550
Average or total	9	1,568	-	10.98	17.76	1.617
Office	2	366	-	47.05	16.96	.360
General superintendent	1	183	-	94.10	34.43	.366
Average or total	40	7,203	-	2.39	20.18	8.439
Payroll taxes and insurance .						1.472
Loss on boardinghouse						.150
Total						10.061

1/ Tons broken - 18,439 (estimated).
2/ Tons milled - 17,220
(From July through December 1954 the company hired 16 and terminated 15 workers.)

Mining

A complete and satisfactory breakdown of mining costs by labor-supply units and operations could not be made, because the records were incomplete; however, a general breakdown, in dollars, of stope development or ore breaking in stopes and of exploration drifting and crosscutting is presented in tables 11 and 12.

To supplement the general cost data in the tables, Bureau of Mines engineers made first-hand studies of several basic underground operations; from these studies it was possible to obtain useful representative costs. The data obtained are tabulated below:

Drifting

195 level: Nearly unaltered limestone in footwall of dike, February 1955; advance, 95 feet

Item	Units	Units per foot	Unit price	Total cost	Cost per foot
Steel:					$2.51
Drill steel (7/8 inch)	84 feet	-	$1.82 foot	$153.30	
Carset bits (1-5/8 inch)	5	-	17.11 each	85.55	
Timber	727 bd.-ft.	-	.131 bd.-ft.	95.25	1.00
Explosives:					2.11
45 percent gelatin dynamite .	800 pounds	8.42	.224 pound	179.20	
No. 6 caps	323	3.40	.023 each	7.43	
Fuse	1,940 feet	20.40	.0123 foot	23.86	
Pipe, with fittings:					.81
1 inch victualic	60 feet	-	.26 foot	15.60	
3 inch (o.d.) victualic	100 feet	-	.61 foot	61.00	
Rail and fittings:					
30 pound rail	90 feet	-	.19 foot	17.10	
Fishplates	160	-	.116 each	18.56	
Spikes	120	-	.09 each	10.80	
Labor:		4.42			15.08
Drill-blast-muck	307 man-hours	-	13.00 foot	1,235.00	
Tram	1/113 man-hours	-	1.75 hour	197.75	
Direct cost per foot (excluding power)					22.00

1/ Average length of tram - 321 feet to shaft.
Note: Steel costs represent amount taken out of stock; therefore they exceed actual use cost. Pipe and rail costs do not represent 95 feet of drift.

365 level: Highly altered limestone and granite near contact, March 19-31, 1954, 62 feet

Item	Units	Units per foot	Unit price	Total cost	Cost per foot
Steel:					$0.58
Drill steel	(1/)	-	-	-	
Carset bits	2	-	$12.92 each	$35.84	-
Timber	2/497	-	.127 bd.-ft.	63.39	1.02
Explosives					1.47
45 percent gelatin dynamite.	350 pounds	5.64	.224 pound	78.40	-
No. 6 caps	163	2.63	.023 each	3.75	-
Fuse	740 feet	11.90	.0123 foot	9.10	-
Pipe with fittings:					.30
1 inch victualic	70 feet	-	.26 foot	18.20	
Rail and fittings:					.73
30 pound rail	120 feet	-	.19 foot	32.80	-
Fishplates	42	-		4.87	-
Bolts, nuts	84	-		7.56	-
Ventpipe, 12 inch diameter ...	62 feet	-		70.06	1.13
Labor:					
All operations3/	319 hours	5.14	15.00 foot	930.00	15.00
Direct cost per foot (excluding power)					20.23

1/ No identifiable record.
2/ 18 feet of drift timbered.
3/ Includes drilling and blasting, mucking, timbering, and tramming 600 feet to shaft.

Stope Development

365 E. raise, with subdrifts to vein, January-April 1954; 255-foot raise, 209-foot subdrift

Item	Units	Units per foot	Unit price	Total cost	Cost per foot
Explosives:					$1.33
Dynamite	2,495 pound	5.38	$0.224 pound	554.40	
Caps	1,369	3.00	.023 each	31.40	
Fuse	9,820 feet	21.2	.0123 foot	31.05	
Timber	13,662 bd.-ft.	28.3	.118 bd.-ft.	1,614.51	3.48
Pipe:					.37
1-inch	235 feet	.51	.24 foot	55.90	
2-inch	262 feet	.56	.44 foot	115.07	
Direct cost per foot .					5.18

Note: Cost of labor, steel, power not recorded in identifiable form.

Stoping

Stopes: A-1-W, A-1-E, A-2-E, and A-3-E; 1954; 32,500 tons ore broken

Item	Units	Units per ton	Total cost	Cost per ton
Explosives:				
Dynamite	23,240 pounds	0.72	$5,127.75	$0.198
Caps	8,767	.27	614.19	
Fuse	39,280 feet	1.20	513.76	
Timber	3,765 bd.-ft.	.116	491.56	.015
Labor:				
Drilling / Blasting	1/ 5,200 man-hours	1/ .16	21,125.00	2/ .65
Direct cost of breaking ore in stopes (excluding power and steel)				.863

1/ Based upon time study of breaking 6,000 tons.
2/ Contract rate.

Shaft Sinking 1/

4- by 8-foot vertical winze, adit level to 365 level, 1951-52

Item	Units per 10 feet	Units per foot	Cost per 10 feet	Cost per foot
Labor:				
Mine-muck-timber	128 man-hours	12.8	$396.80	
Timber framer	32 man-hours	3.2	91.84	
Hoistmen	64 man-hours	6.4	183.68	
Compressor-mechanic	64 man-hours	6.4	128.09	1/ $118.18
Timber:				
Lumber	1,781 bd.-ft.	178.1	2/ 267.15	
Hanger, bolts, etc.			9.72	27.69
Fuel:				
Diesel 3/	320 gallons	32.0	86.40	
Gasoline 3/	16 gallons	1.6	5.92	9.23
Explosives:				
Dynamite	150 pounds	15.0	33.60	
Electric detonators	72	7.2	16.24	
Lead wire	200 feet	20.0	2.50	5.32
Steel:				
Drill steel (broken)	4 feet	.4	8.75	
Carset bits	1	.1	21.56	3.03
Pipe:				
1 inch water pipe with	10 feet	1.0	2.60	
3 inch pipe	10 feet		12.50	1.51
Repair parts, carbide, etc.	-	-	15.00	1.50
Direct cost of sinking shaft per foot				$166.46

1/ Includes 20-percent payroll charges and boardinghouse loss, plus 4 percent estimated time lost repairing broken timbers and similar lost-time accidents.
2/ At $150 per M bd.-ft. for shaft timber.
3/ Power for drilling and hoisting provided by portable diesel compressor at portal; power for blower fan provided by small gas engine.

1/ Based on cost study by Bureau of Mines of 100 feet of shaft sinking. The 10-foot interval is the minimum distance in which all operations were performed.

Diamond Drilling

Underground on 365 level,
June-August 1955; 1,984 feet (AX) in 8 holes,
137 drill shifts

Item	Cost	Cost per foot
Direct labor:		
Drillers and helpers (2,192 man-hours)	1/$6,848.36	$3.45
Supplies:		
Casing and bits	1,301.34	.66
Equipment rental and freight	1,165.07	.59
Power (by distribution)	2,683.97	1.35
Direct cost	11,998.74	6.05
Distributed labor:		
Hoist, pump, warehouse, office	2/6,219.00	
Distributed supplies:		
Shop, camp, mobile, equipment, miscellaneous	926.00	
Distributed general supervision	1,130.00	
Total distributed costs	8,275.00	4.17
Total cost		10.22

1/ Base rates for labor: Lead driller, $4 per hour; driller, $3 per hour; 2 helpers at $2 per hour. Work performed 2 shifts daily, 7 days a week.

2/ Entire cost of operating and maintaining shaft and 365 level charged to drilling, because there were no other operations below adit level.

Milling

The dollar cost of milling, in terms of labor, supplies, and power, is presented in table 13. Cost records that would permit segregation of units of labor and supplies or crushing, grinding, and concentration were not maintained. Nevertheless, warehouse records did provide some data on the consumption of grinding balls, grinding rods, and screens; these data are tabulated below:

Item	Ore, tons	Units expended	Units per ton	Tons per unit	Cost per unit	Cost per ton
Grinding balls:		(pounds)				
3-inch ball	24,586	2,000	0.0813	-	$0.088	$0.007
4-inch ball	17,220	5,040	.292	-	.122	.036
Grinding rods	17,220	18,270	1.061	-	.094	.100
Total grinding mediums except liners	-	-	-	-	-	.142
Screens:		(screens)				
1/2-inch	17,220	1	-	17,220	22.12	.001
6-mesh	17,220	12	-	1,435	27.47	.019
8-mesh	17,220	3	-	5,740	27.25	.005
10-mesh	17,220	4	-	4,305	32.95	.008
20-mesh	17,220	16	-	1,076	22.74	.021
Total screen cost						.054

TABLE 11. - Stoping costs, July-December 1954

Operation	Total	Labor (mining)	Explo-sives	Steel bits	Timber and miscel-laneous	Compressed air and power	Hoist and pump	Muck and tram	Shop and mobile service	Super-vision	Total
Stope drifts and cross-cuts .... feet	739										
Per foot advance		$14.90	$1.09	$0.53	$1.34	$1.59	-	$4.08	$0.46	$1.56	$25.55
Per ton broken		.59	.04	.02	.05	.06	-	.16	.02	.05	1.00
Stope raises .... feet	576.5										
Per foot advance		15.26	1.06	.48	1.40	.71	-	3.78	.36	1.52	24.52
Per ton broken		.48	.03	.02	.04	.02	-	.12	.01	.05	.77
Stope development .. feet	1,315.5										
Per foot advance		15.05	1.07	.51	1.36	1.20	-	3.95	.39	1.54	25.07
Per ton broken		1.08	.08	.04	.10	.08	-	.28	.03	.11	1.80
Ore breaking .... ton	18,439										
Per ton broken		.65	.18	.08	.20	.60	-	.64	.13	.25	2.73
Stoping .... tons	18,439										
Per ton ore broken		1.73	.26	.11	.29	.69	1/ (0.17)	.92	.16	.36	4.52
Stoping .... tons	17,220										
Per ton ore milled		1.85	.28	.12	.32	.74	1/ (.19)	.98	.17	.39	4.85

1/ Charge for maintaining lower levels, inactive during this period, not included in totals.

TABLE 12. - Drifting costs, April-December 1953, January-March 1955

Month	Advance, feet	Direct costs per foot: Labor	Explo-sives	Timber	Steel bits	Power and compressor	Sub-total	Service costs per foot: Hoist	Shop	Tram and haulage	Super-vision	Total cost per foot
April	155.3	$16.09	$3.05	$2.10	$0.78	$14.23	$36.25	$6.79	$4.06	$4.61	$1.33	$53.04
May	226.2	16.10	3.25	.80	.65	6.45	27.25	-	.72	4.42	2.21	34.60
June	100.4	44.06	2.34	1.68	-	7.72	55.80	-	1.25	-	2.06	59.11
July	144.4	46.48	3.45	14.07	-	4.85	68.85	-	2.17	-	1.43	72.45
August	350.9	23.82	3.35	5.64	1.07	3.76	37.64	1.66	1.71	2.50	.59	44.10
September	393.1	20.70	2.74	1.60	1.57	4.43	31.04	2.05	1.37	3.98	.53	38.97
October	263.1	22.71	3.03	2.73	2.67	4.60	35.74	3.55	.38	3.83	2.89	46.39
November	-	-	-	-	-	-	-	-	-	-	-	-
December	16.7	46.50	1.72	9.73	-	6.39	64.34	11.65	-	4.44	2.93	83.36
January	28.5	17.07	5.07	3.96	5.40	9.09	40.59	7.15	2.29	2.21	4.81	57.05
February	94.9	13.33	2.30	6.05	-	7.73	29.41	6.44	2.05	11.56	11.56	60.09
March	163.0	12.96	1.52	6.26	1.26	10.10	32.10	2.96	2.19	11.29	11.29	58.44
Total	1,936.5											
Weighted average .		22.92	2.91	4.08	1.50	6.47	37.88	2.71	1.59	4.20	2.80	49.18

Explanation. - Labor: Direct cost of drilling, blasting, mucking, timbering, and (during May, June, and July) cost of hoist labor; for December, also cost of shop labor. Timber: Includes miscellaneous (rails, pipe, etc.) charges and cost of steel and bits for June, July, December, and February. Power and compressor: Estimated percentage of total cost of producing electric power and compressed air during period. Service costs: Distributed from total camp cost on same basis as power and compressor.

TABLE 13. - Milling costs, July-December 1954

Month	Ore milled tons	Labor per ton: Operators	Labor per ton: General labor	Labor per ton: Total	Supplies per ton: Basic	Supplies per ton: Miscellaneous	Supplies per ton: Parts	Supplies per ton: Total	Power per ton	Service per ton	Total per ton
July	2,541	$1.15	$1.68	$2.83	$0.11	$0.02	$0.17	$0.30	$1.73	$0.84	$5.70
August	2,893	.68	1.72	2.40	.14	.12	.51	.77	1.62	.53	5.32
September	2,247	1.05	1.54	2.59	.20	.70	.36	1.26	2.10	.65	6.60
October	3,182	.92	1.38	2.30	.14	.03	.25	.42	1.62	.55	4.89
November	3,318	.44	1.32	1.76	.21	.20	.18	.59	1.25	.51	4.11
December	3,039	.46	1.38	1.84	.05	.14	.25	.44	1.44	.22	3.94
Total	17,220										
Weighted average		.76	1.48	2.24	.14	.18	.28	.60	1.65	.54	5.03

Explanations. - Labor: "Operators" include mill superintendent and chief operators on afternoon and night shift (superintendent acted as chief operator on day shift). "General labor" includes crusher men and all common labor. No chief operator maintained on night shift in November and December. Mill-labor costs do not include salary of assayer. Supplies: "Basic" includes balls, rods, liners, screens, lubricants, and flotation reagents. Power: Distributed percentage of total cost of power, based upon approximate proportion of "connected" mill power to total "connected" power. Service: Includes shop service and service supplied by camp mobile equipment. Amount charged by distribution, not direct accounting.

Shipping and Smelting

Concentrate derived from operations during the cost-analysis period (July through December 1954) were shipped and smelted with concentrate derived from other periods of operation; in fact, it was impossible to determine the relationship between a particular shipment of concentrate and the ore from which it was produced; therefore, the costs shown in table 14 were derived by applying the average of the actual unit costs of several shipments to the calculated recoveries during the cost-analysis period. The results, although not precise, are believed to be accurate within the requirements of practical cost analyses.

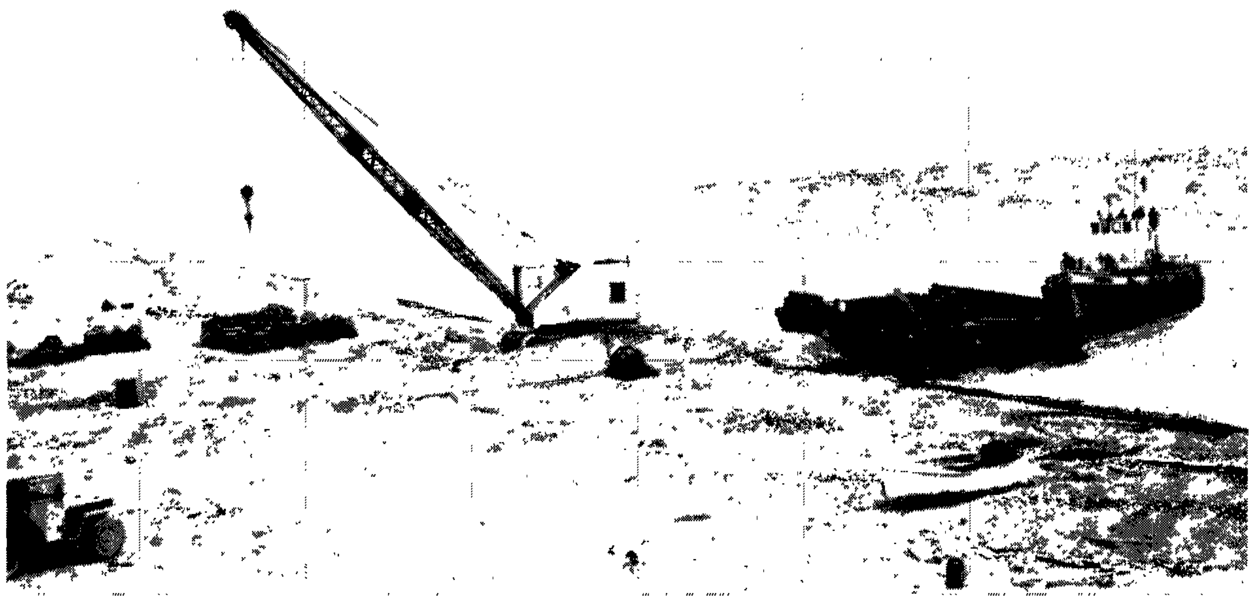

FIGURE 17. - Loading Tin-Concentrate Drums, Lost River Beach. The barge and tug transport oil and other freight between Nome and outlying settlements on Bering Sea. Bulk oil is carried in the hull of the barge; other freight is carried as deck load. The same type of equipment is used for direct ship-to-shore lighterage.

Combined Operations

Actual dollar costs of various phases of the operation were maintained with considerable accuracy; those for July through December 1954 are presented in table 15. Inasmuch as these costs represent the only records of cost distribution maintained by the company, they constitute the basis from which all other costs were computed; unfortunately, they do not provide a factual basis for segregating costs of the basic requirements of mining and milling operations.

The results of several attempts to distribute costs to the mining and milling operations are presented in other parts of this section. Table 16 summarizes these approximate distributions cost of the overall operating costs. Precise reconcilation of all cost items throughout the tables was not attempted, because the distributions were based upon estimates or incomplete data. Also, it should be noted that these costs were calculated on the basis of company figures of tonnage treated and concentrate recovered. This was necessary because they were the only complete figures on mill performance available for the period on which the costs were based. Because of errors in weighing gravity concentrate, the tonnage figures calculated by the company probably were high (see Milling); therefore, the total costs of $17.65 per ton of ore milled and $1.33 per pound of tin recovered, as reported in table 16, probably are low. If an overall correction can be applied to the specific period of July through December 1954, the calculated costs would be about $18.16 per ton of ore milled and $1.47 of tin recovered. Such an assumption, however, fails to consider that the greatest discrepancies probably occurred during the early stages of operation. Actual costs, therefore, cannot be calculated with complete accuracy and must be considered to be between the two extremes.

TABLE 14. - Calculated cost of shipping and smelting, July-December 1954

(Based upon calculated recoveries and typical smelter settlements, Texas City smelter)

Item	Cost per ton of wet concentrate	Total cost	Cost per ton of dry concentrate	Cost per ton of ore milled	Cost per pound of tin
Freight:					
Haulage to beach	$1.74	$410.69	$1.98	$0.02	-
Lighterage	7.65	1,805.60	8.75	.10	-
Ocean freight	21.73	5,128.84	24.82	.30	-
Rail freight	12.03	2,839.39	14.21	.16	-
Total	43.15	10,184.52	49.76	.58	$0.045
Smelting:					
Base charge	-	7,527.97	36.36	.44	.033
Price deduction	-	4,561.92	22.03	.27	.020
Assay deduction[2]	-	4,298.15	20.76	.25	.019
Impurity penalties	-	1,614.91	7.80	.09	.007
Total	-	18,002.95	86.95	1.05	.079
Total freight and smelter	-	28,187.47	136.71	1.63	.124

Calculated recovery = 228,096 pounds tin in 236.026 short tons of wet concentrate[1] and 207.040 short tons of dry concentrate

1/ Includes weight of drum containers.
2/ Deduction of 1.12 percent at $0.927 per pound.

Costs obtained by the United States Tin Corp. during the 6 months of uninterrupted operation indicated that a return of approximately $20 per ton of ore milled was necessary to assure an adequate operating profit. The grade of ore necessary to return $20 per ton of ore milled at various prices for tin and at mill recoveries of 60 and 80 percent is shown graphically in figure 18; this graph illustrates the importance of closely controlled selective mining and improved recoveries. Suggestions for attaining the objectives in any future operation at Lost River are outlined in the appendix.

TABLE 15. - Production costs by accounting units, July-December 1954

Item	Wages and salaries	Supplies	Total cost	Cost per ton milled	Cost per pound Sn recovered
Mine[1/]	$58,598.04	$12,301.07	$70,899.11	$4.118	$0.311
Mill	38,630.68	11,677.34	50,308.02	2.921	.221
Power and air	6,680.85	31,266.02	37,946.87	2.204	.166
Shop	11,696.77	1,049.07	12,745.84	.704	.056
Mobile service	4,978.89	2,747.05	7,725.94	.449	.034
Camp	4,496.51	3,755.33	8,251.84	.479	.036
Boardinghouse loss	2,582.28	-	2,582.28	.150	.011
Payroll, taxes and insurance	25,352.43	-	23,352.43	1.472	.111
Total direct	153,016.45	62,795.88	215,812.33	12.533	.945
General superintendent	6,300.00	-	6,300.00	.365	.028
Engineering and assaying	7,743.06	701.54	8,444.60	.491	.031
Clerical	6,206.76	-	6,206.76	.360	.027
Miscellaneous office, travel, etc.	-	-	6,870.44	.396	.030
Total mine overhead	20,249.82	701.54	27,758.80	1.612	.116
Total operating cost	173,266.27	63,497.42	243,571.13	14.145	1.061
Seattle office expense (estimated)	-	-	22,011.06	1.278	.097
Shipping and smelting			28,187.47	1.636	.124
Total production cost	-	-	293,769.66	17.059	1.282

Calculated recovery = 228,096 pounds tin from 17,220 tons ore milled.

1/ Includes cost of stope development and ore breaking in stopes; excludes cost of mine-level development.

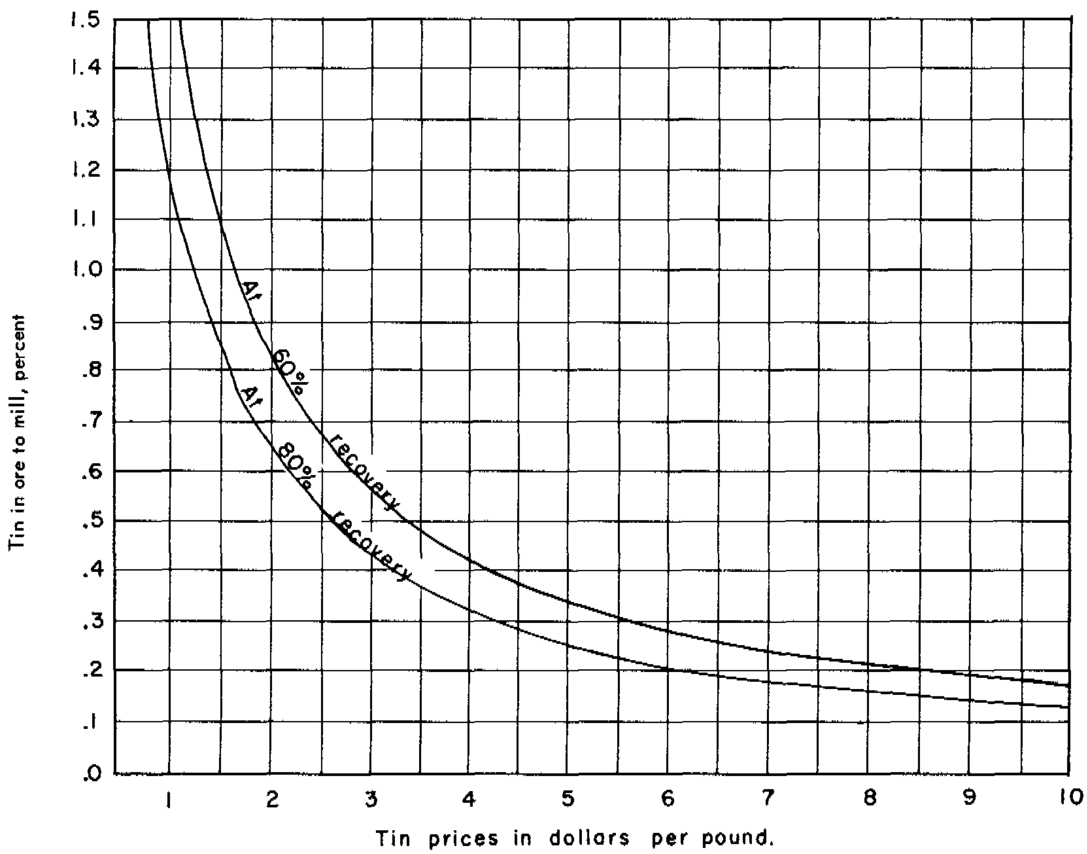

FIGURE 18. - Grade of Ore and Price of Tin Required to Yield $20 Per Ton.

TABLE 16. - Costs per ton milled, by operations, and costs per pound tin recovered, July-December 1954

Item	Direct labor	Direct supplies	Power	Mine haulage	Direct supervision	Payroll, taxes, and insurance	Total direct	Shop and mobile service	Engineers, Clerks, General superintendent	Total per ton	Total per pound
Mining:											
Level development 1/ .	$0.23	$0.09	$0.07	$0.04	-	$0.04	$0.47	$0.02	-	$0.49	$0.04
Stope development ...	1.16	.20	.08	.28	$0.11	.37	2.20	.03	$0.35	2.58	.20
Ore breaking	.69	.46	.60	.64	.25	.23	2.87	.13	.54	3.54	.26
Total	2.08	.75	.75	.96	.36	.64	5.54	.18	.89	6.61	.50
Milling	2.24	.60	1.65	-	(2/)	.78	5.27	.54	.96	6.77	.51
Camp operation	.26	.22	-	-	-	.09	.57	.49	.20	1.26	.10
Shipping and smelting..	-	-	-	-	-	-	-	-	-	1.64	.12
Total	4.58	1.57	2.40	.96	.36	1.51	11.38	1.21	2.05	16.28	1.23
Seattle office (estimate)	-	-	-	-	-	-	-	-	-	1.28	.10
Total	-	-	-	-	-	-	-	-	-	17.56	1.33

Calculated tin recovery = 13.246 pounds per ton.

1/ No level development during this period; costs computed on basis of current drifting costs applied to development footage required between adit level and surface and proportioned to tons mined during period.

2/ Included in cost of labor.

APPENDIX

Introduction

The following proposals for improving mining and milling procedures are offered for consideration in the event operations at Lost River are resumed. The proposals are based upon experience and observations by Bureau of Mines engineers in all phases of exploration, development, and operation from 1942 to 1955.

Proposed Stoping Method

Summary

A method of shrinkage stoping, applicable to any scale of operation, is suggested as a means of gaining optimum control of the grade of ore mined and milled from the Cassiterite dike. Essentials of the proposed method and its basic differences from methods used by previous operators are summarized as follows:

1. A greater proportion of stope development and of preparatory headings would be in ore.

2. The length of stope blocks would be reduced to uniform dimensions of 80 feet between centers of main raises in ore, and the intervals between haulage levels would be reduced to 100 feet.

3. All ore in stopes would be broken by breasting from main raises with horizontal longholes not more than 35 feet in length. The back of the stopes would be nearly horizontal, and the stope fill would be maintained at uniform height close to the back.

4. Ore would be drawn from the stope by overcast loaders from drawpoint crosscuts uniformly spaced 20 feet apart and at right angles to a main haulage drift in the limestone footwall of the dike.

5. Systematic sampling of development headings and stopes would be facilitated and required.

Development and Preparation

Development of a block of ore and subsequent stoping would require the following sequence of level development and stope preparation:

1. A pioneer drift, 5 by 7 feet in cross section, would be driven in the dike along the footwall. This drift would be sampled at regular intervals of not less than 10 feet, utilizing longholes from the side of the drift to the hanging wall to obtain samples the full width of the dike. Track (and temporary timbering if required) would be recovered from this drift before stoping or as stope blocks were delineated.

2. The drift in the dike could be advanced at least 100 feet before a main haulage drift in limestone (6 by 7 feet in cross section) was driven parallel to and a uniform distance of 20 feet from the centerline of the haulage drift to the footwall of the dike.

3. Drawpoint crosscuts (5 by 7 feet in section), at right angles to the direction of the haulage drift, would be driven to the dike at 20-foot intervals along the haulage drift. The drawpoint crosscuts, at 80-foot intervals (main-raise drawpoints), would be driven before the drawpoints between this interval are completed.

4. The main-raise drawpoints, at 80-foot intervals along the haulage drift, would be enlarged to a width of 10 feet. For 15 feet from the footwall of the dike, this drawpoint would be divided by timber into two compartments of equal width. One compartment of the drawpoint crosscut would serve as a manway; the adjacent compartment would be used as a shovel-loader drawpoint. This type of drawpoint would permit raises to be driven without chute lips and would permit unobstructed access to stope raises above and below the level.

5. Main raises to the level above would be completed and connections made to the manway compartments of the drawpoint crosscuts at this level. Main raises would be of two compartments, that is, a chute and a manway timbered with 6- by 6-inch posts and caps. The raises would be driven the full width of the dike or at least 10 feet in width by slightly more than 10 feet in length. A raise would be advanced in the dike to within 10 feet of the level above; starting a few feet below this point, only the manway compartment of the raise would connect to the drawpoint crosscut above by an inclined raise in the limestone footwall. The manway connecting raise need not exceed 5 by 5 feet in cross section and could generally be inclined at slightly more than 45°. The principal units of stope development and the sequence of operations are illustrated in figure 19.

Breaking Ore

Stoping would be started by breaking to the pioneer drift with horizontal or nearly horizontal holes drilled from the main raises. Stope backs then would be advanced upward about 10 feet per round in horizontal slices, which usually would be drilled 35 feet in each direction from protected drill stations in the raise. Better control of the grade of ore broken and more economical consumption of powder is to be expected by using essentially horizontal longholes of limited length, drilled predominantly parallel to the strike of the dike and at uniform spacing to give a burden of 5 feet. The use of spacers in blasting would reduce dilution by preventing excessive shattering of the walls and at the same time would insure good fragmentation of the dike ore.

The rate of drawing ore from a stope would be controlled to permit sampling backs from the top of the broken ore and to permit orientation of holes in the round to be drilled by inspecting the local changes in dip or strike of the dike revealed by the round blasted previously.

The only ore pillar not recovered would be the block (10 by 10 feet) above every third main raise, which would be left to support raises intended as permanent ventilation raises. Ventilation raises at intervals of 240 feet along the main haulageway are adequate for this purpose.

Drawing ore

Broken ore would be drawn from the stope through drawholes and discharged by overcast loaders into cars in the haulage drift. A system of drawpoint crosscuts at right angles from the haulage drift minimizes the total amount of drawpoint crosscuts

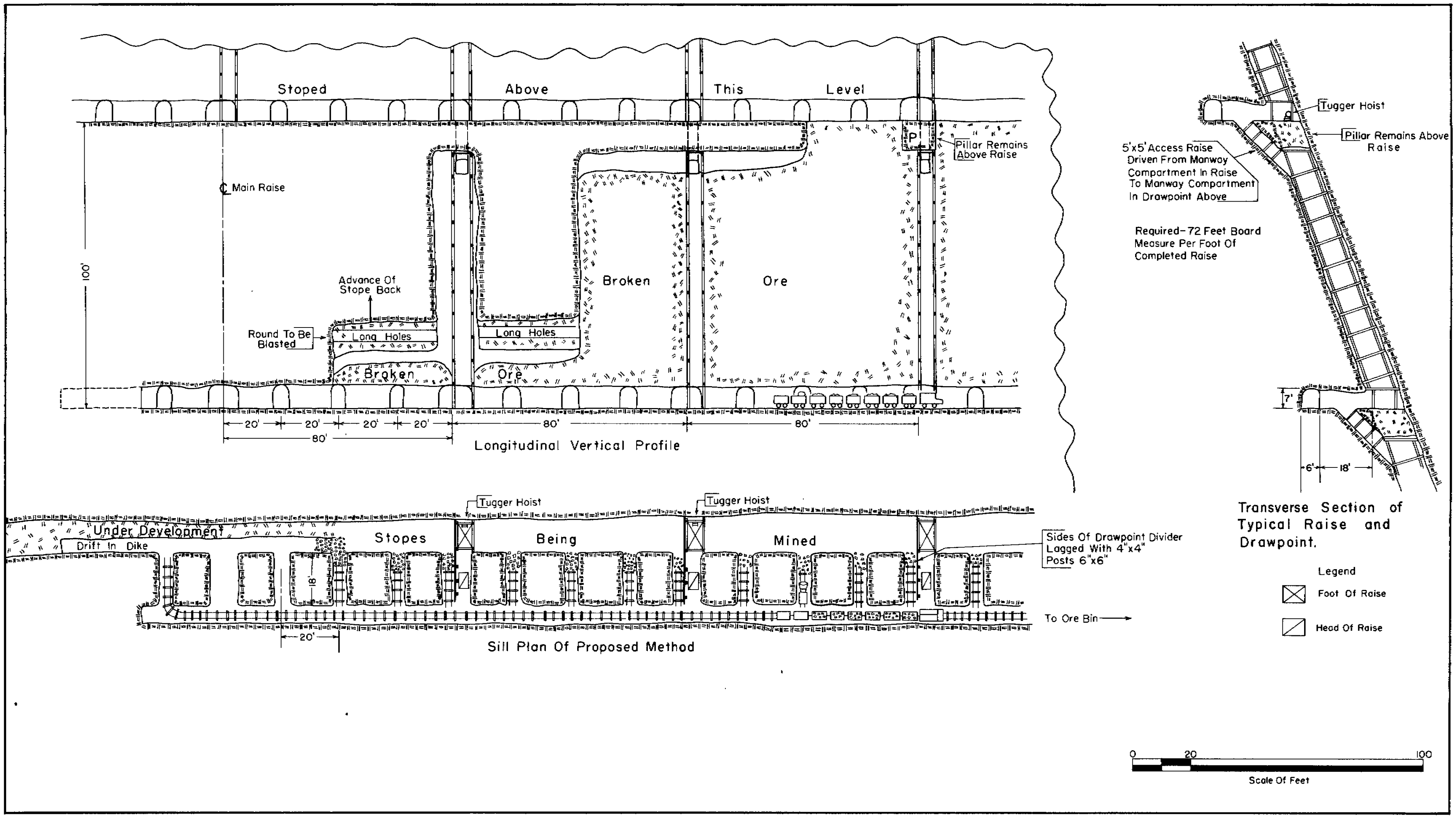

FIGURE 19. - Proposed Stoping Method, Lost River Mine.

for a given spacing of drawpoints. This system generally requires that the track in the drawpoint crosscut be about 7 inches above the track in the main haulage drift to permit efficient transfer of the loader on a dolly from one drawpoint to the other; however, the system has a high loading efficiency, because it eliminates the need for switching every car to be loaded. The drawpoint spacing is based upon an average angle of repose of the broken dike ores of almost 70° and would assure rapid, uniform drawing of ore from the stope.

Sequence of Operations

To permit disposal of development waste into empty stopes, several levels should be developed and mined simultaneously; and each successive upper level should be advanced farther than the level below.

Sampling

Systematic sampling of raises in ore, of the sill drift in the dike, and of stope backs would guide mining and permit blending the different grades of ore from the stopes.

Time and Cost Estimates

Units developed and time required to prepare each block for stoping are estimated as follows:

Development heading	Units developed, linear feet	Average number of drill rounds	Sets of timber	Elapsed time, days		
				1 shift	2 shifts	3 shifts
Pioneer sill drift in dike .	100	15	-	15	8	5
Haulage drift	80	12	-	12	6	4
5 drawpoints (18 feet each).	90	12	-	12	6	4
1 drawpoint, enlarged	(1/)	1	-	2	1	1
Subtotal	270	40	-	41	21	14
Raise:2/ Main raise	90	15	24	27	14	9
Inclined manway raise (5 by 5 feet)	20	4	4	4	2	2
Total	380	59	28	72	37	25

1/ Enlarged drawpoint to 10 feet in width by slabbing from side of crosscut.
2/ If raises advanced only slightly ahead of stope backs, production from stope is possible after completion of drawpoints. Adequate ventilation is necessary, especially if working 3 shifts per day.

The proposed method would require the following estimated units of labor and supplies for stope development and preparation. The selected size of stope block would contain 6,700 tons at the average dike width of 10 feet.

Heading	Units required, linear feet	Labor required, man-shifts	Supplies: Explosives pound	Timber, bd.-ft.	Rail, linear feet	Pipe, linear feet
Drifting:						
Pioneer sill drift in dike.	80	80	560	(1/)	(1/)	(1/)
Haulage drift	80	80	560	200	1/160	1/160
5 drawpoints (18bfeet each)	90	90	630	656	2/180	1/84
Subtotal	250	250	1,750	856	340	244
Rising:						
Main raise in ore	90	60	900	6,480	-	2/180
Manway connection	20	5	200	448	-	2/ 40
Total	360	315	2,850	7,784	340	464
Tons developed per unit, direct estimate3/	18.6	21.3	2.4	.86	19.7	14.5
Cost per ton developed	-	$0.83	$0.09	$0.15	$0.02	$0.05

1/ Installed temporarily; to be salvaged and reused.
2/ Can be salvaged and reused. Main air line, 3 inches; water line, 1 inch; feeder air lines to headings, 2 inches.
3/ Estimated from 1956 costs for supplies. Labor estimated at the following contract rates per foot: Haulage and sill drift, $12.00; drawpoints, $13.50; main raise, $25.00; manway connecting raise, $10.00.

In addition to the supplies shown in the above table, the cost of compressed-air drills, steel, bits, etc., is estimated at approximately $0.10 per ton of ore developed. Total direct cost per ton of ore developed by the proposed method is estimated to be $1.24 per ton.

The cost of breaking ore is largely determined by the wage that is necessary to attract and hold capable miners when required at Lost River. The following estimate is based upon an average annual wage of $7,800 for miners on a 6-day-week work schedule. The productivity of the method of breaking ore is based upon an estimated average of 70 feet of hole per machine-shift. With uniform spacing and burden of 5 feet per hole, the holes would produce 2.1 tons per foot of hole, or 147.0 tons per machine shift of 2 men per crew. With this production and at the annual wage of $7,800, drilling and blasting labor could be contracted at $0.34 per ton broken. Uniform spacing and burden of 5 feet per hole, plus the use of spacers between dynamite charges, are estimated to require about 0.4 pound of explosives per ton of ore broken. The following estimate is based upon current costs for supplies and an annual wage for miners of $7,800.

	Cost per ton of ore broken
Labor, drilling and blasting	$0.34
Explosives, spacers, primers, etc.	.12
Compressed air, drills, steel, etc.	.25
Sampling and supervision	.15
Total	.86

Estimated, direct stope-development and ore-breaking costs on the above basis are summarized as follows:

	Cost per ton of ore broken
Stope development and preparation	$1.24
Ore breaking	.86
Total	2.10

No attempt has been made to estimate total mining costs because of the unpredictable variables that would be involved in any future operation.

Disseminated Ores

Disseminated reserves have not been explored nor tested in enough detail to permit a definite delineation of ore blocks; consequently, a definite mining method cannot be proposed. It is probable that a considerable part of the very low grade material that comprises part of the granite cupola and of the overlying limestone might be mined by open-pit or "glory hole" methods during the summer, but open-pit mining during the winter would not be practicable; therefore any year-round operation would have to be based upon underground mining. Underground mining of these highly kaolinized ores would necessitate square-set stoping or some form of caving. The choice of caving method would depend largely on the dimensions of the ore blocks.

Recommendations for Future Milling Operations

Limiting Factors

Without knowledge of conditions under which operations at Lost River may be reactivated, discussion of future operations must be confined within the limits imposed by grade and character of the ore, proposed date of operation, and proposed tonnage to be treated.

Laboratory beneficiation studies and milling have been confined to treatment of ore from the Cassiterite dike; thus, no metallurgical data or knowledge exists concerning treatment of material from other sections of the tin-bearing area. For discussion, therefore, it must be assumed that the ore to be treated will be similar both in mineral association and grade to ore milled previously. Similarly, it must be assumed that work will begin before the present plant and equipment have deteriorated beyond usefulness.

The proposed daily tonnage would be influenced by a number of factors: Price of tin when the property is reactivated, knowledge of the size and average grade of the workable ore body, the necessity of immediately producing a small quantity of concentrate, the necessity of producing the maximum quantity of concentrate over a relatively short period. Discussion will be directed toward the supposition that the factors and conditions existing at the time of reactivation will allow one of the following procedures: (a) Immediate operation of the mill using present treatment and flowsheet; (b) partial remodeling of the mill to eliminate major operational difficulties and to improve recovery of tin; (c) complete rebuilding of the mill, either at present or increased capacity.

Operation Without Remodeling

When operations were suspended, most of the machinery in the mill was in operating condition or could be made usable by minor repairs. Operating the mill according to the present flowsheet, however, would necessitate replacement of two major items of equipment - the rod and ball mills - responsible for a large part of the mill repair costs and interrupted operation. Mere replacement of these items would not solve any metallurgical or operational problems nor effect an increase in the recovery of tin.

Operating the existing mill without remodeling should be considered only if circumstances allow no other course of action.

Partial Remodeling of Mill

General

The importance of the relationship of mill recovery to unit cost is illustrated by the data tabulated below. These data are based upon total production costs from July through December 1954. (See table 15.)

Recovery vs. unit cost

Tin recovery, percent	Tin in concentrate, pounds	Total production cost per pound tin recovered
60.0	216,834	$1.356
1/ 63.1	228,096	1.289
65.0	234,490	1.254
70.0	252,973	1.162
75.0	271,043	1.085
80.0	289,112	1.017
85.0	307,182	.957

1/ Calculated from production July through December 1954.

Since it is evident that economic operation of the mill is dependent to a large degree on treatment designed to yield maximum recovery of tin, serious consideration should be given to making changes in the present plant and flowsheet to allow such treatment. Additional changes designed to eliminate major operational difficulties should be made if circumstances permit.

Revamping and enlarging the crushing and grinding section are considered of primary importance. For good recoveries it is imperative that cassiterite particles be removed from the grinding circuit as soon as they are liberated from the gangue. Laboratory beneficiation testing has shown that up to 85 percent of the tin can be recovered from Lost River ore (1- to 2-percent tin grade), provided the ore is disintegrated so as to prevent sliming of cassiterite. Limited mill experimentation did not disprove laboratory data; on the contrary, mill testing proved the metallurgical feasibility of grinding at low pulp densities with a light load of grinding mediums. The mechanical limitations of existing equipment, however, precluded such treatment.

Remodeling should include a rod mill and oversize return equipment, large enough to allow grinding with low pulp density and a high circulating load. A smaller rod or ball mill should be installed for regrinding the table middling.

Practice at Lost River has included wet screening the ore fed to the grinding circuit in an attempt to remove fine sand and clay slime. Much of the ore consists of rock and mineral particles cemented by clay; these "clay balls" are not disintegrated easily by wet screening alone, and they pass over the screen to the rod mill. More complete disintegration doubtless could be effected by using a log washer or similar device. Removing the additional fines before grinding would minimize overgrinding the cassiterite. It is suggested that the log-washer discharge material be routed to a cyclone or bowl classifier to remove the bulk of the clay from the circuit immediately and assure no overload on the spigot-type hydroclassifier. Ideally, the disintegration-fines removal unit should be placed ahead of the secondary crusher, so that it would eliminate the packing and plugging troubles that have been experienced with that unit.

Supplementary remodeling to increase operating efficiency should include enlargement of the coarse-ore bin to allow storage of mill feed for 12 hours and installation of a vibrating grizzly feeder to transport ore from the coarse-ore bin to the primary crusher. These changes would reduce operating troubles and labor costs by requiring the primary crusher to operate only one shift for a full day's supply of ore to the mill.

A suggested schematic flowsheet for the crushing and grinding section is shown in figure 20.

Lack of headroom and crowded equipment present operating and maintenance difficulties in the concentration section of the present mill. The flowsheet and treatment, however, basically are sound. It is doubtful that any major changes in this section would be warranted unless complete rebuilding of the plant is considered. A conveyor to transport jig concentrate to the settling box would partly ameliorate the operating problem.

Complete Rebuilding

Only by completely rebuilding the Lost River mill can all its deficiencies be corrected or the tonnage treated be increased materially. If such action is taken, emphasis should be placed on planning the equipment layout to allow easy operation and maintenance. Overhead cranes should be installed to facilitate handling the heavy equipment. A crushing and grinding circuit (see fig. 20) is suggested. A concentrating circuit similar to that previously employed should be satisfactory, but space should be allowed for easy experimental rerouting of flow and for possible limited addition of equipment to any section. Handling concentrate mechanically should be considered to minimize labor expended for that work. A concentrate drier utilizing waste heat from diesel generators might be feasible.

Continued Research

The preceding recommendations are based only upon laboratory bench-scale tests and limited observations of practice in the existing mill. It is almost imperative, even if future operation is only a remote possibility, that research on the following scale be conducted to obtain enough detailed data for intelligent planning.

1. The reported success of the Aerofall mill as a single-stage unit for reducing the product of the primary crusher down to size of liberation, with production of a minimum of fines, warrants investigation, as this mill probably could be used for disintegrating Lost River ore.

2. Additional bench-scale beneficiation testing on Cassiterite-dike ore, supplemented by continuous testing on a laboratory scale.

3. Investigations of the nature and amenability to concentration of ore minerals from the contact zones and from dissemination in both granite and limestone.

4. Studies to determine the applicability of various dispersants to facilitate separation of fine cassiterite and clay minerals.

5. Investigation of various flocculants to increase the efficiency of water reclamation. This phase of research on Lost River ore is particularly significant because of new solids-liquids separation techniques developed recently and because of the advent of new, effective, flocculating agents.

6. Investigation of methods for economic recovery of byproduct minerals, such as those of tungsten, lead, and zinc.

7. Continued basic research to develop a method for recovering cassiterite by flotation.

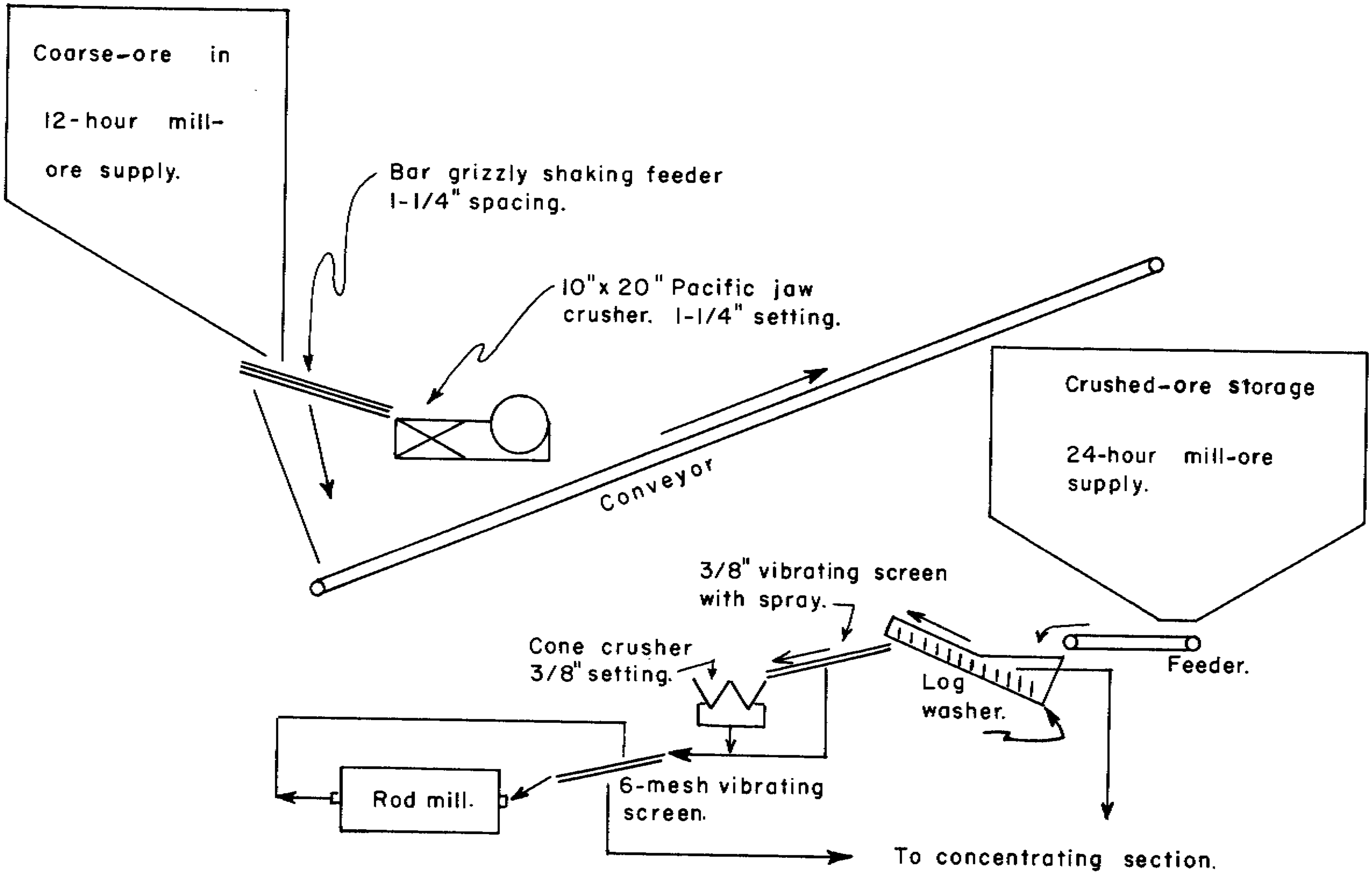

FIGURE 20. - Proposed Crushing-Grinding Circuit.

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