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MINING AND FURNACING METHODS AND COSTS, ABBOTT MINE, COG MINERALS CORP., LAKE COUNTY, CALIF.

By A. C. Johnson and F. D. Hanson

UNITED STATES DEPARTMENT OF THE INTERIOR
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by

A. C. Johnson[2] and F. D. Hanson[3]

SUMMARY AND INTRODUCTION

This circular is one of a series published by the Bureau of Mines describing mining and beneficiation methods and costs at various mining operations throughout the United States.

Inclusion of the Abbott mine in this series is of particular interest, as the complex geology and the character of the deposits make their exploitation a unique operation. The mine operates on a high-cost, comparatively small-scale basis, and only the persistence of an aggressive management has enabled it to maintain its rank among the major mercury producers of California. Rising costs and decreasing metal prices, which continuously threaten the low profit margin, are reflected in increased efficiency of operation.

This report summarizes past operations and describes, in detail, the geology and ore deposits; exploration and mining methods; and furnacing and condensing operations. It covers the entire operation from initial exploration to the actual production of mercury. Comparative production costs for 1944 and 1958 are of special interest.

ACKNOWLEDGMENTS

The authors are grateful to C. O. Reed, general manager of the Abbott mine, for his cooperation and assistance in preparing this report, and to the officials of the COG Minerals Corp. for supplying and permitting the use of the contained data, maps, and pictures.

LOCATION AND PHYSICAL FEATURES

The Abbott mine and furnace plant are in secs. 30, 31, and 32, T. 14 N., R. 5 W., Mount Diablo base meridian, Lake County, Calif., about 24 miles west of Williams (fig. 1). The property, comprising about 400 acres, is in an area of

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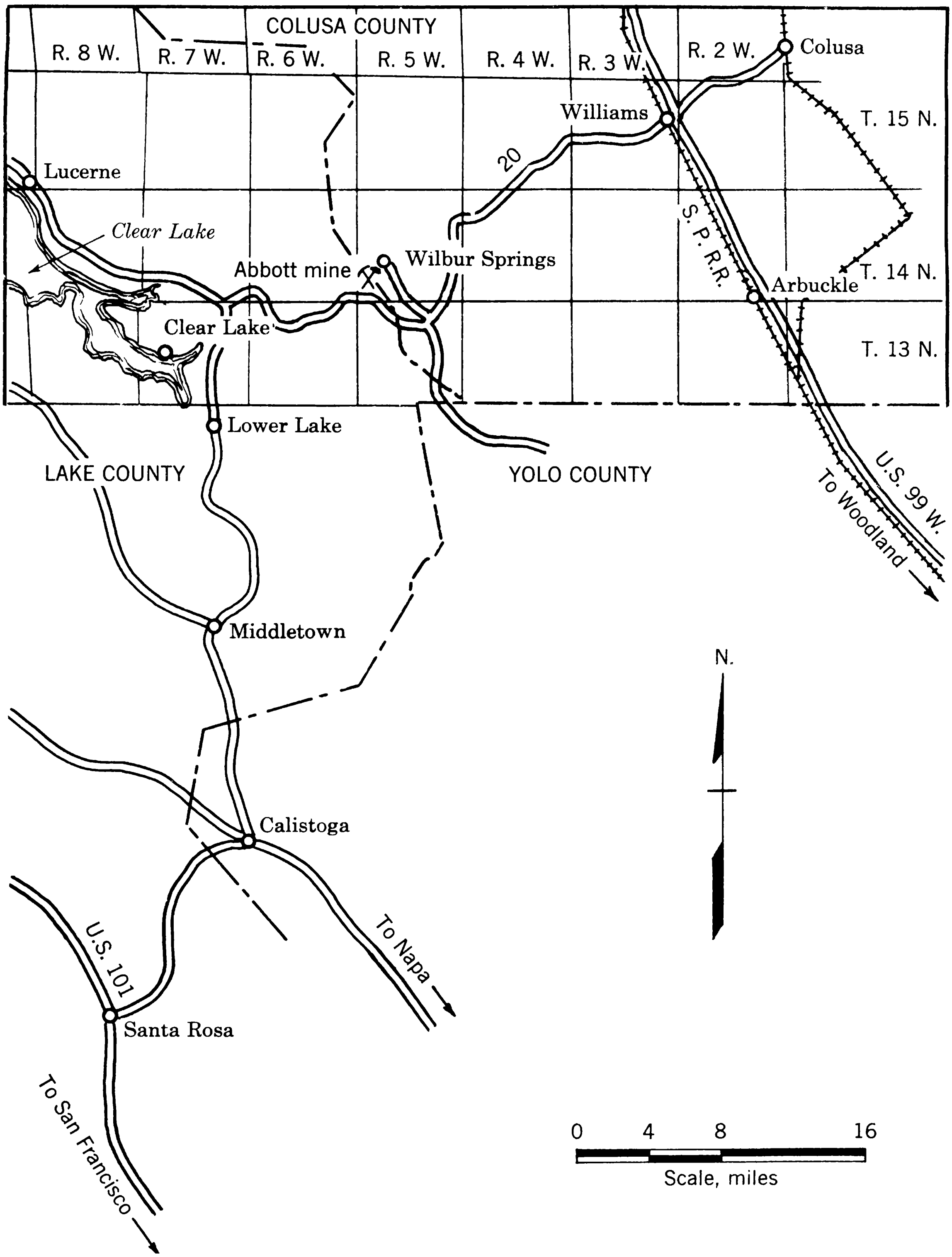

FIGURE 1. - Location Map.

moderate relief and lies in the southwest corner of the Wilbur Springs quadrangle along the western slope of the ridge separating Lake and Colusa Counties. Elevation of the mine is about 2,000 feet. Paved California State Highway 20 connects the mine to Williams on U.S. Highway 99W and the Shasta Route of the Southern Pacific railroad.

A temperate climate prevails with heavy rains during the winter and early spring months. Vegetation comprises oak trees, manzanita, and other evergreen shrubs. A demarcation of the shale-serpentine contacts is clearly visible, as only brush and manzanita grow along the serpentine outcrops.

Telephone facilities are available at the mine. Telegraph, truck transport, and rail facilities are available at Williams. Sacramento, 90 miles to the southeast, and San Francisco, 130 miles to the south, are the nearest labor and supply centers. Timber, equipment, and supplies are trucked to the mine.

Power was formerly supplied by diesel generators, but in 1959 a branch line was constructed from the Pacific Gas and Electric Co.'s transmission system, about 10 miles southwest of the mine. Water for mine and domestic use is pumped from shallow wells on the property.

Company housing is available for 6 families and about 30 single men. Remaining employees live at Williams, Clear Lake, and other small towns within commuting distance of the mine.

HISTORY

Cinnabar was discovered in the Abbot mine in 1862. Early operations during 1870-79 and 1890-1906 yielded 30,000 flasks of mercury. Ore was selectively mined, hand-sorted underground and on surface, and treated in a 10-ton capacity Knox and Osborn[4] furnace and a 48-ton capacity Scott furnace. A period of inactivity continued until 1916-17 when a small amount of mercury was recovered from cleanup operations around the old Scott furnace. Intermittent work by various owners and lessees during 1927-40 also resulted in a small production. The International Metals Development Corp., Seattle, Wash., purchased the property in 1940 and operated it until 1946 when it was acquired by the Abbott Mines, Inc. During this period the mine was developed extensively, a rotary furnace was installed, and 6,623 flasks of mercury were produced. In 1948 the Bureau of Mines explored the property by diamond drilling, with encouraging results.

The company name was changed to California Quicksilver Mines, Inc., in 1951, and the main office was moved to San Francisco, Calif. During 1951-57, extensive exploration under Defense Minerals Exploration Administration (DMEA) contracts resulted in the discovery of new mineralized sections and aided materially in maintaining a continuous operation with a resultant output of 10,147 flasks of mercury from ore averaging 10.36 pounds of mercury per ton.

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[4] Reference to specific makes or models of equipment in this report is made to facilitate understanding and does not imply endorsement of such brands by the Bureau of Mines.

GEOLOGY AND ORE DEPOSITS

General Geology

The ore bodies are contained in dikes and sills of serpentine breccia intrusive into Lower Cretaceous shale and sandstone on the southwestern flank of the Wilbur Springs anticline. This northwest-trending serpentine complex is 2-1/2 miles long and varies from 125 feet to one-half mile in outcrop width. The complex is divided longitudinally into the Front dike system on the southwest side and the Back dike system on the northeast side. Both systems are complex and consist of main-stem footwall (northeast) members with as many as five lateral or up-dip branches. Southeast of the transverse preintrusive A-B fault zone, the serpentine contacts and the bedding planes of the peneconcordant hanging wall shale dip to the southwest at medium-to-steep angles. Away from the serpentine, the shale beds flatten southwestward towards the synclinal axis. Northwest of the A-B fault steep reverse (northeast) dips predominate from the surface down to the 350 level, where the dip reverts to its normal southwest direction and flattens downward to medium angles. On the northeast side of the serpentine, the shale contains much intercalated sandstone, locally predominant, and has a nearly east-west strike and low-to-medium southerly dips, and attitudes which continue westward past the northwest end of the serpentine toward the synclinal axis.

The resurgent cold intrusion of the serpentine as breccia into the incompetent and easily displaced shale was guided by seven sets of preintrusive faults and by the bedding planes of the shale. Southeast of the A-B fault, the primary control consisted of west-northwest-striking reverse faults which dip to the southwest and represent a late phase in the horizontal compression that produced the northwest-trend folds of the region. Contact deflections which are strongest near the A-B fault and decrease in extent to the southeast are very important economically but do not obscure the general trend as they do on the northwest side of the A-B fault.

Four sets of essentially vertical faults with undulatory dips conform in strike to the regional fracture pattern. Two complementary sets strike north-northwest to northwest and east-northeast to northeast. Faults of the first set were important contact deflectors and controlled the general trend of the intrusions for some distance northwest of the A-B fault. Faults of the northeast set, exemplified by the Turner cut, A-B (fig. 2), and Disturnell (fig. 3) fault zones are the most important transverse structural features. On none of the faults have postintrusive movements extended completely across the serpentine complex, although this effect is simulated by preintrusive displacements of the susceptible shale horizon, by intrusive termination of some of the branch dikes against the faults, by sharp changes in thickness of the through-going dikes, and by intraintrusive displacements of the branch dikes produced by differential widening of the larger dikes on one or both sides of a fault zone or by branch dikes within the fault zone. Similar effects are produced by two complementary sets of faults with nearly north-south and east-west strikes. The east-west faults have produced the most favorable contact deflections for ore body localization. The north-south faults are important only in the wide Boggess fault zone at the west end of the productive area.

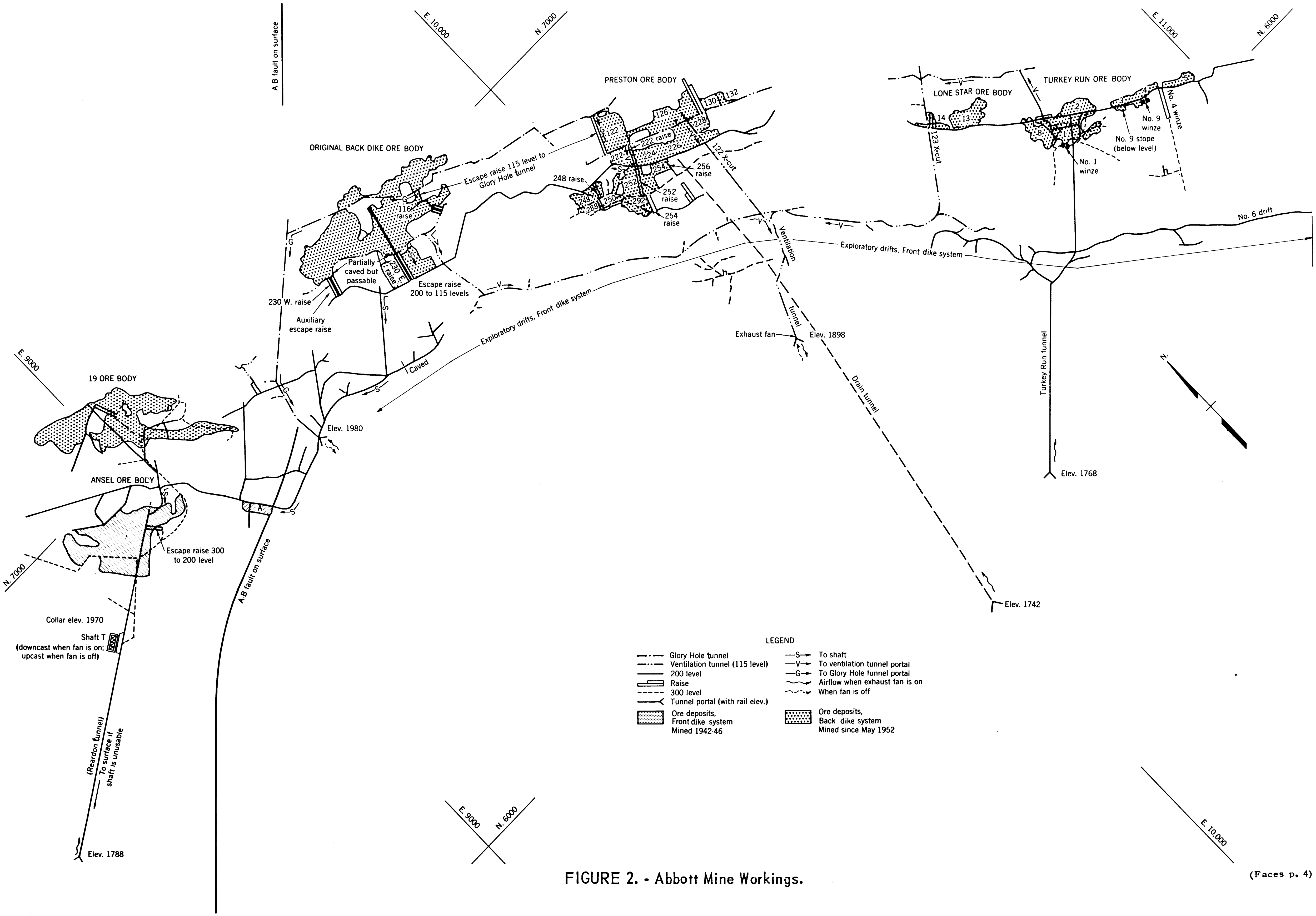


FIGURE 2. - Abbott Mine Workings.

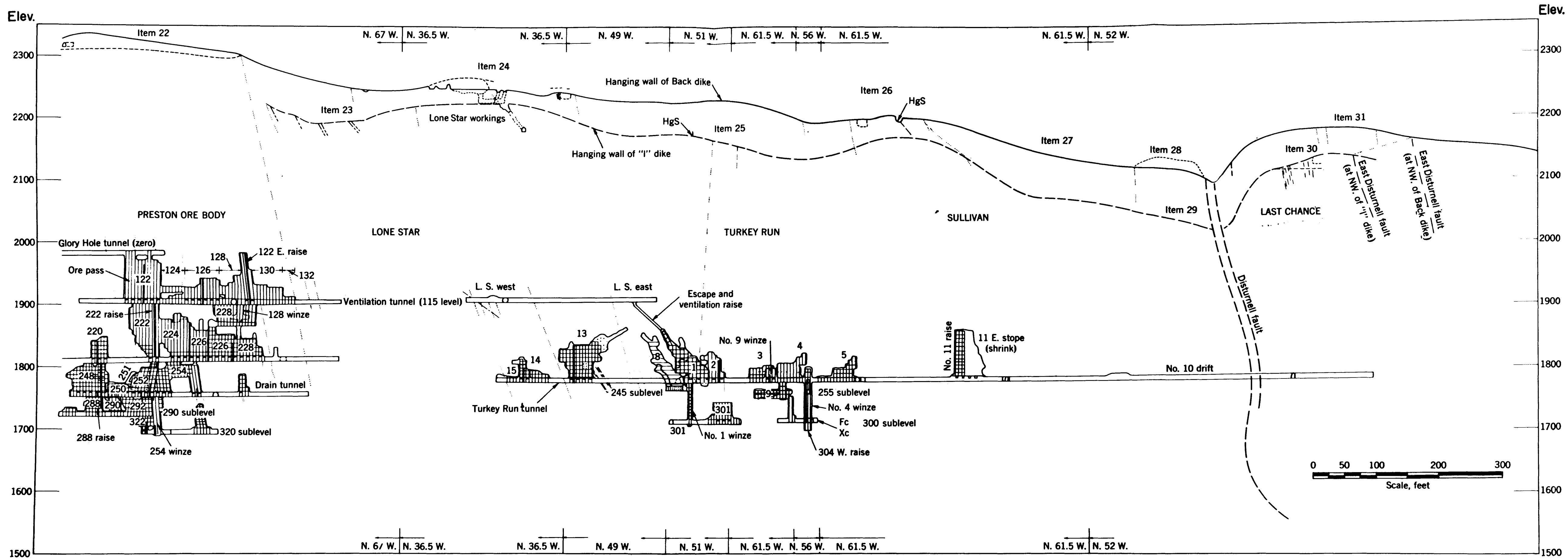


FIGURE 3. - Longitudinal Projection of Southeast Portion of Abott Mine.

(Faces p. 4)

Two sets of low-angle faults, striking east-northeast to northeast and dipping in either direction at average angles of from 20° to 30°, have been important in localizing sections of flattened dip on the hanging-wall contacts of large dikes and in terminating the upward ascent of small dikes which, at places, have silled out beneath the faults. In the former role, faults of these sets have been involved locally in many ore bodies and the northwest-dipping No. 9 fault was the primary control of the Turkey Run ore body. In the latter role, as terminators of small dikes, two southeast-dipping faults (possibly segments of the same fault) localized the 19 and the old B ore body.

Mineralization

Early stages of the mineralization process altered the serpentine breccia to a variety of silica-carbonate rock which was either tough or brittle, according to the relative amounts of magnesian carbonates (dolomite and magnesite) and silica, and could support the tensional fractures that localized the ore bodies. Where both main gangue constituents occur, the highly sheared serpentine was carbonated while the hard, bouldery residuals were partially or completely replaced by dark opaline silica, derived from the carbonated serpentine. This type of rock is locally called tuffoid, a hangover from the days when it was interpreted as a volcanic agglomerate. An increase in the opalite content is first shown by replacement veinlets along the cross fractures and the hanging-wall contact. This grades into masses of essentially solid opalite which house many of the better ore bodies.

The dominant sulfides are marcasite and cinnabar, deposited zonally upward in that order but with an intervening zone of metacinnabar at some places. Other sulfides are rare: One specimen of stibnite has been seen, and several specimens of chalcopyrite and bornite have been observed between the marcasite and metacinnabar. Gold assays up to 0.04 ounce per ton have been secured from massive marcasite, and one high-grade specimen in a narrow quartz veinlet has been reported. Native mercury is absent, but small quantities of mercury oxides or halides are suspected at the bottom of the oxidized zone in some vehicles. The richest ore usually occurs immediately above the downward zoning into marcasite. Locally there is some ore leakage into the overlying shale, and where this shale thins and disappears, the ore spreads into the next overlying dike.

Most of the sulfides were deposited in open fractures or rock interstices, but some replacement probably occurred in the richest ore. The top of the cinnabar zone is usually marked by fracture fillings of vesicular opal-carrying liquid hydrocarbon. Still higher zonal indicators of ore are narrow veinlets of calcite, magnesian carbonates, and chalcedony, sometimes with sparse cinnabar. These zonal relationships have been valuable guides to exploration.

The most prominent secondary minerals are limonite, hematite, sulfur, and various sulfates such as epsomite, jarosite, and melanterite. Workings in altered serpentine are soon covered with epsomite needles and, more locally, by stalactitic melanterite. Much of the liquid hydrocarbon leaks downward from the vesicular opal and collects in rock openings. In some areas the rock and ground water contain and exude H_2S, CO_2, NH_3, and CH_4. Concentration of the latter in a dead-end drift caused an explosion in 1946 that killed two men.

Ore Body Localization and Classification

Abbott ore bodies are found where the serpentine breccia has been sufficiently fractured to provide adequate solution channels and sulfide deposition sites, and where one or more impervious shale walls have impeded the flow of the solutions and confined them to the fractured opalite or tuffoid. Most of the ore seams occupy steeply dipping tensional cross fractures, produced by the horizontal component of contact fault movements resulting from resurgent intrusive activity and later block uplift of the serpentine breccia dikes. Local lateral bulging has produced tensional arch fractures in rounded-off bends in both strike and dip. Generally the position of an ore body is determined by irregularities in the strike, while the location of the best ore is determined by changes in dip; the most favorable spot is the lower end of a section with relatively flat dip. Outwardly pointing contact protuberances or bends are favorable, but concavities are unfavorable. Some of the strike patterns are illustrated in figure 4. In the case of the original Back Dike and Preston ore bodies, the localizing strike deflections were absent or poorly developed on the upper levels. The beneficial effect of dip flattening is illustrated in figures 5, 6, and 7.

The four main types of ore occurrences have different mining characteristics. The Ansel and the Main Boggess ore bodies, with 50,000 tons of ore having a grade of 15 to 16 pounds of mercury per ton, were most easily mined. They were relatively concentrated pipelike ore bodies with unusually steep dips which permitted orderly development from nearby vertical shafts without excessive crosscuts and drifts. The number of nonstandard sets was reduced, and the need for slushing of the ore to drawpoints was eliminated. No ore bodies of this type were mined during the period of this report.

Another type is represented by the Original Back Dike, Preston, Lone Star, and Turkey Run ore bodies, which are alined along the hanging-wall contact of the Back Dike proper in the order named from northwest to southeast, and which have yielded about 83 percent of the post-1951 production. These types have average dips of 52°, but include sections of very flat dip which necessitate slushing, increase the difficulty of holding the gougy shale hanging wall, and complicate timbering by the need either for more nonstandard sets or for a change to inclined sets to facilitate slushing because the lowest girts are on the footwall itself and the next higher girts are all the same distance above the floor. The larger ore bodies of this type exhibit a rough pine tree structure with a more or less centrally located main solution channel, which makes commercial ore throughout the vertical extent of the ore body, but away from which the commercial ore is restricted to nearly horizontal flattenings in dip. This structure increases the development cost per ton of ore mined by necessitating much subsidiary development in the form of sublevels from raises or winzes, and increases the amount of low-grade material which is mined in order to avoid sublevel development.

A third type of ore occurrence is represented by the old B ore body and the 19 ore body, mined in 1952-54, which is shown in cross section in figure 8. This ore body, localized by the upward termination and silling-out of small dikes

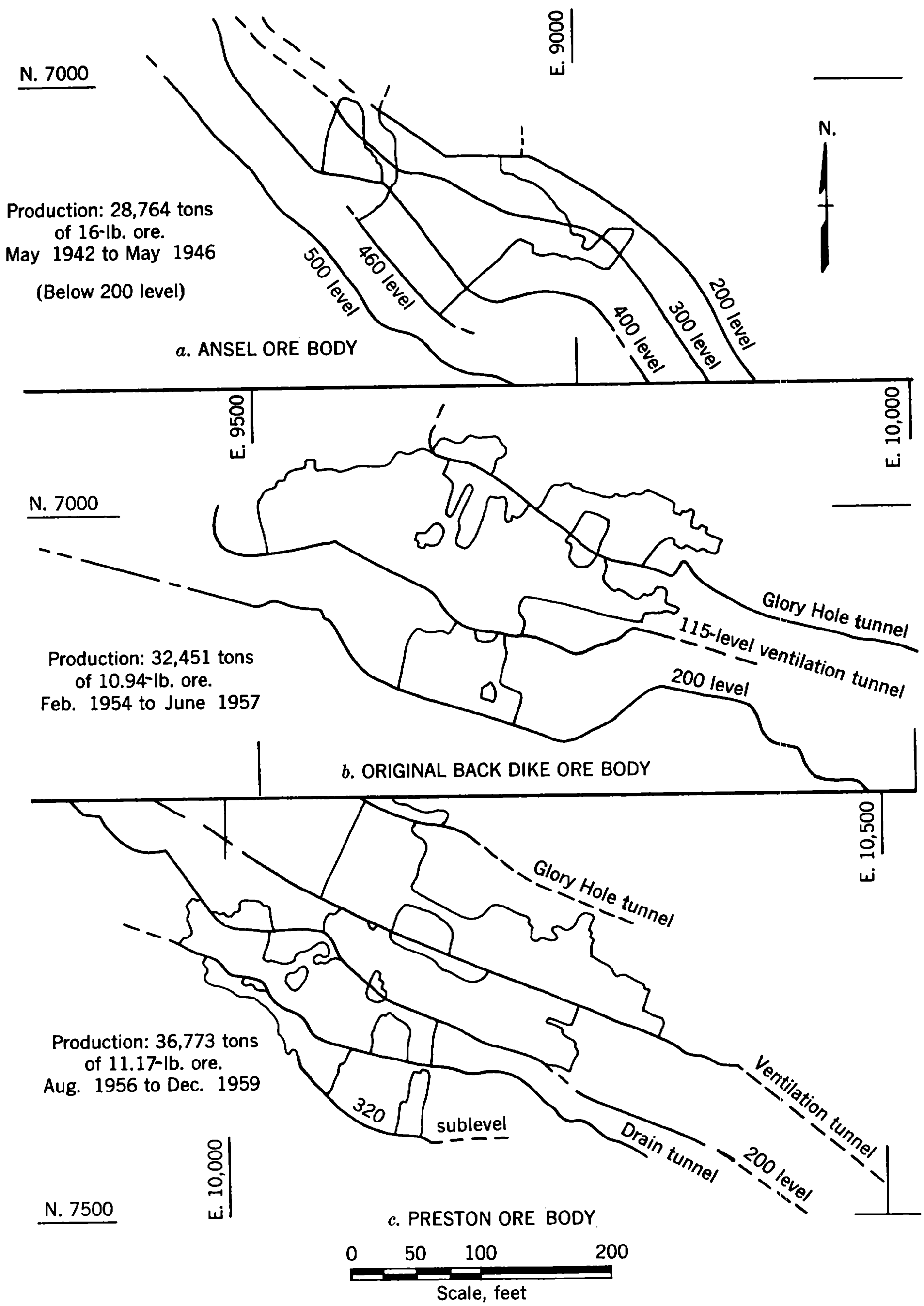


FIGURE 4. - Composite of Three Major Ore Bodies Showing Relationship to Irregularities in Strike of Hanging-Wall Contacts, Shown on Several Levels.

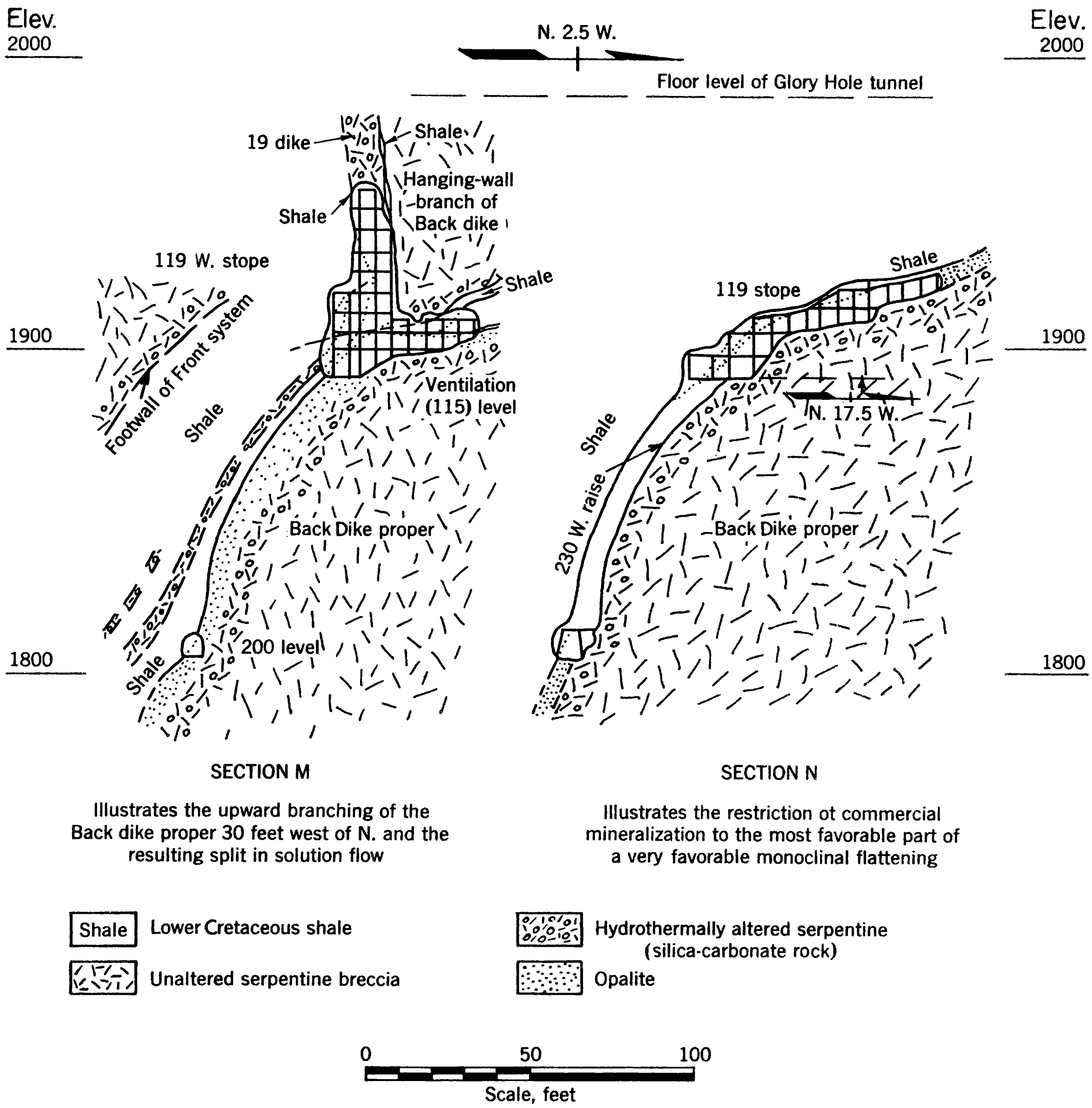


FIGURE 5. - Section Through West End of Original Back Dike Ore Body Showing Stoping Conditions.

beneath a preintrusive thrust fault, has an average dip of 30° or less, and the difficulties of flat-dip mining are extended to the whole ore body. Despite partial filling from waste raises, 219 stope at the top of the ore body caved before diamond drilling could be completed to test for possible upward continuation of the ore body.

Many small, low-grade ore shoots are grouped under a steep-dike category. In several instances, the containing dike was terminated along its strike by one of the undulatory-dipping northeast faults, with favorable terminating dips

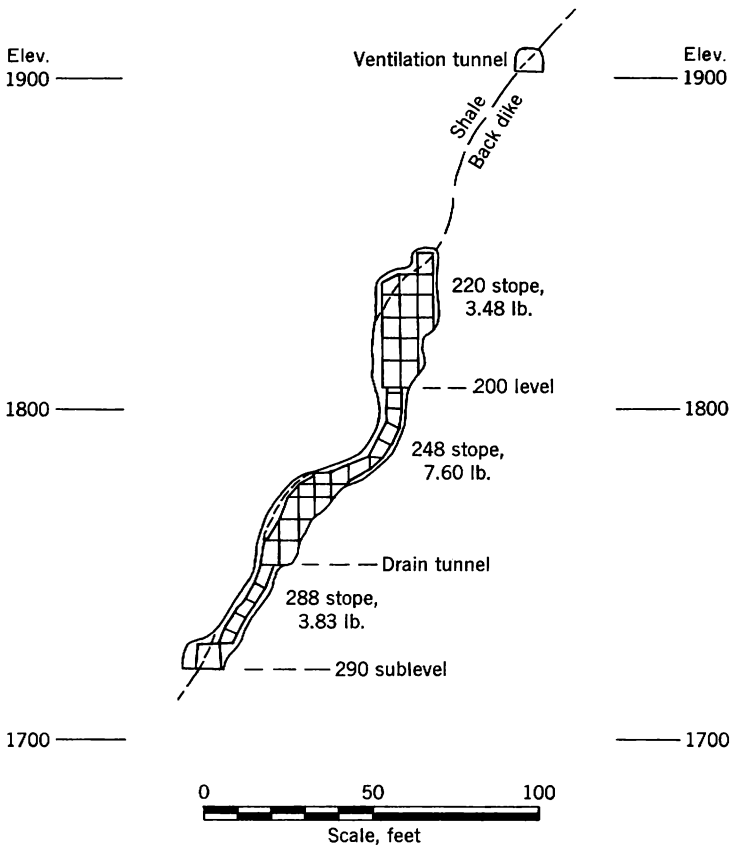


FIGURE 6. - Cross Section Through West End of Preston Ore Body, Good Structure but Limited Vertical Extent.

throughout part of the vertical extent. In general, the small and discontinuous ore shoots were localized by very minor strike and dip fluctuations, or by discontinuous fracture zones with en echelon patterns along a vertical plane.

Small bodies of ore have also been found where small dikes pinch out upward in the shale and in upward-apexing fault slices. All of the profitable production from the mine has come from the major ore bodies with productions of over 15,000 tons, and most of it has come from the largest ones with productions in excess of 30,000 tons. The average grade of an ore body increases with its size. There are no isolated high-grade bodies of ore of minable size.

Table 1 lists ore body dimensions and characteristics for the four major ore bodies mined since 1952 and for that portion of the Ansel ore body mined during the World War II operation.

METHODS OF PROSPECTING AND EXPLORATION

The methods of prospecting and exploration at the mine are limited by the comparatively small size of individual ore bodies and their erratic horizontal and vertical distribution. These irregularities tend to increase exploration and development costs and the time required to bring new ore bodies into production.

Surface prospecting consists of detailed geologic reconnaissance and mapping followed by exploration of geologically favorable areas by surface and underground diamond drilling and driving underground openings.

The exploration work is designed to investigate virgin areas by a combination of diamond drilling, crosscutting, drifting, raising, and winze sinking. Diamond drilling has proved effective in the discovery of new ore bodies,

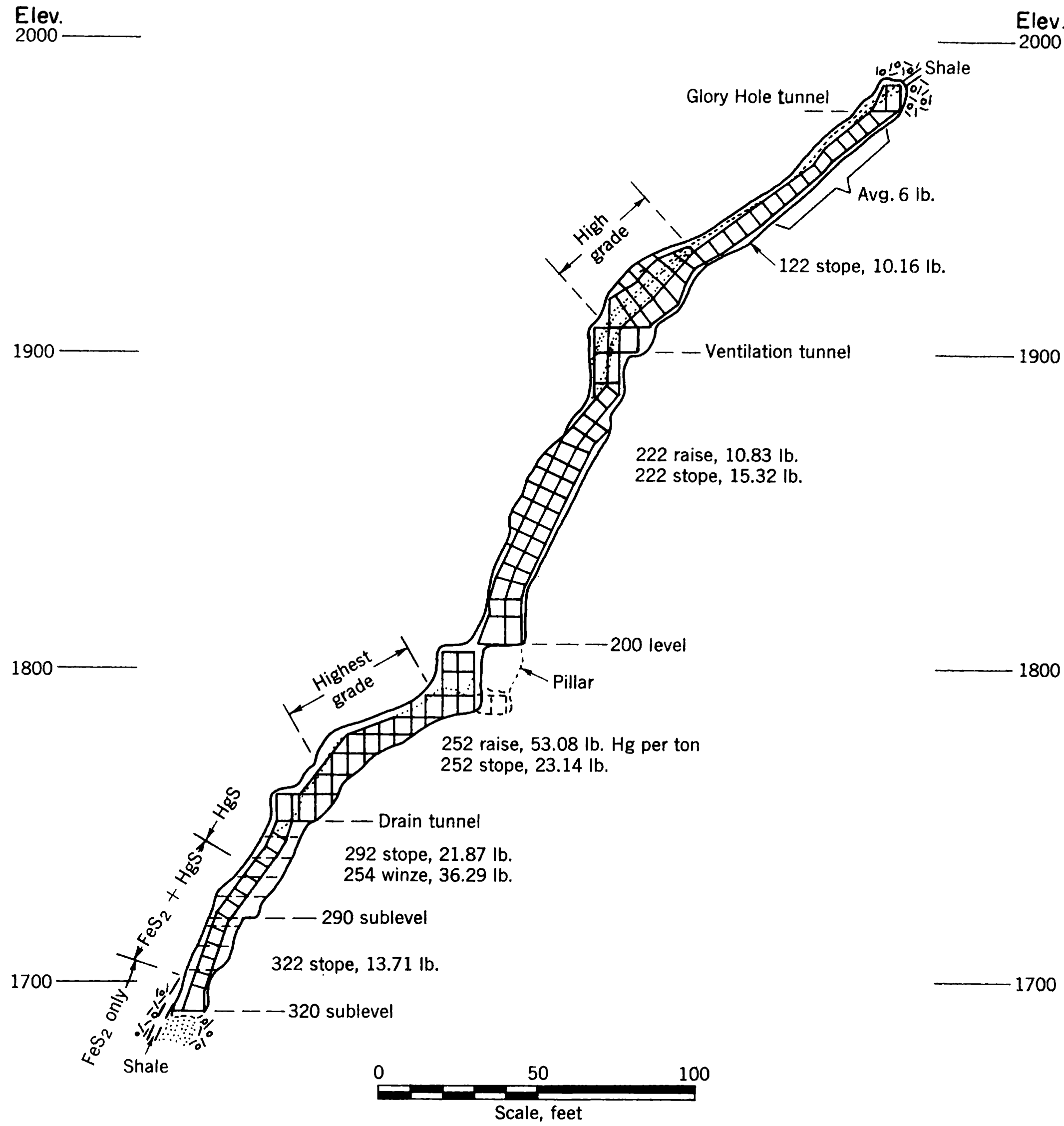
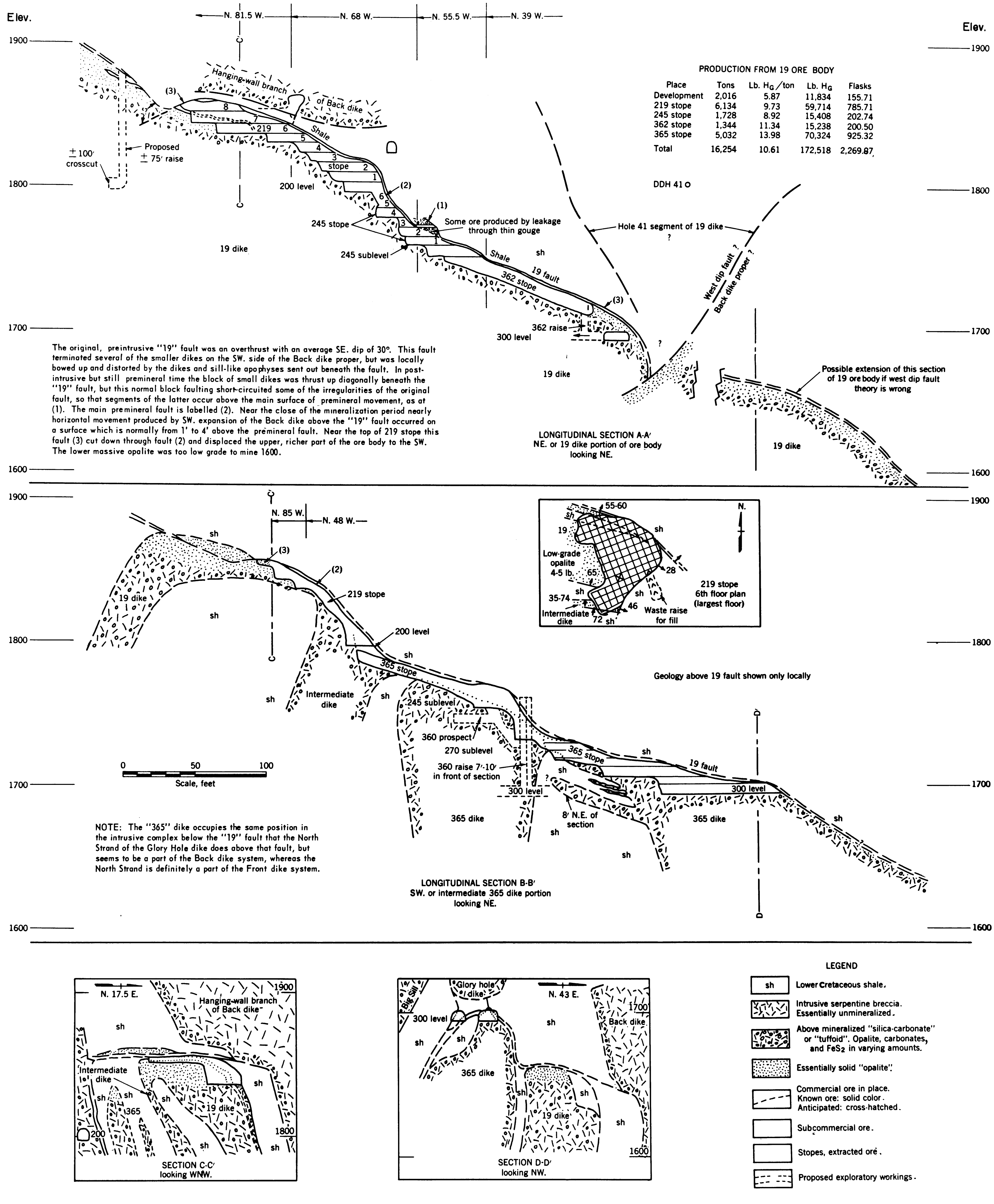


FIGURE 7. - Cross Section Through Preston Ore Body Showing Vertical Continuity of Good Ore in Zone of Strongest Solution Flow.

delineating mineralized areas, and obtaining geologic data. Driving underground openings serves to determine the extent of the mineralized area and permits its subsequent evaluation.

Diamond drilling is done on company account. Surface drilling utilizes a 20-horsepower gasoline-driven drill and a demountable tripod constructed of a 1-1/2-inch-diameter aluminum tubing and universal-jointed clamp fittings which can be attached at any point and at any angle. It is similar to equipment widely



PRODUCTION FROM 19 ORE BODY

Place	Tons	Lb. Hg/ton	Lb. Hg	Flasks
Development	2,016	5.87	11,834	155.71
219 stope	6,134	9.73	59,714	785.71
245 stope	1,728	8.92	15,408	202.74
362 stope	1,344	11.34	15,238	200.50
365 stope	5,032	13.98	70,324	925.32
Total	16,254	10.61	172,518	2,269.87

FIGURE 8. - Section of "19" Ore Body.

(Faces p. 10)

TABLE 1. - Production and characteristics of ore bodies

	Tons	Flasks	Hg, pounds per short ton (average)	Dimensions, feet: Strike length, Maximum	Strike length, Average	Dip length	Maximum vertical extent	Horizontal thickness, Maximum	Horizontal thickness, Average	Horizontal thickness, True
Ansel[1]	28,764	6,056	16.00	218	123	280	274	47	11.1	10.8
19	16,254	2,270	10.61	138	71	[2] 380	166	67	17.9	7.8
Original Back Dike	32,451	4,672	10.94	325	170	335	240	37	10.3	7.4
Preston	36,896	5,412	11.15	275	142	388	295	42	11.4	8.7
Turkey Run	18,718	2,369	9.62	245	109	165	153	55	14.6	13.5
Average (excluding Ansel) .	26,080	3,681	10.73	246	123	317	214	50	13.5	9.3

	Degree of dip[3]: Maximum	Degree of dip: Minimum	Degree of dip: Average	Elevation, feet: Top of ore	Elevation, feet: Bottom of ore	Vertical distance, top of ore to surface, feet	Tons per foot of- Dip length	Tons per foot of- Maximum vertical extent	Tons per foot of- Average strike length	Flasks per vertical foot
Ansel[1]	97	43	69	1,798	1,529	252	103	105	234	22.1
19	70	0	25	1,861	1,693	339	43	98	229	13.7
Original Back Dike	107	0	50	2,042	1,802	[4] 295	97	135	191	19.5
Preston	95	0	52	1,987	1,700	349	95	125	260	18.3
Turkey Run	95	0	52	1,860	1,707	387	112	122	172	15.5
Average (excluding Ansel)..	92	0	45	1,938	1,725	342	87	120	213	16.8

[1] Figures for the Ansel refer only to the portion of the ore body mined below the 200 level. The total tonnage may have been 50,000 tons. The top of the ore was at an elevation of about 2,020 feet or about 30 feet below the surface. The total vertical extent was 491 feet.

[2] This figure is an average for the two portions of the ore body which had individual dip lengths of 340 and 420 feet. This ore body has not been bottomed and may have some upward extension.

[3] Dip figures over 90° indicate that vertical dips have turned over into steep dips in the reverse or northeast direction.

[4] Maximum.

used for the construction of scaffolding. Experience has indicated it to be equally as strong as a heavy wooden tripod and its use greatly reduces setting-up, tearing-down, and moving time. Underground drilling is done with a column-mounted air-driven drill.

Holes drilled to maximum depths of about 350 feet are started BX size with the placing of a few lengths of BX casing, depending upon near-collar rock conditions. If drilling troubles are encountered, AX casing fitted with a casing shoe is run in the hole and reamed down as far as practicable. In case of additional trouble, EX casing is introduced, and the hole is completed with EX core barrels. Double-tube rigid and swivel-type core barrels are used, with core recovery averaging 59.5 percent for 15,382 feet of drilling. Average efficiency for the same footage was 15.4 feet per shift. Drilling conditions and core recovery are usually good until an ore zone is approached. Drilling water converts the already brecciated and gougy shale hanging wall into squeezing ground, which is very difficult to penetrate. Water return is lost in the fractured opalite of the ore zone itself, and core recovery drops to 15 percent or less. For these reasons it is not feasible to try to keep holes within an ore zone, or to make acute-angled intersections. However, because the occurrence of most of the cinnabar in cross fractures, wide-angle crosscutting holes may pass through an ore body without encountering cinnabar, or may raise false hopes by following an isolated seam. These factors enhance the advantages of actual entry by underground workings; the information obtained is much more complete and decisive.

The principal mechanical trouble encountered during drilling was with the bearings in the double-tube swivel-type EX barrels.

In addition to diamond drilling, underground exploration includes driving 5 by 7 foot crosscuts and drifts and two compartment timbered raises and sinking 1-1/2-compartment inclined and vertical winzes.

Diamond-drill holes and underground workings have been both competitive and complementary modes of exploration. Where a choice is required, several factors besides the primary ones of distance and ore probability must be considered. These are: (1) The relative availability among the crew of drift miners and diamond-drill operators, (2) the relative pressure for new ore; if this is badly needed the time advantage of actual entry may be a decisive factor, (3) a psychological factor; for some time after an extensive drilling program, dissatisfaction with drilling delays, poor core recovery, and indecisive results encourages exploration by underground workings without preliminary drilling. When enough barren crosscuts and drifts accumulate on the map, a reaction sets in, and any advance information seems desirable.

During the earlier stages of the exploration of the Back dike ore zone, diamond drilling and underground openings worked very well together as complementary methods of exploration. Several empirical rules were formulated: First, drilling should be stopped as soon as one good ore intersection was secured. This would inspire a crosscut or drift regardless of the negative results of subsequent holes. Second, a virgin stretch of contact merited actual entry regardless of the absence of cinnabar in the core if three horizontal holes showed strong mineralization over a strike length of 100 feet or more, and if two angle holes indicated a favorable dip structure. A minimum program for one area thus consists of a vertical fan of three holes with two additional horizontal holes. This assumes that the area to be drilled is considered to be favorable.

Exploration and development performance for a 10-year period is listed below:

Year:	Footage, underground workings	Footage, diamond drilling
1951	-	5,418
1952	916	2,390
1953	1,299	188
1954	1,603	-
1955	2,002	-
1956	2,866	4,546
1957	3,093	465
1958	2,041	328
1959	1,677	-
1960	993	1,971
Total	16,490	15,306

Production for this period was 111,520 tons. This gives the following per ton figures, at average costs per foot of $31.25 for underground workings and $6.50 for diamond drilling:

	Tons per foot	Cost per ton
Diamond drilling	7.28	$ .89
Underground workings	6.76	4.62
Total	3.51	$5.51

METHODS OF SAMPLING AND ESTIMATION OF ORE RESERVES

Ore occurrences are evaluated by visual inspection. On fresh exposures, cinnabar is easily visible in rock assaying less than 1 pound per ton, and within the critical range of from 3 to 10 pounds per ton, experienced personnel can estimate average grade within a pound. Above this grade range, visual inspection to determine grade of ore becomes increasingly uncertain, but it is sufficient for operational accuracy.

Visual estimates are checked by sampling whenever any doubt exists as to whether a stope should be started or stopped, and are constantly calibrated by comparison with the daily tonnage, mercury production, and average grade figures which are available in the afternoon of the following day. There are periods when all of the ore comes from one working place, and at other times satisfactory grade figures can be assigned to separate tonnages by taking into account the time lag between ore delivery and reduction and the known grade of the ore from one source before or after the period of admixture.

Similar methods are used in compiling the monthly production record. Tonnages are distributed on the basis of the mine car count from each working place and the mine car tonnage factor for that month. In distributing the mercury production the smaller ore sources are considered first and are assigned average grades on the basis of the visual estimates and analyses of the daily production record. The average grade of the largest producer is then found by calculation.

Grab smapling of cars is not done routinely, so that when the need for confirmation sampling arises there is ordinarily no opportunity to sample enough cars of development muck to secure a satisfactory average. The method is unsuitable for determining the grade of the back of a stope because of broken ore storage and hang-up within the stope. Cut samples are required, and the mode of occurrence of the cinnabar determines the manner of their taking. In ore of the grade requiring sampling, most of the cinnabar occurs in rather regularly spaced cross fractures lying at angles of from 60° to 90° with the hanging-wall contact. Within each cross fracture the cinnabar thickens towards the hanging wall, and near the latter may spread into subsidiary fractures. Face samples in drifts are valueless because the round invariably breaks to one of the cross fractures. This distribution of the cinnabar also eliminates the possibility of cutting horizontal wall samples or back samples at right angles to the strike of the contact. The method used in both drifts and stopes is to cut diagonal back samples in the direction at which they cut the cross fractures in the most informative way. An en echelon sample pattern is the result.

Ore reserve estimates are prepared at the first of each year and occasionally at other times. Standard practice is followed as closely as possible. Three classes of reserves are recognized. The names of the second and third-class reserves have been changed from probable and possible to indicated and inferred, although little semantic gain is observed. Both the proven and measured appellations for first-class reserves have seemed to connote a higher degree of tangibility than is usually present, and, in addition, first-class reserves are frequently no more measurable than second-class blocks. The terms assured or positive have seemed to reconcile this aspect with the fact that first-class reserves are almost guaranteed to be realizable.

If no reliable information is available from raises or adjacent stopes about the vertical extent of an ore showing, positive reserves are restricted to the height or depth which would be stoped before any diminution in grade could be considered permanent, for example, from one to four sets - depending on the strength of the mineralization in the drift, on its zonal position, and on the presence or absence of favorable strike and dip features.

Since early 1957 the inferred class has included very speculative areas. After the Preston ore body was discovered beneath the first of a chain of barren outcrops of altered serpentine, it became desirable to formally recognize the possibilities beneath the others. In one report the inferred reserves were divided into two subclasses. This seemed unwieldy, so in subsequent reports a statement was appended that these reserves had no tangible evidence of existence and no standing as reserves from an engineering viewpoint, and that they were merely an attempt to give relative quantitative standings to prospecting possibilities. In these areas the tonnage assigned was usually 50 percent of what an ore body of the size indicated by the outcrop could be expected to yield. Subsequent adjustments were made according to current exploration results and expectations.

Strike-length measurements are usually available on at least one level for both positive and indicated reserves. Since vertical square-setting is the predominant stoping method and since sharp changes in dip may make dip-length estimates unreliable, horizontal and vertical measurements or estimates are used for thickness and height. However true thicknesses and dip-lengths have been used where adjacent stopes were mined by inclined sets. A volume factor of 13 cubic feet to the ton is used.

Accessibility or availability of reserves is not ordinarily taken into account since the amount of work required to get into different areas, or back into them, is known to readers of the report. However, when operations have been restricted to certain portions of the mine, separate summaries of reserves available in that portion have been made.

Grade figures are not assigned to reserves; it is assumed that they will approximate the average grade for the mine. Thus, the average grade of the four major ore bodies mined since 1952 varies from 9.62 pounds per ton to 11.15 pounds, with an average of 10.73 pounds. Some ore blocks have been dropped as the price of mercury has declined, and other blocks have been placed in an inferior class if there is any doubt about their commercial grade.

The mine cannot be operated on a rigid cutoff basis if continuous furnace operation is to be maintained. Too much ore would be missed, since local structural features too small to show up in adjoining stopes or on other levels can produce sufficient high-grade ore to bring an apparently low-grade block up to mine average. The pressure for tonnage has started many blind stopes in subcommercial rock where zonal and structural considerations indicated a chance for upward improvement. The following figures show the percentages of the tonnage from four major ore bodies in different grade ranges, as computed from the production figures for individual stopes and development headings. It will be noted that only 50 percent of the ore falls in the 7-1/2 to 12-1/2-pound range. Some low-grade development rock is included and some low-grade ore has been mined to maintain production when better ore was not available, but much of this material was mined to find the ore bodies of plus 15-pound ore.

Grade range, pounds of mercury per ton:	Percentage of total tonnage
Less than 5	11.8
5-7-1/2	15.9
7-1/2-10	34.1
10-12-1/2	15.6
12-1/2-15	6.0
More than 15	16.6

Table 2 presents ore reserve figures as actually reported, along with yearly production figures which apply to the year starting with the date on the same line. Because of the highly speculative nature of much of the inferred reserve, a total of positive and indicated reserves is given as a figure to be used for the total reserves of the mine as that term is commonly used. Until 1959 this total was usually realized, although seldom exactly in the places predicted. Enough ore was found in inferred blocks to make up for disappointments in indicated blocks. The drop in both production and reserves since 1958 is at least partially correlated with a corresponding drop in exploration and development performance, as illustrated in table 2, although the work that was done did not meet with as much success as in previous years.

TABLE 2. - Ore reserves, tons

Year and month	Mined	1 Positive	2 Indicated	Total 1 and 2	3 Inferred	Grand total
May 1						
1952	3,814	-	-	-	-	-
January 1						
1953	9,673	9,200	8,000	17,200	58,000	75,200
1954	9,764	2,825	9,925	12,750	55,700	68,450
1955	13,461	10,200	10,160	20,360	36,150	56,510
1956	14,994	7,250	21,140	28,390	16,120	44,510
1957	22,595	18,030	36,360	54,390	68,620	123,010
1958	18,066	18,200	50,770	68,970	263,190	332,160
1959	11,880	10,260	19,760	30,020	159,080	189,100
1960	7,273	6,050	20,580	26,630	115,000	141,630

DEVELOPMENT METHODS

Development comprises more than 3 miles of underground workings extending over a vertical range of 500 feet (fig. 9). The earlier mine workings, covering mostly the central and northwest sections of the ore zone, include the 500-foot vertical Main shaft with levels at 200, 300, 400, and 500 feet; the vertical 250-foot Boggess shaft, long since caved, at the northwest end of the developed zone; the Reardon tunnel, a crosscut adit which connects with the 200 level of the Main shaft and the equivalent 150-foot level of the Boggess shaft; and the Glory Hole and Ventilation (Lightner) tunnels with portals southeast of the Main shaft at elevations corresponding to zero and 115 levels of the Main shaft. Stopes served by these workings were exhausted by July 1958, but many exploration possibilities still exist within their limits.

The Glory Hole and Ventilation tunnels and the 200 level of the shaft overlap part of the workings from the Drain and Turkey Run tunnels, two crosscut adits near the southeast end of the property which serve as main haulage ways and provide access to the most recently mined ore bodies. The workings involved in their development have a lateral extent of 2,100 feet and a vertical range of 300 feet.

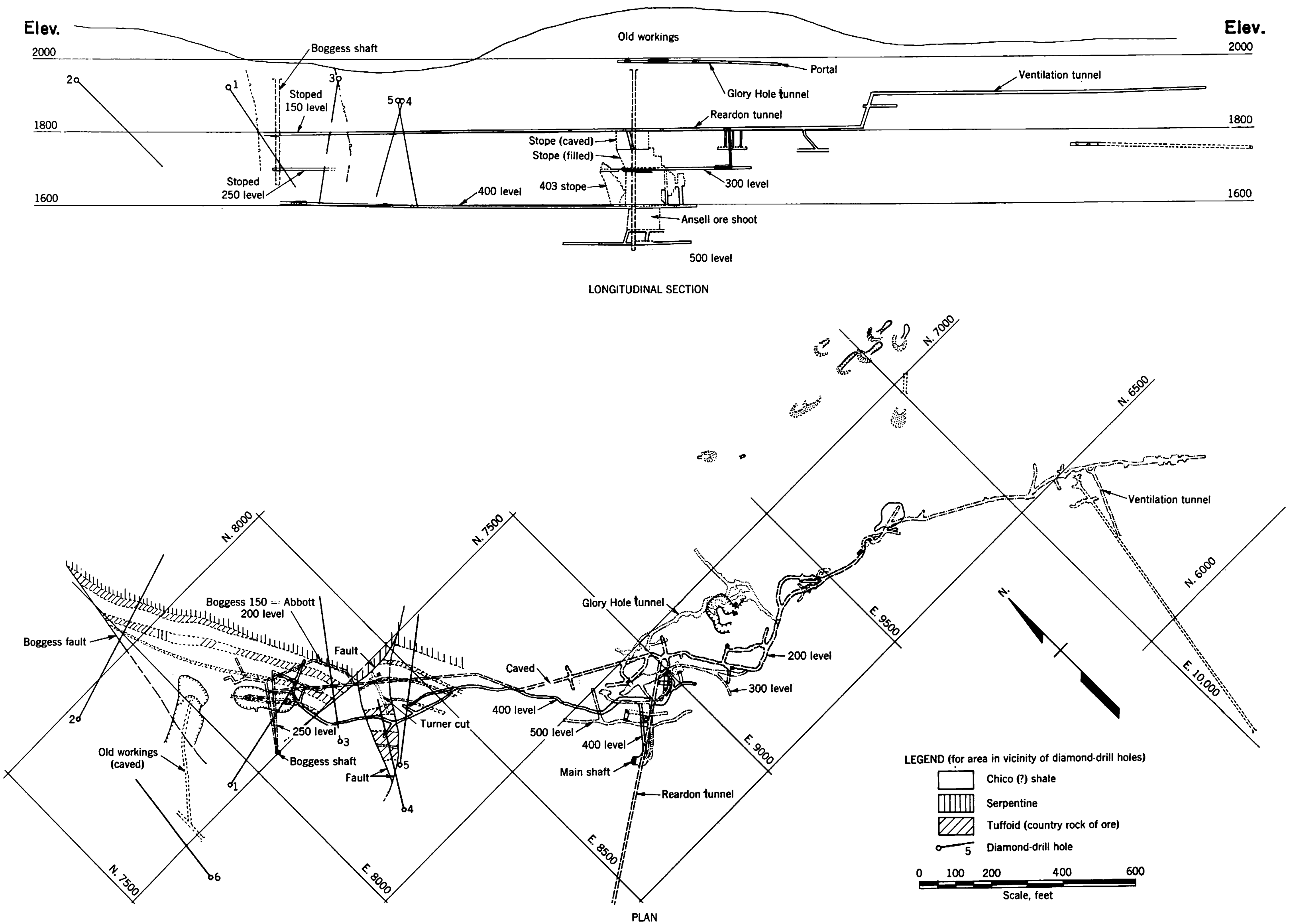


FIGURE 9. - Plan and Longitudinal Section of Abbott Mine, Lake County, Calif.

The Drain tunnel is the lowest adit entry on the property and corresponds to the 265 level of the shaft. In 1941-42, the tunnel was driven 850 feet to the Front dike system but no ore bodies were found and the tunnel was abandoned. However, it pointed directly toward the new Preston ore body which was first opened by a crosscut from the ventilation tunnel and later by a southeastward extension of the 200 level. When a still lower level was required, the Drain tunnel was reopened and extended 200 feet. The bottom of this ore body was mined from a winze sunk below the Drain tunnel level.

The Turkey Run tunnel is the farthest southeast mine entry and in the last 3 years has been the most active. It is also the first entirely new entry to be driven since 1942. In elevation, it corresponds to about the 235 level of the shaft which lies about 1,850 feet to the northwest. The crosscut portion is about 700 feet in length, and workings in the Front and Back dike systems have lateral extents of 750 and 1,400 feet, respectively. Sublevel workings about 66 feet below the tunnel level are serviced by a vertical winze equipped for one-compartment cage hoisting. The Turkey Run tunnel is connected only with the Ventilation tunnel, but the latter connects also with the Glory Hole and Drain tunnels and the 200 level of the shaft (fig. 2).

Ore from both the Drain and Turkey Run tunnels is hauled to the furnace plant ore bin by dump truck. The Drain tunnel has an ore bin, but the ore cars from the Turkey Run tunnel are hoisted to a loading ramp by a tugger hoist and dumped directly into the truck.

Shafts and Winzes

The Main shaft was completed in 1945 and used until July 1958 for access to the Ansel ore body, 19, the original Back dike, and the Preston ore body above the 200 level. In 1955, the shaft caved below water level and had to be filled up to the 300-foot level to save the 300-foot station. It is functional above the 300-foot level, is kept unwatered, and will probably be used again as an operating entry in the future. The shaft contains three compartments, two hoisting and one manway, 4 by 4 feet inside timbers. Outside timber dimensions are 14 feet 8 inches by 5 feet 4 inches; 8-inch timbers were used with sets placed on 5-foot centers. Two-inch outside lagging was used in heavy ground.

Winzes in the drain tunnel and Turkey Run workings include two-compartment inclined and vertical winzes and a one and one-half-compartment vertical winze (fig. 3). The two-compartment type consists of hoisting and manway compartments, 4 by 4 feet inside timbers; the one and one-half compartment type includes a 4- by 4-foot hoisting and a 2- by 4-foot manway compartment. Timbers used are 8- by 8-inches placed on 5-foot centers. Two-inch outside lagging is used when necessary. A 24-hole, V-cut round is used in the larger winzes with an average advance of 5 feet per round; a 20-hole, V-cut round, used in sinking the one and one-half compartment winze, averaged 5 feet per round.

Winze sinking is done with hand-held sinker-type drills, using 7/8-inch collared, hexagon steel and throwaway detachable bits. Forty-percent strength nitrostarch explosive in 1-1/8- by 8-inch cartridges is normally used for blasting. In wet areas, 40-percent strength ammonia dynamite is used. Detonation is by electric primers. Mucking is done by hand into sinking buckets or 3/4-ton end-dump cars which are hoisted by air-driven hoists.

Tunnels, Drifts, and Crosscuts

Main haulage tunnels, drifts, and crosscuts are driven 5 by 7 feet in cross section. In ground requiring support, the cross section is increased to 7 by 9 feet to accommodate necessary timbering.

A burn-cut round is used, the number and depth of holes depending upon the character of the ground. An average round uses 15 holes and pulls 5 feet. Drilling is done with pusher leg-type drill, using 7/8-inch collared hexagon steel and throwaway detachable bits. In exceptionally hard ground, hand-crank drifters, using 1-1/4-inch lugged steel, are used. Blasting is done with 40-percent strength nitrostarch explosive detonated electrically. A compressed-air-driven shovel loads broken material into 1-ton capacity, V-shaped, side-dump mine cars pulled by storage battery locomotives.

Peeled round timbers used for posts and caps are 10 to 13 inches in diameter; girts range from 4 to 6 inches in diameter. Timbers are framed. Posts are battered, about 1 inch per foot of vertical height, to resist side pressure. Sets are placed on 5-foot centers. Round and 2-inch plank back and side lagging is used when driving through brecciated shale.

Raises

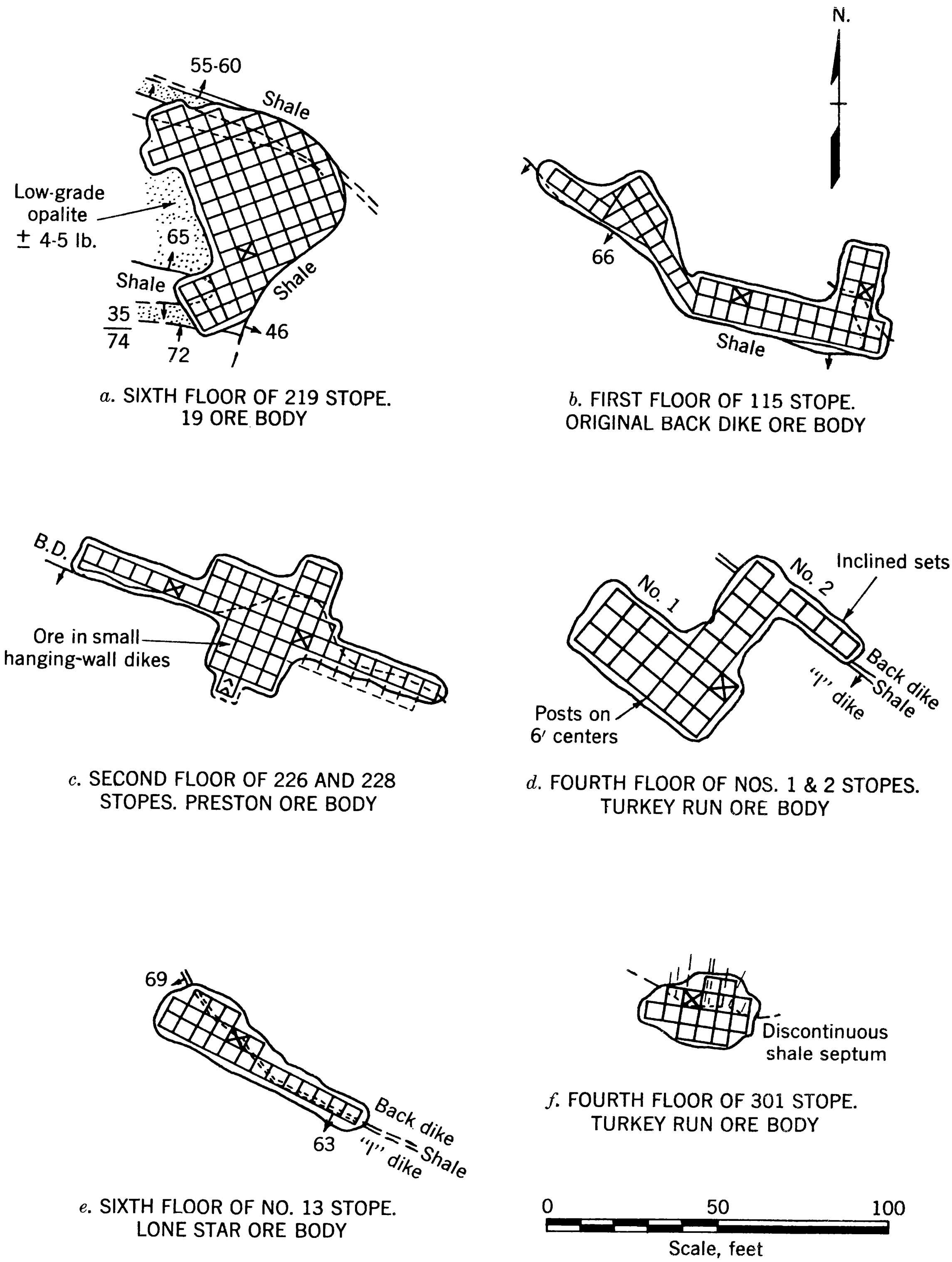
Both two-and three-compartment raises are used. Exploratory raises are usually two-compartment, serving as a chute and manway. Development raises comprise two chutes separated by the manway compartment. Framed round, peeled timbers are used for ground support. Sets are placed on 5-foot centers. The inside dimensions range from 4 by 8 feet to 4 by 12 feet.

A V-cut round is used; the number and depth of holes needed is determined by the character of the ground. An average round for a 2-compartment raise uses 12 holes and pulls 5 feet; a three-compartment raise round averages 18 holes with an advance of 5 feet. Drilling is done with stopers using 1-inch quarter octagon steel and throwaway detachable bits. Forty-percent strength nitrostarch explosive, detonated electrically, is used for blasting.

In steep raises, the muck drops into the chutes; in flatly inclined raises, slushing is ocassionally used.

Stoping

The erratic character of the individual ore bodies, their comparative small size and scattered horizontal and vertical distribution, and heavy ground all contribute to a complicated and expensive system of mining. Generally, the ore bodies dip about 50° south, but strikes and dips change tremendously within a few sets, necessitating a modified system of square-set stoping with many local variations (figs. 4, 6, 7, and 10). Stope sizes vary greatly; typical examples



a. SIXTH FLOOR OF 219 STOPE.
19 ORE BODY

b. FIRST FLOOR OF 115 STOPE.
ORIGINAL BACK DIKE ORE BODY

c. SECOND FLOOR OF 226 AND 228
STOPES. PRESTON ORE BODY

d. FOURTH FLOOR OF NOS. 1 & 2 STOPES.
TURKEY RUN ORE BODY

e. SIXTH FLOOR OF NO. 13 STOPE.
LONE STAR ORE BODY

f. FOURTH FLOOR OF 301 STOPE.
TURKEY RUN ORE BODY

FIGURE 10. - Representative Stope Floor Plans.

are the Back Dike ore body with a maximum strike length of 320 feet, a dip length of 320 feet over a vertical extent of 235 feet, and an average thickness of 10 to 12 feet, and the 19 ore body, which has a strike length of 120 feet, a dip length of from 320 to 405 feet, and an average thickness of 8 to 10 feet.

Stopes are started from the drifts by silling out to the footwall, placing 7- to 8-foot sill posts with caps extending to the footwall, and commencing stoping operations from a pioneer cut above the drift, or laterally from a previously driven raise. In small stopes, which do not extend more than a few sets in height, a pioneer or raise round is drilled into the ore body above the drift and, from the resultant cut, stoping continues by a series of slab rounds drilled horizontally along the ore body. The excavated section is timbered and stoping is continued by raising from each successive floor and continuing the horizontal slab rounds until the ore body is mined out. Mining of large ore bodies by the square-set method starts from a raise. The first floor over the sill is mined out by a series of slab rounds; mining of successive floors is then continued horizontally from the raise (fig. 11).

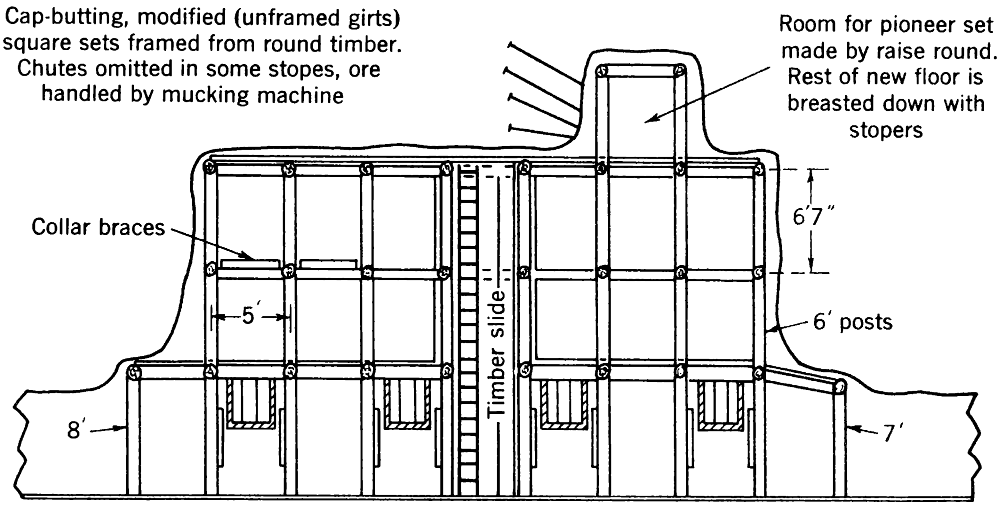
Drilling in the square-set stopes is done with stopers using 1-inch quarter octagon drill steel and throwaway detachable bits. Overhead stoping is accomplished by drilling flat holes and using a slab round system.

Heavy ground in the stopes is supported by a system of square-set timbering. Framed, peeled, round timbers are used with 6-foot posts and 5-foot caps and girts in regular vertical sets. Diameter of the timbers varies from 8 to 10 inches for posts and caps and 6 to 8 inches for girts. Caps are placed normal to the strike of the ore body or from hanging wall to footwall. In some narrow stopes, the sets are placed normal to the dip with an inclined post placed against the hanging wall and the caps cut to fit the ground. This latter method facilitates the use of slusher-scrapers in flat-dipping stopes. Loading chutes are ordinarily spaced on 10-foot centers.

During the stoping operations, the hanging wall of the ore body is probed continuously to explore for possible occurrences in parallel hanging wall dikes. Excavations into the hanging wall require the placing of many special timber sets. In addition, the tendency of the shale hanging wall to squeeze and slough makes close timbering necessary.

Broken ore from the pioneer, or first floor, cut is handled through stopboards into the mine cars or is blasted to the sill level and handled by mucking machines. In the stopes, the ore drops by gravity into chutes or is mucked into the chutes by slusher scrapers. Gravity fall is characteristic of the steeper dipping ore bodies; in the flat-dipping stopes, two-drum electric and air slushers, using a 30-inch standard hoe scraper, are generally used. Ore is scraped down the footwall into the loading chutes, and then drawn into the mine cars.

Filling is required in stopes where extreme caving conditions exist. Waste rock is dumped into the stope from the level above through raise chutes, or other openings to the stope, and is distributed by hand mucking or slusher scraping. Occasionally, filling is obtained by driving waste raises into the hanging wall of the stope.



a. BLIND STOPE WITHOUT PREEXISTING RAISE

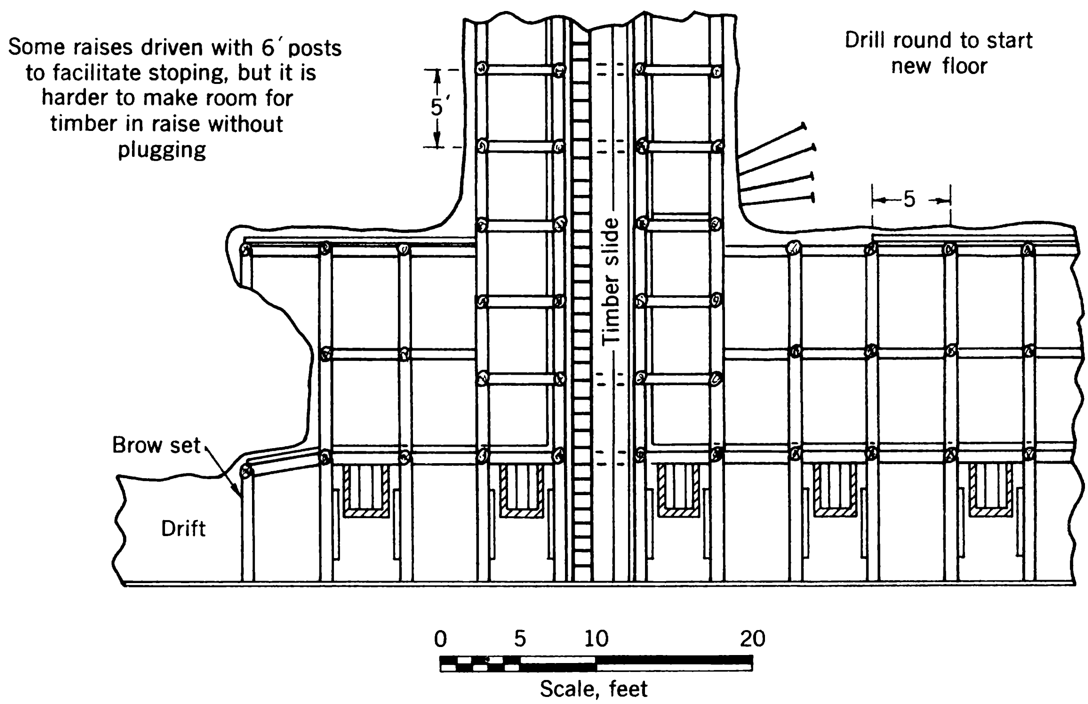


FIGURE 11. - Longitudinal Section of Vertical Square-Set Stopes Showing Methods of Starting New Floors.

Drill steel, timber, and necessary supplies are taken into the stopes by the miners; no definite system of supply distribution is used.

In the flat-dipping square-set stopes, where there is sufficient room to handle waste, some handsorting and blockholing of broken material is done. Ordinarily, all broken rock is mucked into the loading chutes and transported to the main ore bins. Large pieces of nonmineralized rock are sorted out when the cars are dumped. In stopes where the hanging wall consists of highly brecciated shale, broken material is impossible to sort. A section of ore above the Turkey Run tunnel recently was mined successfully by a shrinkage stope. The stope had a uniformly steep dip and a thin shale hanging wall (fig. 3).

UNDERGROUND TRANSPORTATION

In sections of the mine serviced by the main shaft, rock was transported in standard 3/4-ton-capacity end-dump cars hauled by battery locomotives. Cars were caged, hoisted to surface by a 75-hp. double-drum, electric-driven hoist, and hand-trammed and dumped into the ore bin above the rotary furnaces.

Ore in the southeast section of the property served by the Ventilation, Turkey Run, and Drain tunnels is transported underground from working faces and stopes in 1-ton-capacity side-dump cars hauled by battery locomotives to outside loading bins. Ore from the No. 1 winze is hoisted in 3/4-ton end-dump cars which are caged, hoisted to the tunnel level, and trammed by battery locomotives to surface. Ore is hauled in 5-ton-capacity end-dump trucks from tunnel ore bins to the furnace bin.

Raise and stope chutes are lined with 3-inch vertically placed lagging. Chute gates are the two-stop-board type which adequately control the flow of ore into the cars. Chute lips are lined with sheet steel to prevent excessive wear.

AUXILIARY OPERATIONS

Shops in the Main shaft area include the compressor plant, timber framing shed, lamp, and change houses (fig. 12). Other structures in the area include the furnace plant and several storage buildings. The timber framing shed and a portable diesel electric generating unit are located at the portal of the Drain tunnel. Dynamite and blasting caps are stored in magazines located in an unused tunnel and a small building, respectively. Figure 13 shows the surface plan at the Abbott mine.

Waste rock from underground operations is disposed of in dumps adjacent to the shafts and access tunnels; calcines from the furnace plant are stored in waste dumps near the plant and are used for road material.

Compressor Plant

Compressed air for the mine is supplied by 315-, 365-, and 500-c.f.m. compressors, driven by diesel engines. A 2,315-c.f.m. compressor is maintained as a standby unit.

FIGURE 12. - Surface Plant.

Power Plant

Electric power is purchased from the Pacific Gas and Electric Co. It is conducted to the property at 12,000 v. by a 10-mile, wooden-pole, copper power line constructed by the mining company in 1958. The main hoist, shop equipment, reduction plant, and the mine pump on the 300 level are supplied by a bank of 440-v. transformers located in the hoist house. A similar bank near the portal of the Ventilation tunnel services the nearby mine exhaust fan and the Turkey Run and Drain tunnels. Smaller transformers supply 110-v. power for lights and small tool and domestic use.

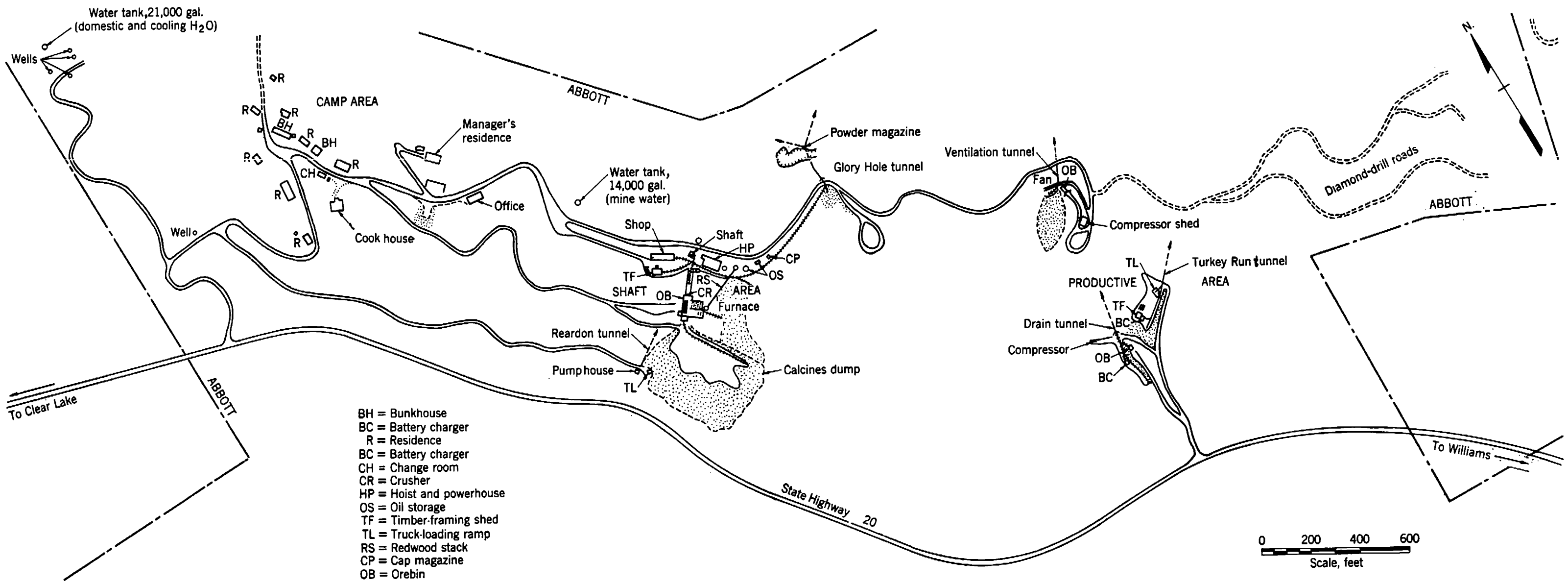


FIGURE 13. - Surface Plan of the Abbott Mine Operations.

Monthly power requirements vary from 22,000 to 30,000 kw.-hr. and cost from $350 to $440. The average cost per kilowatt-hour is about 0.015 cent and is calculated from a sliding schedule which varies from 0.0264 cent to 0.0065 cent per kilowatt-hour.

Standby power is provided by a 75-kw. diesel-generator set, in the hoist house, and a 50-kw. set at the portal of the Turkey Run tunnel. A 100-kw. generator driven by a two-cylinder semi-diesel engine of 100 hp. is still in place. and functional. Diesel fuel costs about 15 cents per gallon f.o.b. mine.

Water Supply

Water for domestic and plant use is pumped to storage tanks from three wells.

FURNACING

Ore is hauled to the coarse ore bin above the mill by mine cars from the Main shaft and Glory Hole workings and by dump truck from the Ventilation tunnel. Mine ore goes to an 8- by 15-inch jaw crusher, set with a 1-inch opening, and is fed by bumper feeder to two 50-foot 45-ton-capacity oil-fired rotary furnaces (fig. 14). Preheated fuel oil was burned under 70-p.s.i. pressure at a temperature of 1,700° F. with oil consumption of about 11 gallons per ton. A changeover has been made to a low-pressure system which utilizes a cheaper grade of oil (PS 300) at 10-p.s.i. pressure with a consumption of about 8 gallons per ton. Air for the low pressure system is supplied by a 5-hp. blower to Hauck burners.

The furnaces are lined with Nos. 2 and 3 arch brick of flintacid composition which are separated from the steel shell by sheet asbestos. Brick alinement has been improved by replacing soft, compressible 1-inch asbestos with thinner but much harder sheets. No reliable figures can be given on liner life since replacements are made in sections, as required. The section just below the end of the feeder tube gives the most trouble. Abrasion and small-scale spalling occur at the lower, firing end of the kilns, but the erosion is quite uniform with little tendency for individual bricks to fall out until the whole section is worn down to the replacement point. One furnace was completely relined by professionals at the time of its purchase in 1954, but the work was no better than that done by experienced members of the furnace and maintenance crews. Any necessary trimming of the final brick in a ring is done by cutting with a circular saw. Partial rings are held in place during installation by a homemade expansible circular jig. Where the gases have had access to the asbestos lining for any considerable period, the latter is converted into slabs of nearly solid secondary cinnabar.

Gas temperatures are taken by recording pyrometers inserted into the goosenecks between the tops of the dust boxes and the 10-hp. fans which exhaust the furnaces and force the gases through cyclone-type dust collectors into the condensing systems. The normal temperature range is from 600° to 650° F. Coarse dust from the dust boxes is shovelled into wheelbarrows; dust from the cyclones is removed hydraulically when the water supply permits. The dust is panned periodically, but the mercury content is not high enough to justify any form of treatment.

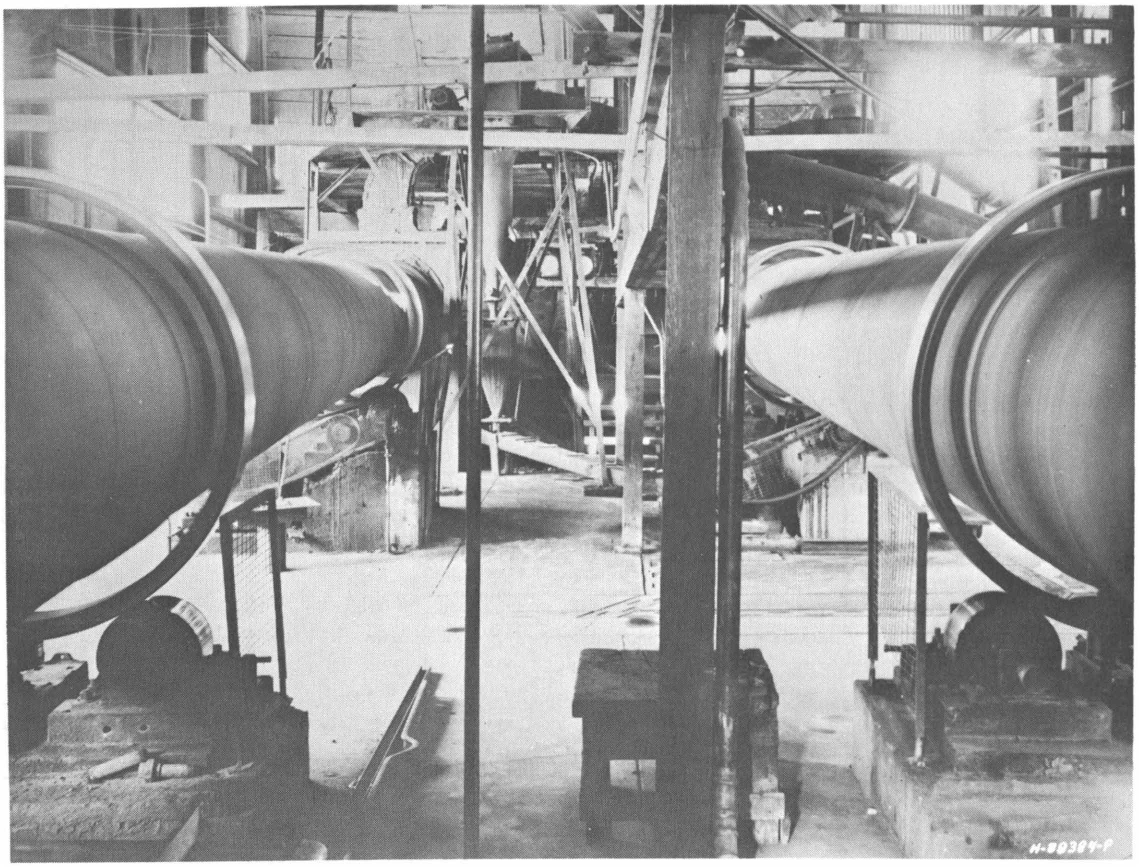


FIGURE 14. - Furnaces.

Each furnace has its own condensing system consisting of two parallel rows of 12-inch steel pipes with walls about five-eights of an inch thick, which are cast in Birmingham, Ala. The first eight pipes in each row are 16 feet high, while the last eight are 12 feet high. Gases are conducted to the lower ends of the first condenser pipes in each row by a steel hot pipe which has two diverging branches a short distance below the fan. The condenser pipes in each row are connected in series by U-shaped bends at the top and by open-bottom Y's at the bottom. The U-bends have clean-cut holes which are stoppered with redwood plugs. The open stems of the Y's discharge under water in redwood troughs which extend for the whole length of each row. Gases from the last condensers pass through Transite pipes into a circular redwood tank with a conical bottom which discharges into a water-filled trough. This tank serves both furnaces as a scavenger condenser and as a surge tank for the flue system which consists of an inclined redwood flue leading up to the concrete base of a vertical redwood stack. Some mercury condenses in the stack system. The inclined flue is hosed down periodically through regularly spaced washdown holes. This mercury collects in the launder of the circular redwood tank. The vertical stack is cleaned less frequently, but some mercury is recovered from the concrete base. Some mercurial soot builds up in the inclined flue and escapes the washdown.

This potential loss of mercury was discovered when the upper portion of the flue was destroyed by fire. Any darkening of the normally white smoke indicates a poorly adjusted fire and a stack loss of mercury. Stack temperatures are not taken routinely, but are accepted as satisfactory when the hand can be held comfortably on the last condenser pipes. Overall recovery is believed to vary from 92 to 95 percent.

Condenser life is variable. Most replacements are partial since the pipes at the hot end last much longer than those at the wet end where the SO_2 has hydrolyzed to acid, and the partially submerged Y's corrode more rapidly than the U-bends. The lives of both the pipes and the Y's have been greatly extended by the use of fiber glass patches and windings. The average life for a system might be from 18 to 20 months, and the cost of a complete system for one furnace approximates $10,000. A completely satisfactory sealing material for pipe connections has not been discovered. Ceramic-type mixtures involving fire clay, cement, lime, or ashes become so indurated that joint breaking becomes the most expensive part of a replacement job. This is particularly true when either a pipe or a connector can be reused, if it is not broken. Calking with metallic lead is now being tried.

The condensers are washed down daily. In the older system the condensate (mercury and soot) is collected in steel or rubber buckets beneath each Y. In the other system the bottom of the trough slopes toward a central sump with an inclined lip on one side and a mercury trap in the bottom. Condensate is pushed to the sump and scraped up the inclined lip by shovels into a high steel cart with a tilting box from which the mud can slide into the hoeing machine. Under satisfactory furnace operation, nearly all of the mercury is recovered from the third, fourth, and fifth spouts. Appreciable production from the first two Y's indicates too low a furnace temperature; from the last two Y's, it indicates too high a temperature.

The hoeing machine is located in the patio. The machine is covered by a hood with a short stack, but is otherwise not enclosed. The present machine was designed by the resident manager, C. O. Reed, who supervises furnace operations. It consists of a circular steel pan about 4 feet in diameter with vertical sides some 8 or 10 inches high. A mercury outlet and a trap are near the center of the flatly conical bottom. A central vertical shaft, driven by a motor beneath the pan, rotates two arms to which are attached staggered rakes that work the condensate and admixed quicklime away from and towards the center. A reversal in direction of two rakes permits the machine to discharge itself through a removable port in the side. The silver man wears rubber gloves and apron and is provided with a respirator. Smoking is prohibited while on the job; a change room and shower is attached to the furnace building. Voluntary rotation to underground and nonfurnace surface jobs is practiced, and such rotation may become compulsory.

Tailings from the hoeing machine, already dried by quicklime, are shovelled into three pans on a car (a retort load) that is trammed some 75 feet or so to one of two Square D retorts. The car is of a height that pans can be pushed into the retort without lifting. The ordinary burning time is 12 hours, but with some material this can be reduced to 8 hours. At times, the material has been put back through the furnaces, but the retort recovery is usually at least a flask a day, and it has been better practice to recover this mercury separately and not build up a circulating dust load.

Mercury from the hoeing machine and the retort is carried in steel pots to a locked, rectangular steel storage and bottling tank. The mercury is poured onto the sunken, inclined top of the bottling tank and flows through a bath of quicklime into an entrance hole at the low corner. The bottom of the tank is also inclined and has a discharge pipe and valve about 1 inch above the low corner. The capacity of the tank is about 32 flasks. At bottling time, mercury is drawn through a small rubber hose into flasks sitting on a platform scale. Empty flasks are purchased at prices varying from 75 cents to $1.50 each. Flasks are cleaned by tumbling with steel shot on a set of rubber rolls. They are then blown out, painted, and stencilled. Filled flasks are stored in a concrete, steel-doored vault in the furnace building until shipping time. Most of the mercury is hauled by company truck to San Francisco or Oakland, Calif., for shipment to the east coast by steamship or truck. Powder is purchased in this area and usually is timed so that the truck has a return load to the mine. Most sales involve the company payment of freight, and are based on the current E & MJ price minus a discount which is usually in the $5 or $6 range. About 1,000 tons of ore is put through each furnace per month. From 1890 to the present time the average monthly production of mercury has been 151 flasks. During the current operation the highest monthly production was 504 flasks in March 1958. Attempts to stabilize monthly production have been unsuccessful.

The concrete floors of the furnace building and patio are inclined and channeled so that wash water drains to a series of wooden settling tanks. Mud from these tanks, the round redwood tank, and other cleanup operations is either put through the hoeing machine or dried in the patio and retorted, according to its grade.

Calcines drop into a steel bin beneath the lower end of the furnaces and are hand-trammed to the edge of the dump. Plans are to mechanize this operation, probably with locomotives. The calcines are used as road material on the company roads and occasionally by other local users. The material powders easily and is not ideal for road material, but winter operations on the shale-based roads would be impossible without using this material. Truck loading is facilitated by a slusher and loading ramp. This removal permits occasional periods of short-haul tramming of current calcines. The calcines are apparently not suitable for lightweight aggregate or building blocks.

During periods of two-furnace operations the crew consists of three furnace men, one silver man, two slag and crusher men, and one relief man who rotates through the various jobs. One-furnace operation eliminates one slag and crusher man. During most of 1959, furnace operation was not continuous and the reduced crew consisted of men who could work either underground or on the furnace.

VENTILATION

All of the active working levels are interconnected, and there is enough difference in elevation so that natural circulation provides ample primary ventilation during the cooler months. This can be augmented when necessary by a 7-1/2-hp., 4-foot-diameter axial flow fan which is installed in the concreted double portal of the Ventilation tunnel. This fan exhausts about 16,000 c.f.m. and makes all other mine openings incast. The fan can be reversed by a simple change in the electrical connections. Opening the steel mesh service gate automatically stops the fan.

Dead-end workings, particularly raises and stopes, become hot and humid with rock temperatures in the 92° to 96° F. range. In these workings, secondary ventilation is required to remove powder smoke, dilute and remove gases, and supply the men with the legal minimum of 200 c.f.m. of fresh air per man. Air is taken from the main stream by 5- and 7-1/2 horsepower electric-driven blowers and delivered to the working face by 10-inch galvanized ventilating pipe and 10-inch Ventube. An air-driven blower is used occasionally. Long drifts require booster fans.

MINE DRAINAGE

Mine water in the shaft area is collected in a rectangular sump on the 300 level station. The water is pumped 105 feet vertically by a float-controlled 15-hp., two-stage centrifugal pump through a 2-inch line from the 300- to 200-foot levels, where it flows through the Reardon tunnel to the surface. Here it is collected in a pond and pumped by a 4- by 4-inch horizontal piston pump to a storage tank above the level of the shaft collar. The water is then redistributed to fire hydrants, to the mill for washdown purposes, and back underground for drilling use; it cannot be used for cooling purposes.

Water in winzes below the local water level is controlled by air-driven single-stage sump pumps. At the 254 winze in the Drain tunnel, a sump pump was retained for dewatering as long as the winze was in use. A shallow sump was maintained in the floor of a drift not in immediate use. Ordinarily, a winze is extended from one to three sets below the sublevel floor for a sump and a float-controlled electric-driven centrifugal pump is installed as soon as possible. Five- and ten-horsepower single-stage pumps are used.

Over long distances--up to 2,250 feet-the water table has a gradient of from 2.5 to 3.0 feet per hundred, but large quantities of water can be dammed up locally in the serpentine behind impervious shale septa. In the crosscut from the bottom of No. 4 winze from the Turkey Run tunnel, water was controlled with an intermittently operating centrifugal pump. However, a diamond-drill hole through a shale band released a dammed-up flow of up to 900 g.p.m. which flooded the crosscut and rose in the winze to within 20 feet of the original water level. When pumping ceased in the 254 winze, the water level rose a little over 0.5 feet per day.

FIRE PREVENTION

The mine is located in an area where grass and brush fires are prevalent in the summer so that this hazard must be provided for as well as the usual fire dangers around buildings and machinery. Grass is burned off around buildings as soon as it becomes dry enough in the late spring. No fires have started underground, but, at the Ventilation tunnel, a welder's spark ignited dry weeds that spread into the buried and partially buried timbering of the old portal.

Besides fire extinguishers, fire control equipment consists of two hydrants and hoses in the camp area, two more near the collar of the shaft, and at the latter area a 3-hp. fire pump. A truck originally designed for well sinking is used as a fire truck. It has a 700-gallon tank and a pump. Another gasoline-driven fire pump with a 150-gallon tank can be quickly installed in a jeep. Fire control equipment is in charge of a mechanic who is a member of the fire department in a nearby town.

SAFETY METHODS AND FIRST-AID ORGANIZATION

Safety conditions are controlled by the mine foreman and the manager and are under surveillance by the State's mining inspector and the engineer for the insurance carrier. Formal first-aid training has not been held for some time, but a safety committee meets twice a month and makes once-a-month inspection trips throughout all active working places. A staff member acts as secretary, and the manager usually attends. The mine foreman and night shift boss are permanent members, and temporary members of the committee are assigned from the underground and surface crews. Minutes of all meetings are sent in to the insurance carrier.

SCALE OF WAGES

The foreman receives a salary of $550 per month and the shift boss receives $500 per month. Other wages are as follows:

	Per 8-hour day		Per 8-hour day
Shaftmen	$21.00	Timber framers	$15.60
Miners	17.33	Truck drivers	15.60
Helpers	15.60	Diamond driller ...	17.33
Mechanics	17.33	Furnace men	15.60
Motorman	17.33	Cook	[1] 11.70
Electrician	17.33	Cook's helper	11.00

[1] And board.

TABULATION OF DATA

Information concerning cost distribution for the various phases of operation at the Abbott mine for mining and furnacing 10,956 tons of ore from January 1, 1959, through November 30, 1959, is shown in table 3 and other tabular work.

TABLE 3. - Personnel

	Men per shift	Shifts	Total per 24 hours
Underground:			
Mine foreman	1	2	2
Miners	6	2	12
Helpers	6	2	12
Motorman	1	2	2
Surface:			
Timber framers	2	1	2
Mechanics	1	1	1
Electrician	1	1	1
Truck driver	1	2	2
Miscellaneous	1	1	1
Mill:			
Furnace men	2	3	6
Miscellaneous	1	1	1

TABLE 4. - Costs in dollars per foot of raise

	Per foot
Labor	$ 33.12
Supervision, including office	4.23
Explosives	3.50
Air and water pipe	.85
Timber	5.04
Ventilation	1.00
Compressed air, fuel for diesels	1.85
Drill steel and bits	.30
Depreciation of equipment	3.85
Compensation insurance and payroll taxes	7.47
Total	61.21

TABLE 5. - Costs in dollars per foot of winze

	Per foot
Labor	$ 52.82
Supervision, including office	5.22
Explosives	5.25
Timber	5.10
Ventilation	.52
Compressed air, fuel for diesels	2.78
Drill steel and bits	.40
Depreciation on equipment	4.96
Miscellaneous supplies	.52
Compensation insurance and payroll taxes	11.61
Total	89.18

TABLE 6. - Costs in dollars per foot of drifting and crosscutting

	Per foot
Labor	$ 19.64
Supervision, including office	2.00
Rails, etc.	1.09
Explosives	4.50
Air and water pipe	.85
Timber	1.02
Ventilation	1.00
Compressed air, fuel for diesels	1.91
Drill steel and bits	.30
Depreciation on equipment	2.98
Compensation insurance and payroll taxes	4.33
Total	39.62

TABLE 7. - Performance records and general data

Mine: Abbott
From: January 1, 1959, to December 1, 1959
Total days: 334

Days worked	286
Hours per day	16
Progress made, total feet	1,597
Progress made, average feet per day	5.58
Total number of drill rounds	320
Average number of rounds per day	1.12
Average feet advance per round	5.00
Total number of man-shifts, drilling	2,903
Average number of holes drilled per round	18
Total drill steel used, pounds	992
Average pounds of steel used per round, pounds	3.1
Total number of bits used	1,914
Average number of bits used per round	6
Average footage advanced per man-shift	2.5
Total number of man-shifts, blasting	429
Total number of man-shifts, loading	1,144
Excavation, cubic feet	101,511
Cubic feet per ton of ore or waste	13
Total number of man-shifts, hoisting	218
Total number of man-shifts, ground control	2,088
Total number of man-shifts, rock disposal	572
Total number of man-shifts, drainage	36
Total number of man-shifts, ventilation	36
Total number of man-shifts, underground	8,008
Total number of man-shifts, surface	2,002

TABLE 8. - Unit cost of supplies

Dynamite	per 100 pounds	$ 22.00
Electric caps	each	.30
Throwaway bits	do.	.22
Timber (round)	per linear foot	.25
Timber (lagging 2x12 and 3x12)	per thousand	60.00
Drill steel, quarter-octagon	per pound	.30
Rails, 5 spikes and belts	per ton	215.00
2-inch pipe	per foot	.54
1-inch pipe	do.	.31
Ventilation pipe	do.	1.10
Diesel fuel	per gallon	.14
Lubricants	do.	.60

TABLE 9. - Summary cost data, January 1959 through November 1959

	Per ton
Exploration and development:	
Drifts and crosscuts: Labor, timber, dynamite, caps, pipes, rails	$ 1.99
Raises: Labor, timber, dynamite, caps, pipes	1.01
Winzes: Labor, timber, dynamite, caps, pipes	.49
Total	3.49
Mining:	
Stoping: Labor, timber, dynamite, caps	6.86
Underground maintenance and repairs: Labor, timber, dynamite, caps	.89
Other underground expense: Foreman, steel sharpening, timber framing, maintenance, equipment repair, miscellaneous supplies	3.21
Hoist and power plant: Labor, compressed air, power, fuel	3.33
Total	14.29
Indirect operating costs:	
Shops: Labor, maintenance, repairs	1.14
Automotive equipment: Labor, maintenance, repairs	.96
Surface yards: Labor, building maintenance and repairs, domestic water, bunkhouse	.38
Cook house: Labor and supplies	.75
Mine office: Labor, telephone, telegraph, supplies	.58
General operating expense: Salaries (engineering and management)	1.87
Depreciation	1.04
Depletion	.23
Amortization	.36
Total	7.31
Furnacing:	
Furnacing: Labor, fuel, lubricants, maintenance, repairs, equipment, miscellaneous supplies	5.33
Assaying and sampling	.006
Flasks	.177
Total	5.51
Marketing cost	.57
Total cost per ton of ore	31.17

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