CONTACT INFORMATION
Mining Records Curator
Arizona Geological Survey
416 W. Congress St., Suite 100
Tucson, Arizona 85701
520-770-3500
http://www.azgs.az.gov
inquiries@azgs.az.gov

IC's – GOLD.

IC 6786. Placer Mining in Western US – Pt 1.
6787 " " " " " – Pt 2
6788 " " " " " – Pt 3.
6843 Prospecting for Lode Gold.
6891 Operation of a 4-tph Au-Ag ore-sampling plant.
6901 Gold Mining in Black Mtns, W Mohave Co., AZ.

I. C. 6786

SEPTEMBER 1934

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

PLACER MINING IN THE WESTERN UNITED STATES

PART I. GENERAL INFORMATION, HAND-SHOVELING, AND GROUND-SLUICING

BY

E. D. GARDNER AND C. H. JOHNSON

I. C. 6786,
September 1934

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

PLACER MINING IN THE WESTERN UNITED STATES[1]

Part I. - General Information, Hand-Shoveling, and Ground- Sluicing

By E. D. Gardner[2] and C. H. Johnson[3]

CONTENTS

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6786."

2 Supervising engineer, U.S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

3 Assistant mining engineer, U.S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

INTRODUCTION

Placer mining is the mining and treatment of alluvial deposits for the recovery of their valuable minerals. The method has been used principally for mining gold, but a large proportion of the world's production of tin, platinum, and diamonds and other gem stones and minor quantities of other heavy minerals have been won in this manner. In the United States, as in the world at large, gold has been the principal mineral obtained by placer mining. Minor quantities of metals of the platinum group are recovered with the gold in some localities. Important quantities of sapphires have been produced at placer mines in Montana, and tungsten minerals have been obtained on a commercial scale from placer deposits in California and Colorado. Other heavy minerals or gem stones, however, have not been mined to any important extent by this method in the United States.

The search for placer gold and the working of the deposits when found have had much to do with the early development of the West. Placer mining has been gradually overtaken and surpassed in importance by lode-gold mining, until in 1932 less than a quarter of the country's total gold production was from placers or about an eighth, excluding Alaska. In 1932 about 76 percent of the placer gold produced in the United States was recovered by dredging. Although other forms of placer mining still are important, they have been declining for many years, as the richest and most readily mined deposits of gravel along the stream courses have been exhausted. During 1931 and 1932 there was a revival of small-scale mining, but few new deposits were discovered.

California has ranked first in the production of placer gold since the discovery of gold on Sutter Creek in 1848. In 1932 the relative importance of the other Western States in gold production by placer mining was as follows: Oregon, Idaho, Nevada, Montana, Arizona, Colorado, New Mexico, South Dakota, Washington, Utah, and Wyoming.

This paper deals with the history of placer mining and production of placer gold, geology of placer deposits, location of placer claims on public lands, sampling and estimation of gold placers, and the classification of placer-mining methods, together with discussions of hand mining and ground sluicing.

Two subsequent papers[4] deal with other phases of placer mining. All phases of placer mining are discussed in the three papers and current practices are illustrated in descriptions of individual mines.

ACKNOWLEDGMENTS

The authors have drawn freely upon the available literature on placer mining, geology, engineering, and other allied subjects; they have endeavored to make suitable reference throughout the text.

G. A. Bigelow of San Francisco furnished data on methods and costs of sampling placer ground. Victor C. Heikes, Clarence N. Gerry, and Chas. W. Henderson of the Economics Branch of the Bureau of Mines kindly provided the authors with lists of the producers of placer gold; this information facilitated field investigations.

The authors also wish to acknowledge their indebtedness to the operators of placers in the Western States who generously supplied information without which this paper could not have been written.

4 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part II. - Hydraulicking, Treatment of Placer Concentrates and Marketing of Gold, and Part III. - Dredging and Other Forms of Mechanical Handling of Gravel, and Drift Mining: Inf. Circs. 6787 and 6788, Bureau of Mines, 1934.

HISTORY OF PLACER MINING AND PRODUCTION OF PLACER GOLD IN THE WESTERN UNITED STATES[5]

The earliest large placer-gold production in what is now the western United States was from the Old and New Placer diggings, near Golden, Santa Fe County, N. Mex., which were worked as early as 1928 and have yielded probably 3 or 4 million dollars in gold. Several smaller deposits were likewise known and worked, but the first discovery of major importance was made by James W. Marshall on January 24, 1848.[6] Marshall was building a sawmill on the south branch of the American River at Coloma, 40 miles west of the present site of Sacramento, and found gold nuggets in the tailrace of the mill. The news spread too slowly to benefit Mexico, which on February 2, 1848, upon payment of $15,000,000, ceded not only California, but nearly all the present Southwest to the United States as war indemnity. The rapid settlement of the West beganwith the great gold rush to theCalifornia fields.

The excitement over gold spread through central California in 1848. In that and the next year the placers of the Trinity and Klamath Rivers, in northern California, were discovered. The total California production of these 2 years was $40,000,000.

Oregon settlers, moving to the southern gold fields, found rich placers in the Grants Pass district in 1852. A brief pause ensued, but when gold was found on the Fraser River and the Caribou, in British Columbia, in 1858 and 1860, the consequent stampedes rapidly opened northern and central Washington.

In 1860, E. D. Pierce discovered gold in the Clearwater country, now part of Idaho. Within a year Pierce City, Oro Fino, Elk City, Florence, and Warren, in Idaho, were founded; gold was also discovered during this period on the John Day and Powder Rivers in eastern Oregon. The Boise Basin, Idaho, was entered and rich placers found, in 1862; others were discovered at Silver City, Idaho, in 1863. The Gold Creek, Bannack, and Alder Creek mines were discovered in Montana in 1862.

The Territory of Idaho was created early in 1863, with Boise as its capitol, and Helena, another placer camp, became the capitol of the Territory of Montana, organized in 1864.

The Gila City placers in Arizona were found in 1858 and the greater ones of the La Paz district in 1862. A year later the Weaver and Lynx Creek deposits were booming.[7]

In Colorado a placer-mining expedition found some gold on Cherry Creek, Ralston Creek, and the Platte River in 1858 but nothing of importance until the next year. The change from the first skepticism of 1848 to the Nation-wide gold fever of the following period is shown interestingly by the fact that the mere presence of prospectors in Colorado gave rise to the wild rumors of rich diggings "near Pikes Peak", and brought a large population to the winter camp on the Platte, where Denver was founded, before a single valuable deposit was discovered.[8] In 1859, rich placers were located on Clear Creek, in the South Park, and on the upper Blue and Arkansas Rivers. Colorado was made a territory in 1861, with the seat of Government at Denver.

Nevada's first placer mining was done by settlers in Carson Valley whose chief livelihood was in supplying the California wagon trains. From 1849 on sporadic placering was done in Gold Canyon on the side of Mount Davidson. Gold was found in a near-by canyon, Six-Mile Creek, in 1857 and 2 years later placer workings there uncovered the outcrop of the world-famous Comstock lode. Further placer discoveries took place in the Sierra district soon after 1863, in the Tuscarora district in 1867, at Copper Mountain (Charleston district) and Osceola in 1876 and 1877, and at Spring Valley in 1881.[9] Other discoveries of some import-

---

5 Chiefly from Bancroft's works, except as otherwise acknowledged.

6 Rickard, T. A., The Discovery of Gold in California: Univ of California Chronicle vol. 30, no. 2, April 1928, pp. 141-169.

7 Wilson E. D., Arizona Gold Placers and Placering: Univ. of Arizona, Arizona Bureau of Mines, Bull. 135, Aug. 15, 1933, pp. 13, 14.

8 Henderson, C. W., Mining in Colorado: U. S. Geol. Survey Prof. Paper 138, 1926, pp. 1-8.

9 See Smith, A. M., and Vanderburg Wm. O. Placer Mining in Nevada: Univ. of Nevada Bull., vol. 26. no. 8, Dec. 16, 1932.

ance have occurred even more recently in this State, as for instance, at Round Mountain in 1906; the production has been about $1,300,000 since its discovery. Lack of water may be partly responsible for the fact that many of Nevada's placers were not prospected until long after the heyday of placering in Idaho, Oregon, and California. The most productive placers in the State, those at Spring Valley, were not worked until 1881, although the lodes there were mined as early as 1868.

In Utah placer gold has been of little importance. The largest production from any district, about $1,500,000, came from Bingham Canyon, where lode discoveries in 1863 were followed a year later by placer locations.[10]

Placer gold was discovered in South Dakota in July 1874[11] by Horatio Ross, a prospector who had accompanied General Custer's reconnaissance expedition into the Black Hills. In 1875, as a result of the inevitable stampede, the rich placers of Deadwood Gulch were found, leading shortly to the location of the Homestake lode mine.

The only New Mexico placers credited with a large production, other than those of the Golden district already mentioned, were discovered in the Elizabethtown district in 1867. These were rich enough to encourage extensive hydraulic installations, including a ditch 41 miles long.[12]

The principal yield of gold from most districts has been in hand methods during the first few seasons following discovery. Hydraulicking has prolonged the period of activity in many districts, but in only a few instances has resulted in greater yields than handworking in the first few years. Regular production nearly comparable to lode mining has been established by dredging in a few fields. This sequence of phases of placer mining is shown by the history of the Boise Basin district in Idaho. Gold was discovered there in 1862; and, according to one estimate, $3,000,000 worth of gold was taken out in 1863, $4,000,000 in 1864, and $5,000,000 in both 1865 and 1866.[13] Hydraulicking was introduced in 1867; the yield, however, dwindled nearly 20 percent annually for the next 10 years and more slowly thereafter, until it was only $200,000 in 1898. Dredging which began late in 1898 doubled the annual production the first year. Again after 1910, when the placer yield was only about $130,000, a new dredging project boosted production to a peak of nearly $500,000, although hydraulicking continued to dwindle. There was virtually no dredging from 1916 through 1925; in 1920 only $1,600 worth of placer gold was mined. A dredge started up again in 1926, and for 5 years, until the boat burned in 1930, the annual yield was $40,000 to $60,000.

The Manhattan district, in Lyon County, Nev., is an example on a small scale of the trend of production in most placer camps, except that here growth was relatively slow due to most of the gold being produced from drift mines. The first recorded placer production was in 1908, when $16,000 was produced. The peak was reached in 1912 and 1913 when each year was credited with a yield of about $165,000. Since then production has fallen off an average of about 20 percent each year, until in 1931 it was only $1,400.

---

10 Butler, B. S., The Ore Deposits of Utah: U.S. Geol. Survey Prof. Paper 111, 1920, p. 340.

11 Lincoln, F. C., Half A Century of Mining in the Black Hills: Eng. and Min. Jour., vol. 122, Aug. 7, 1926, pp. 205-206.

Also O'Harra, C. C., Early Placer Gold Mining in the Black Hills: The Black Hills Engineer, South Dakota School of Mines Quarterly, vol. 19, 1931, pp. 343-361.

12 Lindgren, Waldemar, and others, Ore Deposits of New Mexico: U S. Geol. Survey Prof. Paper 68, 1910, pp. 92-105.

13 U.S. Geol. Survey Mineral Resources of the United States, 1914, part I, p. 616.

TABLE 1.– Production of gold in the United States, by dredges, and number of dredges producing, by States, 1896-1930
(Rearrangement of tabulation in Mineral Resources, 1914, pt. I, p. 855; 1915-30 data from Mineral Resources for respective years)

Year	Alaska		California		Colorado		Idaho		Montana		Oregon		Other States		Total	
	Value	No.	Value	No.	Value	No.	Value	No.	Value	No.	Value	No.	Value	No.	Value	No.
1896	.....	.....	$2,000	1	.....	.....	.....	.....	$42,000	1	.....	.....	.....	.....	$44,000	2
1897	.....	.....	5,000	1	.....	.....	$11,436	1	102,120	4	.....	.....	.....	.....	118,556	6
1898	.....	.....	18,887	3	.....	.....	13,920	1	154,893	4	.....	.....	.....	.....	187,700	8
1899	.....	.....	206,302	8	.....	.....	62,436	5	165,440	5	.....	.....	.....	.....	434,178	18
1900	.....	.....	200,929	16	.....	.....	129,443	6	189,665	5	.....	.....	.....	.....	520,037	27
1901	.....	.....	471,762	22	$6,000	1	116,117	6	146,134	5	.....	.....	.....	.....	740,013	34
1902	.....	.....	867,665	29	10,000	1	101,257	10	318,914	7	.....	.....	[1]$71,686	1	1,369,522	48
1903	$20,000	2	1,475,749	31	15,000	1	86,113	6	229,332	4	.....	.....	[2]89,870	1	1,916,064	45
1904	25,000	3	2,187,038	42	65,594	3	99,110	7	245,700	2	.....	.....	[3]101,275	4	2,723,717	61
1905	40,000	3	3,276,141	50	33,342	3	34,336	3	275,542	5	.....	.....	[4]28,015	4	3,687,376	68
1906	120,000	3	5,098,359	59	48,343	3	38,340	3	397,030	4	.....	.....	[5]19,322	4	5,721,394	76
1907	250,000	4	5,065,437	57	35,235	3	74,438	6	197,141	4	$23,191	2	[6]10,260	3	5,655,702	79
1908	171,000	4	6,536,189	69	141,773	4	77,189	5	402,667	4	.....	.....	[7]24,852	2	7,353,670	88
1909	425,000	14	7,382,950	63	404,636	4	101,7[illegible]4	8	426,649	5	42,667	2	.....	.....	8,783,606	96
1910	800,000	18	7,550,254	72	344,211	6	91,247	6	473,318	7	34,010	6	.....	.....	9,293,040	115
1911	1,500,000	27	7,666,461	65	272,173	4	258,791	7	597,778	8	14,575	3	[8]16,591	5	10,326,369	119
1912	2,200,000	38	7,429,955	65	384,748	3	481,077	8	710,387	6	.....	.....	[9]12,744	4	11,218,911	124
1913	2,200,000	36	8,090,294	63	372,288	4	561,876	6	685,210	5	.....	.....	[10]317,268	2	12,226,936	116
1914	2,350,000	42	7,783,394	60	602,655	5	568,989	4	835,615	5	.....	.....	[11]372,130	4	12,512,783	120
1915	2,330,00[illegible]	35	7,796,465	58	672,386	5	486,541	7	861,626	5	.....	.....	[12]336,107	4	12,483,125	114
1916	2,679,000	34	7,769,227	60	695,265	6	327,696	4	642,572	5	670,415	3	2,539	1	12,786,714	113
1917	2,500,000	36	8,313,527	55	647,270	6	59,446	4	409,455	3	618,922	4	1,805	1	12,550,425	109
1918	1,425,000	28	7,431,927	48	522,921	6	239,762	5	334,750	3	387,740	3	.....	.....	10,342,100	93
1919	1,360,000	28	7,716,919	46	542,103	6	164,854	5	265,590	3	296,750	3	.....	.....	10,346,216	91
1920	1,129,932	22	6,900,366	40	512,876	5	101,679	3	255,550	3	358,884	4	[13]27,169	1	9,286,456	78
1921	1,582,520	24	7,756,787	35	337,950	3	151,762	3	190,416	1	381,960	4	[14]134,173	2	10,535,568	72
1922	1,767,753	23	4,999,215	35	346,327	4	158,827	3	36,941	1	269,994	4	[15]110,211	1	7,689,268	71
1923	1,848,596	25	6,065,735	29	358,864	4	469,900	4	.....	.....	224,117	3	[16]31,835	1	8,999,047	66
1924	1,563,361	27	4,305,521	27	412,080	4	340,462	2	.....	.....	291,557	3	.....	.....	6,912,981	63
1925	1,572,312	27	4,750,842	25	141,103	4	229,489	2	.....	.....	137,282	2	.....	.....	6,831,028	60
1926	2,291,000	32	4,950,545	23	38,860	2	141,160	3	.....	.....	74,191	2	[17]27,029	1	7,522,785	63
1927	1,740,000	28	5,461,929	25	86,902	1	114,116	3	.....	.....	112,643	2	.....	.....	7,515,590	59
1928	2,185,000	26	4,430,913	24	51,019	1	133,418	3	.....	.....	90,103	2	[18]1,878	1	6,892,331	57
1929	2,932,000	30	3,589,259	25	38,497	1	60,143	3	.....	.....	205,464	3	[19]5,938	1	6,831,301	63
1930	3,912,600	27	3,451,801	24	130,824	1	68,527	3	.....	.....	174,470	5	.....	.....	7,738,222	60
1931	3,749,000	28	3,619,355	22	8,793	1	80,352	3	.....	.....	138,155	3	.....	.....	7,595,655	57
1932	4,293,000	25	3,903,481	22	23,194	1	171,130	5	.....	.....	160,848	4	.....	.....	8,551,653	57
1896 [illegible] 1932	50,962,074	.....	174,528,580	.....	8,303,232	.....	6,407,083	.....	9,592,435	.....	[20]5,815,684	.....	[21]634,951	.....	256,244,[illegible]	.....

See page 9 for footnotes.

1 New Mexico. 2 New Mexico. 3 Oregon 3, New Mexico 1. 4 Georgia, North Carolina, Oregon, and New Mexico. 5 Georgia, Oregon. 6 Georgia 2, New Mexico 1. 7 Georgia, Oregon. 8 Nevada 2, South Dakota, Georgia, North Carolina. 9 South Dakota, North Carolina, Georgia, Wyoming. 10 Oregon, Nevada, Alabama, North Carolina. 11 Oregon, Nevada, Alabama, North Carolina. 12 Approximately $335,000 from 2 Oregon dredges. 13 Nevada. 14 $133,020 from 1 dredge in Nevada. 15 Nevada. 16 Nevada. 17 Washington. 18 Washington. 19 Washington. 20 Includes $1,107,746 not segregated from Other States, 1904-15. 21 Excludes $1,107,746 produced by Oregon, 1904-6, 1908, and 1913-15, but credited to Other States.

Coarse gold commonly is found in strata that contain a large percentage of material coarser than sand, as the high specific gravity of gold gives it a resistance to transportation nearer to that of large pebbles than to that of sand. Similarly, fine gold or flour gold is found associated with strata of sand and small pebbles rather than in the layers of silt or clay that mark a low stage of the river.

In prospecting a stream it is reasonable to test the gravels on the convex banks or short sides of the bends and near the heads of the bars. It should be remembered, however, that most of the transporting work of the stream is done during flood times, and the location and shape of the pay streaks in the gravel therefore are fixed during a high stage of the river.

In working placer ground, especially old channels, it is often advantageous to know the direction of flow of the stream that laid down the deposit. This may be indicated by the position of flat pebbles and boulders. These usually are "shingled", that is, lie with a distinct tilt downstream; otherwise the current would have tended to lift or turn them over rather than to press them down onto the stream bed.

Gold placers have been classified by Brooks[17] and more elaborately by Mertie.[18] Gilbert[19] presents a detailed discussion of the eroding and transporting action of natural streams. Although his paper is not concerned with the formation of placers it is helpful in understanding the structure of all stream deposits. The most important type of placer deposit from the standpoint of total production is the simple stream placer in which the gravels of present streams, whether permanent or intermittent, contain gold. The gold lies chiefly along narrow pay streaks within the wider channel or at favorable points in bars which may be exposed at low water. Bars form at points of relatively low velocity, hence are found not only adjacent to the convex bank of the stream at curves but also as transverse bars or crossings where the current swings from one bank to the other between curves in opposite directions and in the straight stretch below any curve. Gold will also be dropped where flood conditions permit deposition of part of the stream's load of sand or gravel.

Streams that are depositing sediment rather than eroding their beds tend to meander so that their banks usually consist of earlier-deposited material similar to their present bars. If erosion of a stream bed is renewed because of a later uplift, or a general lowering of the drainage system, some of the older deposits once forming the banks may be left high above the reach of the stream, which now may be sunk into a rock-walled channel. Even after the stream ceases to erode and again meanders through a gentle, flat-bottomed valley filled with new gravel deposits, remnants of the old bars or "benches" may be found high above water level.

17 Collier, A. J., Hess, F. L., Smith, P. S., and Brooks, A. H , The Gold Placers of Parts of Seward Peninsula, Alaska: U.S. Geol. Survey Bull. 328, 1908, p. 115.

18 Mertie, J. B. The Occurrence Mf Metalliferous Deposits in the Yukon and Kuskokwim Regions: U.S. Geol. Survey Bull. 739, 1923, pp. 160-162.

19 Gilbert, G K., The Transportation of Debris by Running Water: U.S. Geol. Survey Prof. Paper 86, 1914 pp. 219-233.

TABLE 2.- Placer gold production of United States, by States, before 1901 and 1901-32, by years[1]

Year	Alabama	Alaska	Arizona	California	Colorado	Georgia	Idaho	Maryland	Montana	Nevada	New Mexico
Through 1900	[2]$300,000	[3]$14,315,000	[4]$8,200,000	[5]$1,032,827,480	[7]$21,294,219	[8]$12,000,000	[9]$90,000,000	....	[10]$165,000,000	[11]$25,000,000	[12]$14,000,000
1901	1,385	4,980,000	105,034	3,951,049	87,324	18,047	753,716	0	522,700	33,509	59,721
1902	517	5,887,000	10,274	4,247,602	118,774	21,395	365,767	$765	447,046	15,649	130,481
1903	310	6,010,000	11,742	4,052,761	129,049	25,426	378,853	455	481,447	36,424	114,605
1904	(6)	6,025,000	16,848	4,985,290	193,068	[6]52,000	493,002	(6)	478,565	30,192	149,424
1905	1,034	12,340,000	42,667	5,892,076	99,984	29,995	340,465	0	396,901	8,274	99,335
1906	0	18,607,000	40,502	7,375,925	106,019	17,354	353,481	0	521,815	52,838	26,807
1907	42	16,491,000	44,891	6,840,695	97,219	23,413	356,905	0	348,667	55,275	19,340
1908	945	15,888,000	30,937	8,231,187	184,457	11,201	285,643	0	549,995	79,751	23,198
1909	69	16,252,638	28,648	9,104,433	457,085	16,433	281,727	0	543,372	82,965	22,010
1910	357	11,984,806	25,990	8,888,795	389,828	18,211	242,546	0	575,917	162,371	26,094
1911	0	12,540,000	23,641	8,986,527	319,038	23,738	404,327	0	684,801	210,461	18,714
1912	0	11,990,000	43,046	8,645,663	423,865	6,846	632,029	0	806,419	231,653	16,926
1913	0	10,680,000	30,691	8,836,177	408,007	8,570	694,053	0	801,002	305,442	7,861
1914	500	10,730,000	30,140	9,030,849	642,360	11,043	700,454	0	942,217	377,262	29,152
1915	59	10,480,000	35,248	8,608,617	693,310	15,256	584,890	0	949,248	395,319	9,242
1916	777	11,140,000	14,281	8,575,657	712,924	7,626	449,093	0	723,159	354,313	11,116
1917	0	9,810,000	17,214	9,074,030	661,028	2,811	135,231	0	467,063	292,584	12,179
1918	55	5,900,000	4,234	7,838,779	526,202	4,905	276,410	0	396,232	218,380	3,118
1919	0	4,970,000	4,694	8,033,076	550,562	715	190,752	0	291,430	132,288	4,959
1920	0	3,873,000	4,567	7,060,613	514,588	0	113,814	0	288,946	152,639	2,188
1921	0	4,226,000	12,524	8,154,824	344,640	711	181,600	0	227,161	363,142	8,281
1922	0	4,395,000	11,981	5,499,855	356,403	1,723	183,972	0	71,786	239,842	3,932
1923	114	3,608,500	8,854	6,522,583	364,429	513	498,709	0	40,779	81,485	4,218
1924	0	3,564,000	3,139	4,583,372	418,506	79	358,121	0	27,361	27,369	3,639
1925	0	3,223,000	4,267	5,096,144	150,318	68	262,386	0	39,385	52,435	2,018
1926	0	3,769,000	7,007	5,228,403	46,954	1,038	172,826	0	22,828	59,249	2,687
1927	0	2,982,000	6,257	5,837,313	94,434	1,043	155,459	0	22,325	37,400	5,808
1928	0	3,347,000	6,400	4,850,629	61,406	256	169,336	0	17,884	38,266	1,347
1929	203	4,117,000	5,652	3,870,607	45,850	1,928	85,373	0	12,334	43,762	1,650
1930	450	4,837,000	13,057	3,755,143	138,243	243	82,428	0	14,899	38,438	1,316
1931	407	4,842,000	22,103	4,020,746	21,586	781	107,773	0	39,439	59,602	8,405
1932	0	5,522,000	71,933	4,765,475	51,655	3,720	257,151	0	73,125	111,798	26,259
1901-32 (inc.)	7,224	255,010,944	738,463	210,499,895	9,409,115	327,138	10,548,292	1,220	11,826,248	4,380,377	856,030
Through 1932	307,224	269,325,944	8,938,463	1,243,327,375	30,703,334	12,327,138	100,548,292	1,220	176,826,248	29,380,377	14,856,030

See page 12 for footnotes.

TABLE 2.- Placer gold production of United States, by States, before 1901 and 1901-32, by years - Continued

Year	North Carolina	Oregon	South Carolina	South Dakota	Tennessee	Utah	Virginia	Washington	Wyoming	Total	Percent of total U.S. gold production
Through 1900....	[13]$5,000,000	[14]$25,000,000	[15]$1,000,000	[16]$7,000,000	....	[17]$1,000,000	[18]$1,000,000	[19]$5,000,000	[20]$500,000	$1,428,436,699	[21]59.9
1901..................	18,522	1,422,016	7,917	0	0	0	2,646	102,388	41,344	12,107,318	15.4
1902..................	16,599	243,886	4,672	0	$145	0	558	62,016	45,230	11,618,376	14.5
1903..................	9,054	471,020	2,625	0	62	0	0	4,906	8,289	11,737,028	15.9
1904..................	(6)	349,214	(6)	3,614	(6)	1,354	(6)	9,823	2,231	12,789,625	15.9
1905..................	10,005	251,619	0	9,163	207	6,656	806	6,439	2,116	19,537,742	22.2
1906..................	11,906	361,560	270	6,250	1,076	8,613	0	19,209	1,385	27,512,010	29.2
1907..................	9,834	331,406	925	924	0	9,061	117	21,860	4,045	24,655,619	27.3
1908..................	17,555	272,593	810	9,942	612	9,110	661	19,478	820	25,616,895	27.2
1909..................	10,848	221,318	1,445	1,179	625	2,525	876	5,988	1,114	27,035,298	27.3
1910..................	10,281	170,925	2,076	2,972	500	3,980	90	3,859	654	22,510,252	23.8
1911..................	5,111	168,274	261	12,073	0	5,634	808	3,999	7,041	23,414,448	24.2
1912..................	8,752	189,096	419	13,725	0	5,680	0	4,728	766	23,019,613	24.9
1913..................	6,378	450,628	218	1,393	0	1,920	0	4,144	1,407	22,237,891	25.0
1914..................	6,707	548,317	449	1,405	0	1,231	0	5,756	1,841	23,109,683	25.3
1915..................	8,486	482,170	248	1,586	0	958	0	7,160	704	22,272,501	22.7
1916..................	7,893	872,517	320	2,111	0	1,250	0	8,277	349	22,881,663	25.1
1917..................	3,979	727,366	164	924	0	112	0	5,868	34	21,210,587	26.3
1918..................	1,631	498,249	0	431	0	1,368	0	3,430	0	15,673,424	23.6
1919..................	0	380,651	0	396	0	0	0	1,247	0	14,560,770	25.6
1920..................	850	451,117	332	577	0	453	0	1,472	0	12,465,156	25.3
1921..................	830	478,733	50	1,849	0	414	0	3,073	0	14,003,832	28.9
1922..................	535	346,137	32	1,819	0	2,130	0	3,358	1,075	11,119,580	23.5
1923..................	313	276,770	80	0	0	527	0	1,511	0	11,409,385	23.0
1924..................	115	325,582	0	0	0	232	0	698	0	9,317,213	18.4
1925..................	178	186,819	0	0	0	0	0	1,093	0	9,018,111	18.9
1926..................	43	122,758	313	133	0	334	220	28,000	0	9,461,843	20.5
1927..................	1,015	183,697	0	299	0	0	0	389	0	9,327,439	21.4
1928..................	61	120,525	197	230	307	951	0	1,878	0	8,616,673	19.4
1929..................	1,085	246,969	0	0	0	956	0	6,114	0	8,439,483	19.8
1930..................	994	214,419	0	980	0	0	0	3,946	134	9,101,690	20.6
1931..................	1,776	229,851	470	1,988	0	784	0	3,164	623	9,361,498	20.4
1932..................	449	334,923	521	22,639	0	3,143	0	7,999	1,637	11,254,427	23.4
1901-32 (inc.)	171,785	11,931,125	24,814	98,602	3,534	69,376	6,782	363,270	122,839	516,397,073	22.9
Through 1932...	5,171,785	36,931,125	1,024,814	7,098,602	3,534	1,069,376	1,006,782	5,363,270	622,839	1,944,833,772	41.6

See page 12 for footnotes.

1776

1 Data from Mineral Resources of the United States, except as otherwise noted. Mine production is used where available rather than mint returns.

2 Two thirds of total gold production of Alabama through 1900. See Dunlop, J. P., Gold, Silver, Copper, Lead, and Zinc in the Eastern States in 1914: U.S. Geol. Survey, Min. Res. of the U.S., 1914, part I, pp. 139-163.

3 Placer-Gold Production of Alaska, 1880-1923, inclusive, by years, is given by Brooks, A. H., Alaska's Mineral Resources and Production, 1923: U.S. Geol. Survey Bull. 773, 1925, p. 9.

4 See table by Tenney, J. B., and Wilson, E. D., in Arizona Gold Placers and Placering: Bull. Univ. of Arizona, Arizona Bureau of Mines, vol. 4, no. 6, Aug. 15, 1933, p. 15.

5 California placer-gold production from 1848 to 1900, by decades, and 1901 to 1926, by years, is given by Hill, J. M., Historical Summary of Gold, Silver, Copper, Lead, and Zinc Produced in California, 1848 to 1926: Econ. Paper 3, Bureau of Mines, 1929, pp. 10-11.

6 Production of all Eastern States included under Georgia in 1904.

7 Colorado placer-gold production, 1858-67, 1867-1923, by years, is given by Henderson, C. W., Mining in Colorado: U.S. Geol. Survey Prof. Paper 138, 1926, p. 69.

8 About two thirds of total production of gold in Georgia, 1830-1900. See Dunlop, J. P., Gold, Silver, Copper, Lead and Zinc in the Eastern States in 1914: U.S. Geol. Survey Min. Res. of the U.S., 1914, pt. I, pp. 139-163.

9 Total estimated gold production of Idaho, 1860-70, plus one half total gold production, 1871-1900. See Ross, C. P., A Graphic History of Metal Mining in Idaho: U.S. Geol. Survey Bull. 821, 1931, pp. 1-10, for data on total gold production.

10 Montana's placer production, 1862-80, was estimated at $150,000,000. See Report of Director of Mint on Production of Precious Metals in the United States, 1884, p. 286. To this has been added about one fifth of Montana's total gold production, 1881-1900. See Report of Director of Mint, 1900. For Montana placer-gold production by years, 1896-1914, see Heikes, V. C., Gold, Silver, Copper, Lead, and Zinc - Montana: U.S. Geol. Survey Min. Res. of the United States, 1914, part I, p. 772.

11 See Smith, A. M., and Vanderburg, W. O., Placer Mining in Nevada: Univ. of Nevada Bull., vol. 26, no. 8, Dec. 15, 1932, p. 10.

12 The production of placer gold in New Mexico, 1828 "todate" was estimated at $13,000,000 to $15,000,000 by Lindgren, W., Graton, L. C., and Gordon, C. H., The Ore Deposits of New Mexico: U.S. Geol. Survey Prof. Paper 68, 1910, p. 75

13 Approximately two thirds of total gold production of North Carolina, 1799-1900. See Dunlop, J. P., work cited.

14 Approximately two thirds of total gold production of Oregon, 1848-1900, as recorded at U.S. mints and assay offices. See Diller, J. S., Mineral Resources of Southwestern Oregon: U.S. Geol. Survey Bull. 546, 1914, pp. 22-23. The estimate is probably much too low.

15 Approximately one fourth of South Carolina's total gold production, 1829-1900. See Dunlop, J. P., work cited.

16 See O'Harra, C. C., Early Placer Gold Mining in the Black Hills: Black Hills Engineer, vol. 19, no. 4, 1931, p. 361.

17 The early production of the placers in Bingham Canyon, Utah, is estimated at $1,000,000. See Butler, B. S., Loughlin, G. F., Heikes, B. C., and others, Ore Deposits of Utah: U.S. Geol. Survey Prof. Paper 111, 1920, p. 131. The same publication, p. 133, gives Utah placer gold production, 1904-17 by years.

18 Approximately one third of total Virginia gold production, 1828-1900. See Dunlop, J. P., work cited.

19 Approximately one fourth of total gold production of Washington, 1860-1900.

20 Approximately one half of Wyoming's total gold production, 1867-1900. See Henderson, C. W., Gold, Silver, and Copper in Wyoming in 1914: U.S. Geol. Survey Mineral Resources of the United States, 1914, pt. I, pp. 248-249.

21 Total United States gold production, including Alaska but excluding the Philippine Islands, 1801-1900, was 115,275,707 ounces or approximately $2,382,960,000.

22 Total gold production of United States, including Alaska but excluding the Philippine Islands, 1901-32, was 109,117,309 ounces or approximately $2,255,660,000.

On the other hand, a sudden increase in the load of the stream from above, or a raising of base level, may bury the gold-bearing gravels under many feet of barren sediments; or, as in California, volcanic activity may bury gravels under beds of lava or ash. Hundreds of miles of such buried channels have been traced and mined in the Sierra Nevada. The gravels of these deeply buried placers are typically "tight" owing to the pressure of overburden or are cemented by the deposition of lime or silica in the voids. Further erosion may expose such deposits, and sometimes exceedingly rich placers are formed in the new channels. In California the present drainage lines are distinct from the old ones but likewise run generally westward and bear sufficient similarity so that the names of some of the present streams have been applied to the old Tertiary rivers.

Where rivers empty into main valleys or lakes they form alluvial fans or deltas, and extensive placers may be built up; sometimes deposits are continuous from one river mouth to the next. Mertie[20] speaks of such placers as coalescing placers. They also may either be elevated or buried by subsequent events.

Where gold-bearing material is discharged into the ocean, beach deposits sometimes are formed. None has been of great importance in the Western States. The few rich ones were small, and since their exhaustion the lower-grade deposits have been the basis chiefly for intermittent large-scale projects all of which have failed. The famous beach deposits at Nome, Alaska, are typical examples of this class of placer. These were phenominally rich. Both at Nome and on the California-Oregon-Washington coast elevated beach deposits occur several miles inland.

A few other types of placers merit description, although they are of less relative importance. Very rich gold lodes may weather and erode so as to leave a valuable deposit of gold in place in a mass of disintegrated gangue material. This type is known as a residual placer. The broken mass of rock may gravitate slowly down a steep hillside, assisted by frost and by trickling rain waters, thus bringing about a low and imperfect concentration of gold on the bedrock, and forming an eluvial placer. A deposit of this type is the placer at Round Mountain, Nev.,[21] where in 1908 operators with two hydraulic monitors were reported to be washing out $20,000 per month. Current operations at Round Mountain are described in a subsequent paper.[22] The gravel of such deposits is typically angular, very poorly assorted, and generally loose.

Glaciers may play a part in the formation of placers, as at Breckenridge, Colo., where the valley terrane or river wash lying downstream from the terminal moraines has been profitably dredged for many years. In the Weaverville district, Trinity County, Calif., a glacial till that covers large areas is known to be gold-bearing although not sufficiently so to be of economic interest. It is believed, however, to be one important source of the gold of the present and earlier channels of the Trinity River and its tributaries.[23]

Certain of the "dry" placers of the Southwest are so different from typical river deposits as to deserve a separate classification. The gold, instead of occurring in well-defined channels of old or present streams, is distributed in poorly assorted, angular, or subangular gravels over large areas of gulches, hillsides, mesas, and ridges. Usually there is only slight concentration in present gulches, and in some deposits there is little con-

20 Mertie, J. B., The Occurrence of Metalliferous Deposits in the Yukon and Kuskokwim Regions: U.S. Geol. Survey Bull. 739, 1923, pp. 160-162.

21 Ransome, F. L., Round Mountain, Nev.: U.S. Geol. Survey Bull. 380, 1909, pp. 44-47.

22 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part II. - Hydraulicking, Treatment of Placer Concentrates, and Marketing of Gold: Inf. Circ. 6787, Bureau of Mines, 1934.

23 MacDonald, D. F., The Weaverville-Trinity Center Gold Gravels, Trinity County, Calif.: U.S. Geol. Survey Bull. 430, 1910, pp. 48-58.

centration on bedrock. Most of the gold is relatively coarse. These placers are believed to be mainly the result of erosion by floods. Torrential rains produce sudden heavy flows of water that pile up noticeably and often prominent alluvial fans along the mountains at the mouth of each canyon. The water commonly overflows its channel and spreads over the entire fan. The position of the channel itself often is greatly changed during a single flood. The flood waters diminish rapidly, at last depositing a thin coating of silt over the whole area. It is obvious that the channels, if changed frequently, will not be particularly rich, that the swift water will carry most of the fine gold away, and that the opportunity for sorting the gravels and concentrating the gold on bedrock will be too short to be effective. The ground, in fact, may scarcely be wet before the water has stopped flowing. An occasional surface concentration has been noted, which is difficult to explain unless it is a result of wind action.

An unfortunate characteristic of the dry placers from the miner's viewpoint is the common lime-cemented condition of the gravels. This "caliche" often is as hard to mine as and much more water-resistant than any of the "tight" or cemented Tertiary gravels of California.

The thickness and general character of the gravels at individual mines operated in 1932 are given in the chapters on mining methods in the three papers of this series.

The location of the placer-mining districts in the Western States is shown in figure 1.

CHARACTERISTICS OF PLACER GOLD

Placer gold occurs as particles ranging in size from minute grains to nuggets weighing 100 or 200 pounds. Pieces worth more than 5 or 10 cents are spoken of as nuggets; smaller ones are "colors." A scale of sizes, quoted from C. F. Hoffman by Lindgren,[24] is as follows:

Coarse gold, plus 10-mesh.
Medium gold, minus 10-plus 20-mesh.
Fine gold, minus 20-plus 40-mesh.
Powder (flour) gold, minus 40-mesh.

Here, the medium gold averaged 2,200 colors per ounce, or, if pure and valued at $35, about 2/3 of a color to a cent; the fine gold, 12,000 colors per ounce or 3 colors to a cent; and the powder, 40,000 colors per ounce or 10 colors to a cent. Most beach gold and some river gold, such as that of the Snake and Green Rivers, is much finer, ranging from 200 to 1,000 colors to a cent.

Colors and even nuggets almost always are flattened to some extent. Some placer gold occurs as thin flakes, which makes recovery more difficult as the flakes are not separated readily by water action from the more compact rounded grains of heavy minerals such as magnetite or garnet.

Placer gold occurs universally as an alloy with silver. Ordinarily it ranges in fineness from 700 to 950 parts of pure gold to 1,000 parts of the natural alloy, the remainder being chiefly silver. However, lower and higher degrees of fineness are common. Lindgren[25] cites the Folsom dredging field, Sacramento County, Calif., where the gold ranges from 974 to 978 fine. Yale[26] states that some gold from a drift mine near Vallecito, Calif., was 993

24 Lindgren, Waldemar, Tertiary Gravels of the Sierra Nevada of California: U.S. Geol. Survey Prof. Paper 73, 1911, p. 67.

25 Lindgren, Waldemar, work cited, p. 68.

26 Yale, C. G., Gold, Silver, Copper, Lead, and Zinc in California: U.S. Geol. Survey, Min. Res. of the U.S., 1910, pt. I, p. 365.

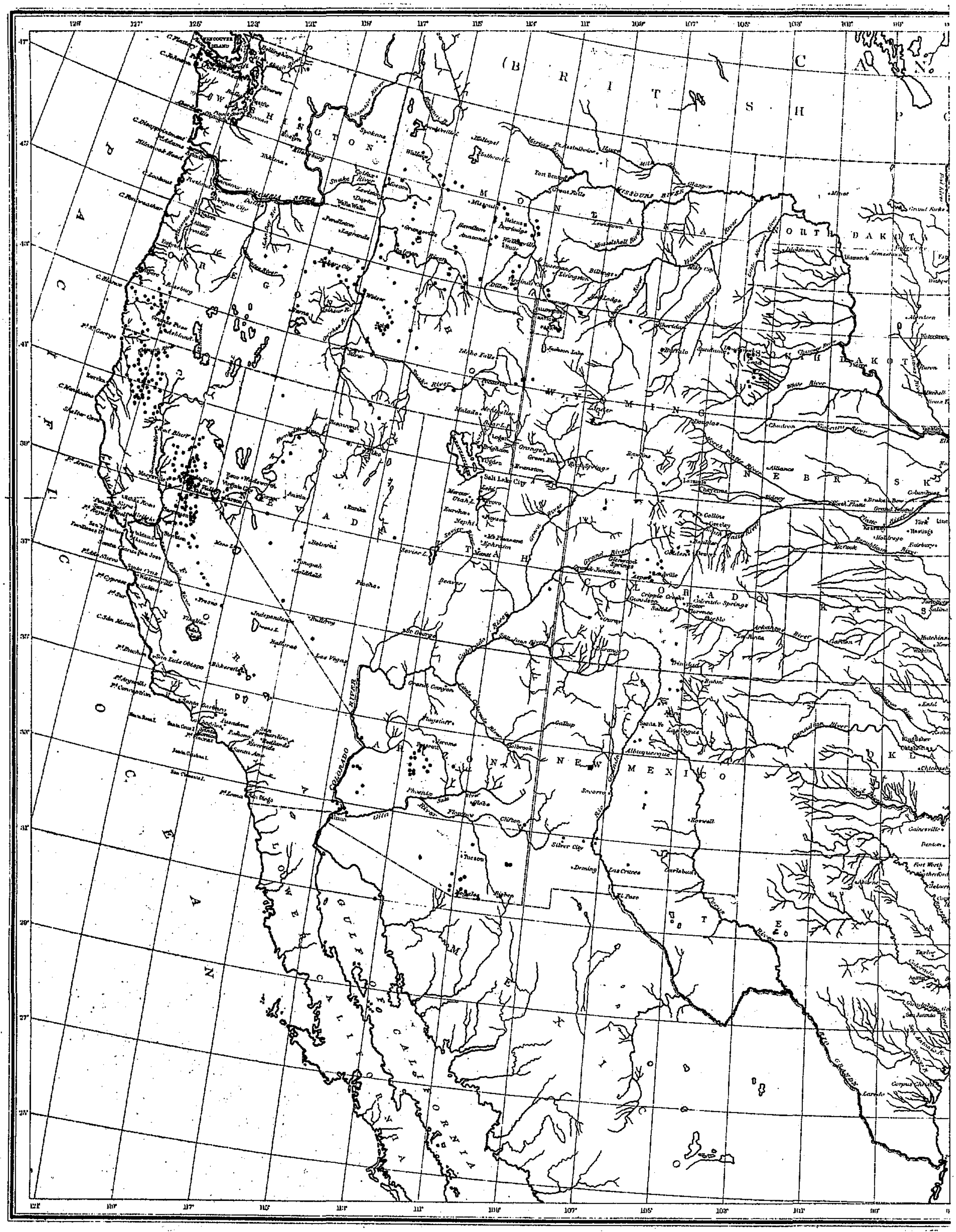


Figure 1.—Location of placer-mining districts in the Western States.

fine, or $20.52 per ounce (at $20.67), and that the gold from this property never fell below 955 fine. In a single small district the fineness of the gold is fairly uniform for any one channel. Some miners consider the fineness a distinguishing feature of a channel in districts where several channels are being explored or mined. This rule, however, is subject to exceptions, because varied sources of gold may contribute to a placer deposit and because the gold appears to lose part of its silver content and hence increases in fineness as it travels farther from its source. According to several authorities this is due to dissolving of the silver by surface waters, an action that would have relatively more effect on fine particles than on large nuggets. Fine or flour gold usually is of relatively high purity.

Associated Minerals

Placer gold invariably is accompanied by other heavy minerals, which comprise the black, white, or yellow concentrates found in the sluice box when cleaning up. Table 3 lists these minerals roughly in the order of their commonness. Some of the characteristics noted under Remarks apply chiefly to the minerals as they are found in sluice-box concentrates.

Magnetite.- Magnetite is by far the commonest mineral constituent of the heavy sands and often is a serious nuisance to the miner because it tends to pack in the riffles. It is almost always present and frequently is as much as 1 or 2 percent of the entire weight of gravel. In some beach and river-bar placers where the mining practice is to "skim" the rich streak, magnetite often amounts to several percent of the material washed. Although a valuable iron ore under certain conditions, magnetite is worthless as a byproduct of placer mines. The impression held by some miners that it contains gold is erroneous. No instance of physical or chemical natural combination of gold and magnetite is known to the authors. The gold content of placer concentrates consisting of magnetite and other heavy minerals is practically all in the form of loose particles of gold. Some of the gold, however, may be attached to quartz particles or other gangue material, particularly if near its source.

Titanium Minerals.- Ilmenite and rutile are the chief ore minerals of titanium but do not have commercial value as gold placer-mine products. It was formerly believed[27] that the black sands of the Pacific coast, which in places contain considerable of these minerals, constituted economically valuable reserves of iron, platinum, and titanium. Recently titanium has been produced commercially from the beach sands of California.[28] Probably no gold is recovered in this operation. Rutile is worth only a few cents per pound in the form of 94-percent concentrate; the demand is supplied amply by some 200 tons produced annually from lode deposits in Norway and Virginia.

Garnet.- Garnet is an abundant accessory mineral in many rocks, particularly if they are metamorphic. It has a higher specific gravity than the usual rock-forming minerals and is present in the concentrates from sluice boxes in many localities. The gem varieties of garnet are the commonest and least valuable of the semiprecious stones; they are used much in cheap jewelry, but the production of gem garnet in the United States, which was never large, has declined to such an extent that production figures are no longer kept. Garnet is one of the most useful abrasive materials, and a considerable production is so utilized. However, this production is from lode deposits exclusively, and no instance is known of the commercial production of either gem or abrasive garnet from gold placers in this country.[29]

---

27 Day D T. and Richards R. H Useful Minerals in the Black Sands of the Pacific Slope: U.S. Geol. Survey Min. Res. of the U.S. 1905 pt. I 1906 pp. 1175-1258.

28 Youngman E P. Deposits of Titanium-Bearing Ores: Inf. Circ 6386, Bureau of Mines, 1930 pp. 3-5.

29 Myers, W. M., and Anderson, C. O. Garnet: Its Mining, Milling, and Utilization: Bull. 256, Bureau of Mines, 1925 54 pp.; also, Aitkens, I. Garnets (Gem Stones): Inf. Circ 6518, Bureau of Mines, 1931, 11 pp

TABLE 3.- Chemical composition and physical characteristics of the chief heavy minerals found in gold placer gravels[1]

Mineral	Chemical formula	Color	Specific gravity	Hardness	Remarks
Gold	Au (+Ag)	Gold-yellow	15.6-19.3	2.5	Very malleable and ductile.
Magnetite	Fe_3O_4	Iron-black	5.2	5.5-6.5	Shiny grains; strongly magnetic.
Ilmenite	(Mg,Fe) TiO_3	do.	4.5-5	5-6	Only faintly magnetic; affects compass needle slightly.
Garnet	$R''_3R'''_2(SiO_4)_3$[2]	Red, brown, various	3.8	6.5-7.5	Vitreous luster; generally in rounded crystals (dodecahedrons).
Zircon	$ZrSiO_4$	Brown, pale yellow, or colorless	4.7	7.5	Adamantine luster.
Hematite	Fe_2O_3	Dark steel-gray to iron-black	4.9-5.3	5.5-6.5	Particles generally smooth, rounded, often red-coated.
Chromite	$FeCr_2O_4$	Iron-black to brown-black	4.1-4.9	5.5	Sometimes feebly magnetic; brown streak.
Olivine	$(Mg,Fe)_2SiO_4$	Olive-green	3.3-3.4	6.5-7	Good cleavage; vitreous luster; transparent to translucent.
Epidote	$HCa_2(Al,Fe)_3Si_3O_{13}$	Pistachio-green	3.2-3.5	6-7	Distinct cleavage.
Pyrite	FeS_2	Pale brass-yellow	4.9-5.1	6-6.5	Usually cubic grains; brittle; metallic luster.
Monazite	(Ce, La, Di) PO_4 + ThO_2	Yellow	4.9-5.3	5-5.5	Resinous or greasy luster; usually in rounded grains.
Limonite	$2Fe_2O_3.3H_2O$	Dark brown	3.6-4.0	5-5.5	Yellow-brown streak.
Rutile	TiO_2	Red-brown to red	4.2	6-6.5	Distinct cleavage; metallic-adamantine luster.
Platinum	Pt (usually also Fe,Ir,Os)	Whitish steel	16.5-18	4-4.5	Malleable; sometimes scales and grains.
Iridium	Ir (Also Pt, etc.)	Silver-white, yellow tarnish	22.6-22.8	6-7	Usually in angular grains; no cleavage.
Iridosmine	Ir, Os	Tin-white to light steel-gray	19.3-21.1	6-7	Usually in flat grains; slightly malleable to brittle; good cleavage.
Cinnabar	HgS	Red	8-8.2	2-2.5	Scarlet streak.
Wolframite	(Fe, Mn) WO_4	Black, dark gray	7.2-7.5	5-5.5	Submetallic luster; good cleavage in one plane.
Sheelite	$CaWO_4$	White, pale yellow, brown, or gray	5.9-6.1	4.5-5	Adamantine, greasy luster; translucent.
Cassiterite	SnO_2	Brown or black	6.8-7.1	6-7	Brittle; rounded grains.
Corundum					
Sapphire	Al_2O_3	Blue, red, yellow, brown	3.9-4.1	9	Adamantine to vitreous luster.
Ruby					
Diamond	C	White, colorless, pale	3.5	10	Adamantine or greasy.
Mercury	Hg	Tin-white	13.6		Small opaque fluid; silvery globules.
Amalgam	Hg, Ag, Au	Silver-white	13-14		Brittle to malleable; rubs silvery coat on copper.
Galena	PbS	Lead-gray	7.4-7.6	2.5-2.7	Metallic luster; lead-gray streak; perfect cubic cleavage; friable.
Silver	Ag	Silver-white	10.1-11.1	2.5-3	Malleable and ductile; tarnishes black.
Copper	Cu	Copper-red	8.8-8.9	2.5-3	Ductile; malleable.
Bismuth	Bi	Silver-white	9.8	2.5	Sectile; brittle; metallic luster.
Cerussite	$PbCO_3$	Colorless or white	6.5	3-3.5	Adamantine luster.
Columbite-tantalite	$(Fe,Mn)(Nb,Ta)_2O_6$	Iron-black to gray or brown-black	5.3-7.3	6	Brilliant to submetallic luster; often iridescent; brittle; good cleavage.
Quartz	SiO_2	Colorless	2.6	7	No cleavage; vitreous to greasy luster.
Feldspars	Silicates of K,Na,Ca,Al,etc.	Colorless, white, pale yellow, or pink	2.5-2.7	6-6.5	Good cleavage; vitreous luster.

1 Dana's Textbook of Mineralogy, 4th ed., by W. E. Ford, 1932, was used for most of the mineralogical data.

2 R'' = [illegible]g, Mn, or Fe; R''' = Fe, Al, or Cr.

Zircon.- Zircon[30] has the highest specific gravity of all gem stones; in addition, certain characteristics of hardness, high refractive index, and color are slowly making it more popular as a gem. Its frequent occurrence in sluice-box concentrates is therefore of interest. It is said to be abundant in the gold-bearing gravels of Henderson County, N.C., and crystals of it are common in most auriferous sands; in some workings it comprises several percent of the concentrates. The crystals are seldom of gem quality, and the usual material is worthless as a zirconium ore because of the low price of the latter - about 10 cents per pound for a semirefined product. Even as a gem uncut zircon brings only a few dollars per carat and seldom over $15 when cut.

Hematite.- Hematite often is found in placer gravels, particularly in the dry placers of New Mexico and Arizona. Sometimes hematite cobbles and small pebbles are said to indicate rich ground, but this idea is likely to be misleading.

Chromite.- Chromite occurs very commonly in black sands, in some deposits greatly exceeding the magnetite in quantity, but has no commercial value as found in gold placers.

Olivine and Epidote.- Olivine and epidote are common dark, heavy, rock-forming minerals and consequently are often present in the concentrates. They are of no value.

Pyrite.- The occurrence of pyrite sometimes has mineralogical interest because it is believed to have formed in place in gold placers. It is often found in the Tertiary gravels in California and sometimes is thought to have resulted from the reducing action of organic matter on iron sulphates in the meteoric waters.[31] Pyrite is also found as detrital grains or masses derived from lode deposits or from the country rock.

Monazite.- Monazite, a phosphate of the rare-earth metals, valuable because of its varying small content of thorium, is characteristic of the Boise Basin placer sands of Idaho but is found in many other districts in Idaho, Colorado, Montana, and Oregon. It is notably lacking in the California deposits, with rare exceptions. A serious attempt was made about 1909 to utilize the monazite contained in the tremendous quantities of hydraulic tailings lying in the Boise Basin fields.[32] A 1,000-ton plant was built, comprising tables, drier, screens, and several magnetic concentrators. A 95-percent monazite sand could be made, containing about 5 percent thoria. None was ever marketed, probably because the cost would not permit competition with the much richer Brazilian and Indian beach deposits.[33]

Limonite.- Limonite is an iron-ore mineral found occasionally in placer concentrates and has no economic importance in this connection.

Platinum and Osmiridium.- Platinum and osmiridium are found in many districts and are almost the only consistently valuable byproducts of American gold-placer mines. They are typical of the beach-placer sands of California and Oregon, where they occur in proportions as high as a tenth or more of the gold.[34] Practically all the Sierra Nevada placers contain platinum-group metals, as do the placers of the Klamath Mountains, particularly those on the Trinity River in the Hay Fork and Junction City districts.[35] None of these deposits, however, have contained enough platinum to be worked for that metal alone. Indeed, the dredge operations are the only ones to produce platinum consistently, even as a byproduct of gold mining, the total annual output seldom being over 300 or 400 ounces. In a recent year one

30 Youngman, E. P., Zircon (the Gem): Inf. Circ. 6465, Bureau of Mines, 1931, 20 pp.

31 Lindgren, Waldemar, Tertiary Gravels of the Sierra Nevada: U.S. Geol. Survey Prof. Paper 73, 1911, p. 76.

32 Sterrett, D. B., Monazite and Zircon: U.S. Geol. Survey, Min. Res. of the U.S., 1909, pt. II, 1911, pp. 898-903.

33 Santmyers, R. N., Monazite, Thorium, and Cerium: Inf. Circ. 6321, Bureau of Mines, 1930, 43 pp.

34 Hornor, R. R., Notes on the Black Sand Deposits of Southern Oregon and Northern California: Tech. Paper 196, Bureau of Mines, 1918, 42 pp.

35 Day, D. T., Platinum: U.S. Geol. Survey, 19th Ann. Rept., pt. VI, vol. 1, 1897-1898, pp. 265-271.

dredging company, which recovered about 50,000 ounces of gold from 10,000,000 cubic yards of gravel, sold about 110 ounces of crude platinum metals; these contained 70 ounces of platinum and 6 or 8 ounces each of iridium and osmium. Very few hydraulic or other placer operators find it worth while to separate the platinum from the gold because of the small quantity involved and the difficulty of finding a purchaser. Although platinum sold for as much as $154 per ounce in 1920[36] its average price in 1932 was only $36.46 per troy ounce of refined metal, and its average price in January 1933 was $26.48. In January 1934 the price was $38.00 per ounce.

The prices of iridium and osmium fluctuate widely, iridium having sold during recent years for $50 to $300 per ounce and osmium for $25 to $115.

Cinnabar.- Cinnabar is found in sluice-box cleanups in a few localities. It is unmistakable because of its brilliant red color. Edman[37] states that minute quantities of cinnabar, in grains and crystals, are found in the platinum-bearing sands of Plumas County, Calif. It was seen recently by the junior author of this paper in pan concentrates from the gravels of Copper Basin Wash near Skull Valley, Ariz., in the form of round pellets the size of bird shot. Cinnabar is known to occur in lode deposits in that locality.

Tungsten Minerals.- Wolframite and scheelite, the most important tungsten-ore minerals, occasionally are found in placer deposits. In Alaska they have been mined from placers in which they were associated with gold. In the States, particularly in the Atolia-Randsburg district of California and in the Boulder County tungsten district, Colo., considerable tonnages of tungsten minerals have been recovered from eluvial or residual placer deposits.[38] At Atolia scheelite was mined and milled on a commercial scale until about 1930 when operations ceased due to the low price of tungsten.[39] In Colorado the mineral was ferberite, the high-iron member of the wolframite series. Huebnerite, the high-manganese end member of the series, has been mined by dry-washing during periods of high prices from shallow alluvial or residual deposits at Round Mountain, Nev.[40]

Cassiterite.- Cassiterite, the principal ore of tin, has been recovered from placer deposits in Alaska and has been noted in a few placer deposits in the States; small quantities have been mined in the Appalachian gold-tin district. None of the occurrences in the Western States has proved to have commercial importance.

Corundum.- Sapphire and ruby are varieties of the mineral corundum; they are translucent or transparent, of a fine blue or red color, and otherwise of gem quality. When the mineral is an opaque light blue, brown, or gray it is known as corundum, and if granular and mixed with impurities such as magnetite it is called emery. Neither corundum nor emery can be produced by placer mining to compete with richer and better-situated deposits or with the increasing production of artificial aluminum oxide abrasives; but sapphires and rubies, especially if of gem quality, are valuable.

It is not known that sapphires or rubies have ever been of much value in this country as byproducts of gold-placer operations. However, in several districts in Montana, notably along the Missouri River near Helena, sapphires were discovered in the course of gold mining, and several attempts were made in 1891 and later years to wash the bars for the gems. These ventures were financial failures, partly because the investment cost was high and partly be-

---

36 Tyler, P. M., and Santmyers, R. N., Platinum: Inf. Circ. 6389, Bureau of Mines, 1931, 69 pp.

37 Edman, J. A., The Platinum Metals of Plumas County, Calif.: Min. Sci. Press, vol. 77, Oct. 22, 1898, p. 401.

38 Hess, F. L., Tungsten Minerals and Deposits: U.S. Geol. Survey Bull. 652, 1917, pp. 44-45.

39 Vanderburg, W. O., Methods and Costs of Concentrating Tungsten Ores at Atolia, San Bernardino County, Calif.: Inf. Circ. 6532, Bureau of Mines, 1931, 12 pp.

40 Ferguson, H. G., The Round Mountain District, Nevada: U.S. Geol. Survey Bull. 725, Contributions to Econ. Geol., 1921, pt. 1, 1922, p. 389.

cause the sapphires were not of good color. In 1899-1900 about 25,000 carats of gems, including a few small rubies suitable for cutting, were selected from about 400,000 carats hydraulicked from the gravels of Rock Creek, Granite County, Mont. Beginning about 1905 these mines were again worked, the chief product being sapphires of a quality suitable for watch jewels and other bearings. Occasional gem stones also were found, and a small quantity of gold was saved as a byproduct. The average selling price of the bearing sapphire was about $1 per ounce. Since 1917 the mines have been operated intermittently on a reduced scale.

From about 1905 to 1915 a bucket dredge was operated intermittently in another Montana gold-sapphire district, on Cottonwood Creek, Deerlodge County. The gulch gravels here were 10 to 14 feet thick and were composed of porphyry cobbles and subangular boulders with 3 or 4 feet of black muck overburden.[41] Both gold and sapphires were concentrated near bedrock. The gold was fine and was said to pay operating expenses. Most of the sapphires were of a quality suitable only for mechanical purposes, although some gem stones were found.

Montana has continued to produce sapphires from the Judith Basin district lode mines. One Montana lode-mining company in 1924, 1925, and 1926 produced an average of about 50,000 carats of sapphires suitable for cutting into gems less than 1 carat in weight. This output was valued at about 50 cents per carat, and the same company's much greater production of industrial stones was valued at a small fraction of a cent per carat.[42] (One carat equals 200 mg.) Large gem sapphires, on the contrary, have sold for as much as $1,500 per carat.

<u>Diamonds</u>.- Diamonds have been reported from gold-placer gravels in California and other districts. Lindgren[43] states that the principal localities in California are Cherokee Flat, Butte County, where 56 specimens, ranging up to 1 1/2 carats in size, were reported, and Placerville, Eldorado County. Tyler[44] gives the placer mines of the Volcano and Fiddletown districts, Amador County, as the chief producers. The gold-bearing gravels of south central Indiana occasionally produce diamonds; one weighing 1 1/2 carats has been described.[45]

It is believed that diamonds originate in the same rocks as platinum -- namely, the serpentines resulting from alteration of peridotites. The diamonds found in gold placers in this country have not been of commercial importance, being small, yellowish, and far from abundant. Doubtless many pass over the riffles because of their relatively low specific gravity (less than 40 percent greater than quartz), and probably many more pass unrecognized into the rejects from the treatment of concentrates. In Arkansas, where diamonds are mined by hydraulic methods, the principal recovery is made on grease-covered vibrating tables.

<u>Quicksilver</u>.- The natural origin of quicksilver in gold placers seldom can be established definitely, because very little strictly virgin ground is now being mined in the United States. Dana's Textbook of Mineralogy states, however, that it does exist in some alluvial deposits. In view of the frequent occurrence in lode deposits of native quicksilver associated commonly with one of the mercury sulphides and the finding of cinnabar in placer deposits, it is not unreasonable to assume that some of the metallic mercury recovered is of natural origin. It is certain, however, that as a rule such quicksilver has been introduced in earlier mining activities. Dredges and hydraulic mines frequently recover considerable mercury in their riffles. In the Skull Valley district of Yavapai County, Ariz., the authors were informed that one plant excavating and treating the shallow gravels of certain

---

41 Sterrett, D. B., Precious Stones: U.S. Geol. Survey Mineral Resources of the United States, 1907, part II, 1908, pp. 821-822.

42 Aitkens, I., Rubies and Sapphires: Inf. Circ. 6471, Bureau of Mines, 1931, 11 pp.

43 Lindgren, Waldemar, Tertiary Gravels of the Sierra Nevada of California: U.S. Geol. Survey Prof. Paper 73, 1911, p. 75.

44 Tyler, P. M., Abrasive and Industrial Diamonds: Inf. Circ. 6562, Bureau of Mines, 1932, p. 14.

45 Schaller, W. T., Gems and Precious Stones: U.S. Geol. Survey Mineral Resources of the United States, 1916, part II, 1917, pp. 892-893.

tributaries of Copper Basin Wash had recovered enough mercury to increase considerably its original stock and to furnish a neighboring plant and a number of "snipers" with several pounds of mercury. This district has been the scene of intermittent 1- or 2-man operations for several decades, yet the occurrence of cinnabar in the gravels makes the natural origin of metallic mercury at least plausible.

Amalgam.- Amalgam is recognized as a mineral only when it is a crystalline combination of mercury and silver having fairly definite characteristics and is thus described in mineralogical textbooks. Clarke,[46] however, gives two analyses of native gold combined with mercury. One from Colombia contained approximately 84 percent of gold, 8 percent of silver, and 7 percent of mercury; the other specimen, described as amalgam from Mariposa County, Calif., contained 39.02 percent of gold and 60.98 percent of mercury (approximating the formula $AuHg_3$, and had a specific gravity of 15.47. Most of the amalgam recovered in sluice boxes, however, undoubtedly is of artificial origin.

Galena.- Galena is an uncommon constituent of placer concentrates, presumably because it is both soft and friable and because it oxidizes readily. It was observed in sluice boxes by the authors in the Cedar Creek district of Montana; in some specimens it still adhered to part of the original matrix of gangue minerals.

Other Minerals.- Although all placer gold is alloyed with silver, no record has been found of the occurrence of native silver, as a separate metal, in placer deposits in the United States, and silver is included in the list only for comparison with gold. It is known, however, that many silver nuggets were found in the Nizina district of Alaska; one nugget, with attached quartz, weighed over 7 pounds.[47]

Copper nuggets have been found in placer gravels in many districts. Like the reported occurrence of bismuth in the gravel of French Creek, Summit County, Colo., this fact is of mineralogical interest only, at least so far as any district in the Western States is concerned. The placers of Chititu Creek in the Nizina district, Alaska,[48] contain enough copper to be a nuisance (several hundred pounds at each clean-up). It is not known that any of this was ever marketed.

In California Gulch, above Leadville, Colo., where placer mining preceded the discovery of valuable lead deposits by several years, cerussite (lead carbonate) was known to the miners simply as a variety of rock that was so heavy as to obstruct sluicing.

The mineral columbite-tantalite occurs in some pegmatite veins. The Black Hills region of South Dakota furnishes fine specimens of this mineral, and it is reported, together with cassiterite and scheelite, in the placer gravels of Bear Creek, in the northern Black Hills.

Quartz and feldspar, which constitute the bulk of placer sands, are included at the end of table 3 for comparative purposes. It will be observed that their specific gravities are one fourth lower than those of olivine and epidote, a third lower than that of common garnet, half those of magnetite and hematite, and only a sixth or seventh that of gold.

Other minerals besides those listed in table 3 are found in sluice-box concentrates, some because of their heavy weight and some merely because of imperfect concentration. Many artificial objects likewise find their way into the concentrates, the typical ones being bird shot and nails—quite often the hobnails of the "oldtimers."

---

46 Clarke, F. W., The Data of Geochemistry: U.S. Geol. Survey Bull. 770, 5th ed., 1924, p. 658.

47 Moffitt, F. G., and Capps, S. R., Geology and Mineral Resources of the Nizina District, Alaska: U.S. Geol. Survey Bull. 448, 1911, p. 105.

48 Moffitt, F. G., and Capps, S. R., work cited, pp. 105-106.

LOCATION OF PLACER CLAIMS ON PUBLIC LANDS

Placer claims containing alluvial deposits of gold or other metals can be located and patented on the public domain, national forests, stock-raising homesteads, and unpatented parts of congressional grants to railroads. Public land temporarily withdrawn from settlement, location, sale, or entry and reserved for water-power sites, irrigation, classification, or other public purposes shall at all times be open to exploration for metalliferous minerals and purchase under the mining laws. However, power or reservoir sites withdrawn by congressional action or Executive order are not subject to mineral location.

Although placer claims can be located and mineral rights obtained on stock-raising homesteads, written permission must be received from the homesteader to enter upon the land, or a bond of $1,000 must be posted to indemnify the agricultural entryman for any damage that may be done to the crops or tangible improvements. Surface rights are limited to the land actually needed for mining purposes.

Mining claims cannot be filed upon patented land except where the minerals have been reserved to the United States, on military or naval reservations, or in national parks or monuments. Land below high tide, lake beds (except Searles Lake, Calif.), or the beds of navigable rivers are not subject to mineral location.

Public land, in the public-land States, valuable for minerals cannot be patented except under the provisions of the mining law, and valid mineral locations take precedence over other forms of land entry.

According to the Federal law, mining locations, both lode and placer, may be made by citizens or those who have declared their intention to become citizens, by an association of qualified persons, or by a domestic corporation. Locations can be made without regard to age, sex, or residence of the locator. No limit is placed by the Federal statutes on the number of locations that may be made in the United States by an individual or a company. A locator may include as colocators other persons who may or may not have seen the ground; also, a person may make valid locations for other parties.

The Federal statutes require that a location notice must contain the names of the locator or locators, the date of location, and a description of the claim by reference to some natural object or permanent monument that will identify the claim. A discovery of valuable mineral within the limits of the claim must be made for the location to be valid.

State laws in most placer-mining States define how the location notice must be posted, the size of the discovery shaft or other discovery excavation that must be sunk if required, and the claim boundaries must be marked. These laws require that the location must be filed with the proper county official.

A single placer claim located by an individual locator is limited to a maximum of 20 acres. The law, however, permits the location of association claims. That is, an association of 2 individuals can file on a 40 acre claim; 3 individuals on 60 acres, etc., up to a group of 8 persons who can locate 160 acres as a single claim.

One discovery of mineral is required to support a placer location, whether it is 20 acres of an individual or 160 or fewer acres of an association of persons. Placer claims should conform as nearly as possible to legal subdivisions of the public land surveys, except where the rectangular subdivisions would necessitate placing the lines upon previously located claims. The smallest legal subdivision is a square tract containing 10 acres and measuring 660 feet each way. Although it has been held that placer claims may be located to conform to their environment where the topography is such that it is impractical to lay out rectangular claims, as in gulchs with precipitous walls, the Land Office regulations require that the entries be as compact as possible and will not permit entries for patent which cut the public domain in long, narrow strips or grossly irregular tracts.

Both lode and placer locations can be amended at any time and the boundaries changed, provided such changes do not interfere with the rights of others.

A placer claim cannot be located over an older lode location if the lode claim is located legally on a deposit of valuable mineral in place. The owner of a valid lode location also owns any placer deposits the claim may contain.

A lode claim can be located legally on known lodes in place containing valuable mineral on an unpatented placer claim held by the lode locator or others in the same manner as if the placer location did not exist. A lode deposit cannot be held under a placer location, but once a placer claim is patented the owner owns and may mine all lodes not known to exist at the time the patent was issued.

Although valid mineral entries have precedence over agricultural or other entries, a placer claim must be shown to be more valuable for minerals than for agriculture in contested cases.

To hold the possessive title to a mining claim not less than $100 worth of work must be performed or improvements made upon or for the benefit of each claim each year, regardless of its size. Where a number of contiguous claims are held in common, the aggregate expenditures for the group may be made on one claim. Locations connecting only at the corners are held to be noncontiguous. The period within which the annual work must be done commences at noon of July 1 succeeding the date of location. Failure to do the annual assessment work will subject a claim to relocation unless work is resumed before such relocation. It has been held that a claim is not subject to relocation if work is being performed on the ground at the end of the required period. In other words, if work is begun on a claim located, say, in September 1928 by noon of July 1, 1930 and diligently carried on thereafter until complete, it is not subject to relocation. Additional work would be required for the period commencing July 1, 1930. Annual expenditure is not required subsequent to making entry at the Land Office for patent. Congress, by the act approved June 6, 1932, relieved claim owners of the necessity of doing the annual assessment work on unpatented mining claims for the period ended July 1, 1932. Congress by the act approved May 18, 1933 also relieved claim owners (except those required to pay the Federal income tax) of doing the annual work for the year ending July 1, 1933.

The annual assessment work may be omitted on a claim for 1 or more years, and the location will still be valid if work is resumed on the ground, provided no interfering interests are affected or no other location has been made on the ground.

Where one of several locators fails to contribute his share of the required expenditures made for the benefit of a claim the co-owners at the expiration of the year may give notice personally or in writing or by advertising in the newspaper published nearest the claim at least once a week for 90 days; if upon the expiration of 90 days after the personal notice or upon the expiration of 180 days after the first newspaper notice the delinquent co-owner shall have failed to contribute his proportion of such expenditures or improvements, his interest in the claim passes by law to his co-owners who have made the required expenditures.

The Secretary of the Interior has been authorized by Congress (act of June 30, 1929) to lease unallotted lands on Indian reservations for mining purposes in Arizona, California, Idaho, Montana, New Mexico, Oregon, Washington, and Wyoming. After declaration by the Secretary that the lands are subject to lease, claims may be located as on the public domain; a duplicate of the location notice must be filed within 60 days with the superintendent in charge of the reservation. The locator has 1 year's preference right to apply for a lease through the reservation superintendent to the Secretary of the Interior. Leases are for 20 years, with provision for 10-year renewals.

Most of the mining States make provision for leasing minerals found on State lands. After discovery, application for a prospecting or mining lease should be made to the authority having charge of State lands. Regulations regarding the granting of prospecting or mining leases vary in different States.

PROSPECTING OUTFITS AND PROVISIONS

The outfit to be taken on a prospecting trip depends upon the mode of transportation, the work contemplated, and the funds available. Enough equipment should be taken, but unnecessary articles make extra work. When a more or less permanent camp is established, added equipment for personal comfort and efficiency can be obtained. Usually a cabin is built for a permanent camp.

Camp Outfit

Most prospectors these days use automobiles and can carry complete camp equipment.

A tent should be carried if an extended trip is to be made. Few prospectors, however, put up a tent for one night unless the weather is bad. Bedding consists of 3 or 4 blankets rolled up in canvas; the number of blankets needed depends upon the climate. A folding cot is advisable; however, in a permanent camp a bunk is usually built. A full-size single-bitted ax should be carried; the one-handed "Boy-Scout" ax is practically useless. A saw and a hammer with 2 or 3 pounds of assorted nails is needed in fixing up a camp and for other purposes. All prospectors and campers carry a good-quality, stout pocket knife. A miner's acetylene lamp provides a good light; a 5-pound can of carbide will last a camp nearly all summer. A flashlight is very convenient, but an ample supply of batteries must not be overlooked.

A 50-foot length of 1/2- or 5/8-inch manila rope is useful for lashing the load to the car or truck, as a tent rope, for getting down into old shafts or cuts, and in making a "Spanish winch" for pulling the car out of a mudhole or for moving heavy equipment. To make this winch, one end of rope is attached to the object to be moved and the other to a tree or other anchor some distance ahead. A 5- or 6-foot length of stout sapling is held vertically or stood in a hole midway between the ends. A loop formed in the middle of the rope is placed around the end of a long pole or pipe which is used as a sweep to wind the rope around the sapling, thereby shortening the rope and pulling the object ahead.

A canvas sheet is handy to protect the load from rain and dust. A canvas war bag is needed for clothing and other articles. A tight, stout, wooden box with a lid saves food by keeping insects and rodents away from it and is a great convenience in making and breaking camp.

A stove is necessary if a camp is to be maintained in cold weather. A combination heating and cooking stove is best, preferably one with an oven in which bread can be baked.

Camp furniture usually is improvised on the spot, the kind and amount depending upon the length of time the camp is to be used or the camper's idea of personal comfort.

A shotgun and a few boxes of shells can be used to provide a change in diet where rabbits or game birds abound; the cost of the ammunition, however, may exceed the money value of the food thus obtained. Many prospectors living in the hills keep rifles to supply the larder with venison. A few fishhooks and a line, or even more elaborate tackle, take little room in the car and provide relaxation and food where streams contain fish. However, too much sporting equipment interferes with the purpose of a prospecting trip, and guns probably would prove a nuisance as they cannot be left unguarded in a camp without the likelihood of their being stolen.

A 2-quart canteen with a shoulder strap usually is needed for carrying drinking water or water for panning. A 2-gallon canteen and a 5- or 10-gallon water keg are necessary in some districts.

Clothing

The most important item of clothing is a pair of stout, thick-soled shoes of good quality, preferably hobnailed. If an extensive trip is planned a second pair may be needed. Other clothing can be patched, but when a prospector's shoes go to pieces his trip is ended. A pair of rubber boots will prove a comfort if much placering is done.

Woolen socks to wear under the heavy shoes help to prevent blisters; several pairs may be worn out in a season.

Other clothes are chosen for the climate and service. A leather jacket is very serviceable and comfortable in cool weather, or a sheepskin coat may be needed when it gets colder. Many prospectors in mountainous regions wear flannel shirts and woolen underwear. Overalls are a common garb.

A complete change of clothing should be taken on all except the shortest trips to permit changing into dry clothing after being caught out in the rain or working in water all day.

Tools

A pick, a long-handled, round-pointed shovel, a gold pan, and a prospector's pick are indispensable. If claims are to be staked, a compass will be needed for running out the lines. A hand magnifying glass is a great help in identifying minerals. If lode deposits are to be sought a mortar and pestle, a horn spoon, or a small pan will be needed for testing rock for free gold or other heavy minerals. A blowpipe outfit and determinative tables are of service to those who can use them. Bags for taking out samples usually are needed. Double paper bags with rubber rings cut from old automobile tubes for closing the bags permit large numbers of samples to be collected with little expense for bags.

A single-jack hammer with 2 or 3 moils will come in handy for taking samples and for loosening rock encountered in making cuts.

Some prospectors carry 1 or 2 sets of hand steel and a few pounds of powder. A few rounds may be drilled and blasted before the steel has to be resharpened. If any extensive rockwork is to be done a forge and a set of blacksmith tools are necessary; these usually are brought in later.

If any placering is contemplated a rocker and enough lumber to build one or two 12-inch sluice boxes may be carried.

Cooking Equipment

For a 1- or 2-man party, a frying pan, a coffee pot, a large and a small stew pan or pot, and a Dutch oven are needed. A knife, fork, spoon, cup, and plate are required for each man. A few extra plates come in handy. A good butcher knife, a water pail, a can opener, and a few tea towels complete the set. Other dishes can be taken according to personal preferences.

Provisions

The variety of food taken on prospecting trips depends upon the method of transportation and the prospector's pocketbook. If the supplies are to be packed on animals, bulky foods such as potatoes and canned articles are omitted. If there is need for economy in making purchases, the list will consist mostly of dried staples and vegetables if available locally.

Bacon, flour, beans, oatmeal, dried or canned fruit, coffee, syrup for hot cakes, and sugar and canned milk for the coffee are the stand-bys in prospectors' camps. As funds get low more beans and less bacon are eaten, and canned fruit is omitted. Canned tomatoes are commonly used; they are cheap and supply needed food elements not contained in dry staples. Where available locally Irish potatoes, onions, and other vegetables are eaten. Fresh meat is not used much in camps in the summer on account of the difficulty of keeping it. In dry climates, however, a quarter of beef can be eaten by a crew of a dozen men before it spoils by using proper care. The meat is hung in the open air during the night and in the morning, while still cool, is wrapped in blankets or a bed roll for the day.

A proper balance should be made in making out a "grub" list so that needed items will not run short. Everyone has preferences for different articles of food which should be followed as much as practicable. It has been found by experience, however, that fancy groceries are the ones left over and the first articles to be used are the bacon, potatoes, and flour. Plain, wholesome fare seems to be preferred in camp, especially where hard work is done.

The old United States army ration for one man for 1 day was: 12 ounces bacon or pork, or 22 ounces fresh beef; 18 ounces soft bread or flour, or 16 ounces hard bread, or 20 ounces corn meal.

For each 30 rations the following items were issued: 5 pounds beans or peas, or 10 pounds rice; 5 pounds sugar; 1 quart vinegar; 1/4 pound hard candy; 1 pound soap; 1 1/2 pounds salt; and 1 1/4 ounces pepper.

The following standard ration list for fireguards is used as a guide by the Forest Service. The articles listed are enough for one man for 30 days.

Article		Quantity
Bacon, salt	pounds	2
Bacon, smoked	do.	10
Baking powder	pound	1
Baking soda	do.	1/2
Beans, pink	pounds	5
Beef, fresh (purchased locally)		....
Butter, best	pounds	2
Candles		6
Cheese, full cream	pounds	1 1/2
Chili powder	small bottle	1
Coffee, best	pounds	4
Corn	cans	6
Dried fruit, variety	pounds	9
Flour	do.	24
Green chili	cans	3
Jam	jar	1
Lard	pounds	4
Matches	large boxes	2
Milk	tall cans	10
Oatmeal	pounds	6
Onions, dry	do.	5
Peas, best grade	cans	6
Pepper, black	pound	1/4
Potatoes, Irish	pounds	15
Raisins, seedless	do.	2
Rice	do.	3
Salt, table	pound	1

Article	Quantity
Sauerkraut....cans	3
Soap, hand....cake	1
Soap, laundry....do.	1
Spinach....cans	3
Sugar....pounds	12
Syrup....gallon	1
Tomatoes....cans	6
Towels, tea....	3

The following weekly allowance of food for one person to give a balanced diet is condensed from suggestions made by Doctor Smith:[49] Three 1-pound cans evaporated milk; 2 pounds potatoes; 4 pounds onions, cabbage, beets, or other vegetables; 3 pounds citrus fruits, or 6 pounds fresh apples, or equivalent dried prunes, apricots, etc.; 3 pounds dried beans; 6 to 8 pounds cereals, whole-wheat flour or bread, rolled oats, shredded wheat, etc.; 2 1/2 pounds dried meat, bacon, ham, or cheese (fresh meat or eggs may be substituted if available); 3 pounds sugar; 1 pound coffee; 1/4 pound salt; 1/2 pound butter; and baking powder.

In many districts of the Southwest water must be carried. The quantity required depends upon the time of year and the amount of work done by the miner. Men working where temperatures range from 100° to 110° drink 2 gallons or more of water per day; under such conditions, a 10-gallon tank would last one man 3 days, allowing for cooking but not for the radiator of the car. In cooler weather a 10-gallon tank should last one man a week or 10 days.

First-Aid Supplies

As prospectors are likely to be away from medical aid, some medical and first-aid supplies should be taken along. These should consist of a laxative (castor oil or salts), iodine or mercurochrome to disinfect cuts or bruises, and a first-aid kit. A snake-bite kit may also prove invaluable.

SAMPLING AND ESTIMATION OF GOLD PLACERS

Failure to sample and estimate properly the available yardage of placer deposits has resulted in a tremendous waste of money and effort. A large proportion of all placer operations has failed because the gold in the gravel was insufficient to repay the cost of even the most efficient mining, not to mention the return of money invested or interest thereon.

Many methods of sampling are available, including the simple panning of gravel from natural exposures, drifting, test-pitting or trenching, shaft-sinking, and churn-drilling. Actual mining on a small scale often is done as a method of sampling prior to investing considerable money in development or equipment. Several examples will be noted later under methods of mining.

The technique of panning and its use for estimating the gold content of gravel are discussed under the head of Panning and Rocking, as these operations are properly considered small-scale mining methods. In the following section the more elaborate methods of sampling are discussed, and costs are given where available.

49 Smith Margaret Cammack Food Suggestions for Prospectors Arizona Gold Placers and Placering: Univ. of Arizona Bull. vol. 3 no. 1 Jan 1 1932 pp 96-98

Weight of Placer Gravel

Placer gravels vary greatly both in weight per cubic yard in place and percentage increase in volume on being loosened. Yet in making estimates of yardage and value it often is necessary to use some factor to convert volume in place into loose volume or into tonnage. In common placer terminology "heavy" gravel indicates coarse rather than weighty material. The weight is greater in tight or cemented than in loose ground, and it increases with the proportion of large boulders and heavy rock material such as diorite, greenstone, or hornblende schist. The amount of moisture present likewise affects the weight.

Three contiguous samples taken by the authors from the same bed of tight, fine clayey gravel overlying a pay streak in the Greaterville district, Arizona, indicated weights of 3,450, 3,540, and 3,000 pounds to the cubic yard, respectively, or an average of 3,300 pounds. A sample of clean gravel with 30 to 40 percent cobbles over 2 1/2 inches in diameter, from another gulch in the same district, weighed 3,600 pounds to the cubic yard in place. The samples contained 5 to 8 percent of moisture. The expansion of the first three samples was 50, 54, and 33 percent, respectively, an average of 46 percent. The last sample expanded 17 percent.

The gold-bearing river gravel at the pit of the Grant Rock-Service Co., Fresno, Calif., weighs 2,850 pounds per cubic yard. Some engineers in calculating tonnage allow 3,000 pounds to the cubic yard, bank measure. Handbooks give weights per cubic yard ranging from 2,600 to 3,650 ponds. An average weight probably is between 3,000 and 3,300 pounds to the cubic yard.

Sampling Natural Exposures

Whenever a gold pan is used the result not only proves or disproves the presence of gold but also usually shows accurately the amount of gold contained in the sample chosen. Panning along a creek bed thus is the most elementary method of sampling a placer deposit. However, the samples commonly are taken so as to render the quantitative results worthless.

A gravel deposit of much size seldom can be sampled directly from its natural exposures; but a few creek banks, steep-sided gulches, or old excavations such as hydraulic pits may be available, in which case certain precautions should be taken to get true samples. First, the vertical extent of gravel to include in a given sample should be determined. If hydraulicking is to be done, the whole depth of gravel ordinarily is included in one sample, except when it is planned to pipe off the barren overburden to waste, in which event it is desirable to know the depth of barren material and samples may be taken of each distinguishable stratum. If drifting is planned, only the lower, economically minable gravel need be sampled. After the location and extent of a sample cut have been decided, care must be taken to have equal quantities of gravel from all points along its length. The best way to do this is to cut a channel or groove of uniform shape and size from top to bottom of the sample distance. Enough such samples must be taken to prove the continuity of the "pay streak."

Mechanically the procedure of sampling a gravel face resembles closely that pursued in lode mines. A pan and a pick are the requisite tools. The bank should be trimmed well and cleaned along the sample line to eliminate effects of surface weathering. The pan may be used to catch the loosened material, or a canvas may be spread on the ground. If conditions favor it, a measurable channel or groove should be cut so that the volume taken can be measured. Otherwise, the only alternatives are to use a factor for pans per cubic yard or to measure the loose gravel in a box which has been calibrated in terms of bank measure.

Drifting

Drifting is a common method of prospecting a deep placer deposit when conditions are favorable. Methods and costs of drifting are discussed briefly under Drift Mining in a subsequent paper.[50] The cost of driving a small drift in placer gravel ordinarily ranges from $2 to $6 per foot. Difficult ground conditions or excessive water may increase the cost to $10 or $15 per foot.

For sampling purposes the gravel usually is taken to the surface and concentrated in sluice boxes. If an old drift is being sampled various methods may be followed. If the ground will permit, the most satisfactory method is to slab 1 or 2 feet from the side of the drift and wash the gravel thus broken. If not, vertical channels may be cut on one or both sides of the drift at intervals of about 5 feet. If the latter is done, the volume of the sample cut may be measured, which is preferable under most conditions, or a factor may be used for reducing loose measures of gravel to solid measure, which facilitates taking the sample but introduces some uncertainty and leads to carelessness in sampling. If values are to be expressed in cents per ton it is still necessary to decide what conversion factor to use in making estimates of tonnage.

Test-Pitting

Test pitting and trenching are applicable only to gravels so shallow that a man can throw out the dirt by hand. The best procedure is to mark out the area of the pit on the surface, making it rectangular, as small as convenient, and preferably in dimensions of even feet such as 2 feet wide and 3 or 4 feet long. Then it should be excavated to bedrock with smooth, vertical walls. Sometimes a cleared space is prepared and the dirt thrown on the bare ground, but in view of the greater difficulty and possible small error involved in rehandling the dirt it is better to shovel it onto a canvas or board platform or into a receptacle. If a large boulder projects into the pit no correction of the theoretical volume of gravel should be made, regardless of whether or not the boulder is removed or allowed to remain in place, as obviously it cannot be ignored in mining operations and is the equivalent of so much barren gravel. The gravel taken from the pit may be thrown into one or more piles, depending on whether or not information is desired regarding one or more individual strata. Sometimes alternate third, fourth, or tenth shovelfuls are used for the sample to reduce the volume to be panned or otherwise concentrated. This should be avoided whenever practical because of the possible error introduced.

The cost of sinking test pits or running trenches in earth and gravel has been noted often enough for fair generalizations to be established. Gardner[51] states that opencast work in placer ground costs from $0.40 to $1.00 per cubic yard, depending on the nature of the ground and on wages. Wages for such work then ranged from $3 to $4 per 8 hours. Furthermore, a man should be able to pick and shovel about 8 cubic yards of fairly loose gravel in 8 hours. In test pits the worker's efficiency would be lowered somewhat by the cramped quarters and by the care necessary to square out the corners and trim the sides to vertical planes.

Near Skull Valley, Ariz., an area of shallow gulch placer ground was being sampled by test pits in 1932. The gravel was moderately fine and loose, angular to subangular wash, 2 to 6 feet deep. Mexican laborers at $3 per day were able to dig an average of five test pits

---

50 Gardner E. D. and Johnson C. H. Placer Mining in the Western United States: Part III - Dredging and Other Forms of Mechanical Handlin.. of Gravel and Drift Mining: Inf. Circ. 6788 Bureau of Mines 1934.

51 Gardner E. D. Cost of Mine Openings: Eng. and Min. Jour. vol 100 Nov 13 1915 pp. 791-794.

per 8-hour man-shift. The pits averaged 2 feet wide, 3 or 4 feet long, and 3 1/2 feet deep. The cost of digging the pits was therefore about $0.65 per cubic yard.

The gravel was shoveled first onto the ground beside the pit. Usually, all gravel was hauled by truck to the washing plant, an average distance of about three fourths mile. If a pit produced more than about 1 yard of gravel the sample was reduced in size by quartering. The sampling was done near a well on Copper Basin Wash. The gravel was shoveled from the truck directly into the hopper of a small screening, washing, and concentrating device mounted on a trailer. A 1-inch centrifugal pump on the trailer took about 15 gallons of water per minute from a large tank on the hillside and forced it through the sprays of a double trommel screen, in which the gravel was washed; all material over about one eighth inch in size was rejected. The fines were concentrated in a patented 12-inch centrifugal bowl. The plant was said to have a capacity of 2 to 3 yards per hour. It was driven by a 1 1/2-hp. gasoline engine. Water was pumped from the well to the storage tank by a 3 1/2-hp. gasoline engine.

About 12 men were employed - 1 at the washer, 2 on each of two trucks, and 7 or 8 digging pits. Wages were $3 per day. About 25 samples were taken and washed per day or 470 during a period of about 3 weeks. The totWl cost was $1,000, itemized as follows:[52]

Labor	$705.70
Trucks	96.00
Lumber	12.85
Sacks, canvas, etc.	24.50
Supplies and repairs for washing plant	25.75
Gasoline, oil, and grease	53.35
Rental of water tank and pipe	15.00
Miscellaneous supplies and expense	66.85
Total	1,000.00

The spacing of test pits depends on the nature of the deposit. If the pay gravel occurs in narrow channels the best plan is to space the pits in lines across the channels, as is done with churn-drill holes when sampling dredge ground. The holes must be close enough to yield an average value which represents the average value of the channel at that point. In practice the spacing ranges from a few to 50 or 75 feet, depending on the uniformity of results. The transverse lines of pits theoretically should be placed close enough to show either fairly uniform values from line to line or a reasonable upward or downward trend of gold content along the channel. Unfortunately, this is seldom possible, and the usual practice of spacing the lines from 100 to several hundred feet apart is a compromise in the engineer's mind between the cost of sampling and the need for accurate results.

Shaft-Sinking

Shaft sinking is the usual method of testing placer ground. Prospect and sampling shafts, unless intended for later use in drifting or mining and unless exceptionally deep (75 feet or more), are sunk as small as practicable. The usual section is rectangular and 3 by 4 to 4 by 6 feet in size. Round timber 4 to 6 inches in diameter which is available in most districts, is commonly used to crib shafts in loose gravel. In gravel tight enough to stand safely without lagging the only timber necessary is stulls set to hold the ladder.

52 Information supplied by W. R. Shanklin Oatman Ariz.

A hand windlass is the usual means of hoisting, using a light steel, 2-cubic-foot bucket and a 3/4- or 1-inch-diameter manila rope, as ordinary wire rope is unsuitable for a windlass; 75 to 100 feet is the maximum depth at which such equipment can be used most efficiently. For greater depths a power hoist of some kind should be installed. Large boulders can be lifted with the ordinary hand windlass if it is provided with long cranks; as much as 800 pounds can be raised by two men. Such feats, however, are considered dangerous because of the general absence of safe brakes or catches on windlasses, and the possibility of killing or injuring the operator if he loses control of the crank handle.

On Bear Gulch, north of Bearmouth, Granite County, Mont., a 4- by 6-foot shaft was sunk to a depth of 32 feet in 1932 at a cost of $3 per foot. The shaft was largely in fine, mucky, loose, angular gravel. It was cribbed solid with 6-inch round timber. A crew of three men averaged 6 feet per day. Wages were $3.50. Little or no water was encountered, and no blasting was required as the ground could be loosened by pick.

On Sauerkraut Creek in the Lincoln district, northwest of Helena, Mont., one man working alone sank a 4- by 5-foot shaft through dry, loose, hillside wash and moderately compact bench gravel at the rate of 3 feet per day to a depth of 30 feet, cribbing the shaft solidly with small round timber.

A 100-acre tract of possible dredging ground in the Pioneer district, Powell County, Mont., was sampled in 1931 or 1932 by sinking shafts to bedrock at 400-foot intervals, or one shaft to 3.6 acres.[53] The gravel was moderately fine, lacked large boulders, and was compact enough to stand without support in 4-foot-diameter shafts, except through a 4- or 5-foot surface accumulation of hydraulic tailings. The depth to bedrock was 10 to 65 feet and averaged 35 feet.

A crew of four men contracted to sink 1,000 feet of shaft at the rate of $1 per foot. They worked very hard and on long shifts, with the result that their earnings averaged $3.50 per day. Two of the men worked individually at starting shafts and putting in the collars and enough board lagging to hold back the loose surface wash. When the shaft became too deep to throw material out by hand the digger would leave it and start on the next. Then the other pair of men installed a hand windlass and sank the shaft to bedrock. The shaft at the surface was cribbed about 4 by 4 feet square; but a short distance down the cross section was converted to circular, with a 4 1/2-foot diameter. Under favorable conditions two men would sink 10 feet per day.

The sampling was done later by an engineer and three assistants. A portable windlass was used, which could be moved in a truck from shaft to shaft. The engineer cut all samples, taking a channel 4 inches wide and 2 to 5 inches deep straight down one side of each shaft. The loosened dirt was caught on a canvas placed in the bottom of the shaft. Distinct strata were sampled separately; the bottom few inches of rather coarse material was relatively very rich. The surface tailings which were consistently barren were not sampled. It was not necessary to dig into bedrock which, although soft, was barren. Usually 3 or 4 samples were taken per shaft. The crew of four men with a truck could sample three 40-foot shafts per day.

Once or twice, where a shaft had caved in before being sampled, it was necessary to sample the dump. This was done by hand-shoveling, throwing every tenth shovelful onto a canvas on which it was rolled and quartered.

The sample was measured in a box about 18 inches square and 18 inches deep, which was calibrated against a measured excavation in solid gravel. It was found that this gravel expanded about 20 percent upon being dug. Each sample finally was washed in a rocker. One rockerman, with the engineer in charge acting as waterman, could wash and clean up about 10 samples per day, averaging about 1/10 cubic yard each.

---

53 Information given by A. V. Corry under whose direction the sampling was done.

This ground was sampled very cheaply. The cost for the shaft sinking was only $10 per acre. The contract of $1 per foot could not be duplicated in normal times. The contractors, however, had almost no expenses except food, as all the timber needed was picked up locally for little or nothing; and their tools consisted of shovels, hammers, and picks. No blasting was done. The sampling cost is not known but from the data supplied could be estimated on a basis of ordinary labor costs as perhaps $0.40 per foot of shaft or $4 per acre.

Many shaft-sinking costs are available in the literature of placer mining. According to Janin,[54] they range from considerably less than $2 to as high as $25 per foot. He cites the instance of a Colorado dredging field, where with labor at $1.50 to $1.75 per day, more than 100 shafts 5 to 30 feet deep were sunk at a cost, excluding sampling, of 24 cents a foot. Representative shaft-sinking costs are given and commented on by Bigelow in the section on churn-drill sampling.

An excellent description of a placer examination is contained in a recent article by D. L. Sawyer.[55] The property was a typical desert dry placer in the Weaver-Rich Hill district, Arizona. The gold of the district ranged from very coarse to fine but averaged about flaxseed size. The gravel was loose and contained so many large boulders that sampling was difficult. It was decided therefore to sink shafts and wash all or most of the gravel excavated. The shafts averaged 3 1/2 by 5 1/2 feet in section and 12 feet in depth. Most of them had to be cribbed, which was done with framed, 2- by 12-inch plank. The shafts were sunk a few inches into bedrock which was cleaned carefully with a whiskbroom and scoop. The cost of shaft sinking was $3 to $6 per foot with hired labor but on contract was $2 per foot for the first 10 feet and $3 for the second 10. The contractors were said to have earned $5 per day.

The dirt from each 5 feet of shaft ordinarily constituted a sample, except where the interval was shortened to separate different strata of gravel. Boulders more than 6 inches in diameter were thrown aside, and the remainder of the gravel was shoveled into 1-cubic-foot wooden boxes and hauled by truck to the washing plant. Careful measurements of the shaft at each sample interval gave the dimensions needed to calculate the yardage of the sample. It was found that in ground lacking boulders over 6 inches in size 1 cubic yard filled 50 boxes, indicating an expansion of 85 percent, which may have been caused partly by failure to fill the corners of the boxes.

The washing plant, said to have a capacity of 6 to 10 yards per day, consisted of a shoveling platform, a hopper and set of screens, a piped water supply from a near-by mine, and a line of four sluice boxes. The boxes were 8 feet long, 10 inches wide, and 6 inches high, made of 1-inch, clear, surfaced lumber, and set in steps so that the material from one dropped into the next. This arrangement obviated the necessity for tight joints and permitted ready adjustments of the grade of each box. The upper box was fitted with a 4-foot riffle consisting of a 1-inch board with 1-inch holes bored at 3-inch intervals. The second and third boxes had transverse riffles 1 1/2 inches high and 8 inches apart. Burlap covered with expanded metal lath was used in the fourth box. The grades of the boxes were adjusted so that the riffles "boiled" properly and the black sand, which averaged about 20 pounds per ton, did not pack; in sluicing one sample the slopes were 1 1/8 inches per foot for the first, second, and third boxes and 1/2 inch for the fourth. The grades were altered to give the best recoveries with different types of gravel. No quicksilver was used. In a run made with a 25-cubic-foot sample, previously tested and found barren, then loaded with about 30 grams of placer gold, 60 percent of the gold was recovered in the first box, 36 percent in the second and third, and less than 0.2 percent in the fourth; about 4 percent was lost.

---

54 Janin Charles Gold Dredging in the United States: Bull. 127 Bureau of Mines 1918 p. 30.

55 Sawyer D. L. Sampling a Gold Placer: Eng. and Min. Jour. vol. 133 July 1932 pp 381-383.

The gravel was shoveled by hand onto the upper screen where it was washed by a flow of 50 gallons per minute coming through a perforated 2-inch pipe. Plus 1-inch material was raked off at this point and plus 1/2-inch material, from the lower screen, the balance falling onto the apron of the sluice, whence the flow of wash water carried it over the riffles. A 1/4-inch screen was used for treating clayey gravel in place of the 1/2-inch screen to insure complete disintegration.

Concentrates from each clean-up were panned and dried; the black sand was separated from the fine gold by a magnet, followed by blowing on a smooth surface or by amalgamation. After the gold was weighed on a balance to the nearest 0.01 mg, that from each sample was bottled and marked with a label showing the number of the shaft, depth in feet represented, volume in cubic yards, weight in milligrams, and value in cents.

At Gold Gulch near Bowie, Ariz., about 200 prospect pits were sunk in 1932 and 1933 to determine the yardage, nature, and value of an area of typical dry-placer ground. The deposit ranged from 3 or 4 to about 30 feet in depth and consisted of a few inches to 2 or 3 feet of clayey soil underlain by 1 to 6 feet of caliche (lime-cemented gravel), which in turn was underlain by tight, angular to subangular gravel containing a large percentage of rock and many boulders 6 inches to 2 or 3 feet in diameter. The bedrock was granite. No water was encountered in the test pits.

A gasoline-driven, 1 1/4-cubic-yard shovel already on the property was used to scoop out a trench at the site of each pit; the trench was 4 or 5 feet wide and about 5 feet deep at the center. From this point a 2 1/2- by 4 1/2-foot shaft was sunk to bedrock. Very little blasting was done, most of the gravel being picked, shoveled into small buckets, and hoisted by a hand line. No timbering was necessary. The shafts were placed 100 feet apart in lines spaced at 300-foot intervals along the gulch. No systematic sampling was done while the pits were being dug. When shaft-sinking was finished samples were taken by cutting vertical channels in one end and one side of each pit.

The work was started on company account, but later the contract system was substituted and progress was more rapid. The power-shovel work was done on a basis of 26 1/2 cents per vertical foot of excavation, the contractor furnishing the fuel for the shovel. Sinking the pits was paid for at the rate of $0.75 per foot to a point 9 feet below the bottom of the shovel cut (that is, the approximate limit of throwing out material by hand-shoveling) and $1.50 per foot from that point to bedrock. Some of the contractors earned $6 per 8-hour shift.

Exploration of the type of placer deposit that is worked by drift-mining often involves sinking one or more shafts. On Quartz Creek, near Rivulet, Mont., in 1932, two men sank a 5- by 7-foot prospect shaft 12 feet in 6 days, cribbing it solidly with 6-inch split cedar. The gravel was chiefly fine, well-washed material, with few boulders too large to handle without blasting. The first shaft was made large enough to use as a development shaft if pay gravel was found. At a depth of 12 feet, where sinking was halted by a considerable flow of water, an ingenious pumping plant was built. Power was provided by a homemade, overshot water wheel and transmitted through an endless wire-rope drive to the pump, which was set up at the collar of the shaft. The water wheel was of wood and steel, 5 feet in diameter and 3 feet wide, and was said to be capable of developing about 15 hp. The pump consisted of rods working in a 2-inch pipe, connected at the bottom of the shaft to a 4-inch pump cylinder with check valve and strainer. The pump rod was given a 10-inch stroke by an arm connected to one crank pin of an old 4-cylinder automobile engine from which the cylinders had been removed. As a separate pump unit could be driven from each crank pin, provision was made for three more pumps should they be required.

The shaft was completed to bedrock at a depth of 18 feet, requiring a total of 15 days, or thirty 8-hour man-shifts. After some drifting on bedrock three more prospect shafts were sunk in a search for the bedrock channel. These were 5 by 5 feet in the clear, cribbed solidly like the first, and 14, 18, and 18 feet deep, respectively. In this work the same crew averaged 2 feet per day.

Before completing no. 1 shaft it was necessary to install a second pump column; the two units then handled easily a flow of about 30 gallons per minute. In no. 2 shaft only one pump was needed, but in no. 3 shaft a flow of about 75 gallons per minute made it necessary to add a third pump and increase the speed to 58 strokes per minute. In sinking no. 4 shaft two pumps with a speed of 32 strokes per minute handled the water. When crosscutting was begun a third unit was added, the speed was increased to 40 strokes per minute, and finally the fourth unit was added. It was found best to keep the speed of the pump low.

A much deeper-lying deposit near Angels Camp, Calif., was prospected by both drilling and underground work.[56] The latter necessitated a shaft 167 feet deep, the sinking of which is described in a subsequent paper[57] under Drift Mining. It cost $39.50 per foot.

Drilling

The sampling of placer deposits by drilling, or, more precisely, drive-pipe sampling, is discussed in detail by Janin.[58] He states that the Keystone no. 3 traction machine is used generally in California. The usual casing is 6 inches inside diameter and 3/8 inch thick, in 5- to 7-foot sections. Drilling without casing is not good practice, and high values in a hole so drilled cannot be accepted. The bit and stem weigh 80C to 1,000 pounds. The cutting shoe usually is about 7 1/2 inches in outside diameter. Theoretically this dimension is the diameter of the cylinder of material excavated, hence it determines the yardage per foot of hole drilled. A sand pump is used to lift the loosened gravel or sludge from the hole, usually after each foot of drilling when in pay dirt. The casing is driven ahead of the tools except when boulders or cemented gravel prevents. The casing is recovered after finishing the hole.

The Empire drill has been used sparingly in this country; in foreign fields, however, it has been used extensively. It is a man-power rig, consisting of a light string of tools working inside a heavy casing (usually of 4-inch pipe) that is fitted with a toothed cutting shoe. Men or animals at the end of a long sweep turn the casing, which sinks into the ground under the weight of four men standing on a platform attached to its upper end and revolving with it. In firm ground this process is hastened by the use of a heavy weight or "jar" operated by the men on the platform. A spring pole may be used to carry the weight of a long string of tools. A sand pump is used to bring the sample to the surface. Peele's Mining Engineer's Handbook (p. 327) cites costs of $1 to $3 per foot for this work. The great advantages of the Empire drill are its light weight, low first cost, and general adaptability to remote regions and unskilled labor. These advantages are offset by the relatively smaller and less accurate samples obtainable with a 4-inch cutting shoe compared with a 6- to 8-inch shoe and by its inability to perform satisfactorily in deep or very tight gravels. With these drills 5-foot samples commonly are taken, and the sludge is measured for correction purposes in a box calibrated on a basis of 25- to 50-percent expansion of the gravel.

In an article on dredging and resoiling in Japan, Little[59] notes that the area being dredged was prospected by Empire drills, with holes at the corners of 360-foot squares. This

---

56 Steffa Don Gold Mining and Milling Methods and Costs at the Vallecito Western Drift Mine Angels Camp Calif.: Inf. Circ 6612 Bureau of Mines 1932 pp. 6-7.

57 Gardner E. D. and Johnson C. H Placer Mining in the Western United States: Part III. - Dredging and Other Forms of Mechanical Handling of Gravel and Drift Mining: Inf. Circ. 6788 Bureau of Mines 1934.

58 Janin Charles Gold Dredging in the United States: Bull. 127 Bureau of Mines 1918 pp. 30-49.

59 Little H. S. Japanese Gold-Dredging Enterprise: Eng. and Min. Jour. vol. 130 Nov. 24 1930 pp. 513-514.

is equivalent to 1 hole to 3 acres. The maximum depth of dredging ground was 33 feet. The sampling evidently was considered sufficient basis for the installation of an expensive plant, comprising a 10-cubic-foot American-built bucket dredge and an electric power plant.

Regardless of the type of drill used, the sludge sample taken from each foot or several feet of drilling usually is treated in a rocker, and the concentrates are panned to recover the gold. The colors are classified by eye according to size, usually into three groups, and a record is made of the number of each. Generally the gold is then amalgamated with a small quantity of mercury, which at the completion of the hole is dissolved in nitric acid. The gold is then weighed to give the total yield of the hole. Frequently the tailings from rocking and panning are saved and reconcentrated at the completion of the hole to detect and recover gold lost during the first washing.

Samples of placer gravel should not be assayed for gold by the usual fire methods used commercially for lode-gold ores. A pan or rocker in expert hands will recover all or more of the gold content of the gravel than can be recovered by any known device so far used successfully in placer mining.

Systematic records of the results of drilling are essential to the subsequent calculation of values and yardages. Figure 2 shows a drill log, or summary, which provided for a complete record of the information gained from a single hole.[60] It was printed on 8 1/2- by 11-inch loose-leaf sheets.

In the work on which this record form was used the panners were trained to classify the gold particles of approximately 1 mg in weight as no. 1 colors. Eight fine traces, or four coarse traces, were supposed to equal one no. 1 color. Four no. 1 colors equalled one no. 2 color, and four no. 2 colors were equal to one no. 3 color, or 16 mg. Colors heavier than no. 3 were weighed separately and calculated accordingly. In practice, the weight of the panner's no. 1 colors, as checked by the final weighing of the gold from each hole, was generally found to be between 0.75 and 1.5 mg. This classification of colors was made as a convenience in estimating the distribution of the gold through the strata. Data recorded by the driller in his drill logs comprised the times of starting and finishing, the number of different sizes of colors found at each pumping, notes on the nature of the gravels, depth measurements, and measurements of core rise and sludge volume. The last was taken by dumping the sludge into a cylindrical container of the same inside diameter as the casing and noting its height. Notations were also made of the colors obtained by repanning the tailings and by panning the settled sludge and the plug of gravel taken from the casing upon pulling it, after the hole was finished. The engineer in charge transcribed these items onto the sheet and proceeded to make his calculations. The measurement of core rise, usually about 15 percent within the casing, and of sludge volume both were used to calculate the "weighted ratio" (0.81), which indicated that an excess of material was taken from the hole; hence, the weighed quantity of gold recovered was reduced in the calculation from 31.55 to 25.55 mg. As 282 feet of hole with 4-inch casing was taken as equivalent to 1 cubic yard, the gold content of the gravel in cents per cubic yard was calculated by dividing 282 by 28.0, the depth to bedrock, multiplying by 25.55 and again by 0.0530, the value in cents per milligram of gold 795 fine (with gold at $20.67 per ounce).

A sketch of the strata penetrated was made by the engineer from the drill crew's notes and his own observation. On it were recorded the numbers of colors recovered from the gold-bearing strata, thus showing at a glance the geology and value of the gravel at that point.

Table 4 contains data on a number of placer-prospecting campaigns and was supplied by G. A. Bigelow, placer-mining engineer of San Francisco. He states:

60 Courtesy of V. V. Clark mining engineer Colorado Springs Colo.

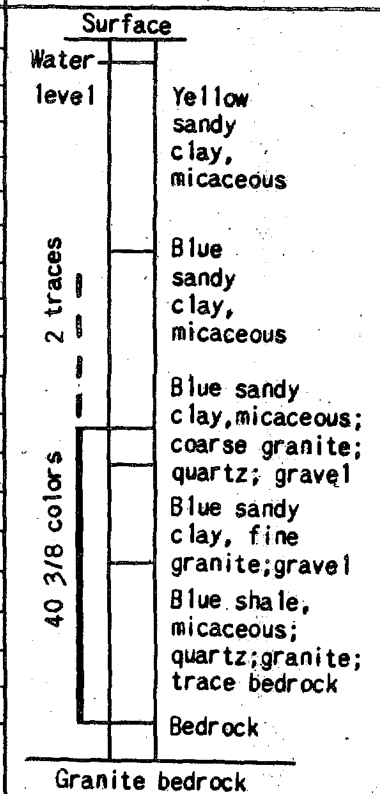
Placer Engineering Co.
Drill-prospecting summary

Property, Paradise Park Dredging Co.
Block no., E
Line no., C 101a
Elevation, 3,675 feet

Driller, F. R. Crawford
Drill, no. 3
Hole started May 19, 3:15 p.m.
Panner, Antonio Perez

Depth, feet	Character of ground	Number of colors			
		Trace	1	2	3
8.0	Yellow sandy clay, micaceous	-	-	-	-
7.5	Blue sandy clay, micaceous	2	-	-	-
1.5	Blue sandy clay, micaceous;				
	coarse granite; quartz;gravel	68	-	-	-
4.0	Blue sandy clay, micaceous;				
	fine quartz; granite; gravel	70	1	-	-
7.0	Blue shale, micaceous; quartz				
	gravel; trace bedrock	89	7	1	-
1.5	Decomposed granite bedrock	-	-	-	-
29.5					
	Repan	7	-	-	-
	Sludge	-	-	-	-
	Plug	-	-	-	-
	Total	236	8	1	

Sketch of strata



Total depth, 29.5 feet; depth to bedrock, 28.0 feet
Weight of gold, 31.55 mg
Number of no. 1 colors, 41.5
Average weight per color, 0.76 mg
Weighted weight of gold, 25.55 mg
Cents per cubic yard, 13.64
Cents per cubic yard at 85 percent recovery, 11.6

Water level, 0.5 feet
Fineness of gold, 795
Cents per milligram, 0.0530
Feet of hole to 1 cubic yard, 282
Weighted ratio, 0.81

Date, May 20, 1929

V. C. Park,
Engineer in charge.

Figure 2.—Drill-log summary sheet.

TABLE 4.- Representative placer prospecting costs: prospecting carried to conclusion and followed by dredging except in two cases

	(1)	(2)	(3)	(4)	(5)	(6)
Style of drill	Keystone no. 3	Keystone no. 3	Keystone no. 3	Shafting	Shafting	Shafting.
Power	Gasoline-driven	Gasoline-driven	Steam	Hand; no timber	Caissons hand-driven	Frozen; sunk by hand.
Year	1931	1931	1908, 1910	1906	1931-32	1907
Location	Folsom, Calif.	Waldo, Oreg.	Sumpter, Oreg.	Salmon, Idaho	Snelling, Calif.	Dawson, Yukon Territory
Months	March	May-June	Aug.-Nov.; July-Nov.	June-September	November-May	June-October.
Formation	Gravel with much clay	Deep gravel with much clay	Loose gravel 16 percent soil.	Shallow gravel	Loose gravel 20 percent soil.	71 percent frozen muck, 19 percent frozen gravel.
Operating conditions	Favorable	Favorable	Favorable	Favorable; water pumped by hand.	Unfavorable; much water; power pumps.	Very favorable; steam thawing.
Wage rate per day:						
Foreman	$9.60	$9.60	$7.67	$7.50	$8.25	$8.00
Driller	8.25	7.25	4.50			
Panner	7.25	6.25	4.50	4.17	7.25	5.65
Helper	4.40	4.00	3.00	3.00	6.45	5.65
Assistant engineer			2.67	5.67	6.45	
Rent of team per day	8.00		5.50	6.00	6.00	4.65
Rent of truck per day	5.00	5.00	[1].50		10.00	
Rent of drill per day	12.50	10.00	5.00			
Board per month	35.00	37.50	30.00	20.00	37.50	50.00
Number of holes	11	16	226	51	46	64
Total depth feet	747.5	1,386	4,400.5	697	778	1,553.5
Number of crew-shifts	37.5	66	320			
Number of man-shifts				354.5	294.5	490
Number of days	22	48	160	75	194	98
Advance per shift feet	20.4	21	12.2	1.97	2.64	3.17
Advance per day do.	34	29	24.4	15.5	4.00	15.85
Average depth per hole do	68	86.7	19.47	13.67	16.9	[2]24.27
Costs: Labor	$1,142.55	$2,671.34		$1,853.00		
Repairs	92.15	82.13				
Freight	70.00	352.65				
Travel expenses	162.67	141.14				
Drill rental	325.00	480.00	800			
Gasoline and oil	70.56	40.66				
Engineering	160.75	1,061.32	562.50			
Teams	8.00					
Rent of truck	200.00	250.00				
Total	2,231.62	5,079.24	10,169.43	2,415.50	7,666.69	5,435.47
Cost per foot	2.985	3.665	2.31	[3]3.46	[4]9.85	[5]3.50

[1] Wagon. [2] 17 muck; 7 gravel and bedrock. [3] $2.01 sinking; $1.45 sampling. [4] $7.48 sinking; $2.37 sampling. [5] Sinking and sampling.

The prospecting was carried to a conclusion in each instance. Dredging followed the prospecting in four of the six cases.

The large difference in cost of engineering between jobs no. 1 and no. 2 was due to the much larger overhead at no. 2 on account of the nature of the deposit. The distance from headquarters at San Francisco also affected the size of the engineering force necessary on the ground. Job no. 3 was in effect two jobs. The work was interrupted for a full year.

Job no. 4 was favored by an ample force of experienced gravel miners, a low water level, and not a difficult quantity to handle. Diaphragm bilge pumps were used and in some shafts too deep for suction, a second shaft sunk 10 feet deep and adjacent to the first was used for increasing the effective depth, by means of two lifts.

Job no. 5 was a very difficult undertaking and required the use of steel telescoping caissons, especially designed for the job. Gasoline-driven pumps of the jackhead deep-well cylinder type proved very awkward but most effective.

In both jobs no. 4 and no. 5 the total contents of the shafts were washed in a long-tom device by hand and in job no. 4 a check sample was cut from the side of the shaft and washed in a rocker.

Job no. 6 is a typical case of shaft prospecting in frozen ground where the gravel deposit is unusually thin or shallow. Here the conditions for shaft work were very favorable, but the high cost of living was reflected in the unit cost. Where experienced men are available, as in this case, and equipment developed by the miners for inaccessible places is at hand for their use, a very low unit cost is obtained.

Drilling in frozen ground is also very economical, owing to the speed with which the work is accomplished and the absence of casing costs. The volume of sample is quickly and accurately obtained by water measurement after completion of the hole.

The unit cost or cost per foot for placer prospecting is usually uncertain, since it depends upon the total footage. The number of holes or cross sections as the case requires may prove to be very few, and the total cost of starting and clearing up the job falls upon a small total footage. Two cases can be cited as follows: One in Colombia, S.A., cost about $25,000 for about 1,500 feet of drilling where the equipment was left behind and never salvaged. Another in central Alaska cost about the same for less than 500 feet of drilling where the equipment was not salvaged on account of cost. In such cases and in many others that constantly arise costs can be reduced to a very low figure by a preliminary examination made by an experienced and reliable placer-mining engineer and are usually represented by the engineer's fee and expenses. The number and distribution of prospect holes needed to provide the essential information can be readily determined by the preliminary survey.

Rich, deep-lying channels, such as those in California, which are worked by drift-mining, usually are prospected and developed by drifting, because the pay streaks are too small in area and too erratic in value to be amenable to drill-sampling. However, drilling may be of great value in determining the nature of the gravel, depth to bedrock, and location and grade of the deep channel. At the Vallecito Western drift mine near Angels Camp, Calif., drilling was applied in this way. Steffa states:[61]

---

61 Steffa Don Gold Mining and Milling Methods and Costs at the Vallecito Western Drift Mine Angels Camp Calif : Inf. Circ. 6612 Bureau of Mines 1932 pp. 5-6

The broad Tertiary valley east of Six Mile Creek was prospected by churn drilling to discover the location of the actual channel. A north-south line of holes was drilled across the valley, spaced as shown in table 1 and figure 1 A.[62] The holes were not drilled in regular order across the valley. The first was drilled north of the valley center in order to determine the bedrock gradient of the north rim and struck bedrock at a depth of 92 feet. This depth and the distance of the hole from the outcrop of the north rim of the channel permitted an approximate calculation of the slope of the north rim. Known elevations of the channel floor at points exposed by mining operations 2 miles west of this section indicated a depth of overburden here of about 150 feet. The second hole, drilled 281 feet to the southward, reached bedrock at a depth of 144 feet. The third hole was intended to be on the south rim and, in fact, struck bedrock at a depth of 135 feet. The fourth hole, with the information then available, was aimed for the channel trough, and the fifth and sixth were drilled to explore the width and possible pay area of channel gravels. The seventh hole was drilled north of the first hole to test for a possible split in the ancient river trough, but none was found, the hole bottoming on bedrock at 72 feet.

The drilling order, location, depth, and size of churn-drill holes are shown in the following tabulation.

Cross-section no.	Order in which drilled	Location	Lateral interval, feet	Depth, feet	Size, inches
1	7	North rim..	....	72	6
2	1	do.	135	92	6
3	2	do.	281	144	12
4	5	do.	43	142	6
5	4	Trough........	30	145	6
6	6	South rim..	31	142	6
7	3	do.	67	135	6
....	8	Shaft site	....	187	12

The drill sludge from the first 100 feet of holes in the vicinity of the channel proper was sampled intermittently. From that depth to bedrock each 5 feet was sampled carefully. The sludge was weighed first and then passed through a rocker. The concentrate was panned and the gold weighed.

The drill rig used was an assembled outfit purchased by the company and was powered by a 10-hp. gasoline engine. The crew consisted of a runner and a helper. Six and twelve inch "Mother Hubbard" type bits were used.

All of the seven prospect holes were drilled without casing, except one where it was intended to sink a prospect shaft; the plan was later abandoned, however. Two more holes were lost, one at 82 feet, due to casing trouble, and one at 57 feet when a large quartz boulder was encountered which neither hammering nor blasting would dislodge. A tenth hole was sunk at the present shaft site, as described later. All drilling was done on company account. The total footage was 1,198 feet, drilled in 278 working days, or an average of 4.3 feet per day. The average cost was $6.11 per foot. The costs of drilling the 6-inch and 12-inch holes were practically the same.

62 Table 1 and Figure 1 A are not reproduced here

As it happened, the estimates based on the results of this drilling and sampling were proved by later underground development to be fairly accurate. Drilling, however, should not be depended on for the exploration of such deposits, as the gold concentration is usually erratic. The recovery of gold from the drill sludge should be taken merely as an indication of the presence of ore in the stream gravels, the quantity and distribution of which must be determined more closely by drifting and finally by extraction.

The Continental Dredging Co., at Breckenridge, Colo., in 1930 and 1931 drilled about 1,500 feet of 6-inch holes, averaging 50 to 60 feet deep, in bouldery ground. The cost of this work was $2.87 per foot.

The cost of maintaining a single drill crew for sampling dredge ground at Oroville in 1932 was about $1,000 per month; the cost per foot of hole drilled was $2.50. The gravel was about 50 feet deep.

The property of the York Mining Co. near Helena, Mont., was prospected by drilling. The following data were furnished by N. C. Sheridan, the engineer in charge. The deposit was a stream placer, consisting of about 3 feet of gold-bearing gravel covered by a thick layer of black mud and soil, locally known as "beaver muck", to an average depth of more than 50 feet above bedrock. Holes were drilled 20 feet apart in transverse lines as much as 800 feet long across the bottom of the valley. Five such lines were drilled within 1 1/4 miles along the stream, with a total of 154 holes averaging 54 feet deep. A Star drill rig was used, powered by a 10-hp. gasoline engine which consumed about 1 gallon of gasoline per hour. The holes were cased with 5-inch pipe in 4- to 8-foot lengths. The drill crew consisted of a driller at $4.00 per shift, a panner at $4.50, and a helper at $3.00. An average of one hole per 8-hour shift was drilled at a cost of $0.50 per foot. The overburden was barren and so soft that the casing could be driven to the gravel stratum in about 2 hours. The holes were sunk 2 or 3 feet into the soft bedrock to insure complete recovery of all gold.

A dredging property on the Trinity River, Calif., was drilled about 1922 and again in 1928. The gravel was 10 to 45 feet deep, overlying slate or greenstone bedrock; the pay streak ranged from 100 to 800 feet wide. Large boulders were not numerous, and the gravel was relatively free from clay but was cemented in places. Along the river the gravel was saturated with water, and shaft-sinking therefore had been found impractical. The gold was fairly coarse and was not confined to bedrock but occurred anywhere from 1 to 15 feet above it. Requa states:[63]

> The property was sampled in three stages: (1) The southern end of the property was drilled by the Metals Exploration Co. prior to the time that the dredge was built in 1923. About 100 churn-drill holes were put down at that time. A few shafts were sunk but the greater part of the prospecting was done with a steam-driven Keystone portable rig, using standard prospecting casing with an inside diameter of 6 inches. The holes were spaced 125 feet apart in roughly parallel rows 750 feet apart. (2) The upper ground was prospected by the Shasta Dredging Co. in 1922, using the same rig that was used in prospecting the ground of the Metals Exploration Co. Over 60 holes were put down in rows about 900 feet apart with the holes spaced 150 feet from each other. (3) When the present owners were investigating the possibilities of dredging from the lower to the upper ground, 27 churn-drill holes were sunk in the 8,000-foot interval between the two previously prospected areas. The rows of holes were about 750 feet apart, and the holes were spaced 125 feet from each other in the rows. The same drill rig was used in this prospecting that had been formerly used.

63 Requa, Lawrence K., Description of the Property and Operations of the Lewiston Dredge, Lewiston, Calif.: Inf. Circ. 6660, Bureau of Mines, 1932, pp. 3-4.

The cost of prospecting can be stated only for this last stage of the work, which was done in 1928. The wage scale was as follows:

	Wages per day
Driller	$6, plus board and transportation.
Panner	6, do.
Fireman	5, no board nor transportation.

The cost was $7.23 per foot. This included the salary of the engineer in charge and the entire cost of a new string of drill pipe - which, however, was not worn out in the prospecting.

The relative merits and applications of shaft-sinking and churn-drilling are the subject of some difference of opinion. Usually, where either method can be applied, shaft-sinking is slower and more costly but more accurate; furthermore, it yields more definite information as to the physical characteristics of the gravel. Any one of these factors may assume such importance as to remove all doubt which method to use. For instance, if the values in a tract of dredging ground are close to the margin of profit by dredging, accuracy of sampling becomes of primary importance, and the cost and speed of sampling have less moment. Conversely, if the sampling is being done for purposes of dredge control on land already held, the cost probably will be the determining factor.

Some ground cannot be sampled accurately by drilling, either because it is loose and coarse, which makes it difficult to recover the gold, or because it is so wet and soft that it is impossible to be reasonably sure of the true volume of sample obtained. Under such conditions shafts should be sunk even if caisson methods are necessary. For a small job churn-drilling would be too costly because of such expenses as first cost of drilling tools and casing, moving into district, and similar overhead or general charges, unless the location is such that equipment can be rented cheaply or unless some contract driller will make a reasonably low bid. Shaft-sinking normally requires very little equipment. Any mining district will furnish miners to sink the shafts, and shaft-sampling is relatively simple and subject to repeated checking as long as the shafts remain open. On the contrary, churn-drilling and churn-drill sampling require the services of an organization whose experience with this particular work and whose reliability and integrity will permit confidence to be placed on the returns.

If it is assumed that the drilling and sampling have been done in the most careful and suitable fashion the significance of the recorded quantities of gold washed from each sample depends wholly on the ensuing computations. First, an assumption must be made as to the volume of gravel represented by each foot of hole drilled; second, it is customary to apply, sometimes in the same volume factor, sometimes as a separate calculation in the head office, a correction for losses in dredging. Only wide experience can teach the engineer what these factors should be, as he cannot see what happens at the cutting shoe of the drill, and no satisfactory determination of dredge recovery has ever been made. The drill removes, theoretically, a cylinder of gravel equal in diameter to the outside diameter of the cutting shoe and 1 foot long for each foot drilled. Actually, some gravel may be pushed ahead or aside, or some may run into the hole from outside the casing. In either case, if the deficiency or excess can be detected by observation of the sludge a correction must be applied; this is generally an empirical factor that is accurate only in proportion to the engineer's experience in such work.

Because the outside diameter of the standard cutting shoe for 6-inch casing is 7 1/2 inches the maker of Keystone drills recommends use of 0.3068 square foot to represent the average cross-sectional area of the hole drilled, this being the exact area of a 7 1/2-inch circle. The complete formula reads:

Value of gravel in cents per cubic yard equals weight of gold recovered in milligrams times value of gold in cents per milligram times the conversion factor 27 (cubic feet per cubic yard), divided by the depth drilled in feet times the cross-sectional area of the hole, or briefly,

$$V \text{ (cents per yard)} = \frac{\text{weight of gold (mg)} \times 0.06 \times 27}{\text{depth drilled (feet)} \times 0.3068}.$$

The factor 0.3068 is sometimes changed for various reasons. Other common factors are 0.2700 and 0.333, which represent the sectional area in square feet of 7- and 7 13/16-inch holes, respectively. Use of the smaller figure, known as "the Radford factor", raises the estimated value of the gravel about a tenth, whereas the larger figure has the opposite effect.

In the drilling of the area on Trinity River estimates for part of the field were based on the use of the factor 0.27. As it was believed later that these results were erroneously high the adjacent ground drilled subsequently was estimated by the use of the regular Keystone factor 0.3068.[64]

There is a great difference of opinion among engineers regarding interpretation of the results of drill sampling. Some would take virtually no account of sludge measurements. Others insist that the sludge should be measured in calibrated boxes and a correction applied accordingly. Still others make a practice of settling and measuring the mud from the run-off and at the end of drilling make use of the volume of this slime to correct their sludge volumes. Another procedure is suggested by W. W. Avery,[65] who states that the height of core in the pipe should be measured both before and after driving the pipe and a correction made based on the actual as compared with the theoretical rise. The assumption is that the rise of the core when driving the pipe should exactly correspond to the volume of material displaced by the walls of the pipe; if not, an excess or deficiency in the sample is indicated which should be compensated in estimating the values per cubic yard.

The interpretation of placer drilling and the effect of spacing of holes also have been discussed widely. In a recent contribution to the subject[66] the author argued briefly that some maximum spacing, or maximum undivided liability per hole, should be fixed for all drilling which was to be the basis of an examining engineer's report. The ensuing discussion by a large number of engineers brought out a distinct preference for relying on the engineer's education and experience but also the need for a clear statement of the method of making the estimate. It was said that the application of drill and recovery factors could affect the value of a deposit as much as 25 to 50 percent. One New York mining company was said to have had such difficulty in interpreting its engineers' reports because of the different factors used by each that they finally were instructed to report merely the basic figures, such as are contained on drill logs, after which the calculations were made in the home office.

One theory of drill sampling is that the drill holes should be spaced wide apart at first, then checked by subdividing the original network of lines or squares.[67] If the result of all the drilling is different from that of the first set of holes, still further work at closer intervals is desirable until the addition of holes fails to alter the estimate of value by an important amount. Such a procedure must, of course, be planned carefully in ad-

64 Requa Lawrence K work cited pp. 4-5.

65 Avery W. W. Estimating Drill-Hole Data in Placer Prospecting: Eng. and Min. Jour vol. 129 1930 pp. 493-495.

66 Rumbold W. R. The Valuation of Alluvial Deposits: Trans. Inst. Min. and Met. vol. 37 1928 pp. 437-541.

67 Hutton H. G. Valuation of Placer Deposits: Min. and Sci. Press vol. 123 Sept. 10 1921 pp. 365-368.

vance, and the engineer must use his judgment as to the importance of the possible error and balance this against the known cost and delay of additional drilling.

R. G. Smith believes that average drill results are low because test shafts sunk on drill holes usually give higher returns.[68] He cites 4 sets of 3 to 5 shafts each, which individually indicated values ranging from 96 to 191 percent of the drill-hole returns. By groups, however, the average shaft value was close to 140 percent of the drilling results. This was in ground so firm that the shafts stood 30 to 60 feet deep without timbering. The drill factor 0.30 was used.

At a dredge property in Idaho it was stated recently by the operators that shaft sampling had given results two or three times higher than drilling. The discrepancy was believed to be due to the presence of about a foot of loose sand just above bedrock and to the bedrock being so soft as to hinder drill recovery.

A good discussion of the accuracy of drill sampling of dredge ground is given by C. W. Gardner, a veteran dredge operator.[69] The data contained therein are shown in table 5. Excluding all areas of less than 20 acres and not considering the last four listed, which were sampled by sinking shafts, the average departure from 100-percent recovery is 27 percent plus or minus; the average gain is very nearly equal to the average loss. Gardner states:

> From all the properties above mentioned it is possible to segregate 3,743 acres, to which data given in fairly accurate reports can be applied. This combined area was prospected by means of 1,749 drill holes, or one to every 2.1 acres. The average value per cubic yard obtained by drilling was 15.4 cents and the average dredge recovery 13.55 cents, or 88 percent.

It is apparent that the sampling of dredge ground is far from a precise science. Certainly such large discrepancies would not be expected in sampling low-grade ores, and obviously no refined calculations can be based on such work. On the other hand, it must be remembered that no final proof of the accuracy of placer sampling is ever possible and that dredging involves a metallurgical operation which is subject to many disturbing effects. These facts are discussed further in the chapter on dredging in a subsequent paper.[70]

Geophysical Prospecting

In the last decade geophysical prospecting has been applied to placer mining. Where the bedrock is of homogeneous or uniform magnetic permeability and where considerable magnetite is associated with valuable gold concentrations, magnetometer surveys can outline these leads quite accurately.[71] The instruments used most for this purpose are the Thompson-Thalen, Askania Schmidt field balance, and Hotchkiss Superdip magnetometers, which measure slight changes in the intensity of the earth's magnetic field. Operation of these instruments is not difficult, but correct interpretation of the results requires experience.

Where uniform bedrock conditions do not exist the magnetic work must be supplemented with electrical work to obtain complementary data for use in interpreting the magnetic anomalies. The electrical work gives information on thickness of gravel and depth to bedrock, contour and outline of the underlying bedrock, and other structural features which may be of value in planning the mining operations. It also gives height of the water table.

---

68 Smith R. G The Discrepancy Between Drilling and Dredging Results: Eng. and Min. Jour. vol. 112 Nov. 19 1921 pp. 812-815.

69 Gardner C. W. Drilling Results and Fredging Returns: Eng. and Min. Jour. vol. 112 1921 pp. 646-649.

70 Gardner E. D. and Johnson C. H. Placer Mining in the Western United States: Part III. - Dredging and Other Forms of Mechanical Handling of Gravel and Drift Mining: Inf. Circ. 6788 Bureau of Mines 1934.

71 Laylander K. C. Magnetometric Surveying as an Aid in Exploring Placer Ground: Eng. and Min. Jour. vol. 121 Feb. 20 1927 pp. 325-328.

TABLE 5.- Tabulation of dredge recoveries as compared with estimates based on drill sampling[1]

Name location or description of tract	Date dredged	Acreage dredged	No. of holes	Spacing of holes	Acres per hole	Average depth of gravel, feet	Value cents per cubic yard: Estimated from drilling	Value cents per cubic yard: Recovered by dredging	Percentage recovery	Constant used in estimatng	Remarks
Systematically drilled tract; later two modern dredges.	...	121	50	Rows across dividing into 8 blocks.	2.4	18	16.8	15.63	93	0.333	Use of factor 0.3068 would have given close agreement
One 500-foot block in above tract.	...	...	...	...	...	...	...	...	49.4	...	Wide variation of blocks makes the closeness of average remarkable.
Another 500-foot block in above tract.	...	...	...	...	...	...	...	...	68.2	...	
Adjacent portion of above property.	...	480	130	17 lines across 400 to 700 feet apart.	3.7	16	11.61	16.44	141.6	.333	Yardage dredged exceeded estimate by 10.1 percent. All blocks gave recoveries ranging between the extremes noted
One block at end of above tract.	...	2.5	...	...	...	...	...	...	199.2	...	
Another block in above acreage.	...	55	...	...	...	...	...	...	182.7	...	Similar deposits same engineer; reason for difference in results unknown.
do.	...	8.5	...	...	...	...	...	...	104.3	...	
Property drilled many years ago.	...	180	33[2]	...	...	...	43.74	19.77	45.7	.3068	
Another part of above property.	...	118.5	38[3]	...	...	...	29.88	31.55	105.6	3068	
Small California property (careful tests made on 3 sections).	...	15.3	27	...	...	30.3	3.04	8.7	286	.27	Average estimated value on acreage basis 5.38 cents per cubic yard; dredge returns 9.28 cents per cubic yard; recovery 172.5 percent.
	...	93	42	...	...	22	6.01	10.11	168.2	.27	
	...	23.5	23	...	...	31.5	4.44	6.40	144.2	.27	

1 Tabulated from data by Gardner C. W. Drilling Results and Dredging Returns: Eng. and Min. Jour. vol. 112 Oct. 22 and 29 1921 pp. 646-649; 688-692.

2 About 1/2 not in but adjacent.

3 Including 14 on adjacent ground.

TABLE 5.- Tabulation of dredge recoveries as compared with estimates based on drill sampling - Continued

Name location or description of tract	Date dredged	Acreage dredged	No of holes	Spacing of holes	Acres per hole	Average depth of gravel feet	Value cents per cubic yard: Estimated from drilling	Value cents per cubic yard: Recovered by dredging	Percentage recovery	Constant used in estimating	Remarks
Pato property Colombia.		[4]80	[5]19				[4]38	[4]42.5	[4]112	387	Empire drill used. Factor feet of hole per cubic yard.
Nechi property Colombia.		[4]60	10	Holes in a line through center of area later dredged.			[4]70.5	[4]35.5	51.3	238	Do.
Chicksan property Korea.	January-June 1918						14.9	12.25	82	239	Do.
	July-December 1918						11.35	11.30	99.5	239	
	January-June 1919						5.55	9.15	165	239	
	July-December 1919						7.2	10.0	139	239	
	January-June 1920						7.7	12.9	167	239	
Large California property (3 tracts the third tract having been mined by 3 dredges).		173.5	[6]57		3.2	22.5	6.8	7.82	115	.27	Average (on acreage basis) of drill results 9.48 cents per cubic yard; dredge returns 9.12 cents per cubic yard; recovery 96.2 percent.
		84.0	20		4.2	44.5	5.9	6.7	113	.27	
		183.0	120		1.5	51.8	11.1	9.64	87	.30	
		106.0	41		2.6	60.6	11.2	9.44	84	.30	
		135.0	58		2.3	56.4	11.6	11.30	97	.30	
Alaska Creek		11.76		50 feet apart in lines 300 feet apart.	.25		95.0	96.3	101.4		Used a 6-inch hand drill.
Late report of a California property, one dredge.		157	76		2.1		19.1	10.4	54.4		Values corrected by sludge measurement; factor 0.27 would have given 16.5 cents per cubic yard.
California company		559 0	560		1.0	22.5	7.58	9.61	126.7	.27	
two separate tracts		420.5	146		2.9	35.9	7.25	8.18	112.9	.27	

4 Approximate.
5 In and adjoining.
6 37 shafts 20 drill holes.

TABLE 5.- Tabulation of dredge recoveries as compared with estimates based on drill sampling - Continued

Name location or description of tract	Date dredged	Acreage dredged	No. of holes	Spacing of holes	Acres per hole	Average depth of gravel feet	Value cents per cubic yard: Estimated from drilling	Value cents per cubic yard: Recovered by dredging	Percentage recovery	Constant used in estimating	Remarks
California property.	1918	19.94	11	...	1.8	32.1	10.39	10.64	102.4	...	Average on acreage basis 1918-20; estimated value 10.25 cents per cubic yard; dredge returns 11.39 cents per cubic yard; recovery 111.1 percent. In 1909 dredge recovery was 58.7 percent; in 1910 194.4 percent
	1919	20.90	10	...	2.1	34.4	9.69	9.22	95.2	...	
	1920	20.43	7	...	2.9	29.8	10.69	14.34	134.0	...	
	1908-17	229	173	...	1.3	...	10.86	12.34	113.6	...	
Yosemite Dredging & Mining Co. (Calif.)	...	7	14	In lines across direction of channel.	...	...	7.75	6.74	87	...	
Do.	...	...	66	do.	...	...	9	...	76	...	Holes ranged from 1 cent per yard or less to 66 cents per cubic yard.
Colorado property; a strip including a line of holes.	...	4.5	14	50 feet apart in a single line.	...	46	17.19	8.76	51.1	...	Clean washed sand and gravel; little clay many boulders.
Colorado property one dredge.	12 years	...	...	...	...	...	...	...	165	...	Medium fine gravel few boulders.
Do.	1 year	(7)	...	...	...	38	...	25	290	...	Do.
Do.	...	...	8	8 in a line cut lengthwise by dredging.	...	...	8.6	33	384	...	Do.
Montana property	...	300	77	Irregular	...	40	15.83	13.55	85.6	...	Most of gold in 3 feet of gravel on bedrock.
California property	Early period	152	23	...	6.6	35	38.6	16.3	42.2	.27	Medium-size gravel with much clay which hindered washing.

7 1,300,000 cubic yards.

TABLE 5.- Tabulation of dredge recoveries as compared with estimates based on drill sampling - Continued

Name location or description of tract	Date dredged	Acreage dredged	No. of holes	Spacing of holes	Acres per hole	Average depth of gravel feet	Value cents per cubic yard: Estimated from drilling	Value cents per cubic yard: Recovered by dredging	Percentage recovery	Constant used in estimating	Remarks
Alaskan creek mined by hydraulicking.	....	(8)	[9]17	....	[10].05	6 to 13	51.6	93.5	180	....	Individual shafts ranged from trace to $1.18 per cubic yard. Gold lay on bedrock covered by boulders.
Idaho dredging property.	....	44	[9]10	(11)	4.4	....	9.9	9.9	100	....	Little water in ground.
California property..	....	140	[9]51	....	2.7	....	15	15.9	106	....	
	....	40	[11]22	....	1.8	....	29	18.2	62.8	....	

8 39 000 square feet.

9 Shafts.

10 Shafts to the acre.

11 Two lines of 5 shafts each at either end of property 1 500 feet apart with shafts spaced about 320 feet apart.

12 Mostly shafts.

At one proposed hydraulic operation in northern California[72] application of these methods showed that the bedrock slope was quite different from what had been thought and that a buried fault of considerable displacement crossed the property, both facts being of prime importance to the operators. The cost of a combined magnetometric and electrical survey at this property was approximately $2.00 per acre for 600 acres. The results of the survey, made by 2 engineers and 2 assistants in about 30 days, were set forth in a surface topographic and geologic map, a bedrock contour map, and numerous vertical sections across the channel. From these data yardage estimates were made, also recommendations for sinking a few shafts and driving a tunnel on bedrock at points where a maximum of information as to gold value could be obtained.

It must be emphasized that no geophysical method or combination of methods is offered by reputable engineers as a means for determining the commercial value of placer-gold deposits. However, information gained by geophysical methods as to the physical features of the deposit are of great value both in subsequent testing and in developing and operating the property.

Bibliography of Placer Sampling and Estimation

Among the more important treatises dealing wholly or partly with the sampling and estimation of placer deposits are the following:

JANIN, C. W., Gold Dredging in the United States: Bull. 127, Bureau of Mines, 1918, pp. 26-52.

GARDNER, W. H., Drilling for Placer Gold: Keystone Driller Co., Beaver Falls, Pa., about 1923, 196 pp. Includes reprint of article by N. C. Stines, The Prospecting and Valuing of Dredging Ground: Min. and Sci. Press, Feb. 3 and 10, 1906.

WIMMLER, N. L., Placer-Mining Methods and Costs in Alaska: Bull. 259, Bureau of Mines, 1927, pp. 31-40.

JACKSON, C. F., and KNAEBEL, J. B., Sampling and Estimation of Ore Deposits: Bull. 356, Bureau of Mines, 1932, pp. 6-12.

CLASSIFICATION OF MINING METHODS

Placer-mining methods can be classified according to the several methods of excavating and transporting the gravel, or they may be designated to correspond with the various ways of saving the gold. The actual moving of the gravel from place is always the principal concern of the miner, and often the gold-saving is entirely incidental to the working of the deposit. The following classification, therefore, seems the most logical and is the one generally used by placer miners: (1) Hand-shoveling; (2) ground-sluicing; (3) hydraulicking; (4) excavating by teams or power equipment; (5) dredging; (6) drift-mining.

Combinations of methods 1, 2, 3, and 4 may be used, and one method may graduate into another. The methods may be defined as follows:

Hand-shoveling.- Hand-shoveling comprises picking and shoveling surface-placer gravels and washing the material excavated to recover the valuable minerals. The gravel may be washed at the "diggings" or transported by wheelbarrows, pack animals, carts, or trucks to the nearest available water. The general method of excavating varies little, but it may be subdivided according to the method of washing the gravel: (a) Panning; (b) rocking; (c) use of long tom; (d) shoveling into boxes (sluicing); and (e) dry washing.

---

72 Jakosky J. J. and Wilson C. H. Use of Geophysics in Placer Mining: Min. Jour. vol. 16 no. 14 Dec. 15 1932 pp. 3-5 and 29.

Ground-sluicing.- Ground-sluicing consists of moving and washing gravel by water not under a hydraulic head. The action of the water usually is assisted by picking. Boulders may be removed by hand or by derricks. A minor part of the water under relatively low pressure may be used to assist in cutting the bank.

Hydraulicking.- This method of mining utilizes water under pressure from a nozzle for cutting the gravel and sweeping it into sluice boxes, through which it passes to a suitable dumping ground. Additional water not under pressure generally is used to assist in moving the washed material through the sluice boxes.

Excavating by teams or power equipment.- This heading includes mining by teams and scrapers, power scrapers, power shovels, and drag-line excavators but not dredges. The gold may be washed in standard sluice boxes or in more complicated power-driven screening and washing plants.

Dredging.- Dredging properly is confined to mining placer deposits by excavating and washing machinery mounted on boats. The so-called "land dredges" as well as floating washing plants fed by separate dry-land excavators are included under the previous method. The only type of gold dredge operating successfully in this country is the bucket type, which excavates by means of an endless chain of buckets, screens and washes the gravel, recovers the gold in sluices, and discharges the waste material into the water behind the boat.

Drift-mining.- Drift-mining consists of working buried strata of gold-bearing gravels by underground methods. The gravels are mined in much the same manner as flat lode deposits; the excavated material is taken to the surface and washed in sluice boxes or treated in other gold-saving devices. In some instances the milling practice resembles that used for extracting gold from lode ores.

Choice of Methods

The method that involves the lowest total cost is preferable for mining a given deposit. The method chosen, however, must be compatible with existing conditions; frequently only a relatively high-cost method will be practicable. Usually natural conditions restrict the selection of a method. The most important natural factors are the quantity and pressure of water available; the slope or grade of surface and bedrock; the depth, extent, and value per yard of the deposit; the character of gravel and bedrock; and the thickness of the overburden.

For profitable exploitation of a mine the gold recovered must be of greater value than the cost of mining it. Although in the past extremely rich placer ground has been found and worked, present known surface placer deposits are of relatively low grade. For profitable operation, relatively large tonnages must be handled per man-shift; too much stress cannot be laid upon this item. All phases of the mining must be correlated, and delays must be reduced to a minimum. Complications in any part of the set-up are to be avoided whenever possible in order that no "bottle necks" may exist. The mine and plant must be so designed and laid out that operations can be maintained at full capacity throughout the season, especially where power excavators or mechanical washing plants are used. Often the difference between expected capacity and actual capacity has been considerable and has meant the difference between profit and loss.

HAND-SHOVELING

The first placer mining in the West was done by hand methods. As the rich surface deposits became exhausted other methods were substituted. Under normal economic conditions the hand-shoveling method is applied mainly to deposits that are too small to justify the capital expenditure necessary for working them by other methods. When work is scarce, however, any available gravel that will give a man a bare living may be worked in this manner. Because very little capital is required, men of small means can mine on their own account by hand-shoveling methods; in times of general unemployment, many such men go into the placer fields and search for and work surface gravels. This type of mining commonly is called "sniping."

During the summer of 1932 thousands of men, most of them inexperienced, were mining in the placer districts of the West by hand-shoveling methods. They seldom earned a "going wage" but usually got enough gold to buy food. The average earnings of experienced men were estimated to be less than $1 per day and of experienced and inexperienced together less than $0.50. Occasionally a rich spot left by oldtimers would be found and as much as $100 might be cleaned up in a few days. These rare experiences, however, were offset by many days spent searching for a place to work or by wasted effort in digging to bedrock only to find it already cleaned or barren. During the autumn of 1931 many old bars in the rivers of the Pacific coast were exposed for the first time in years on account of the extremely low water of that season. Many snipers worked profitably on these bars until the fall rains raised the water level.

Snipers look mainly for unworked ground in or around old workings, or they extend old workings that have been discontinued as being unprofitable. They reclean disintegrated bedrock in old workings or clean out crevices in bedrock in river channels. They work old bars in the rivers where reconcentration may have occurred since the early days of placering. Most of the ground worked by snipers, however, is ground that has been left by oldtimers as being unprofitable.

In dry sections of the country old channels, dry creek beds, or other gold-bearing deposits are worked, and the gravels are cleaned with dry washers.

Most of the ground in the Western States known to contain placer deposits is owned privately or held under mining locations. A newcomer into a field generally has to obtain permission from the proprietor of the land before he can begin placer mining. Sometimes a royalty is charged; at other times hand mining is permitted free, the owners of the gravel doubtless considering that the prospecting value of such work equals the small amount of gold taken from the ground.

The water of most western streams has been appropriated for irrigation or power purposes. Placer miners should consider the previous rights of others in a stream before diverting any water for placer mining. The antidebris law of California prohibits uncontrolled discharge of placer tailings into any tributaries of the Sacramento River.

Panning

Panning is both the simplest and most laborious method of washing gold from placer gravel. With a little experience almost any one can recover most of the gold in a pan of ordinary placer dirt; an experienced man can wash about 10 pans per hour. As a man can dig gravel with a pick and shovel many times faster than he can pan it only the highest-grade gravel is washed. The top dirt usually is thrown aside, and a few inches of gravel on bedrock or material scraped from crevices is panned.

The standard gold pan used in the Western States is made of stiff sheet iron; it is 16 inches in diameter at the top and 2 1/2 inches deep. The rim is flared outward at an angle of about 50° from the vertical. Smaller pans are used for testing. Frying pans or other cooking utensils may also be used for washing out gold. Before any kind of a vessel is used for panning it should be cleaned thoroughly; all grease should be burned out. New pans generally are greasy and should be heated over a fire until all grease is burned off. A rusty pan if clean can be used satisfactorily; the roughness due to the pitting of the rust may be of assistance in holding back the gold.

The pan usually is filled even with the top, or slightly rounded, depending somewhat upon the nature of the material being washed and the personal preference of the panner; it is then submerged in the water. Still water 6 inches to 1 foot deep is best. While under water the contents of the pan are kneaded with both hands until all clay is dissolved, and lumps of dirt are thoroughly broken. The stones and pebbles are picked out. Then the pan is held flat and shaken under water to permit the gold to settle to the bottom. The pan is then tilted and raised quickly so that some of the lighter top material is washed off. This operation is repeated, occasionally shaking the pan under water or with water in it until only the gold and the heavy minerals are left; this material concentrates at the edge of the bottom of the pan. Care must be taken that none of the gold climbs to the edge of the pan or gets on top of the dirt.

Nuggets and coarse colors can now be picked out readily. Cleaning the black sand from the finer gold is more difficult but can be carried nearly or entirely to completion by careful manipulation of the pan as described above, always watching carefully to see that none of the colors are climbing toward the edge. This part of the operation usually is done over another pan or in a tub so that if any gold is lost it can be recovered by repanning.

The concentrates can be dried and the black sands removed by a magnet or by gentle blowing on a smooth flat surface. If there is an excessive quantity of black sand the gold usually is amalgamated by putting about a teaspoonful of mercury in the pan. In sampling work great care is necessary that no fine colors are lost, but in mining by panning the extra time needed to make sure that no fine colors escape probably would not be justified. The loss of 50 or 100 fine colors in a day might not amount to more than 1 cent.

A standard 16-inch pan level full of dry bank gravel contains, on an avrage, 22 pounds. The quantity will vary with the moisture content and nature of the gravel. By allowing 3,300 pounds to the cubic yard (in place), 150 pans would be equivalent to 1 cubic yard. The senior author has been accustomed to using this factor in sampling placer deposits. At 3,600 pounds to the cubic yard and 22 pounds to the pan, 164 pans would be required to wash a cubic yard of gravel. At 10 pans per hour a man could pan about 1/2 cubic yard in a day. With exceptionally clean gravel a man will sometimes pan as much as a cubic yard in a day.

Rocking

More gravel can be handled per man-day by rocking, or cradling as it is sometimes called, than by panning. Moreover, the manual labor of washing a cubic yard is less. The same method of excavating the gravel is used whether it is panned or rocked. The rocker, like the pan, is used extensively not only in small-scale placer work but also in sampling and for washing sluice concentrates and material cleaned by hand from bedrock in other placer operations.

One to three cubic yards, bank measure, can be dug and washed in a rocker per man-shift, depending upon the distance the gravel or water has to be carried, the character of the gravel, and the size of the rocker. Rockers usually are homemade and have a variety of designs. A favorite design in the Western States consists essentially of a combination washing box and screen, a canvas or carpet apron under the screen, a short sluice with two or more riffles, and rockers under the sluice. The bottom of the washing box consists of sheet metal with holes about one half inch in diameter punched in it. A rocker in use at Greaterville, Ariz., was 3 feet 4 inches long and 1 foot 9 inches wide on the inside and had a slope of 5 inches. The screen box was 6 inches deep and 20 inches square inside and had a bottom of sheet iron with 1/4- to 1/2-inch holes punched about 2 inches apart. The baffle was 28 inches long and consisted of a piece of canvas. A single riffle 3/4 inch high was used at the end of the rocker. Figure 3 is a drawing of a prospector's rocker made by W. B. Young of Tucson, Ariz. The bottom of a rocker should be made of a single wide board, if one can be obtained, and planed smooth. This will greatly facilitate clean-ups. The cost of building rockers ranges from $5 to $15, depending mainly upon the cost of lumber.

After being dampened the gravel is placed in the box 1 or 2 shovelfuls at a time. Water is then poured on the gravel while the rocker is swayed back and forth. The water usually is dipped up in a long-handled dipper made by nailing a tin can to the end of a stick. A small stream from a pipe or hose may be used if available. The gravel is washed clean in the box and the oversize inspected for nuggets and dumped out. The undersize goes over the apron, where most of the gold is caught. Care should be taken that too much water is not poured on at one time, as some of the gold may be flushed out. The riffles stop any gold that gets over the apron. In regular mining work the rocker is cleaned up after every 2 or 3 hours, or oftener when rich ground is worked, if gold begins to show on the apron or in the riffles. In cleaning up after a run, water is poured through while the washer is gently rocked; the top sand and dirt are washed away. Then the apron is dumped into a pan. The material back of the riffles in the sluice is taken up by a flat scoop, placed at the head of the sluice, and washed down gently once or twice with clear water. The gold remains behind on the boards, whence it is scraped up and put into the pan with the concentrate from the apron. The few colors left in the sluice are caught with the next run. The concentrate is cleaned in the pan.

With skillful manipulation of the rocker and a careful clean-up nearly all the gold is recovered. Violent rocking is avoided so that gold will not splash out of the apron or over the riffles. The sand behind the riffles should be stirred occasionally, if it shows a tendency to pack hard, to prevent loss of gold. If the gravel is very clayey it may be necessary to soak it for some hours in a tub of water before rocking it.

When water is scarce two small reservoirs are constructed, one in front and the other in the rear of the rocker. The reservoir at the front serves as a settling basin; the overflow goes to the one at the rear where the water is used over again.

Power rockers.- The capacity of rockers may be increased by using power drives. The use of such a machine was illustrated by the operation of George Graves in the Lynn district, Eureka County, Nev., during the summer of 1932. The rocker was 49 inches long, 27 inches wide at the top, and 21 inches wide at the bottom. It was 24 inches high in front and 21 inches at the rear. The screen had 5/8-inch round holes. The gold was caught on three aprons of canvas and wood. Riffles of 1/2- by 1/4-inch wooden strips were used on the aprons. The undersize from the screen passed over each apron in turn. Nearly all the gold was caught on the first apron. The slope of the aprons was 3 inches to the foot.

The device was rocked by an eccentric arm at the rate of forty 6-inch strokes per minute. The capacity of the machine with two men working was 1 cubic yard per hour. Where gravel was free of clay the capacity was said to be as great as 3 cubic yards per hour. The cost of the rocker and the engine for driving it was $160. At $4 per 8-hour shift and 1 cubic yard per hour the labor cost of washing the gravel would be $1 per cubic yard.

A number of small machines patterned more or less after the power rocker are on the market. They usually are built of iron or steel and driven by small gasoline engines. Although of various designs they generally consist of a trommel or a shaking screen to remove coarse material, a short shaking sluice to save the gold, and a pump to circulate the water. Some of them contain a settling tank from which the solids are removed by a rake or drag. These machines have an advertised capacity of 1/2 to 2 1/2 cubic yards per hour and cost $225 to $700. No operating data are available.

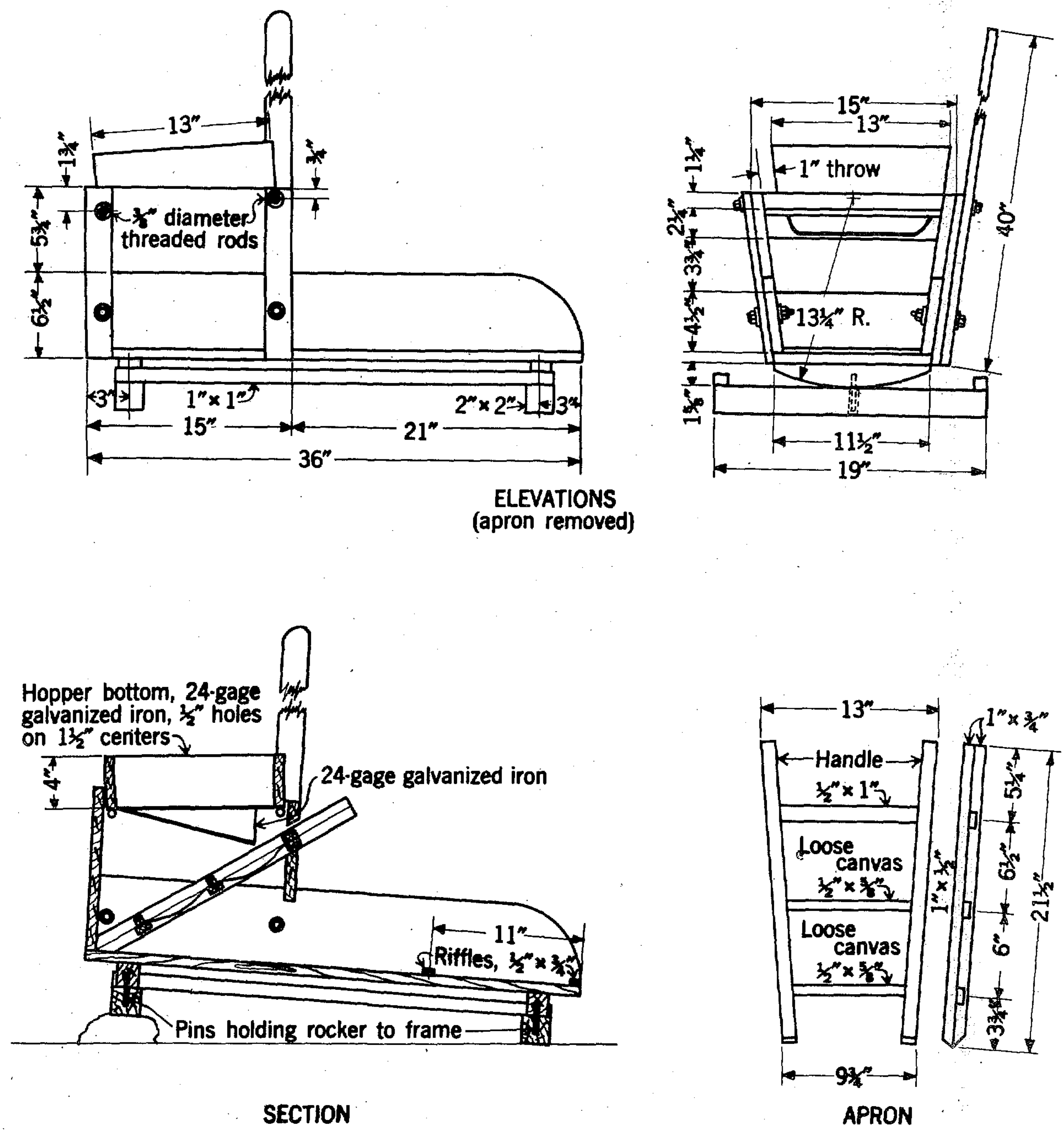


Figure 3.—Prospector's rocker.

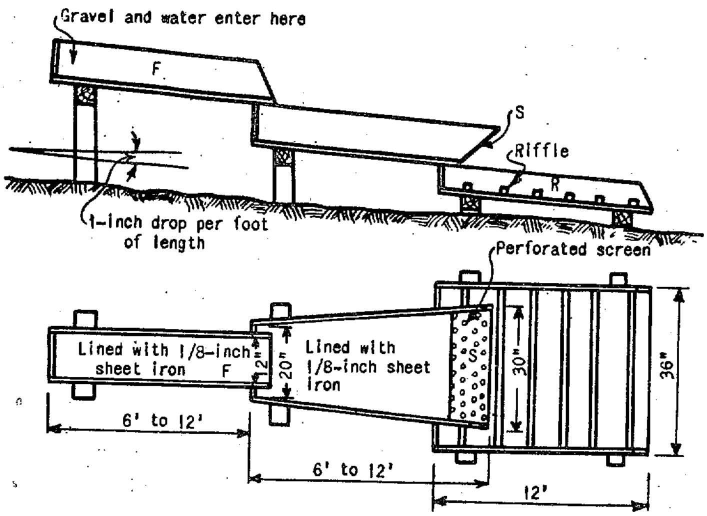


Figure 4.—Long tom.

Long Tom

A long tom usually has a greater capacity than a rocker and does not require the labor of rocking. It consists essentially of a short receiving launder (see fig. 4), an open washing box 6 to 12 feet long with the lower end a perforated plate or a screen set at an angle, and a short sluice with riffles. The component boxes are set on slopes ranging from 1 to 1 1/2 inches per foot. The drop between boxes aids in breaking up lumps of clay and freeing the contained gold.

A good supply of running water is required to operate a long tom successfully. The water is introduced into the receiving box with the gravel, and both pass into the washing box. The sands and water pass through the screen, which has about 1/2-inch openings into the sluice. The oversize is forked out. The gold is caught by the riffles. The riffle concentrates are removed and cleaned in a pan. Quicksilver may be used in the riffles if the gravel contains much fine gold.

The quantity of gravel that can be treated per day will vary with its nature, the water supply, and the number of men employed to shovel it into the tom and fork out the stones. Wilson[73] states that 2 men, 1 shoveling into the tom and 1 working on it, can wash 6 cubic yards of ordinary gravel, or 3 to 4 cubic yards of cemented gravel, in 10 hours. At times the tom is operated by 4 men, 2 shoveling in, 1 forking out stones, and 1 shoveling fine tailings away. Toms are rarely used now in the United States; where running water and grade are available a simple sluice is generally as effective and requires less labor.

A modified form of long tom has been employed for washing beach sands at Nome, Lituya Bay, Yakataga, and Kodiak, Alaska. For saving fine gold the box is set at a high gradient, 3 to 4 inches per foot, and the screened material is passed over riffles and amalgam plates.[74] The water for sluicing generally is bailed with a large dipper. The average duty per man per 10 hours for long-tom or rocker work on beach operations is 3 to 5 cubic yards.

Surf Washers

Surf washers are similar to long toms but are wider and shorter.[75] They can be used only when the surf is of proper height. They are set so that the incoming surf rushes up the sluices, washes material from the screen box or hopper, and on retreating carries it over the riffles and plates. One man can attend to two surf washers, and in one instance 8 cubic yards was handled per 10 hours.

A simple surf washer used about 1902 at Topkuk, Alaska,[76] was a riffled sluice 3 to 4 feet wide and 8 to 10 feet long, set on the sand at the water's edge so that the incoming waves washed through it to the upper end yet retreated below the lower end. The sluice was made of boards nailed to sills at either end which could be weighed down with rocks or otherwise. The sides were only 4 or 5 inches high. The riffles, similar to those used on dredges, were made in sections of about 1- by 1-inch strips and were spaced about an inch apart. The end sections were transverse riffles; the center section was longitudinal. The box preferably was set on a grade of 8 to 10 inches per 12 feet. Best results were obtained by using quicksilver in the riffles. It was stated that when the surf was strong the washer would treat as much as two men could shovel, but that at other times it had to be fed very slowly.

---

73 Wilson E. B. Hydraulic and Placer Mining: John Wiley & Sons (Inc.) New York 3d ed. 1918 425 pp.

74 Wimmler Norman L. Placer Mining Methods and Costs in Alaska: Bull. 259 Bureau of Mines 1927 236 pp.

75 Wimmler Norman L. work cited.

76 Elffner A. E. Beach Mining with a Surf Washer: Min. and Sci. Press vol. 86 June 6 1903 p. 364.

Dry Washers

Dry washers have been used for many years in the Southwest where water is scarce, especially in New Mexico where probably 3 or 4 million dollars in gold has been produced during the last century by dry-washing. A small steady production by dry washers still comes from the Cerrillos, Golden, and Hillsboro districts each year. In years when other employment is scarce the production increases. During 1931 and 1932 a considerable number of men also used dry washers in Nevada, southern California, and Arizona.

Gravel to be treated successfully by dry washers must be completely dry and disintegrated. With dry washers operations must be stopped after rain storms until the ground dries out again. The only successful dry washing up to 1932 has been on a small scale. Plants with mechanical excavators and extensive power-driven dry-washing machinery have been tried, but in the United States none has been commercially successful, partly because in large-scale work the gravel is dug faster than the sun can dry it out. Even in very dry climates the gravel is slightly damp below the surface and must be dried before it can be treated in a dry washer. Spreading the material to sun-dry or putting it through driers adds to the cost of mining. In small-scale work the gravel dries out as fast as it can be treated.

Individual workers select the material they treat with regard to both dryness and probable gold content; it is difficult to do this on a large scale with hired labor. In large-scale work, particularly with mechanical excavation, the cost of sizing the material is an important item; clay and cemented gravel introduce further difficulties.

Dry washers usually are run by hand and have about the same capacity per day as rockers of a corresponding size, but the work of operating the dry machine is much harder. When the gold-bearing material is completely dry and disintegrated, panning tests of the tailing show that a good saving can be made, except perhaps with extremely fine or flaky gold. Completely disintegrated material, however, seldom is obtained. The tops of clay streaks in the gravel are likely to be richer in gold than the gravel itself. Clay or cemented gravel seldom can be broken up sufficiently by hand to free all the gold without the use of some form of pulverizer. In a dry machine all gold included in a lump of waste passes out of the machine. As water usually breaks up all the gravel and separates the gold from the other material a better saving generally can be effected by rocking or in sluice boxes than in a dry machine.

The basic principle of the dry washer is separation of the gold from the sand by pulsations of air through a porous medium. The screened gravel passes down an inclined riffle box with cross riffles; the bottom of the box consists of canvas or some other fabric. Under the riffle box is a bellows by which air in short, strong puffs is blown through the canvas. This gives a combined shaking and classifying action to the material; the gold gravitates to the canvas where it is held by the riffles, while the waste material passes out of the machine.

The gravel is shoveled into a box holding a few shovelfuls at the head of the washer whence it runs by gravity through the machine. A screen with about 1/2-inch openings is used over the box; all stones over about 1 inch in diameter generally are discarded in mining. One man working alone fills the box, then turns a crank which runs the bellows until the gravel is run through; the process is then repeated. With two men working, one shovels and the other turns the crank. One man can treat 1/2 to 1 yard per day with a hand-operated washer where the gravel is handy to the machine. Some dry washers are run by a small gasoline engine which saves the labor of one man; the capacity of such machines is considerably greater than that of hand-operated machines.

In cleaning up, the material back of the riffles usually is dumped into a pan and washed out in water. If water is very scarce the accumulated material from the riffles may be run through the machine a second time and then cleaned further by blowing away the lighter grains of sand in a pan.

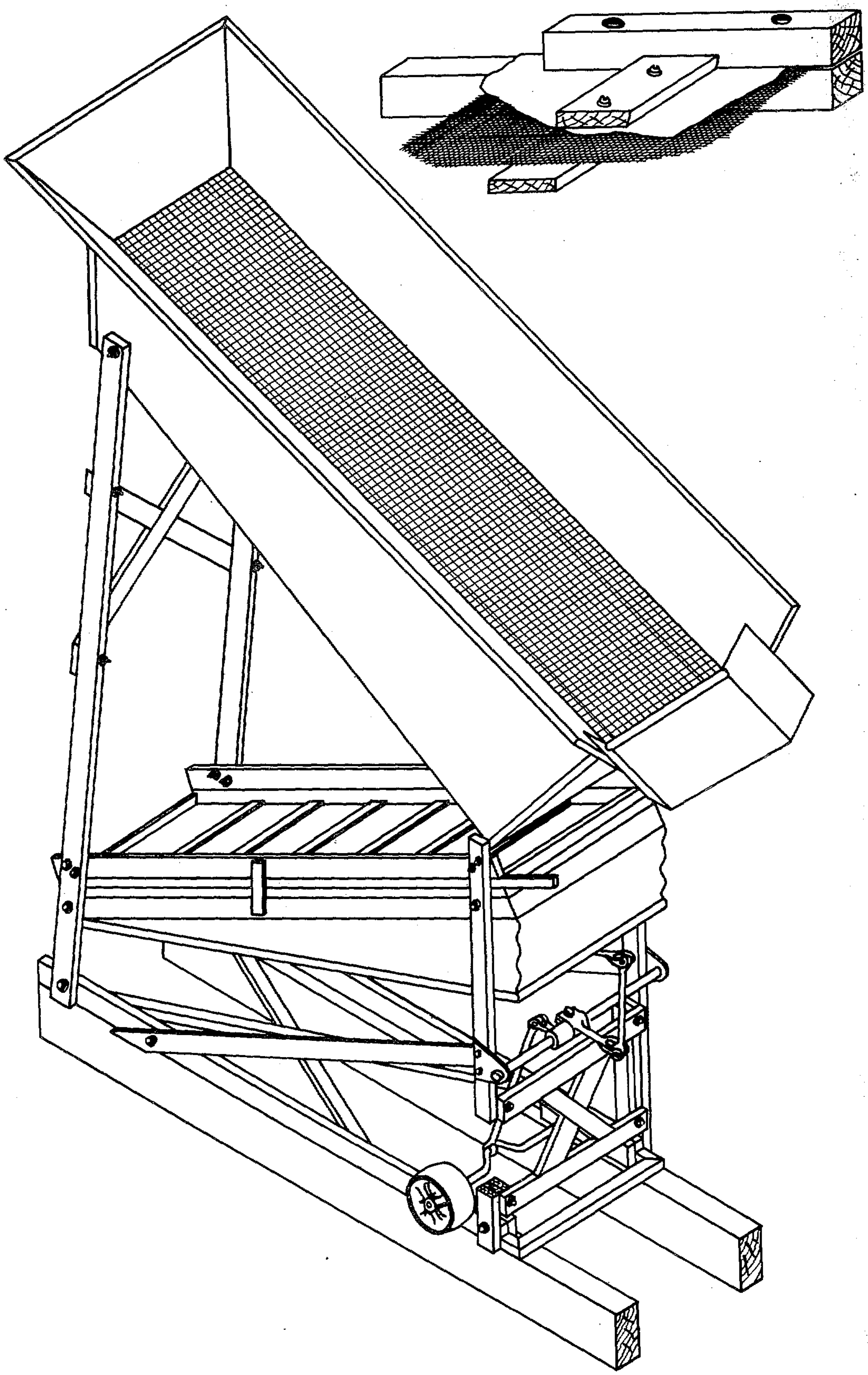


Figure 5.—Power-driven dry washer, Goler Gulch, Calif.

Dry washers usually are handmade and have a large number of designs and sizes. Figure 5 is an isometric drawing of a machine built and run by E. C. Spurr on Goler Gulch near Randsburg, Calif., in June 1932. The bellows of the machine is made of 36-ounce duck and the bottom of the riffle box of 8-ounce single-weave canvas. Spurr has found by long experience that the 8-ounce material is the best for the purpose. Silk or rayon permits a good extraction of gold, but too much dust goes through into the bellows. Heavier canvas is too tight for good separation. Copper-wire fly screen is used under the canvas. The riffle box is 11 inches wide and 40 inches long and contains 6 riffles. The type of riffle is shown in figure 5. The slope of the riffle box is 5 1/2 inches to the foot. The gravel and sand are shoveled onto a screen with 3/8-inch square openings at the top of the washer. The machine is run by a 3/4-hp. gasoline engine. The bellows is operated at 250 pulsations per minute; the stroke is 3 inches. The capacity of the machine was 1 cubic yard in 1 1/4 hours or 0.8 yard per hour. This probably would correspond to 1 1/2 or 2 cubic yards, bank measure, on account of the plus 1-inch material having been thrown out in mining. The gravel had been collected in an old drift mine and carried out in buckets; 3 hours were spent by one man in getting out the gravel.

In cleaning up after running a cubic yard through the Spurr washer the riffle box was lifted out and turned over on a large, flat, baking tin. The concentrate from the upper three riffles was first panned and the gold removed; both coarse and fine gold were saved, the total being worth about 75 cents. The lower four riffles had one good-sized color and a few small ones. Nearly all the gold was caught in the upper riffles; this indicated that a fair saving was being made of the free gold.

Hand-operated machines, however, are usually much smaller and have the riffle box set at a steeper angle.

Shoveling-into-Boxes

Mining.- Shoveling-into-boxes is a small-scale placer-mining method in which the gravel is loosened by picking and is shoveled by hand into sluice boxes. It is particularly adapted to small, shallow, moderately rich deposits where only a little water is available or where the grade is insufficient to allow room for disposing of the tailings unless the gravel is elevated. If the water supply is adequate the ground-sluicing method (described later) should be considered, and if the deposit is large enough to justify the required capital investment some other method such as hydraulicking or dredging would be preferable.

The method of shoveling-into-boxes has its chief application in the working of small deposits by men with little capital but a willingness to work hard.

The quantity of water available will influence the scale of operations and the size of sluice used. A minimum flow of 15 to 20 miner's inches (170 to 225 gallons per minute) is required for a 12-inch sluice box with a steep grade. Smaller flows than this can be utilized by reservoiring the water and using an augmented flow. A common practice followed where the quantity of water is limited is to place a grizzly or screen over the sluice and thus increase the duty of the water. A head box with sloping sides and a grizzly near the bottom may be used for receiving the gravel. In this case the oversize is forked out. In one Nevada operation the gravel first was run through a trommel to wash it and screen out all coarse material. Puddling boxes may be used if the gravel contains much clay. If the ground is of good grade it may be practicable to pump water for the sluice; the feasibility of obtaining a gravity flow, however, should first be investigated, as the mining expense due to pumping may be more than the cost of a ditch distributed over the yardage moved.

Water usually is conducted in a ditch to the sluice which is set up in the most advantageous position to begin work. Boxes are 8 to 24 inches wide and range from 3 to as many as may be required to transport the tailings to a suitable dumping ground; they are laid on sills at approximately the mean elevation of the surface of the deposit. If enough water is available 6 or 8 inches per 12 feet is a good grade with most types of gravel. If water is scarce or the gravel angular a steeper grade will be desirable; the grade should be uniform. The best type of riffle depends on the nature of the gravel and the gold rather than on the method of extraction. An end shake was given to the box at a mine in Arizona to increase its capacity. Water was hauled for washing. Riffles are discussed in a subsequent paper.[77]

The cost of boxes depends chiefly on the cost of lumber delivered to the property, which may range from $15 to $60 per 1,000 board-feet. A 12-foot box, 12 inches wide, constructed of three 1- by 12-inch boards with three sill frames of 2 by 4 lumber, requires about 45 board-feet of lumber. If clear lumber costs $35 per 1,000 board-feet the cost of a set of five boxes would be about $8.00. Material for riffles may cost little or nothing or as much as $1 per box.

After the boxes are set, shoveling begins at an advantageous point. All material of a size that will run through the sluice is shoveled in, and the oversize is thrown to one side. Boulders from the first cut should be stacked outside the pit, on barren ground if possible. The width of a cut usually is limited to the distance a man can shovel in one operation. When shoveling from any distance it is best to set boards above and on the opposite side of the box; these increase the efficiency of the shovelers. The greatest height a man can shovel into a box is 7 to 8 feet; above 5 or 6 feet the efficiency of the shovelers is reduced markedly. If over 3 or 4 feet in depth the gravel usually is excavated in benches to facilitate digging and at the same time to permit the upper layers of gravel to be raised a minimum height. Where the gravel is shallow wheelbarrows may be used. At a Nevada mine the gravel was shoveled onto a conveyor belt which discharged into a trommel. The oversize was discarded, and the undersize ran through a sluice. Where two or more men are working in the same cut the heights of succeeding benches are governed by the character of the material being dug and the distance the gravel has to be lifted. Work may begin at one edge of the deposit and proceed across it by regular cuts. The sluice may be maintained on the surface of the unworked ground or on bents on the opposite side of the cut. After the first cut the boulders are thrown onto the cleaned-up bedrock. Where cuts are run on both sides of the sluice the boxes are supported on bents as the ground underneath them is dug out. At other places the boxes may be set on bedrock and all dirt shoveled into the head of the sluice from short transverse cuts at the upper end of the pit. Work usually begins at the lower end of a deposit so that bedrock may be kept drained. The length and order of making the cuts will depend upon local conditions.

The method of working a deposit near Oroville, Wash., is shown in figure 6, A, and at Blewitt, Wash., in figure 6, B.

Experienced miners work out the ground in regular cuts and in an orderly fashion. Enough faces are provided so that shovelers will not interfere with one another. Provision is made to keep bedrock drained, and boulders and stumps are moved a minimum number of times. Cuts are taken of such width and length as to make the shoveling as easy as possible. The boxes are kept at such a height that the minimum lift of the gravel is required consonant with enough fall for the gravel to run through the boxes with the water available and to allow dump room at the tail end of the sluice. Leaks in the sluice are stopped promptly, and shoveling is done in such a manner that the sluice does not become clogged or water splashed out. Water in the pit hampers shoveling.

Several times as much gravel can be washed per man-shift by this method as by the use of rockers or long toms. Under the best conditions a good workman can shovel 15 cubic yards

---

77 Gardner E. D. and Johnson C. H. Placer Mining in the Western United States: Part II. - Hydraulicking Treatment of Placer Concentrates and Marketing of Gold: Inf. Circ. 6787 Bureau of Mines 1934.

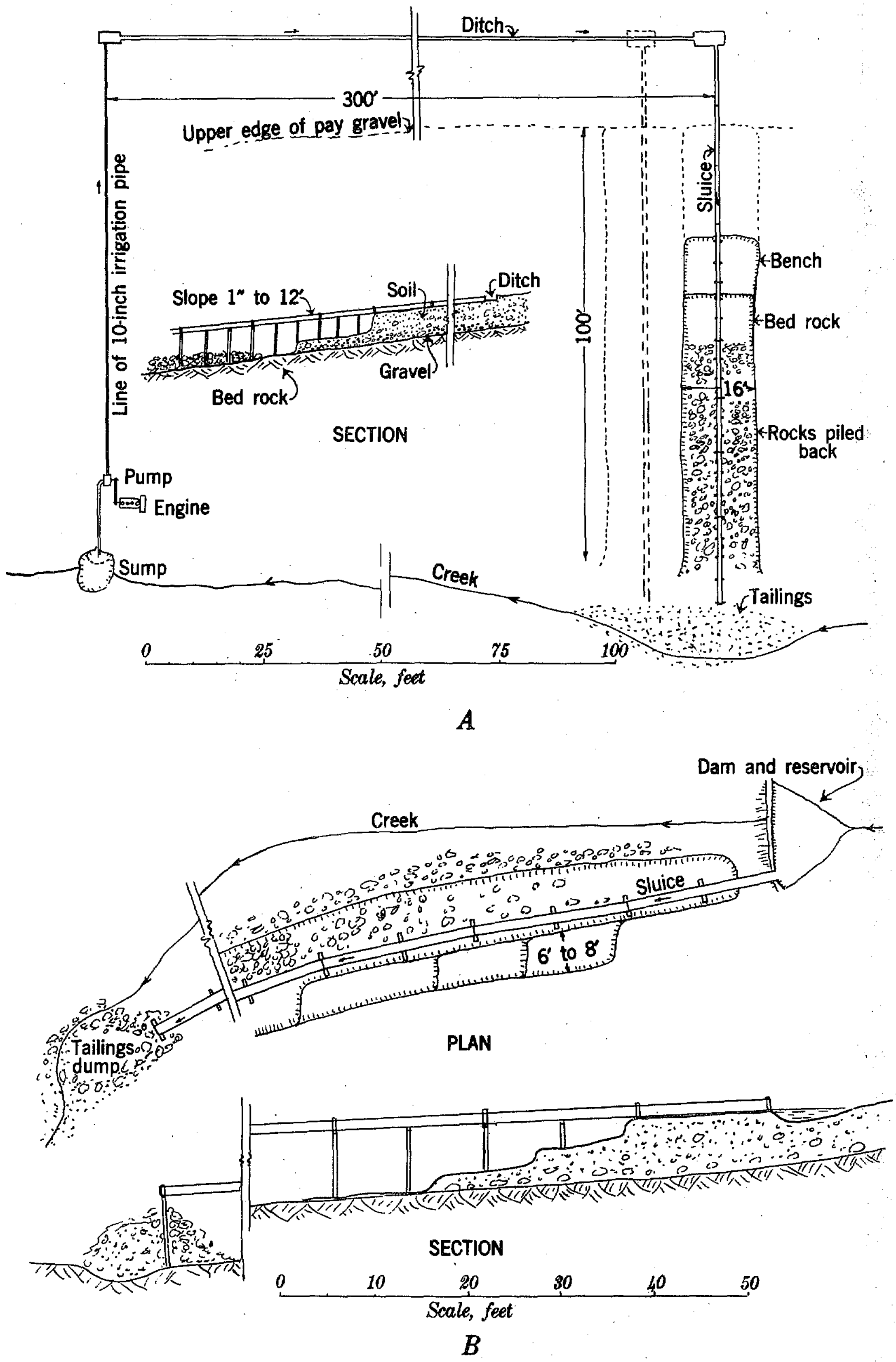


Figure 6.—Shoveling into boxes: *A*, Oroville, Wash.; *B*, Blewitt, Wash.

of gravel per shift into boxes 3 or 4 feet above bedrock. Where the gravel is tight and coarse and the bedrock rough, he may be able to shovel only a yard a day. The time necessary to construct, set up, and clean the sluice boxes and to clean bedrock reduces the average performance of men shoveling into boxes to about 4 yards per shift.

Cleaning bedrock.- When a sufficient area of bedrock is exposed it is cleaned up carefully. If the bedrock is easy to clean this may be done as the work progresses up a cut. In such ground if the gravel is shoveled in benches by two or more men the man who takes up the lower layer of gravel may clean the bedrock as he goes along. If the bottom is difficult to clean a full cut may be taken before the bedrock is given the final clean-up.

Cleaning bedrock is perhaps the most laborious and painstaking task of the placer miner; his success or failure often depends on the thoroughness of this operation.

After all loose gravel is shoveled into the boxes the bedrock should, if possible, be taken up with pick and shovel to a depth of 6 inches or more and likewise put into the sluice. Any large rocks near bedrock should be washed off before they are thrown aside, because clay adhering to them may contain gold. If the bedrock is decomposed or soft, or if there is a false bedrock, this process will recover practically all the gold.

If bedrock is hard, cleaning it is more difficult, as some material is bound to be left on the bedrock in shoveling, and the crevices in bedrock will contain gold.

If water can be taken from the sluice or ditch through a hose under low pressure all loose material missed in shoveling will be washed down to a hollow where it will be collected and placed into the boxes or in a rocker. The bedrock then is scraped and swept, and the c revices are cleaned out; finally it may be washed a second time.

If it is not convenient to use water the bedrock may be cleaned up dry. The bedrock is swept, scraped, then swept a second time; the crevices are cleaned out as before. The sweepings and scrapings, together with the material from the crevices, are washed either in the sluice or in pans or rockers. Large colors or nuggets are picked up as found.

The tools used for the hand work are picks and shovels, brooms, small stiff brushes, perhaps a sponge to dry out hollows, and hand scrapers of various sizes and shapes, ranging from sections of old frying pans to knives, spoons, or bent wire for cleaning narrow crevices.

In small operations, such as the typical 3- or 4-man ground-shoveling venture, cleaning bedrock is carried to greater extremes than in larger mines where labor is hired, as the incentive to careful work is greater and the yield necessary to encourage further cleaning is less. Skilled and careful "crevicers" have been known to make fair wages on bedrock abandoned by organized mining companies. The miner should therefore be sure his cleaning operations have been carried to the point where he is no longer getting enough gold to recompense his efforts.

The concentrates from the sluice usually are panned or washed in a rocker for the final separation of the gold.

Current Practices

During 1931 and 1932 hundreds of men were mining by hand-shoveling methods in the West. The miners worked in crews of 1, 2, 3, or 4 men. In a few instances as much as $3 per man was made, but the average earnings were well under $1 per day. Most of these men were inexperienced and did not work to the best advantage. Boulders would be rolled to one side and sometimes handled 3 or 4 times before the work had advanced beyond them. Again, the ground would be cut up in uneven forms and part of the gravel shoveled twice to get it into the b xes. Boxes were not always well made, and considerable gold would be lost through leaks or cracks.

Table 6 shows seven representative mines worked by or under the direction of experienced placer miners. Where the men were inexperienced the yardage handled per shift was much less. The cubic yards per man-shift and the estimated cost per cubic yard with labor at $3 per shift are included in the table.

TABLE 6.- Representative examples of shoveling into boxes

	Oroville	Jones	Peshastin	Minnick	Blackhawk	Greetan	DeWitt
Operator of mine		J. B. Jones				O. F. Greetan	H. DeWitt.
Address	Oroville Calif.	Oroville Wash.	Blewett Wash.	Blewett Wash.	Blackhawk Colo.	Bearmouth Mont.	Wilcox Ariz.
Location of mine	Feather River	Mary Ann Creek	Peshastin Creek	Peshastin Creek	North Clear Creek	Bear Creek	Gold Gulch.
Number of men working	2	4	4	2	2	2	1.
Hours per shift	9	9	9	8	8	8	8.
Depth of gravel feet	2	[1]6	6	2 1/2	4 1/2	6	
Character of gravel	Loose top dirt	Tight	Tight	Tight	Loose	Medium	Tight.
Percentage of boulders over 6 inches in diameter in gravel	0	30	20	50	10	5	
Character of bedrock	None	Clay	Even	Very rough	None	Even	Clay.
Width of sluice boxes inches	8	8	12	12	10	8	18.
Grade in. per ft.	1	1	7/16	1/2		6/10	
Length of each box feet	12	12	16	12	12	10	22.
Number of boxes	2	8	4	3	3	6	1.
Type of riffles	Hungarian	Steel matting over coco matting.	Hungarian pole	Wire screen over Brussels carpet.	Wire screen over carpet.	Wooden cross	Wooden cross.
Size of riffles inches	1 by 1 1/4		Hungarian 3/16 by 1 1/4; pole 3.			1 by 1	1 by 1.
Spacing of riffles do.	1		1			4	1.
Total length of riffles feet	12	8	64	12	12	28	18.
Height water pumped do.	10	37	0	0	0	0	0.
Percentage of material rehandled	5	10	0	50	0	0	200.
Handled per day cu. yd.	[2]20 to [3]30	25	20	2	6 1/2	9 1/2	1.
Handled per man-shift do.	[2]10 to [3]15	6 1/4	5	1	3 1/4	4 3/4	1.
Gasoline used per cubic yard gal.	3	6	0	0	0	0	1.5.
Estimated cost of labor and gasoline per cubic yard[4]	$0.36 to $0.26	$0.60	$0.60	$3.00	$0.92	$0.63	$3.30.
Remarks	(5)	Good workmen	Good workmen	(6)		(7)	

1 Top 2 feet consists of soil.

2 With 10 percent plus 1 1/2 inch.

3 With no oversize.

4 Assuming labor at $3 per shift.

5 All material over 1 1/4 inches in size screened out at head box.

6 Stumps in gravel and rock ledges above bedrock.

7 Only lower 18 inches of gravel washed.

<u>Oroville</u>.- Two men shoveled into boxes on the bank of Feather Rvier near Oroville, Calif., during the spring of 1932. The material handled in this operation consisted of loose top sand and small gravel on a river bar where gold had been concentrated recently. The gravel and sand were shoveled onto a grizzly with 1 1/2-inch openings at the head of the sluice. As the oversize piled up it had to be shoveled aside. The boxes were moved as required to keep the head of the sluice within easy shoveling distance of the gravel being mined. The gold was very fine, but apparently a good recovery was made. The water was pumped from the Feather River near by using a 4-cylinder automobile engine.

<u>Jones</u>.- In June 1932 J. B. Jones and his three sons were mining a sloping bed of gravel on Mary Ann Creek, near Oroville, Wash. The gravel bench was about 100 feet wide. Cuts were taken at right angles to the creek. The boxes were set even with the surface, the end of the lower box was about 6 feet high at the edge of the flat bottom land.

A 5-inch centrifugal pump driven by an old 6-cylinder automobile engine lifted the water from a sump in the creek below the workings a vertical distance of 37 feet through 10-inch irrigation pipe to a point on the hillside, whence it was led by gravity to the head of the sluice boxes. About 200 gallons per minute was thus provided, without approaching the capacities of either the engine or pump. The water from the sluice box joined the very small flow of the creek and was again caught in the pump sump, having deposited most of its suspended load before reaching that point. The boxes were placed the full length of the cut before shoveling began. The first cut was 16 feet wide, with the sluice in the middle. (See fig. 6, <u>A</u>.) The boxes were supported with trestles as the gravel underneath was removed. Two men worked on either side of the box beginning at the lower end. The forward pair advanced a bench 3 1/2 to 4 feet deep, which included 2 feet of soil on top of the gravel but contained very few boulders that could not be washed. The lower bench consisted of 2 to 2 1/2 feet of gravel which was picked and shoveled into the sluice by the other two men. The boulders over the size of a man's fist were rolled or thrown back from the lower bench as the cut progressed upward. In succeeding cuts the boulders were thrown or rolled into the previous cut.

The riffles consisted of 8 feet of steel matting, such as is used for doormats, laid on coco matting and placed in about the middle of the sluice. In a test run of 50 cubic yards of gravel only 2 cents worth of gold was obtained in two sections of pole riffles placed below the steel matting, after which the poles were discarded. When the gravel contained clay, 1/2-inch cleats were placed ahead of the regular riffles at intervals of a few feet. These tended to break the clay as it went down the sluice.

At the start 12-inch boxes and correspondingly more water were used, but to reduce the consumption of gasoline for pumping the boxes were narrowed to 8 inches. With four men shoveling steadily the sluice box clogged occasionally above the riffles and had to be cleaned out. The tailings had to be shoveled away from the end of the sluice periodically.

<u>Peshastin</u>.- Four men were shoveling into boxes on Peshastin Creek, near Blewitt, Wash., in July 1932. They worked on the same side of the box and took the ground out in four benches. The last man cleaned bedrock as he came along. (See fig. 6, <u>B</u>.) In the first cut all boulders were lifted or tossed over the box. In the next cut the boulders would be rolled under the box into the space made by the previous cut. Two 7-foot sections of iron-clad Hungarian riffles were used for catching the gold. The remaining space in the boxes was covered with longitudinal pole riffles to protect the bottom of the box. Water was taken from the creek where an ample supply was available.

<u>Minnick</u>.- Two men were shoveling into boxes on Peshastin Creek above the operation just described. The gravel was full of large stumps, and rocky ledges projected into the gravel from the bedrock. If the time taken in cleaning the extremely uneven bedrock is considered not over 1 cubic yard could be handled per man-shift. Three 12-inch boxes were used. Riffles consisted of wire screen over carpet.

Blackhawk.- Two men were shoveling into boxes near Blackhawk, Colo., in July 1932. One man dug the gravel and shoveled it into a box 5 feet above the bottom of the pit at the head of the sluice box. He sorted out all material larger than a man's fist and threw it back of him in the pit. The second man pulled the gravel in a steady stream from the box over a 1/2-inch screen set at an angle of 45°. He also shoveled the oversize back into the completed part of the pit. The sluice was 10 inches wide; the riffles consisted of wire screen over carpet.

Greetan.- Two men at Bearmouth, Mont., were loading gravel into a 4 1/2-cubic-foot car which was dumped into a puddling box at the head of the sluice boxes. Only the lower 18 inches of the gravel was gold-bearing; the overburden was cast to one side.

The gold-bearing gravel, which contained about 10 percent of clay, was washed free of clay by hoeing it back and forth in the puddling box. The gate of the puddling box was raised when the box was full of water, and the mud and water and part of the gravel were allowed to run through the sluice boxes. The water supply was insufficient to wash the gravel through the sluice without the surges made by thus impounding it. The water was brought into the puddling box through a fire hose, and the small pressure available assisted in washing the gravel.

The washing arrangement and boxes were similar in design and operation to those used at some small-scale drift mines. The puddling box was made of matched lumber and was 15 feet long, 6 feet wide, and 2 feet high. The lower end was tapered to 8 inches, the width of the boxes.

DeWitt.- During the summer of 1932 H. DeWitt used a dry washer near Wilcox, Ariz. After the winter rains began he used a sluice with an end-shaking movement. The material washed was a red surface clay and gravel. After picking, it was allowed to lie exposed to the sun for a day or more. The drying caused the clay to slack so that it disintegrated in the sluice.

The sun-dried material was pushed to the head of the sluice in a wheelbarrow and fed into the box by hand. The sluice was 18 inches wide, 6 inches high, and 22 feet long and was set on rollers. The shaking movement was imparted by a rod from the head motion of an old Wilfley table. The stroke was about 5 inches and the speed 20 strokes to the minute. The rod connected to the end of the sluice had coil springs on both sides of the end board. The downward stroke was against stiff springs set on either side of the box. On release of the downstream pressure the side springs gave the box a quick flip backward.

The first 5 feet of the box was lined with a sheet-iron plate in which holes had been made with a pick. The projecting rugged edges of the holes tore up the gravel as it passed over them. Riffles consisted of 1- by 1-inch cross strips of wood 1 inch apart. Water was hauled 1 1/2 miles in barrels. About 1 cubic yard could be handled per man-shift. A cubic yard per hour could be wasned in the box when dirt and water were available.

GROUND-SLUICING

General Features and Application

As previously defined, ground-sluicing is a placer-mining method in which gravel is excavated by running water not under hydraulic pressure. In some cases water is stored in reservoirs, and greatly increased flows are discharged during short periods. This variety of ground-sluicing is termed "booming" by placer miners and is so called in this paper. The action of the ground-sluice water may be augmented by hand work or by a jet of water under pressure. However, if hydraulic monitors or "giants" perform most of the work the method becomes "hydraulicking", described in a later paper.[78]

78 Gardner E. D. and Johnson C. H. Placer Mining in the Western United States: Part II - Hydraulicking Treatment of Placer Concentrates and Marketing of Gold: Inf. Circ. 6787 Bureau of Mines 1934.

With a few exceptions ground-sluicing is used in small operations by miners with little capital. Otherwise it is used only where water under a substantial head is not available or where the gravel deposit is too small to justify building ditches or installing pipe lines for hydraulicking. The chief application of ground-sluicing is to deposits in or near stream beds. If applied to higher gravels pipe lines, flumes, or ditches are necessary, and the method loses one of its chief advantages, a low initial cost.

Ground-sluicing is not adapted to mining large deposits of tight or cemented gravel; these demand the tearing, disrupting force of giants. Gravel containing large boulders can be ground-sluiced, but more labor is necessary than if hydraulicking is used; the ground-sluice water seldom loosens and transports as large boulders as a giant or as large ones as will run through the sluice.

At many places no preliminary development work is necessary; at others long cuts must be run to reach bedrock under the deposit. If booming is to be done a dam must be built for a reservoir and perhaps an automatic gate. A sluice box is always used, although in many mines for little more than as a race to carry away the tailings. In bouldery ground derricks or drag lines may be needed to remove boulders.

A large number of small ground-sluicing operations were being carried on during the 1932 season by groups of 1 to 4 men working for themselves. Enough gold to pay regular wages was obtained at very few of the operations. Usually the miners received less than $1 per day for their efforts. A few larger operations have been carried on for a number of years; some of them have been profitable. Essential data concerning the larger mines and representative smaller ones operated in 1932 are shown in table 7. These mines are described in more detail later. They illustrate the applications of the method and general ground-sluice practices.

Types of Placer Deposits Worked

Except for the Osborne lease at Superior, Mont., gravels of moderate thickness along stream beds were being worked by ground sluicing at the mines listed in table 7. The depths of gravel ranged from 5 to 22 feet. Except at two placers very little of the gravel was tight, and none of it was cemented. At these, the Rundle and Morgan placers, Blackhawk, Colo., the gravel was tight and had to be loosened by picking; however, most of the material mined was overburden which was easily ground-sluiced.

Water Supply

The water supply was limited at most of the mines worked by ground-sluicing. At the Rundle and Morgan placers more water was available in the creek than could be used to advantage, because the gravel had to be loosened by picking. As noted in table 7, 60 to 500 miner's inches were used at the various mines listed. The natural flow of the streams ranged from 10 to 500 miner's inches.

Miner's inch.- Western placer miners usually measure and think of water in terms of miner's inches. In California and Montana, as established by law, 40 miner's inches equals 1 cubic foot per second; in Colorado the legal ratio is 38.4 to 1. Forty miner's inches to the cubic foot per second is generally accepted throughout the West; this value of the miner's inch is used in this paper. A miner's inch as used here equals 11.22 gallons per minute; 1 cubic foot equals 7.48 gallons.

TABLE 7.-Representative placer operations using ground-sluice method of mining, 1932

Name of mine	Mine: Operator	Mine: Location	Mine: Address	Gravel: Thickness, feet	Gravel: Character	Gravel: Percentage of boulders over 6 inches in diameter	Gravel: Percentage of clay	Bedrock: Kind	Bedrock: Character
Morgan	Richard Leoncavallo.	Clear Creek	Blackhawk, Colo.	5	Tight, clay-bound	20	5	Clay	Soft.
Ravano	Tony Ravano	Harris Creek	Laurin, Mont.	6	Medium	25	0	Limestone	do.
Bar No. 1		Bar at mouth of Kamloops Creek.	Granite, Colo.	6	do.	10	0	False clay.	do.
Osborne	W. H. Osborne	California Gulch Cedar Creek.	Superior, Mont.	15	Fairly easy to pick.	15	0	Not on bedrock.	
Rundle	W. B. Rundle	Clear Creek	Blackhawk, Colo.	8	Medium	15	2	Porphyry	Rough.
Bar No. 2	Jim Wiley	Bar at mouth of Kamloops Creek.	Granite, Colo.	6	do.	10	0	False clay.	Soft.
Kamloops	Kamloops Placer Gold Mining Co.	do.	do.	18	do.	10	0	Not on bedrock.	
Willow Creek	Lawry, Kennedy, et al.	Willow Creek	Therma, N. Mex.	20	do.	10	1	do.	
Camp Bird	Joe Witherspoon	California Gulch Cedar Creek.	Laurin, Mont.	10	do.	15	0	Limestone	Soft.
Bennet	S. B. Bennet	Quartz Creek	Rivulet, Mont.	15	do.	10	2	Porphyry	Hard, medium rough.
Harvey	C. W. Robertson	Sauerkraut Creek	Lincoln, Mont.	22	do.	35	5		Soft.
Magnus	Magnus & Ole Lindquist, Inc.	Swauk Creek	Liberty, Wash.	20	do.	25	1	Not on bedrock.	

TABLE 7.- Representative placer operations using ground-sluice method of mining, 1932 - Continued

Name of mine	Water									
			Booms				Auxiliary cutting stream			
	Average miner's inches[1]	Capacity of reservoir, acre-feet	Water used, miner's inches	Average length, minutes	Period between, hours	Number per day	Head, feet	Length of pipe, feet	Diameter of pipe, inches	Diameter of nozzle, inches
Morgan	70	None	None	None	None	None	None.	None	None	None.
Ravano	30	1	150	240	20	1	do.	do.	do.	do.
Bar No. 1	60	None	None	None	None	None	do.	do.	do.	do.
Osborne	80	do.	do.	do.	do.	do.	do.	do.	do.	do.
Rundle	60	do.	do.	do.	do.	do.	27	600	10	2.
Bar No. 2	60	do.	do.	do.	do.	do.	6	70	20 to 5	2.
Kamloops	100	do.	do.	do.	do.	do.	15	350	8-6-5	2.
Willow Creek	70	1 1/4	300	27	1 1/2	4	None.	None	None	None.
Camp Bird	70	1/2	650	30	4	2	60	200	6	1.
Bennet	10	1/10	1,600	1 1/2	4	6	None.	None	None	None.
Harvey	[11]300	5/8	7,500	2 1/2	1	24	22	20	18	
Magnus	500	1 3/4	4,000	15	1 3/4	14	None.	None	None	None.

[1] 1 sec.-ft. = 40 miner's inches.
[11] Average 700, first 52 days.

TABLE 7.- Representative placer operations using ground-sluice method of mining, 1932 - Continued

Name of mine	Handling material: Percentage removed by hand or derrick	Handling material: Maximum size of boulders put through boxes, inches	Handling material: Method of handling boulders	Sluice boxes: Width, inches	Sluice boxes: Height, inches	Sluice boxes: Length, feet	Sluice boxes: Number	Sluice boxes: Grade: Inches per foot	Sluice boxes: Grade: Percentage
Morgan	40	6	Hand	18	12	10	2	1 1/5	10.
Ravano	40	3	do.			12	3	1/2	4.2.
Bar No. 1	15	4	do.	12	8	12	3	1 1/2	12.5.
Osborne	15	6	Steam derrick	[4]13	13	12	7	1/3	2.8.
Rundle	30	3	Fork and wheelbarrow	24	12	16	1	1	8.3.
Bar No. 2	15	4	Hand	18	12	12	2	1/4	2.1.
Kamloops	20	2	Power drag line with 3-in. grizzly buckets	30	20	12	42	1/4	2.1.
Willow Creek	10	6	Hand	18	10	12	50	1/2	4.2.
Camp Bird	20	3	do.	22	16	12	3	1	8.3.
Bennet	10	6	do.	36	18	12	3	3/4	6.2.
Harvey	30	9	Hand derrick	38	36	12	16	3/4	6.2.
Magnus	0	15		48	36	12	84	3/5	5.

[4]24 in., when more water was available.

TABLE 7.- Representative placer operations using ground-sluice method of mining, 1932 - Continued

Name of mine	Riffles: Type	Riffles: Width, inches	Riffles: Height, inches	Riffles: Spacing, inches	Riffles: Total length, feet	Labor: Number of men on each shift	Labor: Total number of men per 24 hours	Labor: Total number of 8-hr. shifts during season	Gravel washers, cubic yard: During season	Gravel washers, cubic yard: Per day	Gravel washers, cubic yard: Per man-shift	Costs per cubic yard[2]: Labor	Costs per cubic yard[2]: Supplies	Costs per cubic yard[2]: Total operating
Morgan	Wire screen over carpet.				20	1	1	60	165	2 3/4	2 3/4	$1.27	$0.02	$1.29
Ravano	Pole	3	3	1	36	1	1	[3]84	775	9	9	.39	0	.39
Bar No. 1	Wooden cross, over burlap.	1	1	1	36	3	3	100	300	9	3	1.17	.02	1.19
Osborne	Pole	3	3	1	84	2 and 4	6	156	[5]1,400	53	9	.39	.03	.42
Rundle	Cross, on wire screen over corduroy	3/4	3/4	5	16	1	1	60	174	3	3	1.17	.02	1.19
Bar No. 2	Wooden cross, over burlap.	1	1	1	24	2	2	33	400	24	12	.28	.03	.31
Kamloops	Wooden cross.	1 1/4	1	1	250	2	4	220	[6]4,000	73	18	.20	.04	[7].24
Willow Creek	Wood block, round.	18	5	0	600	4	4	260	[8]1,030	16	4	.87	.04	.91
Camp Bird	Pole	3	3	1	36	2	2	[9]134	2,400	36	18	.20	.02	.22
Bennet	do.	3	3	1	36	1	1	[10]69	1,150	17	17	.21	.01	.22
Harvey	do.	6	6	1	96	1	1	120	3,800	32	32	.11	.03	.14
Magnus	20-lb. steel rails.	1 1/3	2 5/8	2	400	9	9	805	[12]6,000	67	7	.50	.04	.54

[3]Includes 12 shifts cleanin. up. [5]To July 8. [6]To July 17. [7]Not including rental of shovel, or supervision. [8]To July 20. [9]Includes 50 shifts cleaning up. [10]Includes 19 shifts cleaning up. [12]To June 23.

Sluicing

Operations at all ground-sluicing mines listed in table 7 consisted of running cuts preliminary to mining or mining as contrasted to developing.

Ground-sluicing has an advantage for running bedrock cuts in that a relatively small capital expenditure is required to test the gold content of the gravel and to ascertain the mining conditions. Cuts made in this manner also open up the deposit for mining. A relatively low yardage per shift is obtained in such work. A large part of the expense of running such cuts is the construction of the sluice, which will be used if the deposit is mined later. The work at the Rundle, Osborne, Kamloops, Willow Creek, and Magnus mines, at the time the authors visited them, consisted of running bedrock cuts.

At the other seven properties strictly mining operations were under way. Regular ground sluicing can be subdivided further according to the manner in which the water is used: (1) Water poured over the upper face of the cut and (2) water directed along the bottom of a straight or curved bank. Under (1) the cascading effect of the water is utilized to break down the gravel, or the fall of the water from a projecting flume or pipe is used to cut the bottom layer. Under (2) the current is used to undercut the bank and carry away the broken gravel at the face as it breaks down. The depth of the gravel has an important bearing on the practice followed; with thick deposits the cascading or falling of the water over the upper face is likely to give the best results; in thinner gravels the other method is usually advantageous. The width of the deposit is also a factor; in narrow channels the first method would have an advantage over the other. As shown by table 7 the thickness of gravel at the Morgan, Bar No. 2, and Harvey placers where the water was brought over the face was 5, 6, and 22 feet, respectively. At the other four placers, Ravano, Bar No. 1, Camp Bird, and Bennet the depth of gravel was 6, 6, 10, and 15 feet, respectively.

Of the 12 mines listed in table 7 sluicing with the natural flow of the stream was followed at 3. At 5 properties a minor part of the water was used under pressure, and booming was practiced at 5 not counting the Ravano. In one placer (Camp Bird) a hose was used for cutting the bank and the broken gravel removed by booming.

The water may be directed against the bank in a channel made by piled boulders or by boards. It can be diverted conveniently to any desired point or direction by means of what are called shears. These consist of 1- or 2-inch boards 12 feet long nailed to two tripods which slope back from the water flow at an angle of about 60°. The tripods are of pointed poles about 4 feet long and so constructed that boulders may be piled on the base to hold them in place. A row of "shears" may be used to direct the force of the water against a bank, or two rows may be used to form a flume.

Sluicing With Natural Flow of Stream

Of the multitude of "snipers" working on the creeks in the summer of 1932 many washed their gravel by ground-sluicing with the natural flow of the stream. The only equipment essential for this type of work in shallow gravels is a few 12-foot sluice boxes and picks and shovels. Table 7 shows that 2 3/4, 9, 3, and 9 cubic yards, respectively, were handled per man-shift at the first four mines in the table where the natural flow was utilized. At the Morgan placer water running down the face of a narrow cut was used to carry the fine material in the gravel to the box after it was loosened by picking. At the Ravano a cut was made first down one side of the area to be washed during the season and then extended across the channel. The water was directed against the bank by means of dams. Although a reservoir was used and the stream of water was augmented for a half shift each day the mine is placed under this class, as it is representative of a large number of operations where only

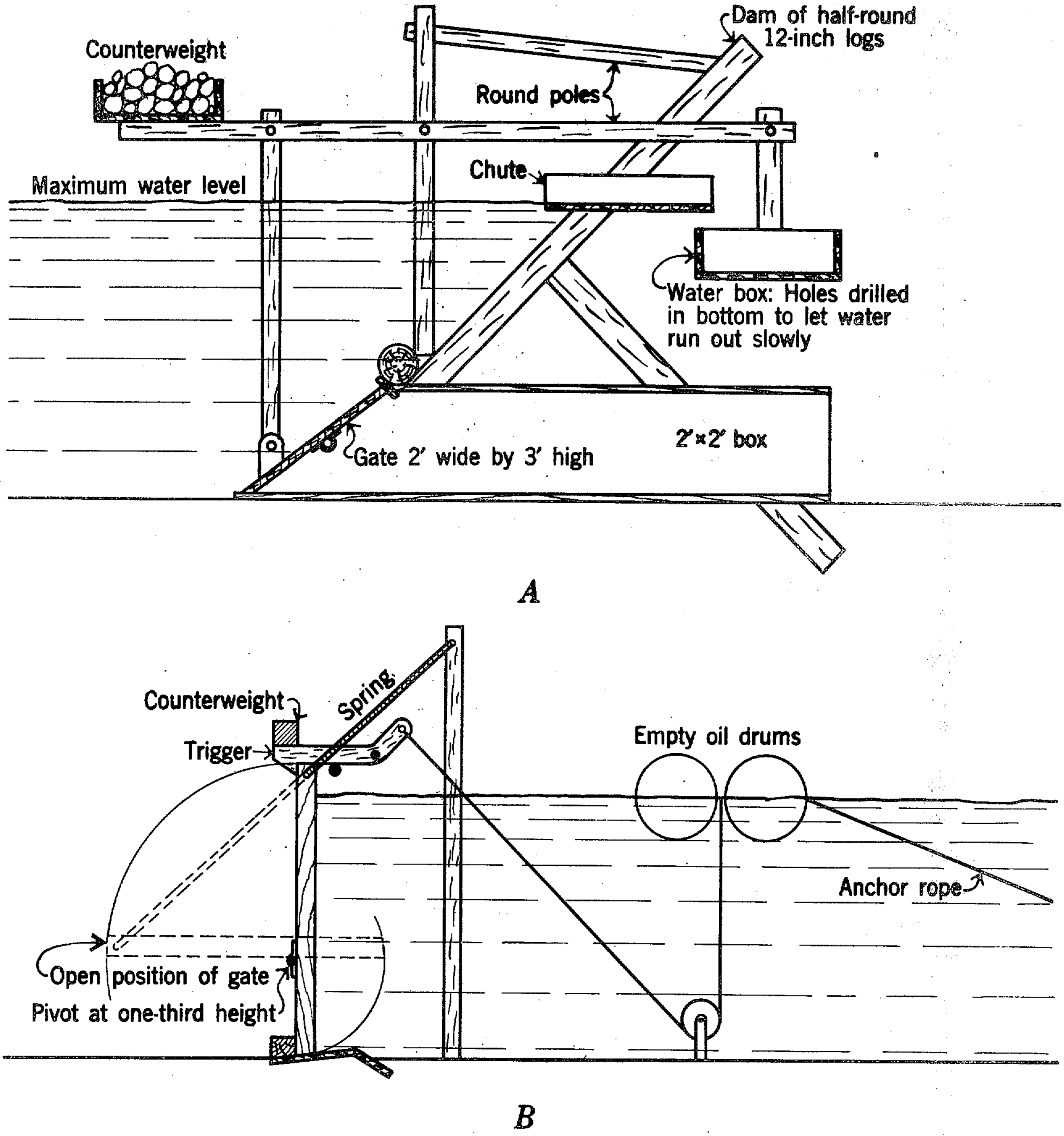


Figure 7.—Two types of automac gates for booming: *A*, Gate used at Bennet mine, Rivulet, Mont.; *B*, gate used at Harvey mine, Lincoln, Mont.

the natural flow was utilized. At the Bar No. 1 the water was brought down over the face. At the Osborne deeper gravels were mined, and the water cascaded down the face of a cut. Boulders were removed by a power derrick.

Sluicing With a Minor Part of Natural Flow of Water Used Under Pressure

Where part of the water can be brought into the workings under pressure through a pipe the nozzle may be used to replace hand-picking in loosening and disintegrating the gravel. It may also be used to replace part of the hand labor in cleaning up bedrock. The ground-sluice water washes the gravel into the boxes; it also breaks down the bank to various degrees at different mines. The actual sluicing operations are quite similar whether or not water under pressure is used.

At the Rundle, Bar No. 2, and Kamloops mines, where pressure pipes were used the head of the water ranged from 6 to 60 feet, as shown in table 7; nozzles 1 and 2 inches in diameter were employed. The yardage moved per man-shift was 3, 12, and 18, respectively.

Booming

Booming utilizes the increased cutting and transporting power of water under flood conditions. As pointed out, water is stored in a reservoir and then released, flowing for relatively short periods. At the beginning of the season, when high water prevails, the booms may occur frequently, and each one may last relatively long. As the supply of water fails booms occur less often until finally there is not enough water to operate. Booming is the most important type of ground-sluicing. A much larger duty can be obtained per unit of water by booming than by other forms of ground-sluicing. The increased volume of water carries boulders into the sluice that otherwise would have to be moved by hand or by power and breaks down the banks against which a smaller stream would be ineffective. Larger sluice boxes must of course be used when booming than when utilizing only the natural flow of the stream.

Booming is used in running development cuts as well as in strictly mining work. Of the three strictly mining operations listed under this head a side cut was used in two and an overcast at the end of the pit in the third.

The average number of booms per day at the five mines (excluding Ravano) ranged from 2 to 24. The duration of booms was 1 1/2 to 30 minutes.

The duration of a boom is governed by the size of reservoir and the flow of water. The capacity of the reservoir should be governed by the character of the ground. In heavy, rocky ground a short period with a correspondingly larger surge is more effective than a longer period with a smaller stream. In other ground, such as that at the Camp Bird mine, longer periods of washing, with correspondingly less water, give the best results. Reservoirs usually are constructed by building a dam across a narrow part of the stream bed or canyon and backing the water up behind it. Ordinarily, earth dams with a board facing on the upstream side are used. Reservoirs with capacities of 1/2 to 1 3/4 acre-feet (table 7) are used at the mines listed. Automatic gates have a double advantage in that no labor is required to operate them and the booming in the pit can continue 24 hours each day. Automatic gates are shown in figure 7. Although the water is not as effective when unattended as when the miners are on shift it accomplishes considerable work during off hours.

Table 7 shows that 4, 18, 17, 32, and 7 cubic yards, respectively, were washed per man-shift at the mines where booming was done.

Sluice Boxes and Riffles Used in Ground-Sluicing Mines

Table 7 also shows that where only the natural flow was utilized boxes ranged from 12 to 30 inches in width and where booming was used from 18 to 48 inches in width.

Wooden cross riffles were used at 4 mines where ground-sluicing was done with the natural flow of the stream. Pole riffles were used in 2, and wire screen over carpet was used in 1 other. Corduroy under wire screen was used under the riffles in 1 and burlap in 2 others.

More substantial riffles are required in the boxes where the gravel is boomed on account of the coarser material and greater volumes put through the sluices. At 3 mines where booming was done pole riffles were used, at 1 mine wooden blocks, and at 1 mine 20-pound steel rails.

The general subject of sluice boxes and riffles is discussed in more detail in a subsequent paper.[79]

Handling Boulders

Boulders generally are handled in ground-sluice mines in the same manner as in hydraulic pits. Derricks are the commonest mechanical device used for the purpose. (See fig. 8.) These may be operated by a hand winch or by steam, gasoline, or electric power. In wide deposits boulders may be removed on platform-skips operated on a cableway. At the Harvey placer, where the ground contains an unusually high percentage of boulders (35 percent over 6 inches in diameter), two hand-operated derricks are used in a pit 90 feet wide. At the Osborne lease a steam-operated derrick is employed. At the other ground-sluice mines listed boulders are moved by hand; large ones are blasted. At the Osborne lease large boulders are drilled with jackhammers, using steam from the boiler which supplies power for operating the derrick. Boulders too large to move by hand are blown from the pit by explosives at the Camp Bird mine.

Bringing the ground-sluice water over the face of the pit has a decided advantage in that boulders need to be handled only once. They are either moved completely out of the workings or piled in the pit on cleaned bedrock, as is done at the Harvey placer. Where a side cut is taken, as at the Camp Bird mine, boulders are moved back over the washed area as washing progresses across the channel. Small stones are tossed free of the pit on the side from which sluicing started. If the bedrock is cleaned only at the end of the season boulders must be moved again.

Cleaning Bedrock

As previously stated, at end-cut mines bedrock is cleaned as the work progresses upstream. Usually a section of the pit corresponding to the length of a sluice box is cleaned at a time. Many miners consider the amount of gold cleaned each box length as the unit of value of the ground. With a side cut the bedrock usually is cleaned at the end of the season. Although frequent clean-ups bring in a current revenue the practice of cleaning up at the end of the season permits full advantage to be taken of the water while it is available. Moreover, cleaning at the end of the season affords an opportunity for a more nearly complete recovery of the gold, as the work can be done in a more leisurely manner.

79 Gardner E. D. and Johnson C. H. Placer Mining in the Western United States: Part III. - Dredging and Other Forms of Mechanical Handling of Gravel and Drift Mining: Inf. Circ. 6788 Bureau of Mines 1934.

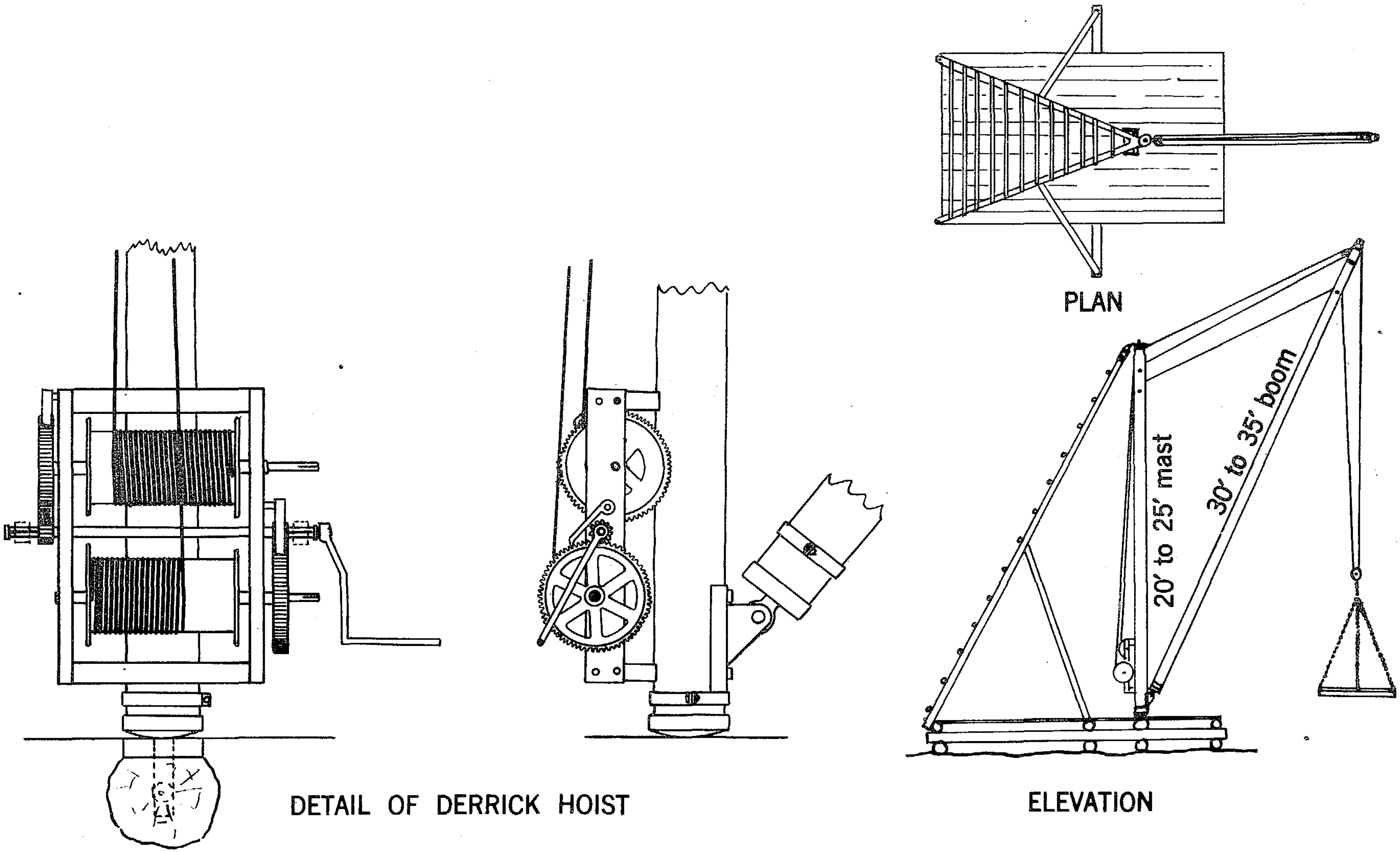


Figure 8.—Derrick used at Harvey placer, Lincoln, Mont.

Bedrock had not been reached in 4 out of 5 places where cuts were run for this purpose. In putting in new boxes, however, layers of gravel on the bottoms of the pits were shoveled carefully into the sluice to save any gold that had been left behind during the washing operations.

Where the bedrock is soft a layer is removed and washed as in other forms of placer mining. The depth taken depends upon how deeply the gold has penetrated into it. The bedrock material may be washed into the main sluice box or shoveled into special clean-up boxes. In mines where a side cut is used a separate clean-up box usually is employed. Advantage is taken of a hose in cleaning bedrock when one is available.

Where bedrock is hard it must be cleaned by hand in all ground-sluice operations. Even if the surface of the bedrock is smooth it almost invariably contains soft seams and cracks which must be dug out with small hand tools.

The sluice-box concentrates usually are panned or washed in a rocker, as described under Separation of Gold and Platinum from Concentrates in a later paper.[80]

Descriptions of Ground-Sluicing Mines Operating in 1932

The mines described are the larger ones and a few typical small ones visited by the authors in 1932. Comparable data concerning them were shown in table 7.

Morgan.- Richard Leoncavallo was working a pit on the Morgan placer on Clear Creek below Blackhawk, Colo., in July 1932. The gravel was tight, coated with clay, and overlain by 2 or 3 feet of recent wash and mill tailings. Cuts about 6 feet wide radiated from the head of the sluice boxes, following rich streaks on a false clay bedrock. All the gravel had to be loosened by picking, which was done while the water was running over the face of the cut. Boulders more than 6 inches in diameter were thrown out by hand. Some of the gravel near the head of the sluice was shoveled in by hand.

About 70 miner's inches of water was used; some of the sediment in the creek water was settled out in a small reservoir above the mine. The sluice consisted of two 10-foot boxes 18 inches wide and 12 inches deep having a grade of 12 inches per box. The first 4 feet was floored with 1-inch screen placed tight on the bottom of the box. Below this 1/2-inch screen was laid over carpet and canvas. A canvas apron was placed in the pit just ahead of and a few inches below the level of the sluice; most of the gold was caught on this canvas. The clay bedrock was cleaned by shoveling the top layer into an 8-inch box built and set up for the purpose.

An average of 2 3/4 yards was washed per day with one man working. At $3.50 per day the labor cost would have been $1.27 per cubic yard. The total operating cost, allowing $0.02 for supplies, would have been $1.29 per yard.

A number of men were conducting similar operations farther down Clear Creek; none were making wages.

Ravano.- About 775 cubic yards of gravel was sluiced by Tony Ravano in 3 months, including 12 days for cleaning up bedrock, on Harris Creek near Laurin, Sheridan County, Mont. The average run-off in the creek during the season was about 30 miner's inches. This water was stored overnight in a small reservoir, the gate of which was partly opened in the morning, allowing a stream of about 150 miner's inches to run until noon. Only the natural flow ran during the afternoon. As the flush water reached the pit it was deflected by boards and rock dry walls against the bank on one side of the pit and thence through a sluice. (See fig. 9, A.) The action of the water was assisted by picking into the bank. During the

80 Gardner E. D. and Johnson C. H. Placer Mining in the Western United States: Part III. - Dredging and Other Forms of Mechanical Handling of Gravel and Drift Mining: Inf. Circ. 6788 Bureau of Mines 1934.

afternoon boulders over 6 inches in diameter were rolled back onto the bedrock previously exposed. Stones 3 to 6 inches in diameter that were not carried out by the water were thrown clear of the pit by hand. All fine material was washed through three 12-foot boxes with longitudinal pole riffles, but nearly all of the gold stayed on the bedrock. Other miners working on the same creek by this method did not use any boxes except for cleaning up; they considered that a good extraction of the gold was made in this manner.

In clearing up bedrock the sluice was extended into the pit one box at a time. The boulders were removed from the bedrock, which was then loosened by picking, and the material was shoveled into the box. As each box was placed the boulders from the next section above were moved back on the part cleaned up where they formed a bed about 2 feet deep. All of the material that was piled on bedrock had been handled twice. A total of 84 shifts was worked, and an average of 9 cubic yards was moved each day. At $3.50 per shift the labor cost would have been 40 cents per cubic yard. There were no expenses other than labor as old timber was salvaged for the sluice boxes.

Bar No. 1.- In July 1932, Bar No. 1 placer on Kamloops Creek near Granite, Colo., was being mined by three men working on one shift per day. About 60 miner's inches of water was used, which came from a ditch at the head of the area being worked. Three 12-foot boxes 12 inches wide and 8 inches high, with a grade of 18 inches to the box, were used for a sluice.

The ground washed was 6 feet deep, comprising a flat bar on a false clay bedrock. There were very few boulders. Conditions were favorable for ground-sluicing. The men, however, were inexperienced and did not work to the best advantage. The water was spread out over too much ground and irregular cuts were taken, leaving small tracts that could be moved only by hand. The boulders were handled 2 or 3 times before final disposal. Gold was lost, as not over one half of the bedrock area was cleaned up.

About 9 cubic yards was handled daily. With wages at $3.50 per day the labor cost would have been $1.17 per cubic yard. Allowing 2 cents for supplies the total cost would have been $1.19.

The adjoining ground was being worked by an experienced man who moved nearly four times as much ground per man-shift with the same amount of water and extracted a larger proportion of the gold in the gravel.

Osborne.- In July 1932, W. H. Osborne with three other men was running a ground-sluice cut in a gravel bench on California Gulch of Cedar Creek near Superior, Mont. This cut was to reach bedrock and was then 120 feet long and averaged 20 feet wide and 15 feet deep. The water cascaded down the face of the cut. Two men loosened the gravel with picks and rolled boulders out of the way. One man worked on the flume and one built boxes and assisted elsewhere when needed. Two men worked on night shift. Small boulders were piled beside the sluice. The ground contained some large boulders which were removed from the cut every other day by a steam derrick working half a shift. Those up to 3 tons in weight could be hoisted with the derrick. Larger ones were drilled with a jackhammer, using steam from the derrick boiler, and blasted.

The water supply was insufficient for good work. On July 8 about 80 miner's inches was available. The boxes had been 24 inches wide, but as the water decreased the width was narrowed to 13 inches. The grade of 4 inches per 12-foot box was too flat for the coarse material being mined; the boxes clogged if not attended continually. Table 7 shows that 9 cubic yards was being washed per man-shift. At $3.50 per shift the labor cost would have been 39 cents per cubic yard. Supplies cost about 3 cents, making a total of 42 cents.

Rundle.- W. B. Rundle was ground-sluicing at Blackhawk, Colo., in July 1932. Part of the water used was from a pressure pipe. The material being washed consisted of 2 or 3 feet of tight creek-bed gravel overlain with 5 or 6 feet of recent wash and mill tailings. A cut was started at the rim of the creek bed and extended upstream; the face had just reached bedrock. The layout of the workings is shown in figure 9, B. The ground was partly loosened, and the boulders were washed clean by water from a 2-inch nozzle under a head of 27 feet. Some picking was necessary to loosen the virgin gravel. The overburden and the loosened material were washed into the boxes by the ground-sluice water which poured over the end of the pit.

All material over 3 inches in size was forked out into a wheelbarrow and taken to a rock dump. The ground sloped steeply upward on the hill side of the pit, and the oversize had been forked out of the pit on the creek side until the rock pile reached such a height that it was easier to use a wheelbarrow.

The riffles consisted of wooden cleats placed 5 inches apart and were of 3/4-inch square material with the downstream edge beveled backward. The bottom of the box was lined with corduroy cloth with the corrugations at right angles to the box; wire screen was placed over the corduroy. The water contained flotation tailings, including a large quantity of pyrite from a concentrator up the creek. The sulphide, however, did not clog the riffles in the sluice box.

An average of 3 cubic yards was washed per day. At $3.50 per shift the labor cost would have been $1.17 per cubic yard; the total cost, with 2 cents per cubic yard for supplies, would have been $1.19.

Bar No. 2.- In July 1932, Jim Wiley, with one helper, was mining a flat grass-covered gravel bar near the Arkansas River at the mouth of Kamloops Creek, Colo. The gravel occurred on a false clay bedrock which sloped about 1 inch to the foot. A 6-inch sand streak occurred just above the bedrock. The ground was worked in cuts 8 feet wide and 150 feet long. There was enough dump room at the lower end of the cuts. The bank was undercut in the sandy layer by a jet of water from a fire hose, using a 2-inch nozzle and a 6-foot head; the overlying gravel broke down and was washed into the boxes by the ground-sluice water. About 60 miner's inches was used. Lumps were broken by picking, and all boulders were thrown back beside the sluice boxes. A tight wing dam built of sod directed the water into the head of the boxes. A 5- by 20-foot steel sheet was placed in the cut 18 inches below the level and ahead of the sluice; the edges of the sheet were turned slightly upward, and the space underneath was packed at the edges with sod. The sheet was moved up to the face of the cut as soon as room was made; the sluice was extended to the lower end of the sheet at the same time. Bedrock was cleaned up by shoveling the top of the false clay bedrock onto the sheet where clay that did not break up was puddled with a hoe or shovel. The material on the sheet was then shoveled into the boxes. Most of the gold was caught on the sheet.

The boxes were 18 inches wide and had a grade of 1/4 inch to the foot. The riffles consisted of 1- by 1-inch transverse wooden cleats placed over burlap. Quicksilver was used in the boxes when cleaning up.

About 16 cubic yards per man per day was washed when ground sluicing. Including clean-up time, the average was about 12 cubic yards per day. At $3.50 per day the labor cost would have been 28 cents. The total operating cost, including 3 cents for supplies, would have been 31 cents per cubic yard.

Kamloops.- The Kamloops Placer Gold Mining Co. was running a cut on a bench to reach some reputed rich ground near Granite, Colo., in July 1932. The cut started at the edge of old hydraulic workings. Bedrock had not been reached. Part of the gravel was tight and contained some boulders 4 or 5 tons in weight. On July 17, 1932 a cut 15 feet wide and 525 feet long had been run. The average depth was about 14 feet and the maximum 18 feet. The sluice was 30 inches wide, 20 inches high, and 504 feet long. Dredge-type Hungarian riffles were used. Boxes were set at the flat grade of 3 inches to the 12-foot box to reach bedrock as soon as possible. To prevent clogging, as much of the washed oversize as could be loaded conveniently was removed from the pit ahead of the boxes with a power dragline. This machine had a 35-foot boom and a 40-foot line. Boulders up to 3 tons in weight could be lifted out of the cut with the dragline by using chains. Boulders over 3 tons in weight were block-holed and blasted. The dragline bucket or dipper was made with a grizzly bottom with 2-inch spacing between the bars. About 20 percent of the material was removed by the dragline.

Water under a 15-foot head was directed against the bank by a 2-inch nozzle mounted on a stand with a "gooseneck." The ground-sluice water flowed down the face. About 100 miner's inches was used.

The crew consisted of two men. One operated the dragline and nozzle, the other attended the sluice and picked down the bank. The dragline operator on day shift acted as superintendent. Two shifts were worked, and an average of 73 cubic yards was handled per day. Allowing a wage of $3.50 per day for all four men the labor cost would be 20 cents per cubic yard. With supplies at 4 cents per cubic yard the total operating cost would be 24 cents, disregarding supervision, rental, and repairs to the dragline.

Willow Creek.- Four men - Laury, Kennedy, Lund, and Neal - were running a cut to reach bedrock on Willow Creek near Therma, N. Mex. On July 20, 1932 the cut was 130 feet long and averaged 24 feet wide and 10 feet deep. A dam had been built above the mine across the creek on bedrock, which raised the underground flow of water above the surface. When ready to boom the gate in the dam was opened by hand. The water poured out of a 10-inch pipe under a 3-foot head and was conducted in a 12-inch pipe to the head of the cut. The pipe extended over the face of the cut, and the stream of water struck the toe of the slope. The pit was boomed four times per day; the length of each booming period was 27 minutes. The rest of the time, while the reservoir was refilling, was spent in throwing out boulders and installing boxes. While booming, two men worked in the face with shovels, assisting the action of the water. The third man watched the boxes to see that they did not become clogged, and the fourth man stayed at the end of the sluice box and pulled away the tailings when they tended to pile up.

The pit had not reached bedrock; when this occurred the face would be widened to include all of the gravel channel. The sluice was 18 inches wide, 10 inches high, 600 feet long, and set on a grade of 6 inches to the 12-foot box. The riffles consisted of round blocks 5 inches long and 18 inches in diameter that just fit in the boxes. A 1- by 4-inch strip nailed on the inside of the box held the blocks in place. In addition to acting as gold catchers the blocks protected the bottom of the boxes.

Lumber cost $22 per M at a sawmill near by. An average of 4 cubic yards per man-shift was being washed; not enough water was available for the economical utilization of the labor. At $3.50 per shift the labor cost would have been 87 cents per cubic yard. With supplies at 4 cents, the total operating cost would have been 91 cents. A large part of the work consisted of building 470 feet of sluiceway down the canyon, below where the cut was started, to provide dump room. Deducting the cost of this, the labor cost would have been about 40 instead of 87 cents.

Camp Bird.- Joe Witherspoon, with one man, was ground-sluicing on California Gulch near Laurin, Sheridan County, Mont., during the 1932 season. The gravel occurred along the bottom of the gulch on the present stream course and averaged about 7 feet deep and 20 feet wide. The pit, an extension of old workings, was 450 feet long on July 5, 1932. (See fig. 9, C.) Overlying the gravel was 2 1/2 feet of loam. The top soil was piped off with a 1-inch nozzle on a firehose connected to a 6-inch pipe with a 60-foot head. The hose also was used for cutting the bank. The pit was boomed on an average of twice a day, the flush water running about 30 minutes each time. The remaining time was used in removing boulders and cutting the bank. Enough water was available in the early part of the season to boom 4 or 5 times per day. The ground-sluice water was deflected and held against the bank by boards and dry walls. A part of the natural flow of the stream ran through the pit while the boulders were being thrown out. Boulders over 6 inches in diameter that could be lifted by hand were rolled or lifted back on the washed bedrock. Stones 3 to 6 inches in diameter were thrown by hand clear of the pit. Boulders too large to lift were blown out of the pit with 40-percent-strength gelatin dynamite. Several sticks of explosive were placed under the boulders so that on detonation the boulders were lifted clear of the pit. The boulders were blasted when partly submerged in water, which acted as stemming for the explosive. Three 12-foot boxes, 22 inches wide and 16 inches high, set on a grade of 1 inch to the foot were used; little gold, however, got into the sluice.

It was estimated that about 50 shifts would be required to clean up the bedrock that was exposed on July 5. In doing this the boulders would have to be moved a second time. The bedrock was soft and would be picked and shoveled into clean-up boxes.

Between 50 and 60 cubic yards was being washed per day. Allowing clean-up time, the average for the period was about 36 cubic yards per day or 18 per man-shift. At $3.50 per day the labor cost would have been 20 cents, and with 2 cents per cubic yard for supplies the total operating cost would have been 22 cents.

Bennet.- I. B. Bennet had been ground-sluicing by booming on a side gulch of Quartz Creek near Rivulet, Mont., for a number of years. The bedrock sloped about 1 1/2 inches to the foot. The gravel was 12 to 25 feet deep and averaged about 15 feet. As the reservoir filled it discharged through a 24- by 24-inch opening, the gate of which operated automatically. Figure 7, A, shows the details of the automatic gate. At the beginning of the season the reservoir filled every 2 hours and for a period of 9 days, when the snow was melting most rapidly, every 20 minutes. At the end of the season the reservoir filled only once a day. The flush water was directed against the bank, which was kept in the form of a semicircle. As the bank was washed away, the coarser gravel, about 3 feet in depth, was left on bedrock. At the end of the season this 3 feet of material was loosened by picking, and the boulders were rolled back onto bedrock; the water carried away the rest of the material. An area 20 feet wide and 100 feet long was washed during the 1932 season of 50 days. About 1 foot of boulders was left to be moved again as the bedrock was cleaned. It was estimated that about 3 weeks would be required for this work. The bedrock was hard and medium rough. It would be cleaned by hand; crevices must be dug out and scraped. An average of 23 yards was mined per 8 hours during the washing period. Although only one 8-hour shift per day was spent on the ground, the booming went on for 24 hours.

At $3.50 per day the labor cost would be 21 cents per cubic yard. The total operating cost was about 22 cents.

Harvey.- C. W. Robertson had been working the Harvey placer near Lincoln, Mont., for the past 15 years. The gravel worked was in the bottom of a gulch along the stream bed. The depth ranged from 18 to 24 feet and averaged 22 feet; the width was 90 feet (fig. 9, D). The gravel contained an unusually large proportion of boulders, some of which weighed as much as 8 tons. The pit advanced the full width of the channel. The washing was done by booming. When work began in the spring the reservoir would fill in 9 minutes, and the full stream would run for 15 minutes. In July a boom occurred once an hour and ran only 2 1/2 minutes. The gate opened automatically (see fig. 7, B). The water was taken from the reservoir in a flume 5 feet wide, 3 feet deep, and 400 feet long. At the head of the pit the flume divided into five branches, the ends of which extended over the face of the pit. During booming the water dropped into the pit through two adjoining branches. The bottom 6 feet of the gravel was fairly tight and partly bound with clay. The overcast allowed the water to drop on this stratum; as the lower gravel was cut away the bank above sloughed down. The sluice was 38 inches wide, 36 inches deep, and 192 feet long and had a grade of 3/4 inch to the foot. Individual boxes were 12 feet long. Pole riffles 12 feet long were used in the boxes. The sluice was cleaned up at the end of the season. It was brought upward in the pit to one side of the center (see fig. 9, D). Twelve-foot cuts were taken from the face of the pit on alternate sides of the sluice. A new box was put in after every pair of cuts.

Boulders were handled by means of two well-built hand-operated derricks and piled back of the ground being worked; they filled the pit to a depth of 10 feet. A dry wall was built up of boulders on either side of the sluice. One derrick stood on a platform across the head of the sluice boxes on the rock walls. The other derrick was on a platform on the main rock pile. The derricks were built of 10- to 12-inch round timber. The boom of the one over the sluice box was 30 feet long and that of the other 35 feet long. The masts were 20 feet long. The winch cables were 5/8 inch in diameter. The derricks were set so that a load from one could be taken up and dumped by the other. A sling platform was used for handling any boulders that a man could roll onto it. A chain was used for larger ones. No blasting was necessary.

Each 12-foot section of bedrock was cleaned up as the cut was completed. The bedrock, which was fairly soft, was loosened by picking and then hosed into the sluice by a 3 1/2-inch firehose connected to an 18-inch pipe from one of the boxes on the bank. The 22-foot head did not give enough pressure to cut the bedrock. After a section was cleaned a team was brought into the pit, and as many large boulders as could be handled from near the face were dragged onto the cleaned-up bedrock. A dry wall was then built to deflect the water to the head of the sluice for mining the next cut. Old rags were used in the wall near the head of the box, and sod was used elsewhere to make the wall watertight for a height of 12 or 18 inches. The wall was raised and the space back of it filled with boulders as the next cut was taken out above.

To July 7, 1932 when the water had materially decreased, 2,520 cubic yards had been washed in 50 working days. A total of 3,800 cubic yards was washed by the end of the season (in the middle of September). Robertson worked the mine alone. Up to July 7, 50 cubic yards had been washed per day. Although only one shift per day was worked the booming went on 24 hours. The labor cost would have been 7 cents; allowing 3 cents for supplies the total cost was 10 cents per cubic yard. For the whole season an average of 32 cubic yards was washed per shift. This would amount to 11 cents per cubic yard for labor, or a total of 14 cents. Some work, such as repairing the dam and flume, was done during the winter; this would increase the total cost per cubic yard 2 or 3 cents.

Magnus and Ole Lindquist, Inc.- This company was running a long cut by ground-sluicing to reach bedrock near Liberty, Wash. After bedrock was reached it was planned to put in hydraulic equipment. On June 23 the cut was about 400 feet long and averaged 20 by 20 feet in cross-section. The sluice boxes were 48 inches wide and 36 inches high. The grade was 5 percent. Riffles were 20-pound steel rails set lengthwise 2 inches above the bottom of the box and 2 inches apart. All boulders up to 15 inches in diameter were put through the boxes. Any over this size were first broken by blasting, then run down the sluice.

Water for booming was stored in a reservoir; the gate was 6 by 6 feet and was opened automatically. The boom lasted 15 minutes, and an average of six booms occurred during the shift; the booming continued 24 hours each day. The water poured over the face of the cut; it broke down the face and scoured out the cut without much assistance from the miners. Boulders occasionally were started rolling and assisted through the boxes by hand.

Of the 9 men employed 3 worked at the sawmill cutting lumber for the sluice boxes; 3 were putting new boxes in the cut as room was made and at the lower end as the tailings filled up the limited dump room; 2 worked at the end of the sluice, leveling off the tailings; and 1 man watched the boxes and supervised the work. The three men who worked on boxes also handled boulders and watched the water during the booming period. The low yardage (7 cubic yards) per man-shift was due mainly to the unusually large number of men employed in cutting lumber and putting in boxes. The total labor cost at $3.50 per shift would have been

50 cents. With 4 cents for supplies the total cost would have been 54 cents per cubic yard. Exclusive of the extra men required to level off the tailing and to extend the boxes at the lower end of the sluice on account of limited dump room, the labor cost would have been about half that indicated.

Summary of Ground-Sluicing Operations and Costs in 1932

Although in the past ground-sluicing has been conducted on a fairly large scale, all operations using this method of mining in 1932 were relatively small. The largest number of men per day (4, 6, and 9) were employed where ground-sluicing was used for developing gravel deposits. Only 1, 2, or 3 men were employed at mines where strictly mining operations were being conducted. An average of 32 cubic yards per day was mined by one man at the Harvey mine, although the gravel contained the unusually high proportion of 35 percent of boulders over 6 inches in diameter. At the other ground-sluicing mines the daily yardage handled ranged from 2 3/4 to 73 and the cubic yards per man-shift from 2 3/4 to 18. The total yardage handled during the entire 1932 season at the mines listed was about 43,000. Although a large number of small mines not listed herein were being worked in 1932 the total yardage moved was not great. If all of them were considered the 43,000 cubic yards listed probably would not be trebled.

The cost of ground-sluicing like that of other methods of placer mining varies directly with the cubic yards handled per man-shift. As shown in table 7 the labor costs per cubic yard for each of the four mines ground sluicing with natural flow of stream were $1.27, $0.39, $1.17, and $0.39, respectively. Supplies were estimated to cost up to 4 cents per cubic yard. The labor costs at the mines using an auxiliary hose with the natural flow were $1.17, $0.28, and $0.20, respectively. Supplies were estimated to cost 2, 3, and 4 cents per cubic yard. Labor costs at five mines where booming was practiced were 87, 20, 21, 11, and 50 cents, respectively. Supplies at the same mines were estimated at 4, 2, 1, 3, and 4 cents per cubic yard.

No supervision or overhead charges were incurred at any of the mines except the Kamloops and Magnus and Ole Lindquist where cuts were being run preparatory to large-scale work. In these two mines working superintendents were employed, and their time at the regular wage rate is included in the operating cost; the charges for supervision would be the salary paid in excess of $3.50 per day.

As stated, the cost per cubic yard of gravel moved in development work was high because of the large proportion of the work involved in placing boxes. Since little equipment was used at most of the mines the amortization charge would be low. The following table shows the estimated total operating costs at the mines where strictly mining operations were being carried on by ground-sluicing.

Mine	Daily yardage	Variation of method	Operating cost per cubic yard
Morgan......	2 3/4	With natural flow..........	$1.29
Ravano......	9	do.	.39
Bar No. 1	9	do.	1.19
Bar No. 2	24	With auxiliary nozzles	.31
Camp Bird	36	Booming..........................	.22
Bennet......	17	do.	.22
Harvey......	32	do.	.14

At the Morgan placer the gravel was loosened with difficulty by picking, hence the high cost per cubic yard. The low efficiency of labor at the Bar No. 1 mine was mainly the result of inexperience. The costs at the other mines listed represent average conditions.

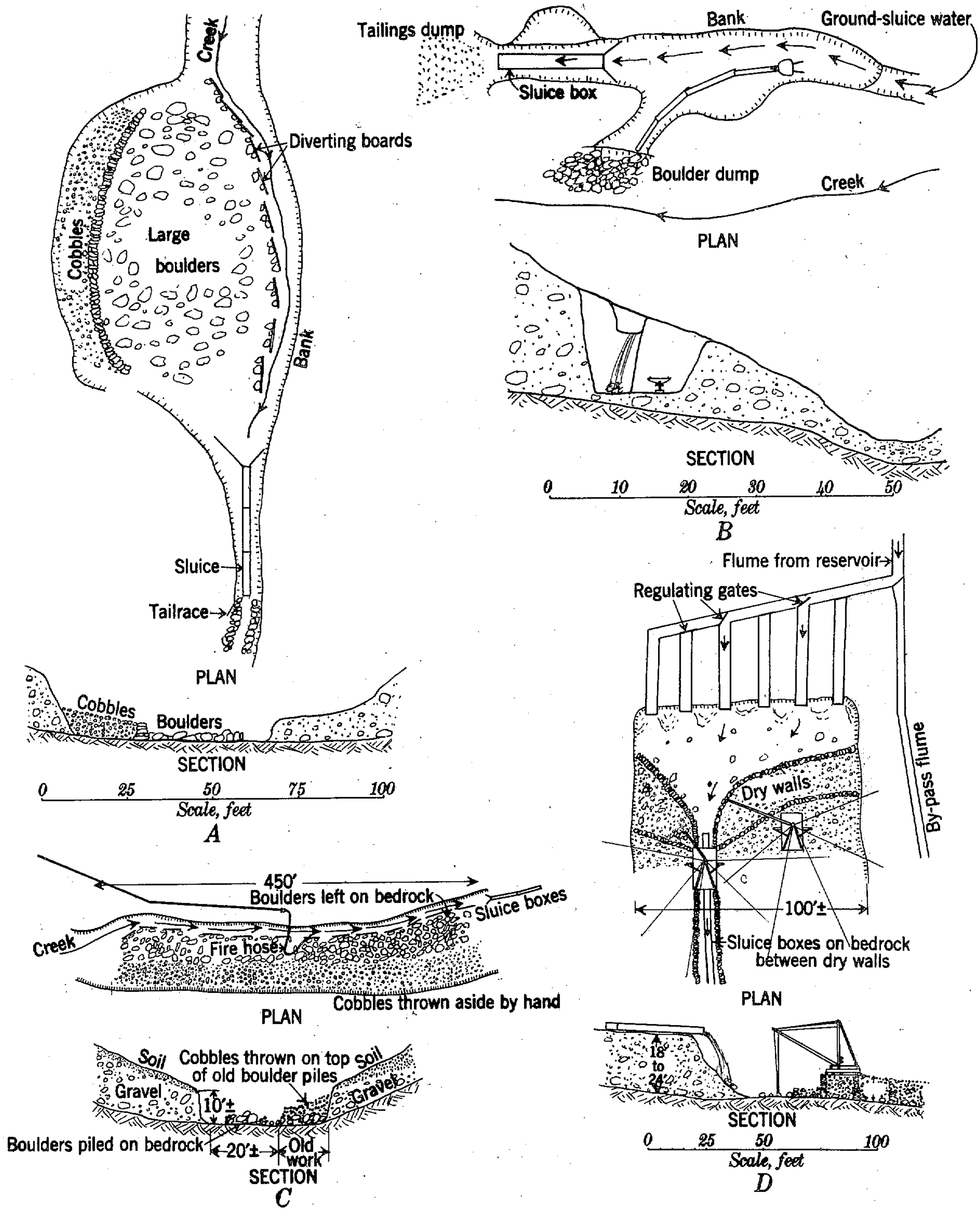


Figure 9.—Lay-outs of ground-sluice mines: *A*, Ravano mine, Laurin, Mont.; *B*, Rundle mine, Blackhawk, Colo.; *C*, Camp Bird mine, Laurin, Mont.; *D*, Harvey mine, Lincoln, Mont.

I. C. 6787 October 1934

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

PLACER MINING IN THE WESTERN UNITED STATES

PART II. HYDRAULICKING, TREATMENT OF PLACER CONCENTRATES, AND MARKETING OF GOLD

BY

E. D. GARDNER AND C. H. JOHNSON

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

PLACER MINING IN THE WESTERN UNITED STATES[1]

Part II. - Hydraulicking, Treatment of Placer Concentrates, and Marketing of Gold

By E. D. Gardner[2] and C. H. Johnson[3]

CONTENTS

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6787."

2 Supervising engineer, U.S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

3 Assistant mining engineer, U.S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

INTRODUCTION

This paper is the second of a series of three on placer mining in the western United States. The first paper[4] discusses the history of placer mining in the Western States and the production of placer gold, geology of placer deposits, location of placer claims on public lands, sampling and estimation of gold placers, and classification of placer-mining methods, together with hand-shoveling and ground-sluicing.

This paper deals with hydraulicking, sluice boxes and riffles, recovery of gold and platinum from placer concentrates, treatment of amalgam, and marketing of placer gold. The discussion of sluice boxes and subsequent subjects in this paper applies to all forms of placer mining.

The third paper deals with dredging and other forms of mechanical handling of placer gravels, and drift mining.

ACKNOWLEDGMENTS

The authors wish to acknowledge their indebtedness to the operators of placers in the Western States who generously supplied information without which this paper could not have been written.

Manufacturers of placer pipe, giants, and power excavators have generously furnished information, including prices, of their products.

Available literature upon placer mining, engineering, and allied subjects has been consulted; the authors have endeavored to make suitable reference throughout the text.

HYDRAULICKING

Application

In hydraulic mining a jet of water issuing under high pressure from a nozzle excavates and washes the gravel. The gold is recovered partly by cleaning bedrock after the gravel has been stripped away but chiefly by riffles in the sluice box through which the washed gravels and water flow to the tailings dump.

Almost all types of placer deposits can be worked by hydraulicking if water is available but certain physical characteristics have an important bearing on the cost of the operation. If the gravel is clayey the washing is more difficult but more important. If the gravel is cemented it can be cut only by high-pressure water. If the grade of bedrock is flat the duty (cubic yards per miner's inch or other unit) of the water is relatively low, and where gravity disposal of water and tailings is impossible or impracticable elevators must be used to raise them from the pit, further decreasing the capacity of the installation.

Apart from the deposit itself, the water supply is the most important factor in determining the application of hydraulicking and the scale of operation. Under any given conditions the daily yardage is roughly proportional to the quantity of water used. The quantity excavated likewise is proportional to the head used on the giants, but the higher pressure is of less value in driving and washing and of none at all in sluicing the gravel through the boxes to the dump. As the cutting and sweeping capacity of the giants usually exceeds the carrying capacity of water a stream of flowing water, known as "by-wash", or "bank water", is directed through the pit and into the sluices. If run over the bank, as in ground sluic-

---

4 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part I. - General Information, Hand-Shoveling, and Ground-Sluicing: Inf. Circ. 6786, Bureau of Mines, 1934, 73 pp.

5 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part III. - Dredging and Other Forms of Mechanical Handling of Placer Gravels and Drift Mining: Inf. Circ. 6788, Bureau of Mines, 1934.

ing, it aids materially in cutting the gravel. The proper relative quantities of high pressure and bank water can be determined only by trial. Frequently the by-wash is supplied by the natural flow of the stream at the mine, the giant water being brought from a considerable distance up the stream or from another source. When an excess of bank water is available it may be used for ground-sluicing, thus increasing the capacity of the plant.

The preparatory or development work necessary to start hydraulicking usually is greater than that for any other form of placer mining except dredging or drift mining. A deposit preferably is opened at the lower end to permit gravity drainage and progressive mining of the entire deposit in an orderly fashion. If the gravel is thick or the grade of bedrock flat a very long cut may be necessary to reach bedrock at the desired point. This may involve the mining of large quantities of barren or at least unprofitable gravel. A more important element of preparatory cost is the water supply. As heads of 50 to 300 or 400 feet are desired, a mile or more of ditch or flume is almost always necessary to bring water onto the property by gravity flow. A single mine may have many miles of ditch, costing perhaps $2,500 per mile, as well as dams and reservoirs and thousands of feet of flumes, tunnels, or inverted siphons. The mechanical equipment of a hydraulic mine ordinarily consists of a few hundred to a few thousand feet of 10- to 30-inch, or larger, iron pipe, one or more monitors, and a varying number of sluice boxes; the cost of equipment ordinarily is small compared to the expenditures necessary for ditches and tailraces.

Although it is obvious that the recoverable gold content of the gravel must pay a profit over operating costs, which usually range from 5 to 20 cents per yard, a surprising number of ventures in hydraulicking have failed because the promoters have not allowed for all the preparatory expenses noted above. Each yard of gravel mined must carry its share of this cost, therefore the size of the deposit is of utmost importance in considering a hydraulic mining venture.

Hydraulicking under suitable conditions is a low-cost method as it yields a larger production per man-shift than any other method except dredging. The initial investment required is less than that for dredging; hence, hydraulicking in small or medium-size deposits may be more economical even though dredging would result in a lower operating cost. When the operations are on a very large scale hydraulicking costs are lower than dredging costs on a comparable basis. Very clayey or bouldery gravels should be hydraulicked as dredging usually is unsatisfactory in such ground.

There is enough similarity in all hydraulic operations that no natural classifications of the method can be made. The methods of attacking the gravel vary too little to make any general distinctions. Factors such as conditions of the gravel, percentages of boulders and clay, grade of bedrock, and quantity and head of the hydraulic water affect the costs, but no general grouping is possible in accordance with any of these heads. In this paper hydraulic mines are placed in three groups: (1) Those where neither elevators nor pumps are used; (2) those using either hydraulic or Ruble elevators or both, and (3) those where water is pumped for washing.

Ditches

Open ditches are used commonly to bring water close to, yet high enough above, the mine to furnish a satisfactory pressure for the giants. At several hydraulic mines in the Western States and Alaska ditches 30 to 40 miles long have been built, and even relatively small operations usually have 5 to 10 miles of ditch line.

Hydraulicking is feasible with heads as low as 40 or 50 feet if the gravel is not tight; however, heads of 80 to 200 feet usually are desired, and if the gravel is cemented it is not uncommon to employ high-pressure equipment and heads ranging from 300 to 400 feet. This consideration fixes tentatively the location of the lower end of the ditch. Its final location may be a matter of compromise, as the head usually can be increased only at the cost of a lengthened ditch or a decrease in the grade. The latter reduces the quantity of water that can be carried in a ditch of given size.

The grades of most hydraulic-mine ditches lie between 4 and 8 feet per mile, or 3/4 to 1 1/2 feet per 1,000 feet. Early Californian ditches were run on much steeper grades, but the consequent high velocities caused erosion of the banks and serious breaks were common. Small ditches may be run at grades of 6 to 12 feet per mile without excessive velocities.

Practical velocities range between limits of which the minimum is determined by silting and the maximum by erosion. If the entering water contains sediment it may be deposited in the ditch. This should be guarded against by installing a sand trap near the intake and by designing for a velocity of not less than 1 foot per second. On the other hand, a velocity of more than 3 feet per second is apt to erode the channel and cause breaks. The following are recommended as maximum mean velocities for ditches in various materials:

Material	Mean velocity	
	Feet per second	Miles per hour
Loose sand	1	0.7
Sandy soil	2	1.4
Loam	3	2.0
Stiff clay, gravel	4	2.7
Coarse gravel, cobbles	5	3.4
Conglomerate, cemented gravel, soft rock	6	4.0
Hard rock	10	7.0

The above figures represent mean velocities, the corresponding bottom velocities being 20 or 30 percent lower and the corresponding surface velocities as measured by floating objects possibly being 25 to 35 percent higher.

The velocity, hence the capacity of a ditch, depends upon its slope, the nature of the walls, the size and shape of the water section, and the straightness and regularity of the channel. All these factors, except straightness and regularity of cross-section, are involved in the well-known Kutter formula:

$$V = \frac{\dfrac{1.811}{n} + 41.65 + \dfrac{0.00281}{S}}{1 + \dfrac{n}{\sqrt{R}}\left(41.65 + \dfrac{0.00281}{S}\right)} \times \sqrt{RS},$$

in which

V = mean velocity (in feet per second),
n = roughness coefficient,
S = sine of slope (fall divided by length),
R = hydraulic radius (area of water section divided by wetted perimeter of channel) in feet.

This formula ordinarily is applied by means of tables or charts. Figure 1 is a chart devised by Fred C. Scobey of the United States Department of Agriculture.[6] The proper value to use for the coefficient n is a matter of judgment. The following values of n are recommended by modern designers.[7]

Values of Roughness Coefficient n

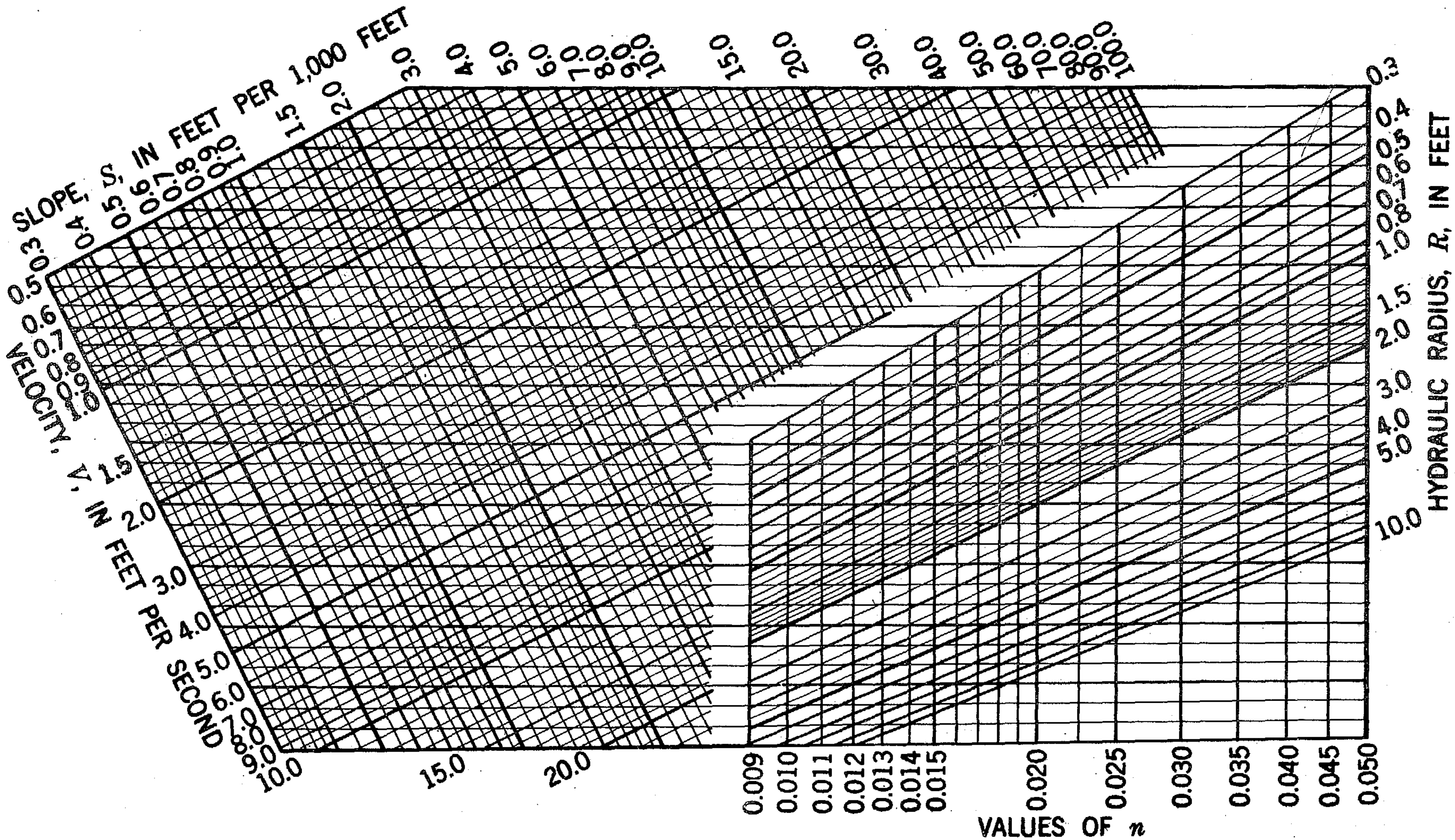
Surface	Best	Good	Fair	Bad
Coated cast-iron pipe	0.011	[1]0.012	[1]0.013	......
Commercial wrought-iron pipe:				
Black	.012	.013	.014	0.015
Galvanized	.013	.014	.015	.017
Smooth brass and glass pipe	.009	.010	.011	.013
Smooth lock-bar and welded "OD" pipe	.010	[1].011	[1].013	......
Riveted and spiral steel pipe	.013	[1].015	[1].017	......
Vitrified sewer pipe	.010-11	[1].013	.015	.017
Common clay drainage tile	.011	[1].012	[1].014	.017
Concrete pipe	.012	.013	[1].015	.016
Wood-stave pipe	.010	.011	.012	.013
Plank flumes:				
Planed	.010	[1].012	.013	.014
Unplaned	.011	[1].013	.014	.015
With battens	.012	[1].015	.016	......
Concrete-lined channels	.012	[1].014	[1].016	.018
Cement-rubble surface	.017	.020	.025	.030
Dry rubble surface	.013	.014	.015	.017
Semicircular metal flumes:				
Smooth	.011	.012	.013	.015
Corrugated	.0225	.025	.0275	.030
Canals and ditches:				
Earth, straight and uniform	.017	.020	[1].0225	.025
Rock cuts, smooth and uniform	.025	.030	[1].033	.035
Rock cuts, jagged and irregular	.035	.040	.045	......
Winding sluggish canals	.0225	[1].025	.0275	.030
Dredged earth channels	.025	[1].0275	.030	.033
Canals with rough, stony beds; weeds on earth banks	.025	.030	[1].035	.040
Earth bottom, rubble sides	.028	[1].030	[1].033	.035

[1]Values most used.

Earth canals for irrigation usually are designed with n = 0.025 or even 0.0225; however, the usual hydraulic-mine ditch is not straight, uniform, nor smooth, and probably the coefficient 0.030 or 0.035 should be applied. The velocities and discharges for a number of ditches of small to medium size shown in table 1 were calculated on the assumption that n = 0.035. Any increase in the assumed value of n results in an approximately equal percentage decrease in the calculated velocity, or a doubled percentage increase in the required slope.

6 Metcalf, Leonard, and Eddy, H. P., Sewerage and Sewage Disposal: McGraw-Hill, 2d ed., 1930, p. 130.

7 Part of a more complete list by Horton, R. E., Eng. News, vol. 75, 1916, p. 373; quoted by Metcalf, Leonard, and Eddy, H. P., Idem, p. 123.



Follow intersection of n and R along horizontal guide lines to intersection of S and V, or vice versa

Figure 1.—Diagram for solving Kutter formula to determine flow of water in open channels or pipes.

TABLE 1.--Calculated velocities and discharges for small and medium-size ditches

Bottom width, b.....feet		1			2			3				4				
Top width, t.....do.		2.0	2.5	3.0	3.0	4.0	5.0	4.0	5.0	6.0	7.0	6.0	7.0	8.0	9.0	10.0
Depth, d.....do.		.5	.75	1.0	.5	1.0	1.5	.5	1.0	1.5	2.0	1.0	1.5	2.0	2.5	3.0
Area, A.....sq. ft.		.75	1.31	2.0	1.25	3.0	5.25	1.75	4.0	6.75	10.0	5.0	8.25	12.0	16.25	21.0
Hydraulic radius, R.....		.31	.42	.52	.37	.62	.84	.40	.69	.93	1.16	.73	1.00	1.24	1.47	1.68
Slope, ft. per mile	Slope, ft. per 1,000 ft.	Velocity of flow, feet per second														
1	0.19	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.564	.....	0.502	0.601	0.687	0.767
2	.38	.....	.....	.....	.....	0.490	0.628	.....	0.528	0.682	.814	0.560	.723	.861	.98	1.098
3	.57	.....	.....	0.524	.....	.602	.771	.....	.652	.841	.999	.690	.890	1.058	1.20	1.346
4	.76	.....	0.505	.605	.....	.699	.895	.....	.758	.973	1.159	.799	1.032	1.230	1.40	1.558
5	.95	.....	.565	.680	0.500	.785	1.005	0.537	.851	1.094	1.299	.901	1.158	1.376	1.570	1.744
6	1.14	.....	.618	.746	.550	.862	1.103	.589	.933	1.199	1.423	.984	1.270	1.511	1.72	1.912
7	1.33	0.517	.670	.806	.595	.931	1.194	.637	1.008	1.296	1.539	1.068	1.373	1.633	1.85	2.066
8	1.52	.552	.717	.862	.636	.997	1.277	.683	1.080	1.390	1.647	1.141	1.468	1.747	1.98	2.212
9	1.70	.586	.763	.918	.678	1.057	1.357	.725	1.148	1.476	1.749	1.214	1.560	1.852	2.11	2.348
10	1.89	.619	.804	.968	.714	1.116	1.430	.765	1.209	1.555	1.845	1.277	1.644	1.955	2.226	2.471
11	2.08	.650	.841	1.015	.749	1.169	1.499	.802	1.270	1.629	1.932	1.339	1.725	2.048	2.33	2.590
12	2.27	.681	.882	1.061	.785	1.225	1.568	.842	1.328	1.704	2.024	1.402	1.805	2.145	2.43	2.714
15	2.84	.762	.986	1.186	.876	1.370	1.756	.941	1.484	1.909	2.263	1.569	2.018	2.397	2.72	3.031
20	3.79	.882	1.142	1.372	1.014	1.584	2.027	1.086	1.715	2.204	2.612	1.811	2.330	2.770	3.152	3.502
		Discharge, cubic feet per second[1]														
1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	5.60	.....	4.12	7.20	11.21	16.2
2	.....	.....	.....	.....	.....	1.47	3.31	.....	2.12	4.59	8.10	2.80	5.94	10.32	15.93	23.1
3	.....	.....	.....	1.04	.....	1.80	4.04	.....	2.60	5.67	10.00	3.45	7.34	12.72	19.50	28.4
4	.....	.....	0.66	1.20	.....	2.10	4.72	.....	3.04	6.55	11.60	4.00	8.50	14.76	22.75	32.8
5	.....	.....	.73	1.36	0.62	2.34	5.25	0.94	3.40	7.36	13.00	4.50	9.57	16.56	25.51	36.5
6	.....	.....	.81	1.50	.69	2.58	5.78	1.03	3.72	8.10	14.20	4.90	10.48	18.12	27.95	40.1
7	.....	0.39	.88	1.62	.75	2.79	6.25	1.12	4.04	8.78	15.40	5.35	11.30	19.56	30.06	43.5
8	.....	.41	.94	1.72	.80	3.00	6.72	1.19	4.32	9.38	16.50	5.70	12.13	21.00	32.18	46.4
9	.....	.44	1.00	1.84	.85	3.18	7.14	1.26	4.60	9.99	17.50	6.05	12.87	22.20	34.29	49.4
10	.....	.46	1.05	1.94	.89	3.36	7.51	1.33	4.84	10.53	18.40	6.40	13.53	23.52	36.24	51.9
11	.....	.49	1.10	2.04	.94	3.51	7.88	1.40	5.08	11.00	19.30	6.70	14.19	24.60	37.86	54.4
12	.....	.51	1.15	2.12	.98	3.66	8.24	1.47	5.32	11.48	20.20	7.00	14.85	25.68	39.49	56.9
15	.....	.57	1.30	2.38	1.10	4.11	9.24	1.64	5.92	12.89	22.60	7.85	16.66	28.80	44.20	63.6
20	.....	.66	1.49	2.74	1.26	4.74	10.66	1.91	6.88	14.85	26.10	9.05	19.22	33.24	51.19	73.5

[1]To convert to miner's inches multiply by 40.

TABLE 1.- Calculated velocities and discharges for small and medium-size ditches - Continued

Bottom width, b .... feet		5							6				8		
Top width, t .... do.		7.0	8.0	9.0	10.0	11.0	12.0	13.0	10.0	12.0	14.0	16.0	12.0	16.0	20.0
Depth, d .... do.		1.0	1.5	2.0	2.5	3.0	3.5	4.0	2.0	3.0	4.0	5.0	2.0	4.0	6.0
Area, A .... sq. ft.		6.0	9.75	14.0	18.75	24.0	29.75	36.0	16.0	27.0	40.0	55.0	20.0	48.0	84.0
Hydraulic radius, R		.77	1.06	1.31	1.55	1.78	2.00	2.21	1.37	1.86	2.31	2.73	1.46	2.48	3.36
Slope, ft. per mile	Slope, ft. per 1,000 ft.	Velocity of flow, feet per second													
1	0.19	0.402	0.525	0.628	0.720	0.802	0.880	0.952	0.653	0.831	0.988	1.124	0.686	1.047	1.318
2	.38	.57	.75	.90	1.02	1.14	1.24	1.35	.93	1.18	1.40	1.58	.95	1.48	1.87
3	.57	.71	.92	1.10	1.26	1.41	1.53	1.658	1.14	1.45	1.72	1.95	1.20	1.82	2.29
4	.76	.83	1.07	1.28	1.46	1.62	1.77	1.92	1.32	1.68	1.98	2.25	1.40	2.10	2.64
5	.95	.934	1.208	1.440	1.642	1.825	1.991	2.152	1.491	1.892	2.223	2.522	1.570	2.352	2.941
6	1.14	1.02	1.32	1.57	1.79	1.99	2.18	2.35	1.63	2.07	2.44	2.77	1.72	2.58	3.22
7	1.33	1.108	1.43	1.70	1.94	2.16	2.35	2.54	1.76	2.23	2.64	2.99	1.85	2.79	3.48
8	1.52	1.18	1.53	1.82	2.07	2.31	2.52	2.72	1.88	2.39	2.82	3.20	1.98	2.98	3.72
9	1.70	1.25	1.63	1.94	2.20	2.45	2.67	2.88	2.00	2.54	2.99	3.39	2.10	3.16	3.94
10	1.89	1.319	1.717	2.043	2.326	2.582	2.822	3.040	2.114	2.676	3.146	3.566	2.221	3.326	4.154
11	2.08	1.39	1.80	2.14	2.44	2.71	2.95	3.19	2.22	2.81	3.30	3.75	2.32	3.50	4.35
12	2.27	1.45	1.88	2.23	2.55	2.82	3.09	3.33	2.32	2.94	3.45	3.92	2.43	3.66	4.54
15	2.84	1.62	2.10	2.50	2.85	3.15	3.45	3.72	2.59	3.28	3.85	4.37	2.72	4.07	5.07
20	3.79	1.878	2.433	2.892	3.292	3.655	3.990	4.304	2.995	3.788	4.454	5.038	3.146	4.705	5.869
		Discharge, cubic feet per second[1]													
1	....	2.4	5.1	8.8	13.5	19.2	26.2	34.2	10.4	22.4	39.6	61.6	13.8	50.4	110.9
2	....	3.4	7.3	12.6	19.1	27.4	36.9	48.6	14.9	31.9	56.0	86.9	19.0	71.0	157.1
3	....	4.3	9.0	15.4	23.6	33.8	45.5	59.8	17.8	39.2	68.8	107.2	24.0	87.4	192.4
4	....	5.0	10.4	17.9	27.4	38.9	52.7	69.1	21.1	45.4	79.2	123.8	28.0	100.8	221.8
5	....	5.6	11.8	20.2	30.8	43.7	59.2	77.4	23.8	51.0	88.8	138.6	31.4	112.8	247.0
6	....	6.1	12.9	22.0	33.6	47.8	64.9	84.6	26.1	55.9	97.6	152.4	34.2	123.8	270.5
7	....	6.7	13.9	23.8	36.4	51.8	69.9	91.4	28.2	60.2	105.6	164.4	37.0	133.9	292.3
8	....	7.1	14.9	25.5	38.8	55.4	75.0	97.9	30.1	64.5	112.8	176.0	39.6	143.0	312.5
9	....	7.5	15.9	27.2	41.2	58.8	79.4	103.7	32.0	68.6	119.6	186.4	42.0	151.7	331.0
10	....	7.9	16.8	28.6	43.7	61.9	83.9	109.4	33.8	72.4	126.0	196.4	44.4	159.8	348.6
11	....	8.3	17.6	30.0	45.8	65.0	87.8	114.8	35.5	75.9	132.0	206.2	46.4	168.0	365.4
12	....	8.7	18.3	31.2	47.8	67.7	91.9	119.9	37.1	79.4	138.0	215.6	48.6	175.7	381.4
15	....	9.7	20.5	35.0	53.4	75.6	102.6	133.9	41.4	88.6	154.0	240.4	54.4	195.4	425.9
20	....	11.3	23.7	40.5	61.7	87.8	118.7	154.8	48.0	102.3	178.0	277.2	63.0	225.6	493.1

[1]To convert to miner's inches multiply by 40.

Thus the velocities and capacities shown in table 1 might be increased 15 or 20 percent for ditches in unusually good condition.

Although the shape of the ditch has a bearing on its capacity, in practice the section is influenced more by the method of excavation. However, for a given area, the section should be so shaped as to have the largest hydraulic radius consistent with economical construction. The usual earth or gravel ditch for hydraulic mines has a trapezoidal section, with a flat bottom 2 to 10 feet wide, sides sloping about 45°, and a water depth of one third to three quarters the bottom width. The sides should be excavated at a slope that will be stable in use, otherwise caving will result in irregularity of section and consequent loss of capacity. The following side slopes are recommended for ditches in various materials:

Material	Side slopes	
	Horizontal to vertical	Degrees
Firm soil, coarse firm gravel	1 : 1	45
Ordinary soil, loose or fine gravel	1 1/2 : 1	35
Loose, sandy soil	2 : 1	25

Wimmler[8] who tabulates data on 35 Alaskan ditches, states that side slopes of 45 to 65° are common but that the higher slopes cut down quickly.

On steep hillsides relatively steeper sides and deeper sections may be cut if the soil is firm to avoid excessive excavation on the uphill side of the ditch. In rock the sides may be vertical; the width should be twice the water depth, as in rectangular channels this results in the least excavation for a given capacity and slope. Likewise, in rock the size may be decreased and the grade increased, thus reducing the yardage of rock excavation. Ditches should be designed to run not more than three fourths full, allowing 1 to 3 feet of freeboard.

In porous soil considerable water is lost by seepage. Peele[9] quotes Etcheverry as stating that seepage losses range from as little as 0.25 cubic foot per square foot of wetted surface per 24 hours in impervious clay loam to 1.0 cubic foot in sandy loam and 2 to 6 cubic feet in gravelly soils. It is easily computed that a medium-size ditch, 5 miles long, carrying 1,000 or 2,000 miner's inches, may lose 5 or 10 percent of the intake water by seepage, even in good soil, and in porous soil, as much as 20 percent. Remedies where the loss is serious are to decrease the size of ditch and increase the velocity; to reduce the velocity to a point at which the silt will deposit and tend to seal the ground; to line the channel with sod, canvas, or concrete; or to substitute flumes for ditches. According to Wimmler, sod lining often is used in frozen muck in Alaska, sometimes with entire success.

Very few ditches have been built in recent years, and no modern costs are available. Many methods are available for such work, ranging from hand-shovel and pick work to excavation by power shovel or mechanical ditchers. A common method is to plow the surface and excavate as near to grade and correct section as possible with teams and scrapers, then finish by hand. Some instances have been noted where hydraulic giants were used for ditch excavation. This, of course, is possible only when water is available from a higher ditch line. Incidentally the hydraulic miner uses high-pressure water for excavating wherever practicable.

The alinement of ditches should be such that excavation to grade will provide just enough bank material to form a channel of the desired size. Wherever the water level is to

8 Wimmler, N. L., Placer-Mining Methods and Costs in Alaska: Bull. 259, Bureau of Mines, 1927, pp. 40-56.

9 Peele, Robert, Mining Engineers' Handbook: John Wiley & Sons, 2d ed., 1927, p. 2147.

be above the original ground surface it is well to plow the surface before excavation starts to form an impervious joint between the bank and ground. If the material is not such as to form tight banks it may be advisable to excavate the entire water section below the original surface. The grade must be maintained exactly and the desired section adhered to as closely as possible, as all irregularities have a retarding effect on the flow. Curves should be made smooth and regular for the same reason.

If there is danger of water from floods or other sources filling the ditch beyond capacity, spillways must be provided at intervals to prevent breaks in the line which would stop operation and be costly to repair.

Measuring weirs.- The simplest method of accurately measuring a flow of water in a stream or ditch is by means of a weir. Numerous types of weirs are used, and there are many formulas for calculating the flow over weirs.

A common type of weir is shown in figure 2. The width of the weir notch should be at least six times the depth of the water flowing over the crest. The bottom of the notch should be level and the sides vertical. The weir notch is beveled on the downstream side so as to leave a sharp edge on the upstream side. The weir should be installed so that the water in the pond above is comparatively still. It must also be high enough so that there is free access of air to the underside of the overflow sheet of water. A stake is driven in the pond 5 or 6 feet above the weir with the top of the stake level with the notch of the weir. The depth of flow over the weir is measured with a rule or square placed on top of the stake. The Francis formula is commonly used for calculating the flow.

$$\underline{Q} = 3.33\ \underline{wd}^{3/2},$$

where

$\underline{Q}$ = quantity of water in cubic feet per second,
$\underline{w}$ = width of notch in weir,
$\underline{d}$ = depth of water going over weir.

The discharge per foot of length of thin-edge weirs is cubic feet per second and miner's inches for depths of 1/8 inch to 24 7/8 inches is shown in table 2. The table is compiled from the above formula.

Flumes

As already stated, most hydraulic-mine ditch lines contain some flume sections. Flumes may be necessary where the line passes around cliffs or over ravines or desirable over porous or shattered ground where a ditch would lose much water or tend to cause slides. On steep hillsides or where ditching would require much costly rock excavation a flume may prove economical; finally, the cost of the line may be lessened and considerable saving made in the total fall by building a flume on trestles across valleys instead of ditching the greater distance around the head.

The same conditions should be considered in designing a flume as in designing a ditch, and the Kutter formula (see p. 5) applies equally to both. The formula is used most conveniently in the form of tables or charts. (See fig. 1.)

The low friction coefficient of board flumes may be used to advantage either by building a flume of smaller section or by decreasing the grade below that of the ditch line. If the latter is done a saving in head may be made at the mine. Usually, however, smaller sections and higher velocities are used than for the ditch line. The width of the flume should be twice the water depth and a freeboard of 1 to 2 feet allowed. According to Egleston[10] the usual water velocity is 3 to 6 miles per hour (4 to 9 feet per second). The same

10 Egleston, Thomas, The Metallurgy of Silver, Gold, and Mercury in the United States: John Wiley & Sons, New York, 1890, p. 152.

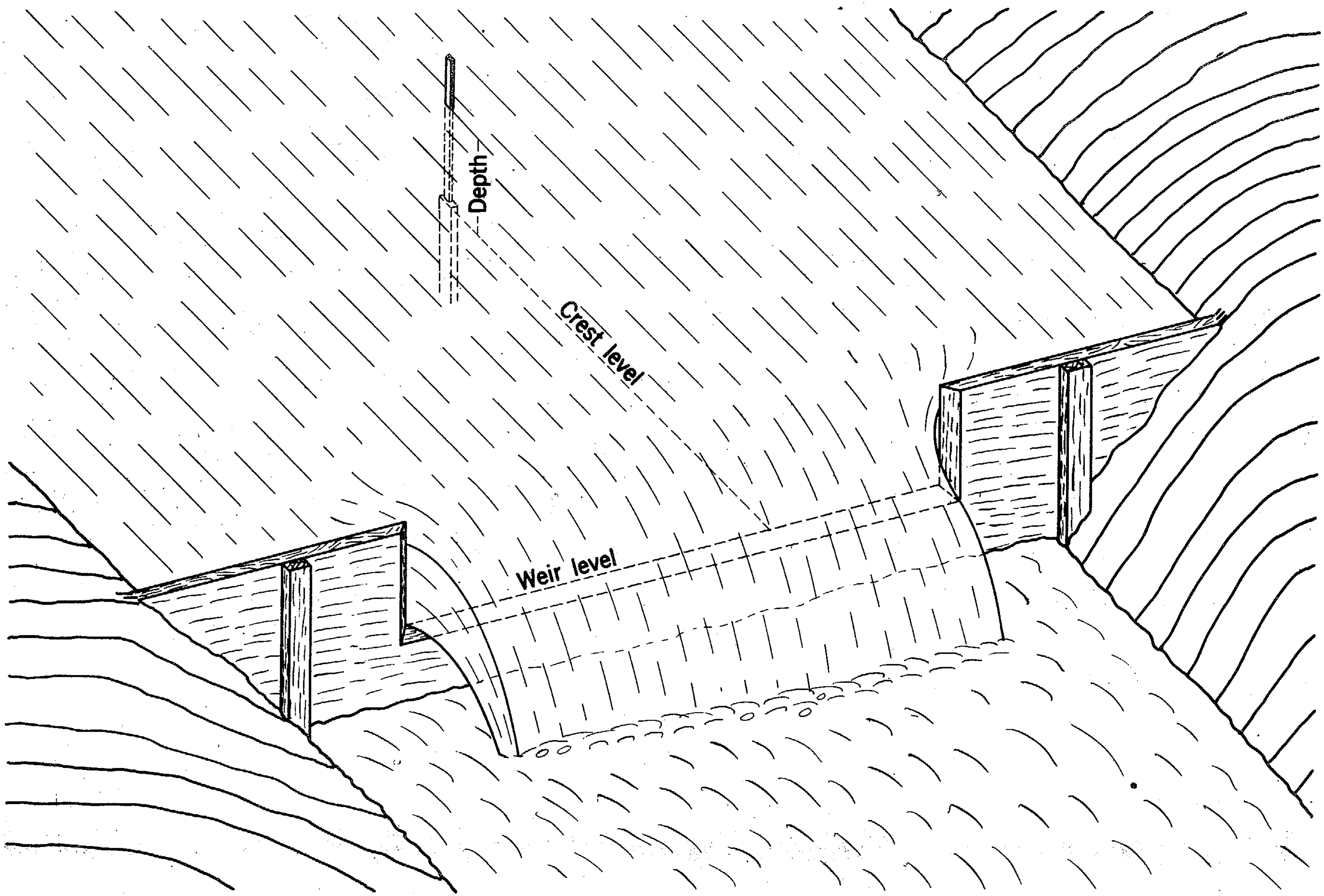


Figure 2.—Weir for small stream.

TABLE 2.- Discharge per foot of length of thin-edge weirs, in cubic feet per second and miner's inches (40 miner's inches equals 1 cubic foot per second); calculated from formula, $Q = 333\ wd^{3/2}$

Head inches	0		1/8		1/4		3/8		1/2		5/8		3/4		7/8	
	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches
0	0	0	0.003	0.1	0.01	0.4	0.02	0.8	0.03	1.2	0.04	1.6	0.05	2.0	0.07	2.8
1	0.08	3.2	.10	4.0	.11	4.4	.13	5.2	.15	6.0	.17	6.8	.18	7.2	.20	8.0
2	.23	9.2	.25	10.0	.27	10.8	.29	11.6	.32	12.8	.34	13.6	.37	14.8	.39	15.6
3	.42	16.8	.44	17.6	.47	18.8	.49	19.6	.52	20.8	.55	22.0	.58	23.2	.61	24.4
4	.64	25.6	.67	26.8	.70	28.0	.73	29.2	.76	30.4	.80	32.0	.83	33.2	.86	34.4
5	.89	35.6	.92	36.8	.96	38.4	.99	39.6	1.03	41.2	1.06	42.4	1.10	44.0	1.14	45.6
6	1.18	47.2	1.22	48.8	1.25	50.0	1.29	51.6	1.33	53.2	1.37	54.8	1.41	56.4	1.44	57.6
7	1.48	59.2	1.52	60.8	1.56	62.4	1.60	64.0	1.64	65.6	1.68	67.2	1.72	68.8	1.77	70.8
8	1.81	72.4	1.86	74.4	1.90	76.0	1.94	77.6	1.99	79.6	2.03	81.2	2.08	83.2	2.12	84.8
9	2.16	86.4	2.21	88.4	2.25	90.0	2.29	91.6	2.34	93.6	2.39	95.6	2.44	97.6	2.49	99.6
10	2.54	102.	2.58	103.	2.63	105.	2.68	107.	2.73	109.	2.78	111.	2.83	113.	2.87	115.
11	2.92	117.	2.97	119.	3.03	121.	3.07	123.	3.13	125.	3.18	127.	3.23	129.	3.28	131.
12	3.33	133.	3.38	135.	3.44	138.	3.49	140.	3.54	142.	3.59	143.	3.65	146.	3.70	148.
13	3.76	150.	3.81	152.	3.86	154.	3.92	157.	3.97	159.	4.03	161.	4.08	163.	4.14	166.
14	4.20	168.	4.25	170.	4.31	172.	4.36	174.	4.42	177.	4.48	179	4.54	182.	4.59	184.
15	4.65	186.	4.71	188.	4.77	191.	4.83	193.	4.89	196.	4.95	198.	5.01	200.	5.07	203.
16	5.13	205.	5.19	208.	5.25	210.	5.31	212.	5.37	215.	5.43	217.	5.49	220.	5.55	222.
17	5.62	225.	5.68	227.	5.74	230.	5.80	232.	5.87	235.	5.93	237.	5.99	240.	6.06	242.
18	6.12	245.	6.18	247.	6.25	250.	6.31	252.	6.37	255.	6.44	258.	6.50	260.	6.57	263.
19	6.63	265.	6.70	268.	6.77	271.	6.83	273.	6.90	276.	6.96	278.	7.03	281.	7.10	284.
20	7.17	287	7.23	289.	7.30	292.	7.37	295.	7.44	298.	7.50	300.	7.57	303.	7.64	306.
21	7.71	308.	7.78	311.	7.85	314.	7.92	317.	7.99	320.	8.06	322.	8.13	325.	8.20	328.
22	8.27	331.	8.34	334.	8.41	336.	8.48	339.	8.55	342.	8.62	345.	8.69	348.	8.76	350.
23	8.84	354.	8.91	356.	8.98	359.	9.05	362.	9.13	365.	9.20	368.	9.27	371.	9.35	374.
24	9.42	377.	9.50	380.	9.57	383.	9 64	386	9.72	389.	9.79	292.	9.87	395.	9.94	398.

author gives the range in grade of 28 prominent California flumes as 9 to 18 2/3 feet per mile. The extreme range of 86 well-known flumes in the Western States was 5 to 53 feet per mile, the usual range 10 to 18, and the average slightly under 14. Bowie[11] states that grades of 25 to 35 feet per mile are used where practicable. Such steep grades would permit the use of a relatively small flume section, but the authors believe that usually they would involve inconveniently high velocities; moreover, a longer flume would be required tM give the same head.

The construction of wooden flumes has changed little since the early days of placer mining in California. Figure 3,A,[12] illustrates the early type of box. This was built in 12- or 16-foot sections of 1 1/2- to 2-inch lumber, 12 to 24 inches wide. The longitudinal joints were made tight by nailing over each a batten 1/2 inch thick and 3 or 4 inches wide. Figure 3,B,[13] illustrates a flume built about 1930 for water power; it carries about 600 miner's inches on the flat grade of one fifth foot per 1,000 feet and would serve excellently for a small hydraulic water-supply line. It differs in construction from the other type illustrated chiefly in having splines between all the boards of the boxes and lacking framing in the sills and caps. It was built over 6,800 feet of rugged country at a total cost of $2.50 per foot.

Where the flume is on grade the box units should be set on stringers laid on a carefully cleared and graded surface or on a bench cut in the hillside. Trees or branches that might fall and wreck the flume should be removed. In cold climates the flume may be covered and heaped with earth to prevent freezing. Where the flume is on trestles a walk must be provided; usually a line of plank is nailed over the caps or on alternate sills extended a couple of feet to one side of the box.

The grade must be uniform, and at curves the outer edge of the flume should be raised sufficiently for the smoothest possible flow of water, the elevation being determined by trial.

Three-foot iron placer pipe cut in two lengthwise has been used successfully for flumes at placer mines in British Columbia.

Diversion Dams and Reservoirs

Diversion dams for hydraulic ditch lines usually are of earth-filled timber cribs or rock-filled cribs faced with boards. Small streams often are dammed by throwing logs across and facing the upstream side with boards. Diversion dams usually are only a few feet high but should be built where possible on solid rock or hardpan, sufficiently wide to be stable and provided with suitable spillways to prevent erosion and scouring out of the foundation.

At mines where the water supply is insufficient for 24-hour operation or where the stream flow is less than is needed to operate at the desired capacity for one shift, reservoirs often are used. If it is impracticable to have the reservoir in the stream itself above the diversion dam, it is usually located at the lower end of the ditch, just above the intake to the pipe lines. Reservoirs may be built by damming a canyon, by excavating a basin on level ground, or merely by enlarging a section of the lower end of the ditch. As a

---

11 Bowie, A. J., Jr., A Practical Treatise on Hydraulic Mining in California: D. Van Nostrand Co., New York, 1889, p. 143.

12 Bowie, A. J., Work cited, p. 143.

13 Carman, J. B., Milling Methods at the Questa Concentrator of the Molybdenum Corporation of America, Questa, N. Mex.: Inf. Circ. 6551, Bureau of Mines, 1932, p. 14.

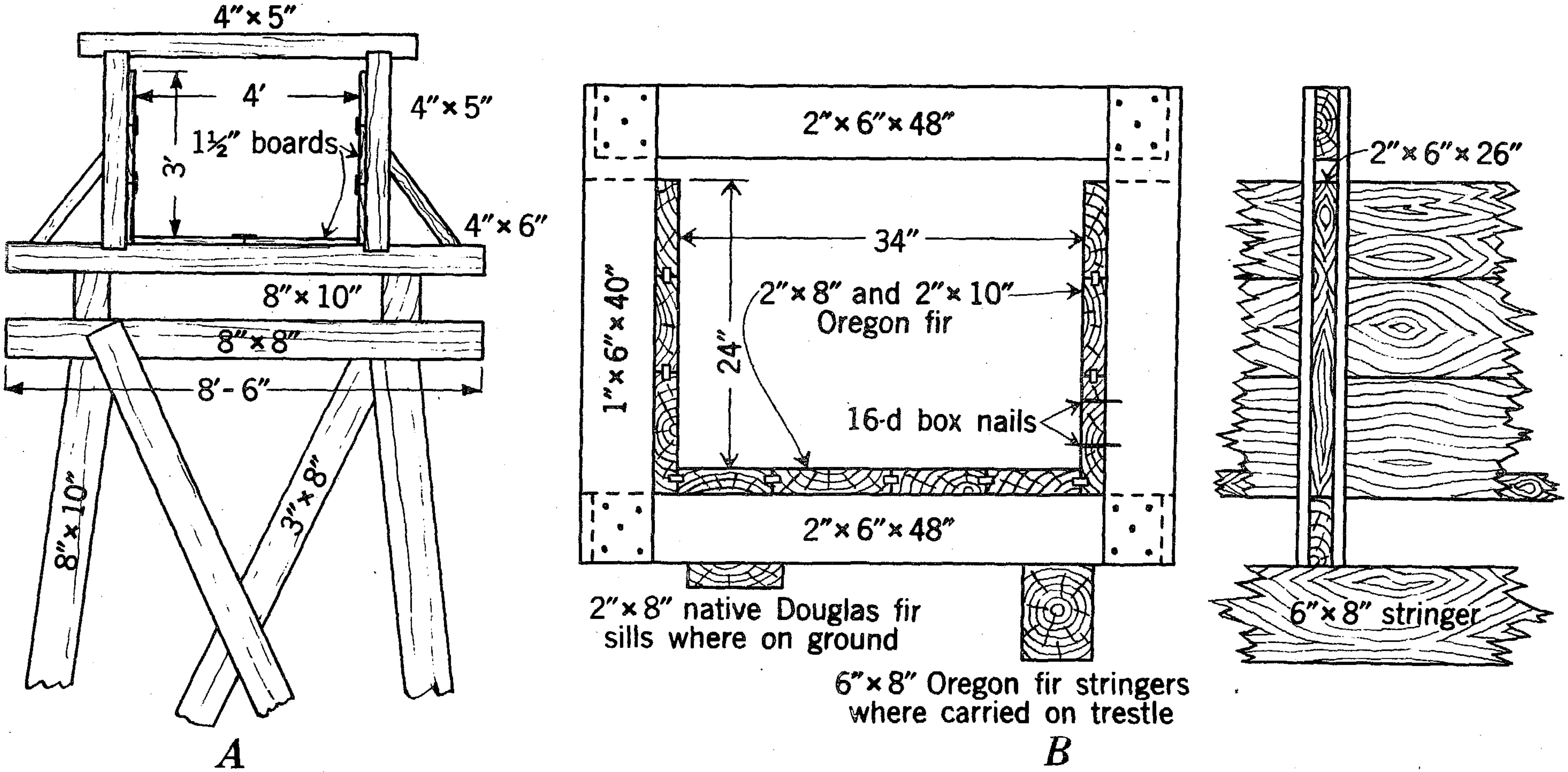


Figure 3.—Flume for hydraulic mines: *A*, Flume and trestle used in California; *B*, flume at Questa, N.Mex.

reservoir break might be disastrous to a mine lying directly below it, the work should be done carefully, all leakage checked, suitable gates and spillways provided, and regular inspection maintained.

As both diversion dams and reservoirs tend to act as settling basins it may be convenient to provide gates close to the bottom through which sediment may be flushed as often as necessary.

Mining Equipment

The chief items of equipment used in most hydraulic mines are pipe lines to carry the water under pressure to the places where it is used; giants or monitors for cutting, washing, and driving the gravel; derricks, winches, or other machinery for handling boulders; and sluice boxes for saving the gold and disposing of the tailings. Picks, shovels, and forks are the common hand tools used at placer mines. Power drills run by compressed air or steam may be used if the gravel contains an excessive quantity of large boulders. However, hand drills are used at most mines to drill boulders and sometimes to drill cemented gravel or hard-clay strata. Churn drills are employed occasionally for drilling cemented gravel ahead of hydraulicking.

Pipe Lines

Pipe.- As described previously, ditch lines are used to bring the necessary water to a convenient point above the mine. From that point a pipe line is laid down the hillside to the pit. Occasionally, where the grade of a creek is steep, the water will be diverted from the stream directly into a pipe line. Although wooden stave pipe is used at a few properties, steel pipe is preferred at nearly all hydraulic mines.

Pipe may be made from steel sheets in the mine shops or bought from pipe manufacturers. Unless a large quantity of pipe is to be used or transportation is difficult it usually is more economical to buy the pipe already made up. Various types of steel pipe are used, but light-weight riveted pipe with slip or stove-pipe joints generally is preferred in the Western States as it is cheaper, lighter, and more easily transported and installed than other steel pipe.

Spiral riveted pipe will stand greater pressures and harder usage than the straight riveted pipe, but it is more expensive. Moreover, flange joints, which are an added expense, generally are used with the spiral pipe. Ordinary riveted pipe of 10 to 16 United States standard gage material 7 to 46 inches in diameter was being used in western mines in 1932; the diameters used most were 36, 32, 24, 22, 18, 15, 11, and 9 inches. Large pipes are easily damaged if made of material thinner than 14 gage. Usually two or more diameters and gages of pipe are used in the same line, mainly as a matter of convenience since this permits nesting in transit. A saving may be made in ocean freight and occasionally in truck hauls by nesting the pipe.

Slip-joint pipe is made in standard lengths of 19 feet 7 1/2 inches each. The sections may be made longer or shorter, however, as required by transportation purposes, provided they are in multiples of 4 feet. The extra pipe required for a slip joint is about 3 inches per section. The standard length of sections of spiral riveted pipe is 20 feet. Placer pipe usually is coated inside and out with an asphalt paint.

A pipe of smaller diameter will withstand a greater pressure than a larger pipe of the same wall thickness; therefore it is common practice to use smaller diameters as the pressure increases. Reducing the diameter increases the friction in the pipe, and a balance must be struck between loss of effective head in the pipe line and first cost of the line. Branch lines usually have a smaller diameter than the main supply lines.

Table 3 shows the weight, strength, and cost of riveted pipe with slip joints. Prices and weights are for pipe double dipped with asphaltum coating.

The number of feet and sections of various diameters of slip-joint pipe required to make a carload are shown in table 4.

TABLE 3.- Weights and prices of slip-joint straight-riveted pipe[1]

Pipe diameter, inches	Gage no.	Weight per foot, pounds	Safe head, feet	Price per foot[2]	Pipe diameter, inches	Gage no.	Weight per foot, pounds	Safe head, feet	Price per foot[2]
6	16	5.3	340	$0.33	16	16	13.4	158	$0.74
6	14	6.4	490	.39	16	14	16.3	198	.90
					16	12	22.3	277	1.25
7	16	6.2	325	.38	16	10	28.2	356	1.60
7	14	7.4	450	.45					
					18	16	15.1	140	.82
8	16	7.0	315	.42	18	14	18.3	175	1.00
8	14	8.4	394	.50	18	12	24.9	246	1.39
8	12	11.6	553	.67	18	10	31.6	316	1.79
9	16	7.8	280	.46	20	16	16.7	126	.92
9	14	9.4	350	.54	20	14	20.3	158	1.13
9	12	12.9	490	.73	20	12	27.5	221	1.54
					20	10	35.0	284	1.99
10	16	8.6	252	.51					
10	14	10.4	316	.62	22	16	18.3	115	1.01
10	12	14.3	443	.84	22	14	22.2	143	1.24
10	10	18.1	568	1.09	22	12	30.1	201	1.69
					22	10	38.5	258	2.15
11	16	9.5	230	.55					
11	14	11.4	287	.66	24	16	19.8	105	1.10
11	12	15.6	402	.91	24	14	24.2	131	1.39
11	10	19.7	517	1.17	24	12	32.6	184	1.82
					24	10	41.9	237	2.31
12	16	10.3	210	.59					
12	14	12.4	263	.71	26	14	26.2	121	1.45
12	12	16.9	368	.98	26	12	35.1	170	1.97
12	10	21.4	473	1.27	26	10	45.3	219	2.51
13	16	11.1	194	.63	28	14	28.2	113	1.57
13	14	13.4	243	.75	28	12	37.6	158	2.13
13	12	18.3	340	1.05	28	10	48.7	203	2.70
13	10	23.1	437	1.35					
					30	14	30.1	106	1.66
14	16	11.8	180	.66	30	12	40.1	147	2.24
14	14	14.4	226	.80	30	10	52.0	189	2.84
14	12	19.6	317	1.12					
14	10	24.8	406	1.44	32	12	42.5	137	2.38
					32	10	55.4	177	3.00

[1]Furnished by Western Pipe & Steel Co., San Francisco, Calif.
[2]F.o.b. San Francisco, as of October 1932.

TABLE 3.- Weights and prices of slip-joint straight-riveted pipe[1] - Continued

Pipe diameter, inches	Gage no.	Weight per foot, pounds	Safe head, feet	Price per foot[2]	Pipe diameter, inches	Gage no.	Weight per foot, pounds	Safe head, feet	Price per foot[2]
15	16	12.6	168	.71					
15	14	15.4	211	.85	34	12	45.0	129	2.52
15	12	20.9	297	1.19	34	10	58.7	166	3.19
15	10	26.5	379	1.52					
					36	12	47.5	122	2.65
					36	10	62.0	156	3.37

[1]Furnished by Western Pipe & Steel Co., San Francisco, Calif.
[2]F.o.b. San Francisco, as of October 1932.

TABLE 4.- Maximum quantity of slip-joint water pipe that can be loaded on flat car 8 feet 6 inches wide by 40 feet long, using side stakes 10 feet high

Diameter, inches	Maximum number	
	Sections	Feet[1]
4	1,152	22,320
5	760	14,725
6	544	10,540
7	420	8,137.50
8	310	6,006.25
9	264	5,115
10	220	4,262.50
11	180	3,487.50
12	144	2,790
13	128	2,480
14	112	2,170
15	98	1,898.75
16	84	1,627.50
18	60	1,162.50
20	60	1,162.50
22	40	775
24	40	775
26	40	775
28	30	581.25
30	30	581.25
32	30	581.25
34	30	581.25
36	30	581.25
48	10	193.75

[1]For weights per lineal foot see table 3.

The minimum carload weight is 20,000 and the maximum 80,000 pounds in California. Carload shipments take fifth class rate. Less than carload shipments of pipe up to 12 inches in diameter take third class and over this diameter one and one half times the first-class rate.

Table 5 shows the weight, strength, and cost of spiral riveted pipe and the weight and cost of two types of flanges used for joining the pipe. The pipe is asphalted inside and out and comes in lengths up to 40 feet. The prices quoted are for pipe with plain ends and in carload lots.

As used pipe is available in nearly all placer districts very little new pipe is purchased except for installations of some magnitude. There are no established prices for used pipe.

The Y's and T's needed for branch lines usually are purchased from the pipe manufacturers. A header box may be used when more than two branches are taken out at one place.

Joints and valves.- In making the pipe with slip joints the diameter of one end is slightly contracted. (See fig. 4,A.) This joint in straight pipe lines will withstand most pressures encountered at placer mines. Slip joints, however, become battered from frequent laying, and other types are desirable where the pipe is moved often.

Flanged or bolted joints are used in some pits for siphons and in very high-pressure lines. Three different types are listed in table 5. The Taylor flanged joint is of forged steel and is welded to the pipe; the price includes bolts and gaskets. The 8-inch size is good for a working pressure of 200 pounds; 8- to 12-inch size, 125 pounds; 12- to 20-inch size, 110 pounds; and above 20-inch size, 75 pounds. The American flange also is complete and attached to the pipe. Most sizes are good for a working pressure of 300 pounds per square inch.

The bolted joint is complete with rolled-steel retaining sleeves and forged-steel flanges. It affords from 1° to 3° flexibility at each joint. Tightness is maintained by adjusting the bolts, even at high pressures. Anchoring of the pipe line is necessary at high pressures and in large sizes to prevent pulling out of the joint. This joint can be used in making moderate curves in lines.

Four different kinds of joints are illustrated in figure 4. As stated before, figure 4,A, is a slip joint commonly used in the Western States. Figure 4,B,[14] shows a forged steel slip-on joint welded to the pipe to form a flush square end. A gasket is used. Similar joints may be riveted to the pipe instead of welded. Figure 4,C,[14] shows a bolted socket joint. This connection permits both deflection and expansion; the pipe, however, must be anchored at curves. When a retaining shoulder is used on the inside pipe a small deflection is possible, but the joint loses the ability to allow expansion. Figure 4,D,[15] shows a bolted elbow joint for making a turn in a pipe.

The collar and ringed castings are identical with those furnished on the regular bolted joint made by the same company. The elbow is made by placing relatively inexpensive angle castings between the rings. The degree of turn is governed by the shape of the center casting. Lengths and list prices of the regular bolt joint, as of January 1933, are as follows:

---

14 From catalog of Taylor Forge & Pipe Co., Chicago, Ill.

15 From catalog of Abendroth & Root Manufacturing Co., Chicago, Ill.

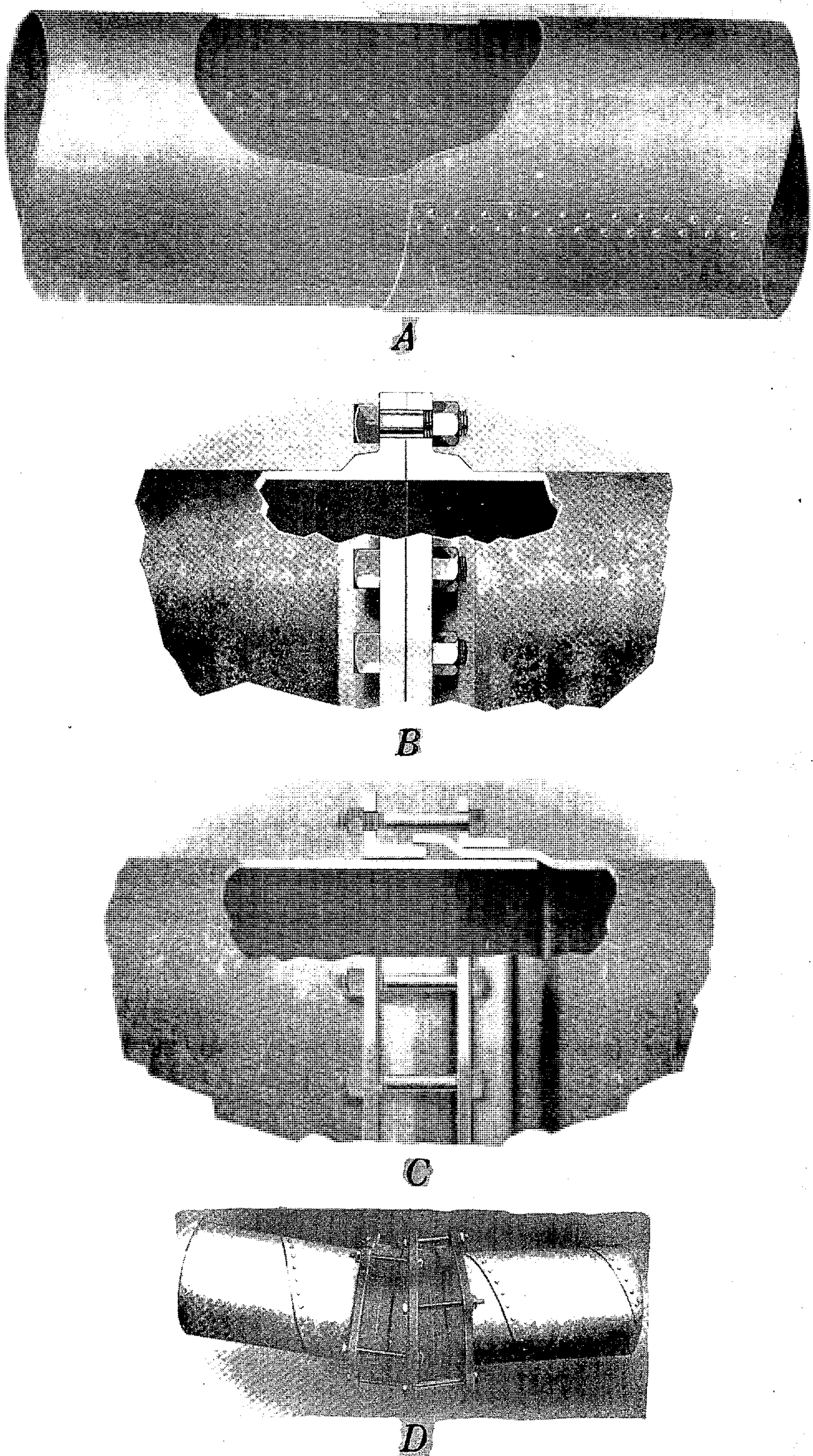


Figure 4.—Joints for placer pipe: *A*, Slip joint; *B*, forged-steel slip joint, on flanges welded to pipe; *C*, bolted-socket joint without retaining shoulder; *D*, bolted joint elbow.

TABLE 5.- Weights and prices of spiral riveted pipe and of connecting joints[1]

Pipe						Joints, complete					
Diameter inches	Gage no.	Weight per foot lb.	Bursting pressure lb. per sq. in.[2]	Flow cu. ft. per min. at 12 ft. per sec.	Cost per foot[3]	Taylor flanged		American flanged		Bolted	
						Cost each	Weight lb.	Cost each[3]	Weight lb.	Cost each[3]	Weight lb.
8	16	7.1	935	250	$0.53	$4.14	16.4	$5.21	45.0	$3.09	25.7
8	14	8.8	1,170	250	.62	4.14	16.4	5.21	45.0	3.09	25.7
10	16	8.8	750	394	.66	6.00	24.2	7.64	69.6	3.54	30.2
10	14	10.9	935	394	.77	6.00	24.2	7.64	69.6	3.54	30.2
12	16	10.5	625	564	.79	6.57	27.0	10.22	104.0	4.62	43.0
12	14	13.0	780	564	.92	6.57	27.0	10.22	104.0	4.62	43.0
12	12	18.0	1,080	564	1.22	6.57	27.0	10.22	104.0	4.62	43.0
14	14	15.8	670	768	1.07	10.03	35.0	15.78	122.0	5.24	54.0
14	12	22.0	940	768	1.48	10.03	35.0	15.78	122.0	5.24	54.0
16	14	18.0	585	1,000	1.26	14.93	61.0	22.69	138.0	7.49	66.0
16	12	25.0	820	1,000	1.68	14.93	61.0	22.69	138.0	7.49	66.0
16	10	31.2	1,050	1,000	2.06	14.93	61.0	22.69	138.0	7.49	66.0
18	14	19.8	520	1,272	1.41	19.94	78.0	27.93	165.0	9.35	80.0
18	12	27.4	730	1,272	1.82	19.94	78.0	27.93	165.0	9.35	80.0
18	10	34.3	940	1,272	2.26	19.94	78.0	27.93	165.0	9.35	80.0
20	14	22.0	470	1,576	1.56	22.75	85.0	30.09	219.0	10.35	101.0
20	12	30.3	660	1,576	2.04	22.75	85.0	30.09	219.0	10.35	101.0
20	10	37.9	840	1,576	2.50	22.75	85.0	30.09	219.0	10.35	101.0
20	8	45.8	1,030	1,576	3.02	22.75	85.0	30.09	219.0	10.35	101.0
24	12	36.2	540	2,256	2.39	27.34	119.0	37.30	278.0	12.22	128.0
24	10	45.3	705	2,256	2.99	27.34	119.0	37.30	278.0	12.22	128.0
24	8	54.7	820	2,256	3.61	27.34	119.0	37.30	278.0	12.22	128.0
24	6	64.0	1,015	2,256	4.22	27.34	119.0	37.30	278.0	12.22	128.0
28	10	51.2	605	3,072	3.38	32.31	210.0	52.60	431.0	14.47	146.0
28	8	63.0	735	3,072	4.16	32.31	210.0	52.60	431.0	14.47	146.0
28	6	75.9	870	3,072	5.01	32.31	210.0	52.60	431.0	14.47	146.0
30	10	56.3	560	3,530	3.71	33.10	225.0	60.92	457.0	17.71	155.0
30	8	68.0	685	3,530	4.49	33.10	225.0	60.92	457.0	17.71	155.0
30	6	79.7	810	3,530	5.26	33.10	225.0	60.92	457.0	17.71	155.0
32	10	61.0	525	4,000	4.03	46.86	237.0	74.37	578.0	18.71	176.0
32	8	73.6	645	4,000	4.86	46.86	237.0	74.37	578.0	18.71	176.0
32	6	86.2	760	4,000	5.69	46.86	237.0	74.37	578.0	18.71	176.0
34	10	64.7	490	4,540	4.28	53.79	247.0	83.73	618.0	19.70	180.0
34	8	78.0	600	4,540	5.15	53.79	247.0	83.73	618.0	19.70	180.0
34	6	92.8	715	4,540	6.12	53.79	247.0	83.73	618.0	19.70	180.0
36	10	68.5	470	5,090	4.52	60.20	264.0	88.38	668.0	20.95	192.0
36	8	82.7	570	5,090	5.45	60.20	264.0	88.38	668.0	20.95	192.0
36	6	96.9	680	5,090	6.39	60.20	264.0	88.38	668.0	20.95	192.0

1 Furnished by Taylor Forge & Pipe Works, Chicago, Ill.

2 A factor of safety of about 4 should be used in placer mining; 1 foot head of water is 0.43 pound.

3 F.o.b. Chicago, as of October 1932.

Diameter, inches	Price, each	Approximate weight, pounds
7	$3.20	18
8	3.40	24
9	3.90	30
10	6.00	42
11	6.45	50
12	6.70	51
13	7.85	55
14	8.70	58
15	10.45	82
16	12.55	90
18	13.35	93
20	13.45	100
22	15.00	111
24	17.00	126
26	19.00	170
28	21.00	180
30	23.00	315

Sometimes a lead joint is used; this consists of a sleeve three fourths of an inch larger than the pipe, placed around the two ends to be connected. The space between the rung and pipe is filled with molten lead.

Riveted elbows furnished by the pipe manufacturers generally are used for making turns in pipe lines. Taper joints are used where reductions are made in lines. Sudden reductions in size are to be avoided because of the loss of head and strain on the line.

Standard valves are used for diverting water or closing off flow in pipe lines. Valves should be closed slowly and with great care in high-pressure lines; the pressure exerted by the sudden stoppage of flow in the water column may burst the pipe.

Air vents are needed at all crests in hydraulic pipes to prevent a vacuum being formed and subsequent crushing of the pipe. Venting also is necessary to prevent air pockets in the line. Figure 5 shows an air vent used at the Salyer mine in Trinity County, Calif.[16] The device consists of a leather-faced flap on a hinge bolted on the inside of the pipe. A bail attached to the flap goes through an oblong hole 1 3/4 by 3 inches in size, cut in the pipe. As the water fills the pipe the flap fits tightly against the inside; as the water falls the flap drops, making a vent.

Pressure boxes.- To give the water entering a pipe line an initial velocity pressure boxes or penstocks are used. A head of 4 to 6 feet usually is provided. A length of large-diameter pipe may be used at the top of the line instead of a penstock. A screen usually is placed at the head of the line to keep out trash. In some installations settling boxes are provided where solid matter may settle out before the water goes into the pipe, as such material may cause rapid wear of the nozzles of the giants.

Laying pipe lines.- Pipe lines are laid by beginning at the bottom and working upward. Sharp curves are avoided wherever possible, and where used the pipe must be anchored securely to prevent the thrust of the water pressure from pulling the joints apart. Curves in a vertical plane are especially undesirable as they may cause air pockets in the pipe. The pipe

16 Engineering and Mining Journal, Air-Vent Valve for Hydraulic Mining: Vol. 131, Feb. 23, 1931, p. 161.

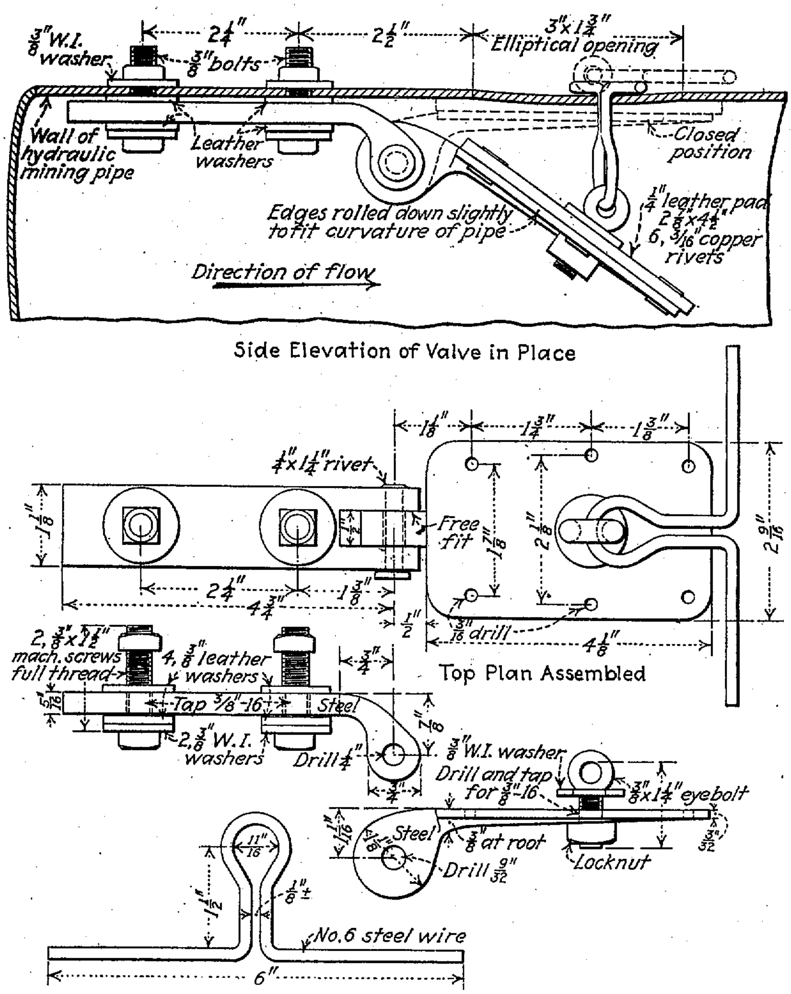


Figure 5.— Air-vent valve for pipe lines, Salyer, Calif.

should be filled gradually for the same reason. In crossing small ravines a trestle should be built first and the pipe laid on plank for the complete distance.

In laying new pipe with slip joints the outside pipe is started over the end of the other, then heated with a blow torch, which expands the outer pipe and melts the tar previously placed on the end of the lower pipe. As the heating is completed the upper pipe is driven home by hammering on a block of wood placed at the upper end. The tar makes a watertight connection. Where the pipe has been battered from previous handling, burlap or sacking may be wrapped around the joint before driving. If leaks develop they may be stopped by driving in wooden plugs; sometimes an outside band is required.

In placing pipes with flanged joints they are laid end to end and the bolts put through and tightened up. The flanges usually are attached to the pipe at the factory. This prevents nesting of the pipe in shipping but permits a better joint to be made.

When pressures are very high or when the pipe has vertical or lateral curves, lugs should be riveted on the ends of the pipe with slip joints and the two pipes wired together after the connection is made to prevent the joint pulling out. Similar lugs can be used for anchoring the line to stumps or posts.

In straight pipe lines expansion joints should be placed at intervals of 100 to 2,000 feet, depending upon the conditions to be met. Where pipe lines have lateral curves expansion joints are not needed, as the expansion or contraction of the pipe is taken up in the curved sections. A long, empty pipe line may contract several feet between a warm day and a cold night, and unless provision is made for this contraction the pipe will pull apart. When the pipes are kept full of water this contraction does not occur. Pipe lines are buried in some locations but seldom at western placer mines.

The cost of laying pipe lines depends upon the size of the pipe and the topography and cover of the country. Ten men working 90 days laid 5,000 feet of 36- to 16-inch pipe at the Browning mine, Leland, Oreg., in open country in the spring of 1932.

Flow of water through pipes.- The quantity of water that will flow through a pipe line at a placer mine depends mainly upon the diameter of the pipe, the effective hydraulic head, and the size of the nozzle used on the giant at the end of the pipe. Generally the nozzle used is of such a size that the pipe will carry the available water. As the water supply is reduced smaller nozzles are used on the giants.

The effective head on a pipe is the static head minus the loss of head due to friction. The loss of head depends upon (1) the velocity of the water, (2) the roughness of the interior of the pipe, (3) the diameter of the pipe, and (4) the length. The pressure available and the amount of flow at the end of a long pipe depends mainly upon the last three items. The pressure of the water in the pipe has no effect, by itself, on the loss of head. Formulas have been derived for calculating the loss of head in which coefficients of roughness are used. These coefficients have been derived by experiment for different types of pipes; specifically, however, consideration must be given to the service conditions encountered. No standard of roughness exists, and the degree of roughness of the interior of a pipe does not remain constant. Usually a pipe is chosen about 20 percent larger than would be indicated if there was no loss due to friction. Flow through an unobstructed pipe line of uniform diameter can be calculated from a number of formulas. The Kutter modification of the Chezy formula appears to be preferred by hydraulic engineers. The Chezy formula may be stated as:

$$V = C\sqrt{RS}.$$

The Kutter modification of the Chezy formula is:

$$V = \frac{\frac{1.811}{n} + \frac{0.00281}{S} + 41.66}{1 + \frac{n}{\sqrt{R}}\ 41.65 + \frac{0.00281}{S}} \sqrt{RS}$$

where

V = mean velocity of flow, feet per second;
C = "coefficient of retardation," so-called;
R = mean hydraulic radius of the pipe, that is, 1/4 the diameter;
S = hydraulic grade or slope, in feet per foot of length of a pipe of uniform size;
n = "coefficient of roughness," so-called.

The value of n for riveted lap-joint pipe up to and including 3/8 inch thick can be taken as 0.015. Graphical solutions of this formula are made conveniently by the use of a diagram such as that shown in figure 6.

Selection of diameter of pipe line for given flow of water.- The chart shown in figure 6 will assist one in making a choice of the diameter of pipe to be used in any given line. As an example, say that 320 miner's inches or 8 cubic feet per second of water is available under a 100-foot head, and the pipe line is to be 1,200 feet long. The use of three sizes is preferable because of saving to be made in freight on the pipe. To solve, start at the bottom of the chart on line 8 and follow it up to where it intersects diagonal lines representing different diameters of pipe. By following the horizontal lines from these intersections to the left margin the friction-head loss may be noted for each diameter of pipe. With 12-inch pipe this loss is 80 feet per 1,000, which would indicate that little if any pipe of this diameter should be used in the supply line. The loss with 14-inch pipe is 33 feet per 1,000 feet of line. If 400 feet of this diameter pipe was used in the line the loss of head would be 13 feet. With 15-inch pipe the loss per 1,000 feet would be 23 feet and with 16-inch pipe 16 feet. The losses for 400 feet of these two sizes would be 9 and 6 feet, respectively. With 18-inch pipe the loss would be 8 feet per 1,000 or 3 feet for each 400 feet. The total loss of head with 400 feet each of 14-, 16-, and 18-inch pipe would be 22 feet. The effective head, therefore, would be about 78 percent of the actual head. By using all 18-inch pipe the total loss of head would be only 10 feet. If the gravel is easy to cut and need not be swept long distances a 22-foot loss of head may not be serious. In tight gravel, however, it probably would be economical to use just the 16- and 18-inch diameters, or possibly to construct the whole line of 18-inch pipe. If the total available head were 200 feet, the smaller pipes probably would prove satisfactory, as the percentage of loss would only be one half as much as with a 100-foot head.

Therefore, larger-diameter pipe is needed for long lines than for short ones as the loss of head is directly proportional to the length of the lines. Moreover, where the loss of head is important relatively larger pipe must be used. If the pipe is dented, rusted, or poorly laid, possibly less water would flow through a given pipe than is shown on the chart. In new straight pipe probably the flow would be more than is indicated on the chart as it has been drawn to cover average conditions.

Giants

A giant or monitor is a device with a nozzle for directing and controlling a stream of water under a hydraulic head. The giant can swing horizontally through a full circle and

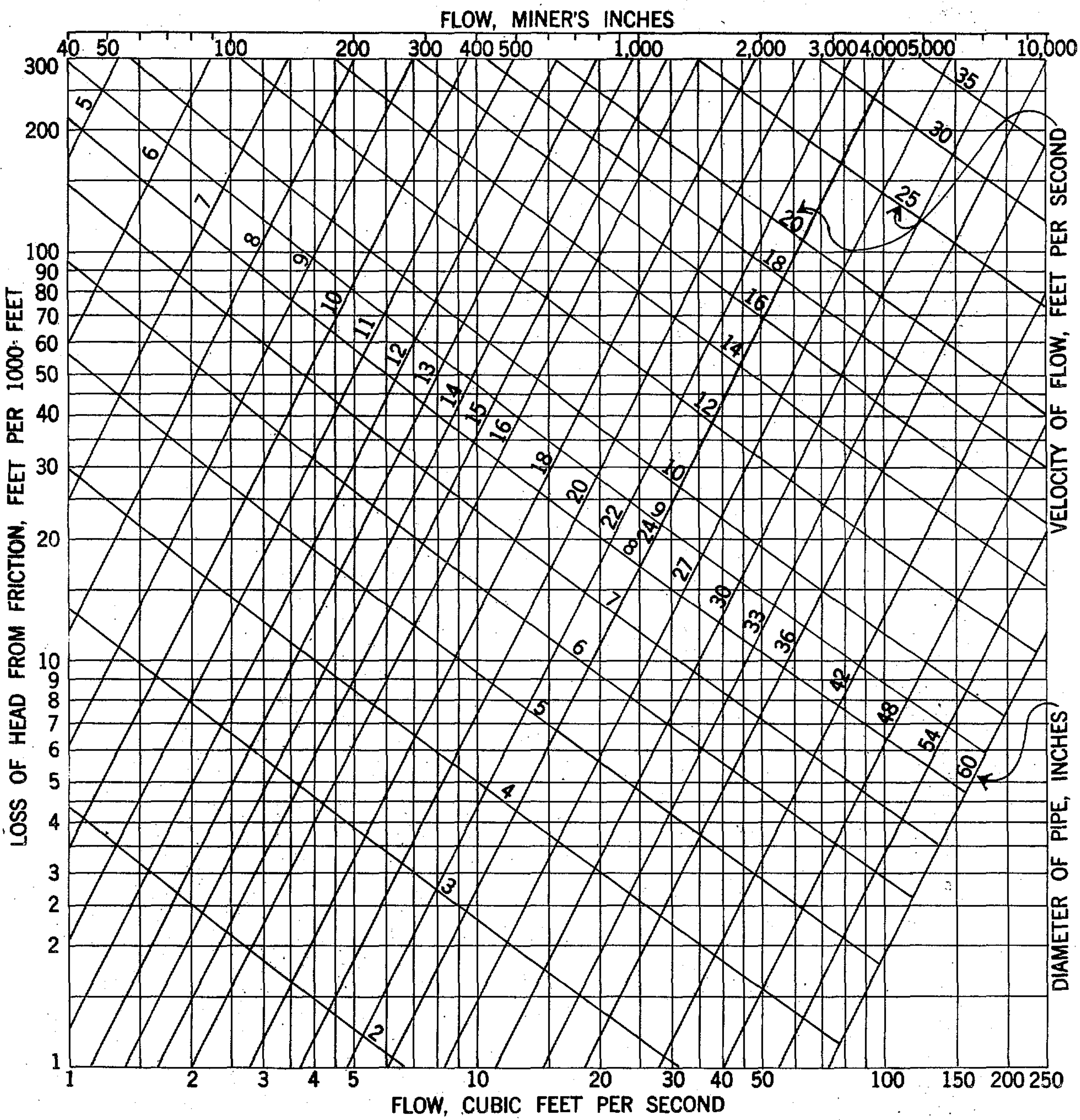


Figure 6.—Chart showing loss of head in pipes due to friction (N = 0.015).

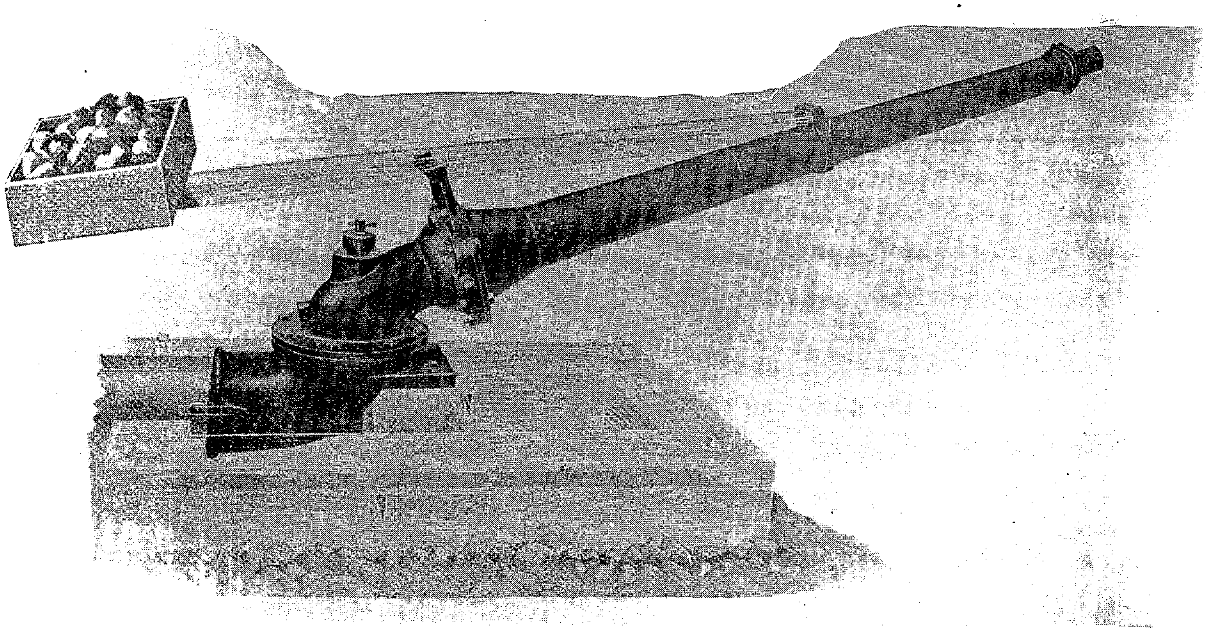


Figure 7.—Hydraulic giant, no. 1, 2, and 3.

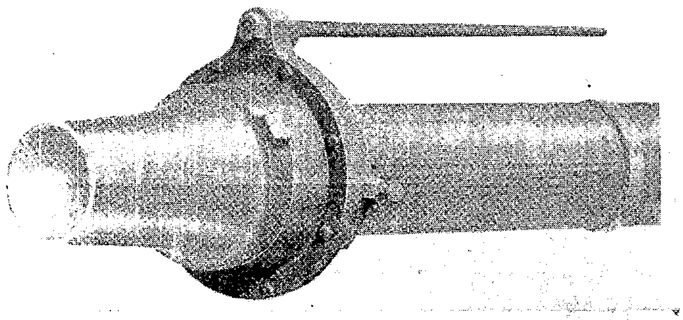


Figure 8.—Deflector for hydraulic giant.

from 11° below to 55° above the horizontal. A standard giant is shown in figure 7. The box of stones is used to counterbalance the weight of the spout. A giant generally is set up in a pit by being bolted to a log or to timbers securely anchored in bedrock. Nozzles of different diameters can be used up to the diameter of the outlet of the giant to make allowance for variation in the quantity of water used. The giant and nozzle are constructed so that a rotary motion of the jet is prevented, and the water is discharged in a solid column. Giants are made for a wide range of service in 10 sizes, numbered 0 to 9, inclusive.

With heads of 100 feet or more deflectors are used for pointing the larger giants. A common type of deflector consists of a short section of pipe that projects over the nozzle. (See fig. 8.) It turns on a gimbal joint and is controlled by a lever. As the deflector is turned against the jet the force of the stream turns the giant in the opposite direction.

Table 6 shows the sizes, weights, and prices of giants and deflectors made by one manufacturer.[17] Other companies make similar equipment at competitive prices.

TABLE 6.- Sizes, weights, and prices of double-jointed, ball-bearing giants and deflectors

Giants							Deflectors	
Size no.	Diameter of pipe inlets, inches	Diameter of butts with nozzle detached, inches	Shipping weight, pounds	Weight of heaviest part, pounds	List price[1] Flanged inlet	List price[1] Slip-joint inlet	Weight, pounds	List price[1]
0	5	3	350	105	....	....	(2)	....
1	7	4	390	120	$180	$165	30	$29.00
2	9	5	520	150	225	210	40	32.00
3	11	6	890	210	320	295	45	37.00
4	11	7	1,075	225	365	330	55	40.00
5	13	8	1,475	335	485	460	70	56.00
6	15	9	1,850	520	620	580	75	62.00
7	15	10	2,100	520	725	685	80	67.00
8	18	10	2,300	600	850	805	80	67.00
9	18	11	2,450	690	925	865	90	71.00

[1]Subject to discount because of fluctuations in prices of iron and steel.
[2]None required.

Giants 1 to 3 are constructed as shown in figure 7. The larger sizes are of more substantial construction and usually are equipped with a ball-bearing kingbolt. For heads of 400 or more feet heavy lugs may be used at the joints as a safety precaution.

Discharge through nozzles.- Table 7 gives the discharge through different sizes of nozzles under heads from 100 to 400 feet. In this table 40 miner's inches is considered as 1 cubic foot per second. The theoretical flow of water through nozzles exceeds the figures in table 7 by about 10 percent; allowances have been made for friction losses. The flow through nozzles not shown in the table or for different heads can be calculated from the equation:

17 Joshua Hendy Iron Works, San Francisco, Calif.

$$Q = 8CA\sqrt{h}$$

where

Q = cubic feet per second,
A = area of nozzle (square feet),
h = effective head at nozzle (feet),
C = coefficient of discharge ranging from 0.8 to 0.94 (usually taken as 0.9, which makes allowance for friction).

To convert cubic feet to gallons multiply by 7.48.

TABLE 7.- Flow of water through giants[1]

Giant no.	Diameter of nozzle, inches	Effective head, feet: 100		200		300		400	
		Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches
0	1 1/8	0.6	22	0.8	31	.....	.....	.....	.....
0	1 3/8	.8	33	1.2	47	.....	.....	.....	.....
1	2	1.6	63	2.2	89	2.7	109	3.1	125
1	3	3.0	120	4.3	173	5.3	213	6.4	257
2	3	3.3	133	4.7	187	5.7	227	6.6	267
2	4	5.6	227	8.3	333	10.3	410	11.9	477
3	3	3.7	148	5.0	200	6.5	245	7.1	283
3	4	6.0	240	8.6	343	10.6	423	12.2	488
4	4	6.3	253	8.9	357	10.9	437	12.6	504
4	6	13.3	535	19.2	770	23.7	950	27.5	1,100
5	5	9.8	395	13.9	560	16.7	670	19.7	790
5	6	13.5	540	19.3	770	23.8	950	27.7	1,110
6	6	13.8	550	19.6	780	24.1	960	27.9	1,120
6	7	18.7	750	26.7	1,070	33.2	1,330	37.7	1,510
7	6	14.2	570	20.0	800	24.5	980	28.3	1,130
7	7	19.0	760	26.9	1,080	33.3	1,330	38.0	1,520
8	7	19.2	770	27.2	1,090	33.8	1,350	38.3	1,530
8	8	15.2	1,010	35.3	1,410	43.7	1,750	48.7	1,950
9	9	32.0	1,280	45.0	1,800	55.3	2,210	63.7	2,550
9	10	39.3	1,570	55.3	2,210	68.2	2,730	78.7	3,140

[1]Adapted from table in catalog of Joshua Hendy Iron Works, San Francisco, Calif.

Derricks and Winches

The same general types of derricks and winches are used as in ground-sluicing, which has been described in a previous paper.[18]

18 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part I. - General Information, Hand-Shoveling, and Ground-Sluicing: Inf. Circ. 6787 Bureau of Mines, 1934, 73 pp.

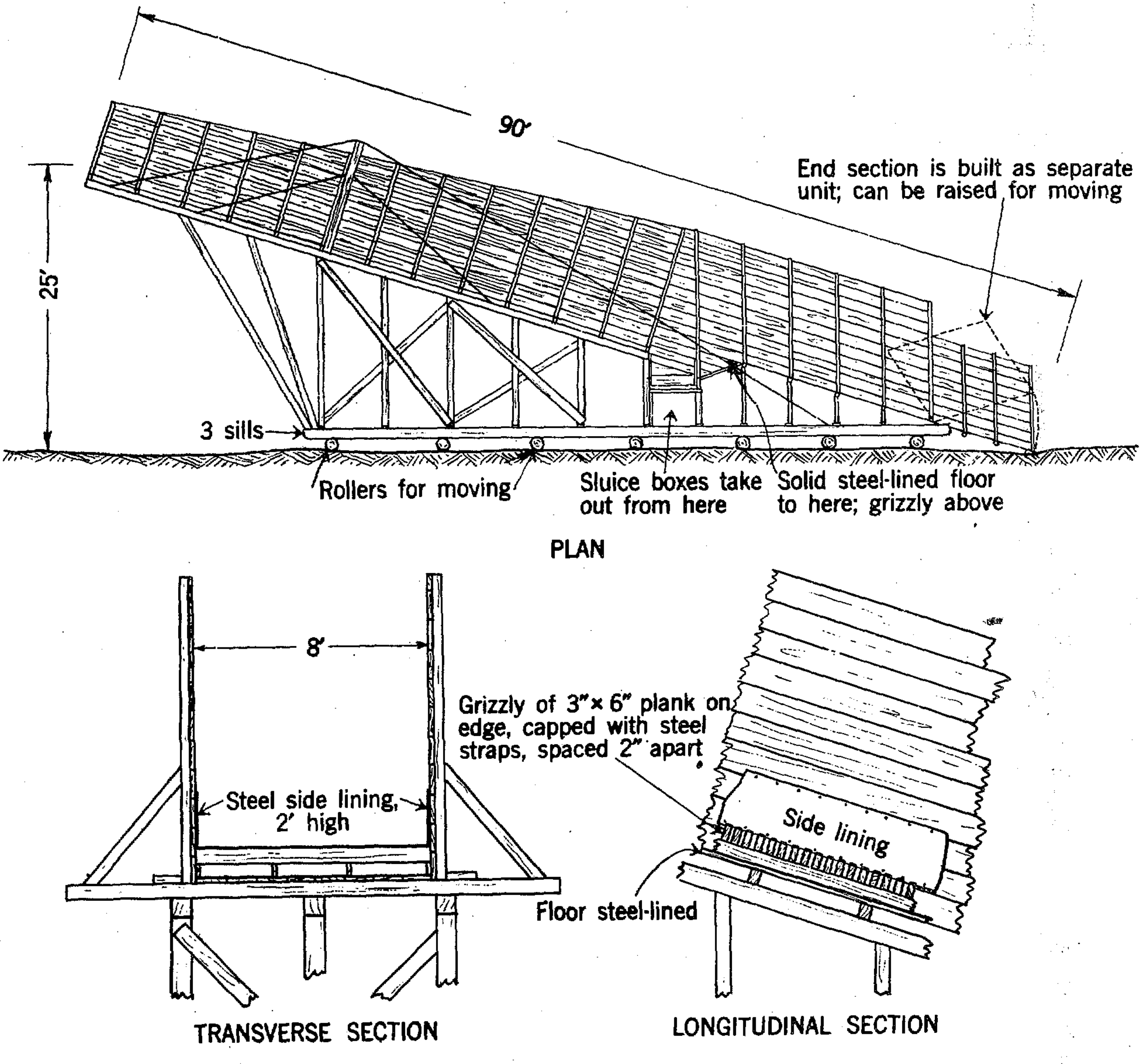


Figure 9.– Ruble elevator used at Redding Creek mine, Douglas City, Calif.

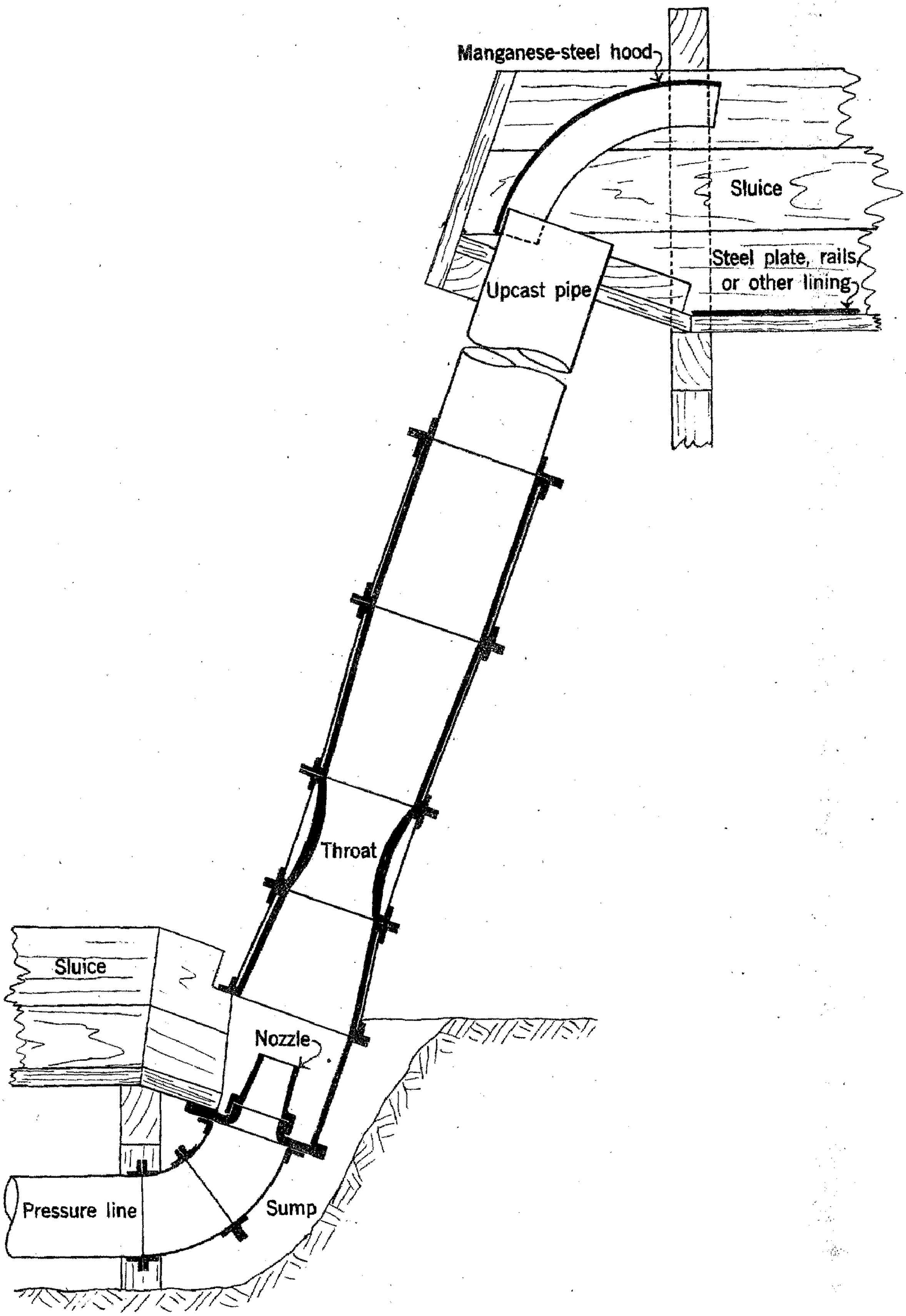


Figure 10.—Hydraulic elevator.

Ruble Elevators

The Ruble elevator is named for the Ruble mine in Josephone County, Oreg. It consists essentially of an inclined grizzly on a pitch of about 17°, up which the gravel is driven by the stream from a giant. The oversize goes over the grizzly to a rock pile, and the undersize runs down a chute under the grizzly and thence into sluice boxes, usually set at right angles to the elevator. The spacing between bars of elevators in use in 1932 ranged from 3/4 to 2 1/2 inches. A 10- or 12-foot apron is used in front of the grizzly. The gravel generally is swept to the foot of the elevator by one giant and through the Ruble by another. The gravel must be washed thoroughly before it is elevated, and the stream of the elevator giant must be used with caution; otherwise, considerable gold may be driven over the top.

Under favorable conditions one giant can handle as much material through the Ruble as another can cut and sweep to it. Under other conditions less than half of the material can be put through the Ruble that one giant working steadily can get to it. Figure 9 is a drawing of a Ruble elevator used at the Redding Creek mine at Douglas City, Calif.

Hydraulic Elevators

Hydraulic elevators are used to raise gravel, sand, and water out of placer pits into sluice boxes. An elevator consists of a pipe with a constricted port or throat and a jet which provides a high-velocity ascending column of water. The relative diameter of pipe, throat, and jet must be proportioned according to the conditions under which the elevator is used. A section of an elevator is shown in figure 10.[19] The elevator may also be used as a water lifter.

The height to which gravel can be lifted is one tenth to one fourth of the effective head of the pressure water at the nozzle of the elevator. Usually the lift will be about one fifth the head.

The volume of gravel that can be handled by an elevator depends primarily upon the head and volume of pressure water available and to a lesser extent upon the quantity of other water that has to be raised by the elevator. The solids in the water usually are 1.7 to 2.5 percent.

Where little drainage water has to be handled and other conditions are favorable the proportion of the water delivered to the elevator and the giant, respectively, should be about equal, provided the pressure is the same in both. Usually, however, about twice as much water or a correspondingly higher head is required for the elevator. The discharge of the elevator should be high enough to provide dumping ground, otherwise a giant may be needed to stack the tailings. Where plenty of water is available a compound or step-lift elevator may be installed in which one third of the pressure water is used in the first lift and two thirds in the second, with a correspondingly larger area of upraise pipe. Thus the height of the lift may be nearly doubled. Double lifts sometimes are used; that is, the discharge of one elevator goes to the intake of another.

The elevator discharges upon a cover plate to take the wear in the head of a sluice. Boxes may or may not be used in the pit. The size of the gravel handled is limited by the size of the throat of the elevator. Grizzlies generally are used at the intake. Coarse material reduces the capacity of the elevator; sometimes a Ruble elevator is used in the pit, and only the under-size is sent to the hydraulic elevator.

---

19 After Joshua Hendy Iron Works catalog.

In clayey ground a hydraulic elevator tends to break up the clay as it goes through the elevator, thus permitting a higher extraction of the gold.

Gravel pumps have been used successfully in alluvial tin mines and in at least one placer mine in British Columbia.[20] As far as known they have not been used successfully in placer mining in the Western States.

Hydraulic Mining Practices

Conditions varied widely at the hydraulic mines operated in the Western States in 1932. The practices at these mines illustrate the different phases of hydraulic mining and are discussed in this paper. In earlier days, however, when the large hydraulic mines of the West were being worked, more elaborate equipment and larger installations were used than at present. Higher banks were worked, and very large daily yardages were washed, with correspondingly lower costs.

Data concerning the principal hydraulic mines being operated in 1932 are given in tables 8 to 13, inclusive. Operating costs are representative for the conditions shown.

Gravels

The gravels being worked at hydraulic placer mines in the summer of 1932 ranged in average depth from 5 to 100 feet (see table 8); at Relief Hill, where an old mine was being reopened, the depth was 200 feet. The condition of the gravel ranged from soft, easily washed material to gravels that had to be loosened by blasting. The percentage of boulders over 1 foot in diameter ranged from less than 1 to 20. Usually, 5 to 15 percent of all material handled consisted of boulders. Boulders up to 20 inches in diameter were put through the sluices. Clay constituted zero to 15 percent of the total material. At one mine, the Elephant, 2 1/2 feet of gravel was overlain with 40 feet of volcanic ash.

Bedrock at nearly all mines was soft, and the top could be piped off in cleaning up. The slope of the bedrock ranged from 1/10 inch to 2 inches per foot.

Water Supply

Very few hydraulic mines can operate the entire year. Advantage generally is taken of high-water periods for working the mine. In California the season may begin in November or December, when the winter rains commence, and continue into the dry season of June or July. At most California and western Oregon mines the winter temperature is not low enough to interfere seriously with placer operations. Elsewhere in the West, however, hydraulic placer mining must cease with the advent of cold weather in October, November, or December. At such places, work can not begin until spring when the snow melts and the ground thaws. In many localities placer mining can be carried on only while the snow is melting on the mountains above during the spring months. The length of the 1932 season at the mines visited by the authors ranged from 25 to 225 days. The precipitation during the winter of 1931-32 was normal or above normal in nearly all districts; immediately preceding years, however, were dry, and the number of days operated at the majority of places was much less than in 1932. In exceedingly dry years some mines do not have enough water to operate at all.

Reservoirs are used at most mines. As the flush supply gives out the water may be stored and used periodically for mining. Usually cutting operations cease when water is not available for piping at least 1 1/2 or 2 hours per day. The dwindling supply then will be used for cleaning bedrock and cleaning up the boxes.

---

20 Operations of B. Boe on Cedar Creek, Quesnel District: Ann. Rept. of the Minister of Mines of British Columbia, 1932, p. A112

TABLE 8.- Hydraulic placer mines being operated in Western States, 1932, and data on gravel, bedrock, and gold

Mine			Gravel				Boulders			Bedrock		Gold	
Name	Location	Operator	Thickness, feet: Maximum	Thickness, feet: Minimum	Thickness, feet: Average	Physical condition	Over 1 foot in diameter, percent in gravel	Maximum diameter through sluices, inches	Clay, percent in gravel	Kind	Physical condition	Character	Value, per ounce (at $20.67 per fine ounce)
Senger	Weaverville, Calif.	M. A. Senger			6	Tight, with roots and boulders.	20	12	5	Gravel	Smooth	Medium	
Elephant	Volcano, Calif.	K. D. Winship estate.	50	40	[1]45	Tight	3		3	Decomposed slate.	Soft	Coarse	
Horton Gulch	Cecilville, Calif.	J.O.McBroom	20	15	17	do.	5		5		do.	Medium	$16.55
Banner	do.	F. S. George and Bros.	8	20	15	do.	10		4		do.	Coarse	17.90
Indian Creek	Douglas City, Calif.	Gribble & Son, lessees.			9	do.	8	18	5		Uneven		
Salmon River	Cecilville, Calif.	A. B. Farnsworth & Bros., lessees			[2]17	Medium	6	18			Soft	Medium	
Jacobs	Junction City, Calif.	H. K. Wilson			[3]40	do.	5	18	5	Slate	Uneven	do.	
Omega Hill	Washington Camp, Calif.	Omega Hill Mining Co.	60	30	[4]45	Tight	8	18		Schist	Soft, uneven.		
Indian Hill	Comptonville, Calif	B. T. Dyer			35	do.	10	14		Slate	Rough		
Depot Hill	do.	Fred Jourbert			60	do.	5	4		do.	Uneven	Fine	19.50
North Fork Placers.	Helena, Calif.	F. W. Reynolds et al.			15 40	Cemented Medium	5	12	2		do.	Coarse	
Salyer	Salyer, Calif.	Salyer Consolidated Mines Co.			20 80	do. Easy breaking						Fine	18.50
Norton and Nelson.	Galice, Oreg.	Norton and Nelson.			12		12		2	Slate	Rough, medium.		18.80
Salmon Creek	Baker, Oreg.	Salmon Creek Placer Co.			25	Tight	8	6	10	do.	Soft		
Blue Channel	Wolf Creek, Oreg.	M. C. Davis	30	12	8 12	Cemented Medium	3	12			Hard, uneven.		
Deep Creek	Lozeau, Mont.	L. E. Frank			15	Tight	10	9			Rough		19.70

1 1 1/2 to 3 feet of gravel overlain by 40 to 45 feet of volcanic ash. 2 Including 11 feet of soil overburden. 3 28 feet of clay soil overlying 12 feet of gravel. 4 Overlain with 40 feet of volcanic ash.

TABLE 8.- Hydraulic placer mines being operated in Western States, 1932, and data on gravel, bedrock, and gold - Continued

Mine			Gravel				Boulders			Bedrock		Gold	
			Thickness, feet										
Name	Location	Operator	Maximum	Minimum	Average	Physical condition	Over 1 foot in diameter, percent in gravel	Maximum diameter through sluices, inches	Clay, percent in gravel	Kind	Physical condition	Character	Value, per ounce (at $20.67 per fine ounce)
Yellowstone Gold.	Emigrant, Mont.	Yellowstone Gold Mining Co.	50	10	30	Loose	5	12		Not reached			
Virginia City	Virginia City, Mont.	Virginia City Mining Co.			30	Medium	5	6					$18.00
Henderson No. 1	Gold Creek, Mont.	Henderson Mining Co.			45	do.	10	10	5	Clay	Soft		18.75
Henderson No. 2	do.	do.			15	do.	2	8	10	do	do		18.75
Wisconsin Gulch	Sheridan, Mont.	Wisconsin Placer Gold Corporation			27	Tight	20	20		Schist	do.	Coarse	17.00
Stemwinder	Superior, Mont.	I. H. Gildersleeve & Bros.			70	Medium	5	8	10		do.		19.80
Diamond City	Townsend, Mont.	Diamond City Mines Co.			12	do.	10	10		Slate	Rough		17.50
Superior	Superior, Mont.	Superior Mines Co.			60	do.	5				Even		
Hockensmith	Leesburg, Idaho	Goff Bros.	20	1	12	do.	5		5				19.20
Golden Rule	Warren, Idaho	L. E. Winkler, et al.			12	do.	5	12			Even		
Fortune	Kokomo, Colo.	Fortune Tarryall Gold Placers Co.			30	Easy breaking	1	18	0	Sandstone	Soft		18.00
Dodman and Weston.	Breckenridge, Colo.	Dodman and Weston			50	do.	5	10		Slate	Rough		
Round Mountain	Round Mountain, Nev.	Nevada Porphyry Gold Mines, Inc.	3	50	30	Partly cemented	15				Uneven	Coarse	12.70
Redding Creek	Douglas City, Calif.	Redding Creek Placers, Ltd.			9	Medium	8	2		Cemented gravel.	Smooth		
Browning	Leland, Oreg.	MacIntosh Bros., lessees.	18	5	12	do.	2	3/4			Even		
Llano de Oro	Waldo, Oreg.	Allen, et al, lessees.	15	30	25	do.	0	3	15		Soft	Very fine	

TABLE 8.- Hydraulic placer mines being operated in Western States, 1932, and data on gravel, bedrock, and gold - Concluded

Mine			Gravel				Boulders			Bedrock		Gold	
			Thickness, feet										
Name	Location	Operator	Maximum	Minimum	Average	Physical condition	Over 1 foot in diameter, percent in gravel	Maximum diameter through sluices, inches	Clay, percent in gravel	Kind	Physical condition	Character	Value, per ounce (at $20.67 per fine ounce)
Platurica	O'Brien, Oreg.	Nelson and Harrison, lessees.			35	Tight	1	8			Uneven	Fine	$18.50
Davis	Centerville, Idaho.	J. J. Davis	4	6	5	do.	15	6					
Gallia	Sawyer's Bar, Calif.	Gallia Placer Mining Co.	30	35	33	Medium	10	2 1/2			Fairly even		18.50
Lewis	Galice, Oreg.	Harry Lewis			18	do.	3	5			Even		
Conners	Bridgeport, Oreg.	J. C. Conners	35			do.	5	4			Rough, soft		18.00
Eldorado Bar	York, Mont.	Eldorado Mining Co.	20		14	Easy breaking	5	8					
Old Garden Gulch.	Centerville, Idaho.	John D. Smith	4	10	7	Medium	1	7	0	Granite	Soft		

TABLE 9.– Water supplies and ditch and pipe lines at hydraulic mines in the Western States, 1932

Mine	Water supply					Ditch lines				Pipe lines			Reservoirs, capacity, acre-feet
	Hours used	Quantity, miner's inches[1]				Width, feet	Depth, feet	Length, miles	Capacity, miner's inches	Diameter, inches	Length, feet	Head of water, feet	
		Natural flow	Used by giants	Bywash	Total								
Senger	5		400	0	400			6		30 to 15		225	
Elephant	4		135	40	175					18 to 8		115	None.
Horton Gulch	9	250	220	30	250			2	400	22 to 13	800	65	None.
Banner	10	300	300	0	300			3	700	18 to 8	1,200	100	None.
Indian Creek	5		500	0	500			5	3,000	22 to 15	2,200	275	1.2
Salmon River	9	1,100	1,100	0	1,100	8	3 1/2	4	3,000	22 to 15	5,000	225	None.
Jacobs	1 1/2	50	460	0	460	0	0	0	0	22 to 15	600	90	2
Omega Hill	10	600	1,300	200	1,500	6	3	16	5,000	30 to 15	1,800	210	
						4	3	8					
Indian Hill	9		430	220	650			9		22 and 15	4,500	130	
Depot Hill			450	150	600			9	700	30 to 11	3,800	[2]160	
North Fork Placers	18	1,000	1,300	0	900	([3])				24 to 15	1,500	400	
	5				400							200	
Salyer	24	2,800	2,800	0	2,800				5,000	Two 16		350	
Norton and Nelson	9	600	210	390	600	[4]2	3	1 1/2		15	300	[5]90	None.
Salmon Creek	8	150	90	60	150			1/2		15 to 11	500	150	None.
Blue Channel	5	240	1,200	0	1,200			4		32 to 18	1,100	360	.2
Deep Creek	5	90	100	100	200					10 and 8	1,000	135	.1
Yellowstone Gold	18	600	240	360	600					20 to 10	2,400	175	None.
Virginia City		65	70	15	85					15	200	65	.01
Henderson No. 1	2	50	600	0	600					18		270	1.0
Henderson No. 2	4	20	150	0	150					18		80	
Wisconsin Gulch	24	2,800	1,300	1,500	2,800			3/4	4,000	46 to 24	2,700	220	None.
Stemwinder	2 1/2	60	130	50	180			3		10		225	
Diamond City			100	800	900					8 and 11	4,500	125	.1
Superior			85							10	750	90	None.
Hockensmith					150	2 1/2	2	5		12 and 7	700	75& 150	.1
Golden Rule	12		275							18 to 9	1,400	200	None.

[1]40 miner's inches = 1 cubic foot per second.

[2]Effective head.

[3]Two flume lines, 12 and 8 miles long, the latter 3 1/3 feet wide by 2 1/2 deep.

[4]Flume sections 2 by 2 and 2 1/2 by 1 2/3 feet; total length 1,000 feet.

[5]Effective head.

TABLE 9.- Water supplies and ditch and pipe lines at hydraulic mines in the Western States, 1932 - Continued

Mine	Water supply					Ditch lines				Pipe lines			
	Hours used	Quantity, miner's inches[1] Natural flow	Used by giants	Bywash	Total	Width, feet	Depth, feet	Length, miles	Capacity, miner's inches	Diameter, inches	Length, feet	Head of water, feet	Reservoirs, capacity, acre-feet
Fortune	20	...	190	...	...	...	...	[6]1 1/2	2,500	24 to 12	2,700	200	None.
Dodman and Weston	8	400	50	350	400	...	...	...	...	10	150	40	None.
Round Mountain	7	...	400	...	400	...	...	...	...	30 to 15	43,000	350	...
Redding Creek	24	1,200	1,200	0	1,200	(7)	0	0	...	24 to 15	3,000	300	15.0
Browning	24	900	[8]900	0	900	...	...	...	...	36 to 16	5,000	300	...
Llano de Oro	24	11,700	[9]1,700	10,000	11,700	...	...	6	520	22 to 15	3,000	[2]360	...
								9	1,800	22 to 11	3,400	[2]125	
								8	10,000	0	0	0	
Platurica	24	650	[9]650	0	650	...	...	11	...	22 to 15	4,000	450	...
Davis	24	240	[9]240	0	240	...	...	1 1/2	...	20 to 11	1,200	125	None.
Gallia	...	...	[8]725	...	[10]2,000	(11)	...	...	4,000	36 to 15	2,000	265	...
Lewis	9	300	[12]300	0	300	...	...	...	...	22 and 15	...	180	...
Conners	...	...	28	0	28	0	0	0	0	5	500	[13]300	None.
Eldorado Bar	...	...	60	290	[14]350	...	...	...	...	12	500	[15]100	None.
Old Garden Gulch	16	400	400	0	400	...	...	...	...	12 to 7	900	[16]100	None.
										12	700		

[6] Flume sections 6 feet wide by 3 feet deep.
[7] Flume line 2 1/2 miles long, 4 1/2 feet wide, by 2 1/2 feet deep.
[8] Includes giant at Ruble elevator.
[9] Including hydraulic elevator.
[10] Including hydraulic elevator and generator.
[11] Flume sections 4 feet wide by 4 feet deep.
[12] Including giant at Ruble and hydraulic elevators.
[13] Water lifted 225 feet and given a pressure of 65 pounds per square inch at nozzle by pump.
[14] Water pumped 120 feet vertically.
[15] 40 pounds to square inch pressure furnished by a booster pump.
[16] 40 pounds to square inch pressure furnished by pumps.

TABLE 10.- Piping equipment and operation at western hydraulic mines, 1932

Mine	Giants (monitors): Cutting: Number used	Cutting: Size no.	Cutting: Diameter of nozzle, inches	Driving: Number used	Driving: Size no.	Driving: Diameter of nozzle, inches	Tailings disposal: Number used	Tailings disposal: Size no.	Tailings disposal: Diameter of nozzle, inches	Ruble elevator: Number used	Ruble elevator: Size no.	Ruble elevator: Diameter of nozzle, inches	Hydraulic elevator, diameter nozzle, inches	Total number of nozzles used at one time	Height gravel elevated, feet: Ruble	Height gravel elevated, feet: Hydraulic	Hours cutting per day	Hours driving per day	Hours piping tailing per day	Method of handling boulders
Senger	1	2	4 1/4				0			0			0	1	0	0	4	1	0	Blast.
Elephant	1	2	3				0			0			0	1	0	0			0	Hand.
Horton Gulch	1	3	5				0			0			0	1	0	0			0	Blast.
Banner	1	2	4	1	3	4	1	3	3	0			0	2	0	0	10	15	5	Power derrick.
Indian Creek	1	4	5	1	4	5				0			0	1	0	0	3 1/2	1	1/2	Blast.
Salmon River	2	5	6	1	4	6	1	5	[1]6	0			0	2	0	0			[2]16	Power derrick.
Jacobs	1	4	7				0			0			0	1	0	0	2	1	0	Blast.
Omega Hill	1		6	1		6	0			0			0	[3]2	0	0			0	Do.
Indian Hill	1	6	4 1/2	1	6	6	0			0			0	1	0	0			0	Hand and blast.
Depot Hill	1	4	4 1/2	1	4	5	0			0			0	1	0	0	2 to 4	2 to 4	0	Do.
North Fork Placers	1	6	7	[4]1	5	5	0			0			0	2	0	0	18	5	0	Do.
Salyer	1		7	1		7	0			0			0	2	0	0			0	Crane and tractor.
Norton and Nelson	1	2	3	1	2	4	0			0			0	1	0	0			0	Gasoline hoist.
Salmon Creek	1	2	2 1/4				0			0			0	1	0	0	16	0	0	Power shovel.
Blue Channel	1	3	4 1/2	3	3	5	1	3	5	0			0	2	0	0	6	5	1	Hand blast.
Deep Creek	1	2	3				0			0			0	1	0	0	3	2	0	Hand.
Yellowstone Gold	1	2	4				0			0			0	1	0	0	5	13	0	Power derrick.
Virginia City	1		2 1/2	1		[5]1 1/4	0			0			0	1	0	0	6	18	0	Hand and car.
Henderson No. 1	1	2	5				0			0			0	1	0	0	2	2	0	Power derrick.

1 Also one 2-inch nozzle used for operating derrick.

2 Works automatically all night.

3 Sometimes three giants with 5-inch nozzles are used.

4 Water from a separate source.

5 In sluice box.

TABLE 10.- Piping equipment and operation at western hydraulic mines, 1932 - Continued

Mine	Giants (monitors): Cutting: Number used	Cutting: Size no.	Cutting: Diameter of nozzle, inches	Driving: Number used	Driving: Size no.	Driving: Diameter of nozzle, inches	Tailings disposal: Number used	Tailings disposal: Size no	Tailings disposal: Diameter of nozzle, inches	Ruble elevator: Number used	Ruble elevator: Size no.	Ruble elevator: Diameter of nozzle, inches	Hydraulic elevator, diameter nozzle, inches	Total number of nozzles used at one time	Height gravel elevated, feet: Ruble	Height gravel elevated, feet: Hydraulic	Hours cutting per day	Hours driving per day	Hours piping tailing per day	Method of handling boulders
Henderson No. 2	1	2	2	...	2	4	0	...	...	0	...	...	0	1	0	0	4	4	0	Hand.
Wisconsin Gulch	1	4	4 and 5	1	4	4 and 6	0	...	...	0	...	...	0	2	0	0	24	24	...	Power derrck.
Stemwinder	1	2	3	1	2	3	0	...	...	0	...	...	0	1	0	0	3	3	0	Hand.
Diamond City	1	2	2 and 3	1	2	3 and 4	0	...	...	0	...	...	0	1	0	0	12	4	0	Power shovel.
Superior	1	2	2 1/2	...	...	...	0	...	...	0	...	...	0	1	0	0	8	8	0	Steam derrick.
Hockensmith	1	1	1 3/4	1	1	2	0	...	...	0	...	...	0	1	0	0	...	...	0	Hand.
Golden Rule	1	3	4	...	...	...	1	2	3 1/2	0	...	...	0	1	0	0	4	8	0	Do.
Fortune	1	2	3	1	2	4	0	...	...	0	...	...	0	1	0	0	...	...	0	Do.
Dodman and Weston	1	2	3	...	...	...	0	...	...	0	...	...	0	1	0	0	4	2	0	Do.
Round Mountain	1	...	...	1	...	...	0	...	...	0	...	...	0	1 or 2	0	0	7	...	0	Derrick and cars.
Redding Creek	1	...	5	1	...	5	1	...	3	1	...	5	0	2	0	25	...	...	...	Blast.
Browning	2	2	4 1/2	1	3	5	0	...	...	1	4	4 1/2	0	2	14	0	12	12	0	Hand.
Llano de Oro	1	3	3 3/4	1	2	3 3/4	1	2	3	0	...	...	3 3/4	4	0	44	24	24	24	None.
Platurica	1	2	3	1	2	3	1	2	3	0	...	...	3 1/2	2	0	54	10	10	4	Hand and blast.
Davis	1	1	1 1/2	...	...	...	0	...	...	...	...	...	3 1/2	2	0	17	12	12	0	Do.
Gallia	(1	2	4	1	4	4 1/2	0	...	...	1	4	4 1/2	4	3	25	30	...	...	0	Derrick.
	(1	2	3 1/2																	
Lewis	1	3	3	1	1	3	1	2	4	1	2	4	3 1/4	2	11	9	4	4	1	Hand and blast.
Conners	1	...	5/8 or 3/4	...	...	...	0	...	...	0	...	...	0	1	0	...	4	4	0	Hand.
Eldorado Bar	1	...	1 1/4 or 1 1/2	1	...	1 1/2 or 2	0	...	...	0	...	...	...	1	0	...	...	...	0	Do.
Old Garden Gulch	1	2	2 1/2	1	2	3	0	...	...	0	...	...	3 1/2	2	0	19 1/2	17	24	0	Derrick.

TABLE 11.- Sluice boxes and riffles at western hydraulic mines, 1932

Mine	Water through sluice, miner's inches	Duty of water: cu.yd. of gravel per 24 hrs. per miner's inch	Sluice boxes: Width, inches	Sluice boxes: Depth, inches	Sluice boxes: Total length, feet	Sluice boxes: Grade, Inches per foot	Sluice boxes: Grade, Per-cent	Riffles: Type	Riffles: Width, inches	Riffles: Height, inches	Riffles: Length, Ft	Riffles: Length, In	Riffles: Center to center, inches	Riffles: Point at which quicksilver used	Under-currents
Senger	400	0.5	36	24	96	3/4	6.2	Wooden cross	2	6	3	0	4	...	0
Elephant	175	.8	18	16	32	...	...	Hungarian	...	...	1	6	...	...	0
Horton Gulch	250	.9	24	18	36	1	8.3	Rock paving	...	6	...		...	...	0
Banner	300	...	26	24	36	1	8.3	Pole	4	4	2	2	4 1/4	...	0
Indian Creek	500	3.7	48	36	48	3/4	6.2	Wooden blocks	12	12	...		12	Undercurrent	1
Salmon River	1,100	...	36	30	150	7/12	4.9	Rock paving	...	7	...		...	do.	1
Jacobs	460	4.2	48	36	120	3/4	6.2	Wooden blocks	...	...	...		...	...	0
Omega Hill	1,500	2.7	48	36	1,700	1/2	4.2	do.	...	12	...		...	First boxes	0
Indian Hill	650	...	40	40	288	1/2	4.2	Wooden blocks and rock paving	12	12	...		...	do.	3
Depot Hill	600	1.0	30	24	3,500	3/14	1.8	Wooden blocks	12 to 24	7	...		12	do.	0
North Fork Placers	1,300	1.0	48	40	168	1	8.3	Rails	...	3 1/2	12	0	4 1/4	do.	1
Salyer	2,800	4.3	60	50	350	3/4	6.2	Wooden blocks	...	...	...		18	do.	1
Norton and Nelson	600	...	20	20	100	3/4	6.2	Hungarian	...	...	2	8	...	...	0
								Pole	...	...	6	0			
Salmon Creek	150	2.6	26	20	180	1 1/4	10.4	Rails	2	2 1/2	10	0	4	...	0
						1 1/4	10.4	Poles	4	4	6	0	5		
						1	8.3	Hungarian	1 1/2	1 1/2	2	2	3		
Blue Channel	1,200	...	36	...	80	...	...	do.	2	4	3	0	6 1/2	...	0
Deep Creek	200	1.3	20	18	300	3/4 to 1 1/2	6.2 to 12.5	Pole	4	4	6	0	15	None	0
Yellowstone Gold	600	5	23	24	500	1/3	2.8	Angle iron	2	1 1/2	1	11	2 3/4	None	0
Virginia City	85	1.4	14.5	18	180	5/12	3.5	Hungarian	2	4	1	2 1/2	8	None	0
								Pole	4	4	6	0	5		
								16-lb. rails	1 3/16	2 3/8	12	0	3		
Henderson No. 1	600	2.0	22	24	120	3/4	6.2	Cast iron	3	1 1/4	4	0	5	None	0
Henderson No. 2	150	4.0	22	24	120	3/4	6.2	do.	3	1 1/4	4	0	5	None	0
Wisconsin Gulch	2,800	.1	44	40	1,900	7/24	2.4	40-lb. rails	1 7/8	3 1/2	30	0	6 1/4	None	0
Stemwinder	180	1.3	24	18	240	7/12	4.8	Pole	4	4	5	6	...	None	0

TABLE 11.- Sluice boxes and riffles at western hydraulic mines, 1932 - Continued

Mine	Water through sluice; miner's inches	Duty of water: cu.yd. of gravel per 24 hrs. per miner's inch	Sluice boxes: Width, inches	Sluice boxes: Depth, inches	Sluice boxes: Total length, feet	Sluice boxes: Grade: Inches per foot	Sluice boxes: Grade: Per-cent	Riffles: Type	Riffles: Width, inches	Riffles: Height, inches	Riffles: Length: Ft.	Riffles: Length: In.	Riffles: Center to center, inches	Riffles: Point at which quicksilver used	Riffles: Under-currents
Diamond City	900	.6	32	36	2,700	1/3	2.8	Pole	5	5	6	0	6 1/2	None	0
Superior			48	60	5,000	11/48	1.9	Wooden blocks		7				First boxes	0
Hookensmith	150	1.6	18	18	600	9/24 to 2/3	3.1 to 5.5	Poles	4	4	2	6	5	None	0
Golden Rule			30		160	3/4	6.2	2 by 6's	2	6	2	0	4 2/7	First boxes	0
								Poles	5	5	12	0	6		
Fortune			26	30	72	2/3	5.5	Rails	1 3/16	2 3/8	12	0	4 1/2	do.	0
Dodman and Weston	400	.8	28	18	132			Cross strips	2	4	2	4	4 3/4	None	0
Round Mountain	400		36	36	5,000	1/3	2.8	Rails	2 3/4	2 3/4	12	0	4 1/2	First boxes	0
Redding Creek	1,200	.5	48		48	3/4	6.2	Hungarian	2	4	4	0	4 1/2	do.	0
Browning	900	.7	48		32	7/16	3.6	Wooden cross	2	4	4	0			0
Llano de Oro	1,300		30	24	700	1/8 to 5/16	1 to 2.5	Steel rails			20	0		First boxes	1
			[1]30	24	180	5/16	2.5								
Platurica	650	.9	32		304	3/8	3.1	Wooden block		6	2	10		do.	1
								Angle iron	4	4					
Davis	240	.5	32	24	72	3/4	6.2	Hungarian	1	1 1/2	2	6	2 1/2	All boxes	0
Gallia	[2]325		24	24	125	9/24	3.1	Angle iron	2	2	2	0			0
								Rails			10	0			
Lewis	300	1.0	30		95	3/4	6.2	Steel rails			3	0			1
Conners	28	1.7	12	10	90	1 1/2	12.5	Pole	3	3	6	0	4		1
								Hungarian	1 1/4	2	1	0	2 3/4		
Eldorado Bar	350	1.4	30	36	400	1/4	2.1	Iron	3	1/2	16	0	4 1/2	None	1
								Wooden blocks	4	4	4	0	4		
Old Garden Gulch	400	.4	30		96	3/4	6.2	Angle iron	2	2	2	6	4	First boxes	0

1 In pit.
2 Box in pit between Ruble elevator and hydraulic elevator.

TABLE 12.- Operating data and costs at western hydraulic placer mines, 1932

Mine	Operating data									Total costs per cubic yard			
	Cubic yards washed			Washing, days, season of 1932	Total man-shifts	Men employed		Daily wage scale	Length of shifts, hours	Labor	Supervision	Supplies	Total operating
	Per day	Per season	Per man-shift			Per shift	Total						
Senger	40	3,600	20	90	180	2	2	$3.50	9	$0.18	0	$0.02	$0.20
Elephant	62	8,000	20	130	390	3	3	4.00	10	.20	0	.02	.22
Horton Gulch	80	8,000	40	100	200	2	2	3.50	9	.09	0	.02	.11
Banner	...	15,000	84	...	180	...	...	3.50	10	.04	0	.01	.05
Indian Creek	[1]175	8,800	[2]88	50	100	2	2	4.00	10	.046	0	.03	.076
Salmon River	223	29,000	56	130	520	4	4	3.50	9	.06	0	.01	.07
Jacobs	240	12,000	120	[3]50	100	2	2	4.00	8	.03	0	.02	.05
Omega Hill[4]	1,700	75,000	280	...	...	6	6	4.00	10	.02	...	.02	[5].04
Indian Hill	...	100,000	...	...	...	7	8	[6]3.50	9	...	...	...	[7].08
Depot Hill	...	40,000	...	...	...	6	6	4.00	...	...	0	...	[7].115
Salyer	7,400	718,900	500	97	...	...	15	4.00	...	...	0	...	[8].0263
North Fork Placers	[9]770	150,000	140	195	1,080	[10]2	4	4.00	9	.03	0	.015	.045
Norton and Nelson	80	12,000	40	150	300	2	2	4.00	9	.10	0	.02	.12
Salmon Creek	260	...	33	...	...	4	8	4.00	8	.12	$0.03	.05	.20
Deep Creek	57	[11]2,100	11	...	185	5	5	4.50	8	.41	0	.01	.42
Yellowstone Gold	100	9,000	31	[12]90	[13]287	2	4	3.50	9	.11	0	.02	.13
Virginia City	140	...	14	[14]36	...	4,3, and 3	10	3.50	8	.25	.10	.02	.37
Henderson No. 1	200	9,000	40	45	225	3 and 2	5	4.00	8	.10	0	.02	.12
Henderson No. 2	320	3,500	64	11	55	do.	5	4.00	8	.06	0	.02	.08
Wisconsin Gulch	[15]283	15,000	13	53	1,166	9,9, and 4	[16]22	4.00	8	.31	.03	.035	.375
Stemwinder	300	18,000	50	60	360	3	6	3.50	10	.07	0	.02	.09
Diamond City	350	...	70	...	...	3 and 2	5	3.50	8	.05	.03	.02	.10
Superior	560	67,000	70	120	960	6 and 2	8	3.50	9	.05	.03	.02	[17].12
Hockensmith	80	10,000	40	125	250	2	2	3.50	8	.09	.01	...	.10

[1]350 when washing. [2]175 when washing. [3]Water available 21 days. [4]All data for 1931 season. [5]Exclusive of ditch repairs, cleaning up, and supervision. [6]Was $4.00 other years. [7]Includes $0.02 for storage of tailings. [8]Excluding general administration. [9]900 for actual days washed. [10]12 on breaks in ditch. [11]To July 9. [12]To July 12; total season 225 days. [13]To July 12. [14]To July 5. [15]Exclusive of 4 men for 9 days cleaning up. [16]10 shifts per day extending boxes. [17]Including $0.02 general expense.

TABLE 12.- Operating data and costs at western hydraulic placer mines, 1932 - Continued

Mine	Operating data									Total costs per cubic yard			
	Cubic yards washed			Washing, days, season of 1932	Total man-shifts	Men employed		Daily wage scale	Length of shifts, hours	Labor	Super-vision	Supplies	Total operating
	Per day	Per season	Per man-shift			Per shift	Total						
Golden Rule	110	7,000	37	63	189	3	3	3.50	12	.095	...	[18].03	.125
Fortune	440	...	73	...	...	3	6	3.50	10	.05	.015	.02	.085
Dodman and Weston	104	[19]8,500	52	...	[20]164	2	2	3.50	8	.07	0	.01	.08
Round Mountain	240to880	128,000	...	...	...	2	20	...	...	...	0	...	...
Redding Creek	540	56,600	25	105	...	7	21	3.75	8	[21].135	.02	[22].035	.19
Browning	667	30,000	111	45	[23]210	4 and 2	6	5.00	12	.045	0	.015	[24].06
Llano de Oro	[25]286	50,000	57	[26]175	875	2	5	[27]4.00	8	.07	0	.01	.08
Platurica	500	100,000	55	200	1,800	3	9	4.00	8	.07	0	.01	.08
Davis	64	...	16	...	...	2	4	4.00	12	.25	0	.02	.27
Gallia	200	12,000	37	60	330	3	6	4.00	12	.11	.04	.02	.17
Lewis	[28]49	7,000	48	[29]141	147	[30]1	1	4.00	9	.085	0	.015	.10
Conners	18	...	6	...	...	3	3	4.00	8	.67	0	.26	.93
Eldorado Bar	500	...	41	...	...	4	12	3.50	8	.09	.03	[31].17	.29
Old Garden Gulch	160	...	27	...	...	2	6	4.00	8	.15	0	[32].23	.38
Boe (British Columbia)	300	...	60	...	...	5	5	4.00	10	.07	0	[33].21	.28

[18]Explosives, $0.02.
[19]To July 18.
[20]Including 32 shifts preparatory work.
[21]Ditch and reservoirs, $0.02.
[22]Explosives, $0.01.
[23]900 extra shifts on construction work.
[24]A construction cost of $0.12 per cubic yard incurred during year.
[25]400 for actual days washed.
[26]125 days actual washing.
[27]Sluice tenders, $3.50.
[28]106 for actual days washed.
[29]66 days washing.
[30]6 extra shifts.
[31]Power, $0.15.
[32]Power, $0.21.
[33]Includes interest and amortization.

TABLE 13.- Undercurrents at western hydraulic mines, 1932

Mine	No.	Distance between grizzly bars, main sluice, inches	Undercurrent tables		Riffles			Gold recovered on undercurrents, percent of total
			Width, inches	Length, inches	Type	Size, inches	Distance, center to center, inches	
Indian Creek	1	3/8	8	24	Hungarian	1 by 4	3 1/2	8
Salmon River	1	1/8	5	11	do.	1 1/4 by 1	2 1/4	3
Indian Hill	3	1 1/4	8	24	Block	6 by 6 1/2	7 1/2	...
			4	20	Angle iron	3 1/2 by 3 1/2	5	
			10	12	Rock paving	...	...	
North Fork Placers	1	...	12	20	Hungarian	1 by 1 1/4	2 1/4	...
Salyer	1	...	1, 12	34	Various	...	...	...
			16, 8	12				
Llano de Oro	1	3/8	8	12	Steel matting on burlap.	...	...	...
Platurica	1	3/8	5	22	do.	...	...	...
Lewis	1	1/4	4	12	Wire screen on burlap.	...	...	...
Conners	1	3/4	1	24	Bored plank	...	...	...
Eldorado Bar	1	...	12	36	Hungarian	1 by 1 1/4	2 1/4	5

Water rights in most of the older placer districts have been adjudicated. The rights of some old placer companies are still intact, and the water can be used without hindrance for operating these mines. However, other water rights in streams have been obtained by power or irrigation companies, and water for placer mining must be acquired from those controlling the rights. In some instances, however, water can be appropriated for placer mining

Table 9 shows the average flow of available water, the hours it was used each day, and the average quantity used in the giants and as by-wash at the principal hydraulic mines being operated in 1932. It will be noted that the average quantity of by-wash water was less than that of pressure water at most of the mines; in four mines, however, the opposite was true. At a few places no by-wash water was used.

As stated before, water under a relatively low pressure may be used for undercutting a bank to assist ground sluicing. Generally, however, a head of at least 40 feet must be available for hydraulicking sand and loam and the easiest cutting gravel. An 80- or 90-foot head usually is required to cut average gravel banks. When the gravel is tight or contains boulders a head of at least 125 feet should be available for hydraulicking. For very tight or cemented gravel, heads over 200 feet should be available. Higher heads give greater cutting and driving power to the giants and thus increase production. High pressures are necessary for high banks, as the giants must be set far enough away that caving gravel will not injure the workmen when the banks are undercut. The extreme range of the heads on the giants at the mines was 40 to 450 feet. The usual range was 100 to 300 feet.

In at least 75 percent of the 40 or more operating hydraulic mines visited in 1932 water was conveyed in ditches dug by the early miners. Often, old pipe lines or salvaged pipe were utilized. Some of the present lines are built of pipe first installed 50 years ago. Water was pumped at four mines described. Pumping water for hydraulicking, however, has not been generally successful.

Data on ditches and pipe lines also are given in table 9. Ditch lines as much as 23 miles long were used by individual mines operating in 1932. Several small mines had no ditches but took the water directly into pipe lines from the creeks above the mines.

The pipe ranged in size from one line with an intake diameter of 46 inches, which was reduced by stages to 24 inches in diameter at the pit, to lines of 10-inch pipe.

Reservoirs where used ranged in size from 0.01 to 15 acre-feet.

Duty of water.- The duty of a miner's inch of water in hydraulicking is defined as the number of cubic yards of gravel which it can break down and send through the sluice in 24 hours. The factors affecting this duty are so varied that it can be compared directly at few mines. An average duty of a miner's inch cannot be calculated for the same reason. The duty of water appears to be highest in large-scale operations. Tight or cemented gravel is difficult to break down; a high bank takes less pressure water per cubic yard than a low one; a flat bedrock requires an excessive quantity of water for sweeping; angular rock and gravel with flat or large boulders requires more water to move it than does small-size, rounded material; clay-bound gravels require excessive washing to free the gold; a high water pressure is more effective than a low one for cutting or sweeping; and the grade and size of sluices govern the daily yardage that can be washed through them. The calculated duty of water at the mines operating in 1932 ranged from 0.4 to 4.3 cubic yards per miner's inch. (See table 11.) In these calculations by-wash water is included.

Conditions at the mines range from the most difficult to at least average. Wimmler[21] reports a duty of as high as 10 cubic yards per miner's inch at some Alaskan placer mines; the usual range, however, was about the same as that shown in table 11.

---

21 Wimmler, Norman L., Placer-Mining Methods and Costs in Alaska: Bull. 259, Bureau of Mines, 1927, p. 139.

Piping

After a mine is opened up the gravel bank is undercut by the giant which allows the overlying material to cave into the pit. The fall breaks the gravel to some extent; it is further reduced by being played upon by the stream from a giant or by by-wash water. As the gravel is being disintegrated it is swept by the giant toward the sluice box. Where the gravel is clay-bound or contains lumps or streaks of clay it may be washed back and forth across the pit bottom one or more times until free from the clay.

A smaller-diameter nozzle generally is used for cutting than for sweeping. As an example, a quantity of gravel may be brought down with a giant with a 4 1/2-inch nozzle. Then the water will be shut off and a 5-inch nozzle put on the giant for driving the gravel to the sluice, or a separate giant with a 5-inch nozzle can be used. Usually two or more giants are set up in a pit even when only enough water is available to run one at a time. One large giant will do more work than two small ones using the same quantity of water. The giants are placed at the most strategic points both to cut the bank and wash the gravel to the sluice box. Where two giants are used at a time one may be used for cutting and the other for sweeping. The cutting giant is set on an angle to the face. At the old La Grange mine the streams from two 9-inch nozzles were used together for both cutting and sweeping. Giants may be set up at the lower end of the sluice to stack the coarse material in the tailings where the grade is not sufficient for it to be disposed of naturally.

Sometimes a pit is laid out so that all of the gravel washed in one season is swept to the head of the sluice. After the clean-up the boxes are extended through the washed-out pit and set up for the next year's work. At other places the boxes are extended upward as room is made.

When a pit is started a cut is taken across the channel, after which a diagonal or square face is advanced upstream. In wide channels or bars two or more parallel cuts may be taken. One pit may be worked while boulders are handled or bedrock is cleaned in the other. At the Ruby Creek mine at Atlin, British Columbia, the channel was 250 feet wide; two 125-foot cuts were made and worked alternately.[22] Wing dams of timber, logs, or boulders generally are built to guide the water and gravel into the head of the sluice. Examples of layouts are shown in figures 11 and 12.

Occasionally the form of the deposit and the contour of the bedrock are such that the gravel is washed over the side of the boxes rather than into the end. Then the sluiceway is sunk into bedrock.

At some mines overburden containing little or no gold may be mined separately. This system has an advantage when dump room at the end of the main sluice is limited, as the higher material may be disposed of elsewhere. At one mine, the Salmon River, the light top material was stripped after the water supply was too low for working the heavier gravels but was still sufficient to supply one giant. The usual practice, however, is to mine the full thickness of gravel at one time. The admixture of the top soil and light gravel with the heavier material from near bedrock may permit moving a larger proportion of boulders to the sluice than otherwise.

The number of giants used at one time in the mines operating in 1932 ranged from 1 to 4. The size of the giants ranged from nos. 1 to 6 and the diameter of the nozzles from 1 1/2 to 7 inches. Table 10 shows the number and size of the giants and the diameter of nozzles used for various purposes at the individual mines. It also contains the size of the nozzles at mines using hydraulic elevators. It will be noted that a larger nozzle is used for

---

22 Lee, C. F., and Daulton, T. M., The Solution of Some Hydraulic Mining Problems on Ruby Creek, British Columbia Trans. Am. Inst. Min. and Met. Eng., vol. 55, 1917, p. 90.

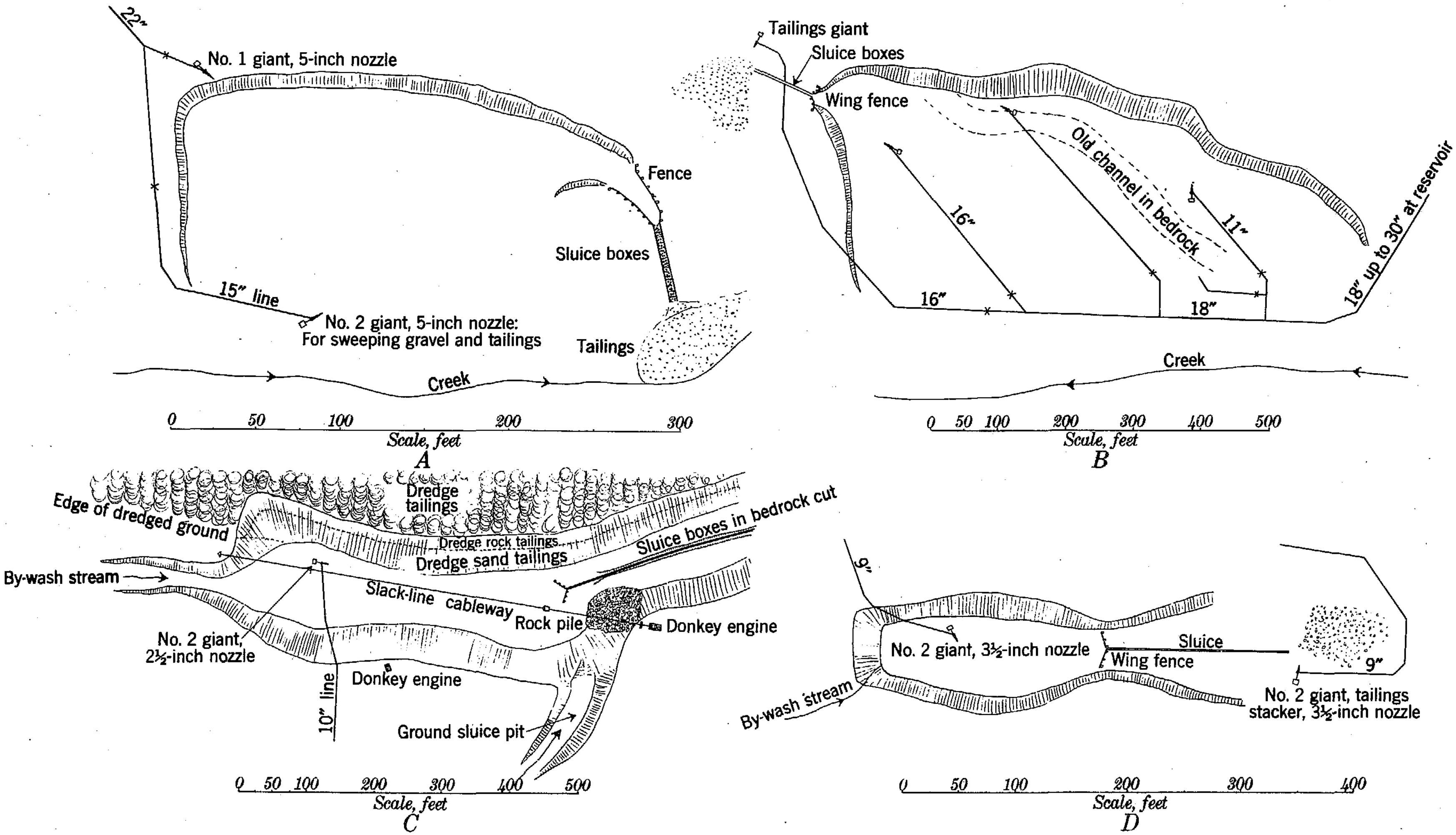


Figure 11.—Lay-outs of hydraulic mines: *A*, Indian Creek mine, Douglas City, Calif.; *B*, Blue Channel mine, Wolf Creek, Oreg.; *C*, Superior mine, Superior, Mont.; *D*, Golden Rule mine, Warren, Idaho.

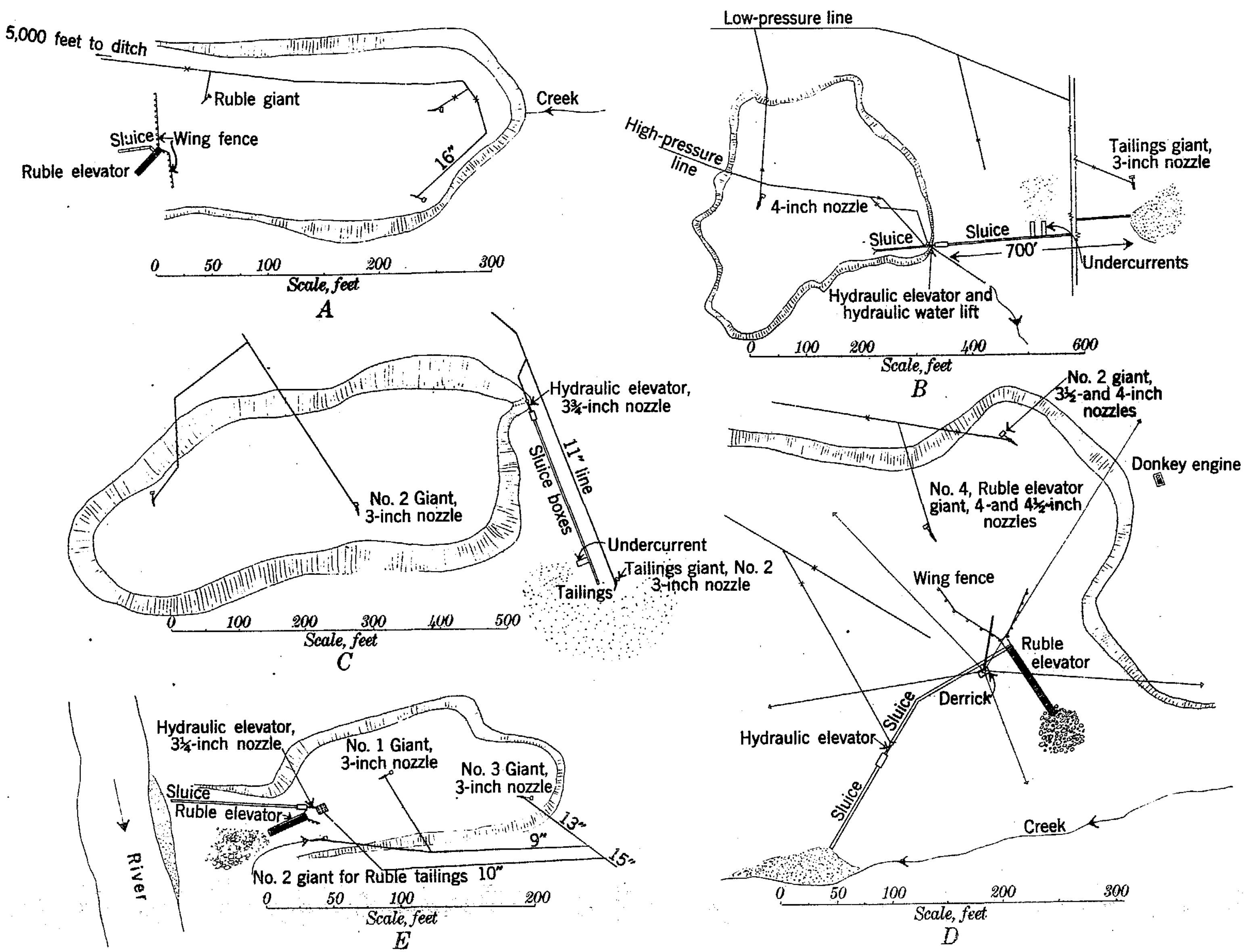


Figure 12.—Lay-outs of hydraulic mines using elevators: *A*, Browning mine, Leland, Oreg.; *B*, Llano de Oro mine, Waldo, Oreg.; *C*, Plataurica mine, O'Brien, Oreg.; *D*, Gallia mine, Sawyers Bar, Calif.; *E*, Lewis mine, Calice, Oreg.

sweeping than for cutting in about half of the mines and the same size in the other half. In one mine, the North Fork placer, water used for sweeping came from a separate source under a lower head; a smaller nozzle was used than for cutting where the pressure was higher. The nozzles used in the elevators ranged from 3 1/4 to 4 inches. The distances that the material was elevated were 25, 44, 54, 17, 30, 9, and 19 1/2 feet. The distances the coarse material was elevated by Ruble elevators were 14, 25, and 11 feet.

Handling Boulders

Where the size and grade of sluices permit, all boulders that can be moved by the giant are run through the boxes. As stated before, the upper limit in size at present mines ranged from 4 to 20 inches in diameter. At some of the early-day large producers boulders weighing 3 or 4 tons were successfully put through the sluice.[23]

In ground sluicing any boulder that can be washed into the sluice by the water usually goes through without trouble. In hydraulicking, however, boulders too large to run through the sluice may be swept into it with a large giant using a high head of water. Boulders too large to be moved by the giant or to run through the sluice are handled in various ways, depending mainly upon the number and size of the boulders encountered and the magnitude of the operations.

In small-scale operations boulders may be rolled by hand to one side or onto cleaned-up bedrock, or dragged away by teams. Occasionally, a boulder too large to handle may be left standing on the floor of the pit and bedrock cleaned up around it. The usual custom when the proportion of boulders is small, however, is to break them up by means of hammers or by blasting and wash the fragments through the sluice. In the larger operations with relatively shallow gravel, as at the Salmon River mine, the boulders may be pulled from the pit by winches or moved by a derrick mounted on a tractor, as at the Salyer mine. At the Diamond City mine a drag line with an orange-peel bucket handled boulders very cheaply under the existing conditions. A relatively narrow cut was being run. The drag line was operated on a bench above the cut and piled the boulders on the bench back of the dragline. The most common method of handling boulders, however, is by means of a derrick. The boulders that can be rolled by hand are loaded onto a sling or a stone boat and hoisted from the pit. Large ones are hoisted by means of chains. At some mines few boulders that can not be moved by the giant are encountered; derricks are used at the head of the sluice for removing those too large to go through. Stumps are handled in much the same manner as boulders.

Cleaning Bedrock

Bedrock usually is cleaned by piping. As much as 2 feet of bedrock may be cut by the giant and the material washed through the sluice. Occasionally a fire hose with a small nozzle may be used for the purpose. When the bedrock is hard and contains crevices, it must be cleaned by hand. The crevices and soft seams are dug out by means of small, flat tools made for the purpose, as described in a previous publication under Hand-Shoveling.[24]

---

23 MacDonald, D. F., The Weaverville-Trinity Center Gold Gravels, Trinity County, Calif.: U.S. Geol. Survey Bull. 430, 1910, pp. 48-58.

24 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part I. - General, Hand-Shoveling, and Ground-Sluicing: Inf. Circ. 6786, Bureau of Mines, 1934.

Sluice boxes and riffles

Sluice boxes were laid on bedrock at the most of the mines being operated in 1932. At a few, where high channels were being worked, cuts had been run to bedrock to permit an adequate grade for the sluices. In one mine, a tunnel was used.

Individual boxes were 12 feet long at the majority of places. In a few districts 16-foot boxes were preferred, and occasionally a 10- or 14-foot box was used. The length of the sluice at various mines ranged from 32 to 5,000 feet. The long sluices generally are used only when they are necessary as tailraces. The width of sluice boxes at these mines ranged from 12 to 60 inches. Data on sluice boxes and riffles at the principal hydraulic mines being operated in 1932 are given in table 11. It will be noted from this table that the extreme range in the grade of boxes was from 1/8 inch to 1 1/2 inches to the foot (1.0 to 12.5 percent). The usual range was from 1/4 to 3/4 inch to the foot (2.1 to 6.2 percent).

Riffles serve a twofold purpose, they protect the bottom of the sluice and catch the gold. Both strength and wearing qualities are required in large-scale hydraulic operations where boulders up to a ton in weight may be put through the boxes. Wooden blocks, rails, rock paving, and iron castings, in the order named, were used at the larger mines operated in 1932. When the service was not so severe, poles, angle iron, and Hungarian-type riffles were used. The Hungarian riffles usually were made of wood and were protected from wear on top by strap iron. The kind, size, and spacings of riffles used at the mines visited in 1932 are shown in table 11.

At all mines most of the gold was caught in the first few boxes of the sluice. The top boxes were cleaned up twice a season, monthly, weekly, or even oftener. In long sluices the lower boxes were cleaned only at the end of the season or when repairs were needed. At the time of the general clean-up worn riffles were replaced and the sluices repaired if necessary. Quicksilver was used in the sluices at the largest mines, but at the majority it was used only in cleaning up.

Although the sluice is an efficient gold-saving device some gold gets away, especially if the gold is very fine and the gravel carries a relatively large proportion of black sand. To further recover the gold, undercurrents were used at 10 mines listed. The term "undercurrent" in placer mining is used to designate a device for catching the gold contained in the fine material drawn out through a grizzly in the bottom of the sluice. The undercurrent usually is placed near the lower end of the sluice. At most mines it is not possible to draw all of the material small enough to go through the grizzly to the undercurrent, as not enough water would be left in the sluice to dispose of the coarse material. The quantity drawn off is controlled by the area of the grizzly and the openings between the bars. As shown in table 11, the grizzly bars are 1/8, 1/4, 3/8, 3/4, or 1 1/4 inches apart. Undercurrent boxes, or tables as they are sometimes called, are relatively wide to permit a shallow depth of the sands.

The same type of riffle generally is used on undercurrents as in sluices where a screened product is treated. As shown in table 11, Hungarian riffles, usually similar to those used on dredges, were favored. Steel matting or wire screen over burlap was used at two mines; planks with holes bored in them, angle iron, and stone paving were used at one mine each; and a variety of riffles was used at another mine (Salyer). Quicksilver was used on undercurrents at nearly all of them. An important function of an undercurrent in placers where quicksilver is used in the main sluice is to catch quicksilver or balls of amalgam that may get away in the sluice. As much as 10 percent of the recovered gold may be saved on the undercurrent, but in most places less than 5 percent is so obtained. At three mines where an estimate was made, 3, 5, and 8 percent, respectively, of the total gold recovered was saved on the undercurrent. At two places so little gold reached the undercurrents that they were not cleaned up at the end of the 1932 season.

Sluice boxes and riffles are discussed further under the general section Sluice Boxes and Riffles. The methods of cleaning up boxes also are described.

Hydraulic Mines Operated In 1932

The mines described in the following pages were visited by the authors in June and July 1932. A few other properties were inspected, but as no operating data could be obtained they are not included; with few exceptions they were unimportant, and the practices followed were similar to those at neighboring mines which are described. Two additional mines, the Round Mountain and the Eldorado, that were operated in 1931, are included.

General and operating data concerning these mines are given in tables 8 to 13, inclusive. A mine in British Columbia not visited by the authors is described in the text.

As already stated, hydraulic mines are placed in three groups: (1) Mines without elevators or pumps, (2) mines with Ruble or hydraulic elevators or both, and (3) mines where water for piping and sluicing is pumped. Within each group the operations are further divided according to States.

Mines without elevators

California

Senger.- M. A. Senger, with one man, operated a small mine near Weaverville in the summer of 1932. The deposit was a 6-foot stratum of recent gravel overlying the famous La Grange Channel at the edge of the old workings. The gravel contained a large percentage of boulders and tree stumps. Some small timber had to be cleared off before the gravel could be worked. The false bedrock (top of La Grange Channel) was relatively steep (2 inches to the foot), which facilitated piping boulders to the sluice boxes. Boulders too large to go through the 36-inch sluice were first blasted or broken by hand. Water under a head of 225 feet was obtained from the old La Grange ditch lines; a 4 1/4-inch nozzle was used. Between 30 and 50 cubic yards was handled per day, depending upon the quantity of boulders to be broken. The operating cost was 20 cents per cubic yard. (See table 12.)

Elephant.- The Elephant mine at Volcano was worked under lease during the 1932 season. The gravel ranged from 1 1/2 to 3 feet deep and was overlain by about 45 feet of white, tough, volcanic ash. The ash was drilled by hand augers and blasted. Then it was partly broken up with picks and washed away by the giant. After the ash was removed the gravel was cut by the stream from the giant with a 3-inch nozzle and swept through a race into a sluice consisting of two 16-foot boxes with Hungarian riffles. About 175 miner's inches of water under a 115-foot head was available for 4 hours each day. The tailings were run into a settling basin formed by an earth-filled dam with a concrete spillway outlet. The ditch and pipe lines were in place and were used in early workings in the vicinity. About 62 cubic yards was washed per day or 20 1/2 cubic yards per man-shift. The operating cost of washing the gravel and overburden was 22 cents per yard.

Horton Gulch.- J. O. McBroom operated the Horton Gulch placers on the South Fork of the Salmon River near Cecilville. The 1932 season extended from January 1 to April 11. The gravel was fairly tight. The grade of bedrock was 1 inch to the foot. One giant with a 5-inch nozzle, working under a 65-foot head, was used for both cutting and sweeping the gravel into the sluice boxes. About 30 inches additional by-wash water was used for moving the gravel through the sluice which consisted of three 12-foot boxes 24 inches wide. The riffles were hard boulders hand-shaped to make a pavement 7 to 10 inches thick. An undercurrent was used for 1 month and then discarded; about 1 ounce of gold was cleaned up from the undercurrent during the month's run. All large boulders were blasted. An average of 80 cubic yards per day was washed during the 1932 season. Two men were employed. At $3.50 per shift the labor cost would be 9 cents; supplies would amount to about 2 cents per cubic yard, making a total of 11 cents.

Banner.- C. A. George and brothers operated the Banner mine on the East Fork of the Salmon River near Cecilville. The gravel was tight and 8 to 20 feet deep. The head on the giants was 100 feet. Besides the three giants used for cutting, sweeping, and tailings disposal, one 2-inch nozzle was used for running a power derrick to lift to one side boulders too large to go through the sluice boxes. Two giants, one with a 4-inch and the other with a 3-inch nozzle, were used at a time. The sluice was 26 inches wide; pole riffles were used.

About 15,000 yards were washed in 1932 in approximately 180 shifts. At $3.50 per shift the labor cost would be about 4 cents; the total operating cost would be about 5 cents.

Indian Creek.- The Indian Creek placer near Douglas City was operated during the 1932 season by Gribble and son. Preparatory work began early in the spring. A new set-up was made, but old ditch lines were used. The pipe line was relaid and a new sluice put in. In 25 days, from May 20 to June 18, about 8,800 cubic yards was washed.

The ground was fairly easy to cut, and nearly all boulders were washed through the boxes. The few large ones encountered were blasted and then piped out of the pit. Water was brought to the mine through a 5-mile ditch and a 2,200-foot pipe line. The head was 275 feet. A reservoir with an automatic gate was used. It filled in about 1 1/2 hours and furnished enough water to run a giant with a 5-inch nozzle for about 1 1/4 hours. Two giants, each with a 5-inch nozzle, were set up in the pit. (See fig. 11,A.) The gravel was cut and swept to the head of the box with a no. 1 monitor. Some gravel, however, was left on the floor of the pit, and a no. 2 monitor was used every third run to wash this material into the sluice and stack the tailings at the end of the sluice box. The boxes were 48 inches wide.

While the actual washing operation was in progress 350 cubic yards was washed per day or 175 cubic yards per man-shift. This would be a labor cost of 2.3 cents per cubic yard. Estimating the preparatory work and time for cleaning up to be the same as that for actual washing, the total labor cost would be 4.6 cents, as shown in table 12; supplies would amount to about 3 cents, making a total of 7.6 cents per cubic yard.

Salmon River.- The Farnsworth brothers operated the mine of the Salmon River Mining Co. on the South Fork of the Salmon River near Cecilville during the 1932 season. The gravel consisted of 6 feet of pay dirt overlain with 11 feet of overburden. The grade of the bedrock was three fourths of an inch to the foot. Water under a 225-foot head was brought to the mine through a 3-mile ditch and 1 mile of pipe line. The diameter of the first 3,000 feet of the pipe line was 22 inches. This was reduced to 18 and then to 15 inches at the pit. The branch line on the floor of the pit to the different giants was of 11-inch pipe. Four giants with 6-inch nozzles were set up in the pit, but only two were used at a time. A fifth giant with a 5-inch nozzle was set up at the lower end of the sluice. Usually one giant cut the bank and one of the same size swept the gravel to the head of the sluice. The river ran alongside of the gravel being washed, and the sluice box emptied into it. The river water carried the sand and fine gravel downstream; coarse material, however, piled up in the stream. The dump giant was used 1 1/2 to 2 hours during the working shift to stack the coarse material at the end of the box. At the end of the washing shift this giant was set with an automatic control so that water played on the boulders until the next morning. A windrow of boulders 30 feet high along the opposite bank of the river had been made by the giant. The largest boulders were washed to the top of the pile. The stream played in a vertical arc; it was depressed slowly and went up faster. About 1 minute was consumed in each cycle. The giant was overbalanced so that the stream was elevated when free. It was pulled downward by means of a 2-inch hydraulic cylinder fed through a hose from the pipe line. At the end of the stroke a trip turned a valve which shut off the water to the cylinder; at the top of the upward swing another trip opened the water valve. Each morning the river bed at the end of the sluice was free of boulders.

The sluice boxes were 36 inches wide by 30 inches high and were set on a grade of 7 inches to each 12-foot box. One setting of the sluice-way sufficed for a season's work. The head boxes were protected by parallel rows of 6-inch poles placed horizontally on either side of the box. The poles were laid on an earth fill, the surface of which slanted upward at an angle of 25° from the edge of the boxes. At the end of the season the poles were removed and the underlying gravel was washed into the boxes. The riffles in the sluice consisted of rock paving. Diorite boulders with one flat side were selected from the washed gravel in the pit. These stones were dressed by hand to make a rigid paving with a fairly smooth upper surface. Formerly wooden blocks were used, but they had to be replaced every 60 to 70 days. The sluice was cleaned up at the end of the washing season. An undercurrent was used at the lower end of the sluiceway. The screen consisted of 3/4-inch round steel rods 15 inches long, placed 1/8 inch apart lengthwise with the sluice. The undercurrent table was 5 feet wide and 11 feet long; wooden Hungarian riffles were used. Quicksilver was used in the sluice box; some reached the undercurrent where it was caught in the riffles.

Boulders up to 18 inches in diameter were put through the sluice. A hand derrick with a 25-foot mast and two 30-foot booms was set at the head of the sluice to remove any oversize boulders that were washed to this point. A derrick hoist was used for dragging stumps and large boulders from the main part of the pit. The hoist pulled over a 25-foot mast guyed with 4 lines; apparently, however, 5 lines should have been used. The cable (1 1/2 inches in diameter) was pulled out by hand; the range was 400 feet from the hoist set-up. The hoist was double-geared and was run by an undershot water wheel driven by a 1 1/2-inch nozzle. A stream from a 1-inch nozzle was used on top of the water wheel for braking. No explosives were used in the mine. The overburden was washed from the top of the gravel and run directly into the river. This work was done during low-water periods when enough water for only one giant was available.

Lumber cost $30 per M. An average of 223 cubic yards was worked per day during the 1932 season. The operating cost was 7 cents per cubic yard with labor at 6 cents.

Jacobs.- The Jacobs mine on the Trinity River at Junction City was worked during the 1932 season by H. K. Wilson and partner. The gravel was about 12 feet deep and was overlain by 28 feet of recent wash. Water from other workings in a side gulch above was conserved in a reservoir of about 2 acre-feet capacity. The reservoir was made by an earthen dam across a depression on a bench. The water was conveyed to the workings through a 600-foot pipe line of 22-, 18-, and 15-inch pipe. The head was 90 feet.

A no. 4 giant with a 7-inch nozzle was used. Enough water was available for washing only 1 1/2 hours per day. The remaining time was used in drilling and blasting boulders. Four 50-pound boxes of explosives were used during the 1932 season. The sluice boxes were 48 inches wide and had 9 inches of fall per 12-foot box. The sluice was placed in an old cut in bedrock. A cut 40 by 40 by 200 feet was washed during the season. The season was 50 days long, but water for piping was available only 21 days. A total of 12,000 cubic yards was washed at a cost of 5 cents per yard.

Omega Hill.- The Omega Hill mine was on an ancient channel above the South Fork of the Yuba River near Washington Camp; it was an extension of old workings. Old ditch lines and water rights were used. The gravel was 30 to 60 feet deep under 10 to 20 feet of volcanic ash. The lower 6 to 8 feet of gravel was partly cemented. Gold-bearing gravel also lay on top of the ash; the top gravel was washed separately from the lower gravel. Water was brought to the mine through two ditch lines. The upper line was 16 miles long and had many flume sections 6 feet wide and 3 feet deep. The lower ditch was 8 miles long with a cross section of 3 by 4 feet. The head in the pit was 210 feet. The water was regulated by a reservoir above the workings. During 1931, an unusually dry season, water was available for 45 days. The usual season had been from February 1 to September 1. Although plenty of water was available in 1932, a very late start was made because the ditch and flume lines required extensive repairs.

One 6-inch giant was used for cutting and another for sweeping the gravel to the head of the sluice. Sometimes three giants with 5-inch nozzles were employed. A varying amount of wash water was used, depending upon how fast the gravel was cut. The sluice, which was 48 inches wide and 36 inches high, was brought into the pit through a bedrock cut. After leaving the cut it was 1,700 feet long and was built along the mountainside to a gulch where the tailings were stored back of a dam. The tailings dam was made by dumping the coarse material on the dam and running the fine material back into the reservoir. At the end of the boxes the stream was run over a grizzly 6 feet wide and 11 feet long. The grizzly spacing was 1 inch wide at the bottom and 5/8 inch wide at the top. The oversize was used to build up the dam. The undersize was run back in another flume parallel to the main sluice and dumped in the reservoir back of the dam. The dam was started with brush and sand, then rock was dumped on top. Before starting a reservoir a culvert 2 feet 4 inches by 3 feet 4 inches was first laid on the bottom of the gulch. The clear water entered the culvert at the upper end of the reservoir and discharged through the dam. When one section of the gulch was filled another dam was built below. The antidebris law in California does not permit placer tailings to be run into the Yuba River.

After the washing season was over the sluice boxes were cleaned up and worn wooden blocks replaced. The company operated a sawmill for cutting riffle blocks and lumber for sluice boxes. An average of about 1,700 cubic yards per day was handled. When operating full force the labor cost per cubic yard for washing was about 2 cents, and total costs were 4 cents per cubic yard.

Indian Hill.- B. F. Dyer operated the old Indian Hill mine near Comptonville during 1931 and 1932. The material washed up to the end of 1932 consisted mainly of slides from the faces of the old workings. The gravel deposit was 35 feet thick; the grade of bedrock was 1 inch to the foot. Water was brought to the mine through a 9-mile ditch and 3,000 feet of 22-inch and 1,500 feet of 15-inch pipe. The head was 130 feet. One no. 6 giant with a 4-, 4 1/2-, or 6-inch nozzle was used.

Boulders up to 14 inches in diameter were run through the sluice boxes. The sluiceway was down a narrow gulch and consisted of six sections of boxes (2 to 6 boxes to the section) and the rock bottom of the gulch between sections. There was a drop of 10 or 15 feet at the end of each section of boxes. The fall and cascading down the rocky gulch between each section broke up all cemented material and washed the gravel free of clay.

The boxes were 40 inches wide and 40 inches high; the grade was 1/2 inch to the foot. The upper five sections of boxes were paved with wooden blocks; the riffles in the lower section were of rock paving. Seventy percent of the gold was caught in the upper two boxes. Three undercurrents were used near the lower end of the line. The discharge of one undercurrent went into the main sluice before the next was taken out. The grizzly opening for an undercurrent in the bottom of the main sluice was 18 by 40 inches. The grizzlies were of 1 1/4- by 3-inch iron bars set on edge 2 1/2 inches center to center. Additional top water was run over the undercurrents from an opening in the side of the sluice. The first undercurrent was 8 feet wide by 24 feet long. The riffles consisted of rows of pine blocks 6 inches thick by 6 1/2 inches deep separated by 1 1/2-inch crossboards. The second undercurrent was 8 feet wide and 20 feet long. The riffles consisted of 3 1/2- by 3 1/2-inch angle iron 7/8 inches thick and set crosswise on 5-inch centers. The third undercurrent at the end of the lowest box was 10 by 12 feet. The riffles were four angle irons 1 inch apart at the head of the undercurrent and rock paving from there down.

A crew consisted of 7 or 8 men. About 100,000 cubic yards was washed during the 1932 season at a cost of 8 cents per cubic yard exclusive of construction work.

Depot Hill.- The Depot Hill mine, near Comptonville, belonging to Fred Jourbert, has been worked continuously since 1855. It was opened up by his grandfather and later worked by his father. The gravel is an old channel running parallel to the present creek; it averages 60 feet deep. The grade of the bedrock is 1/2 inch to the foot. The gold occurs in the lower 10 feet on bedrock. The gravel contains a 2-foot streak of pipe clay, which after being broken down has to be shot or picked before piping. Holes are drilled with an auger.

The number of hours per year that water has been available for piping has averaged 1,100 to 1,400 since 1917. The previous 20 years the average was 2,500 to 3,000 hours. In 1903 water was available for 4,300 hours and in 1931 for only 36 hours. Piping was done about 1,200 hours during 1932. The water supply was brought to a reservoir through three ditches having a total length of 9 miles. A 3,800-foot pipe line of 30- to 11-inch pipe ran from the reservoir to the mine. The effective head was 160 feet. During high water 600 to 700 miner's inches were used. The mine operated as long as 400 inches were available. Toward the end of the season piping was carried on 3 or 4 hours per day. The mine was worked one shift per day.

The crew consisted of 5 or 6 men, including Jourbert, who did the piping. When piping, the pit crew tended the sluice box and piled boulders. At other times they picked up clay streaks on bedrock and blasted the clay stratum as it dropped into the pit; they also broke large boulders.

The sluice was 30 inches wide and 3,500 feet long. This length was necessary to reach a place for depositing the tailing. The grade was only 3 inches in 14 feet, which was considered by Jourbert to be the minimum practical one. The flat grade limited the size of material that went through the boxes and required more water per yard of gravel handled. Riffles were of 7-inch blocks; those made of Douglas fir lasted longer than those of other native woods.

The gravel in the gold-bearing part of the bank is clay-bound. The rock must be washed free of clay to make a satisfactory saving of the gold. After being cut the gravel was swept back and forth across the pit to the head of the sluice box to break it up thoroughly and to free the clay. The gravel was washed until the quartz pebbles showed white.

A no. 4 giant with a 4 1/2-inch nozzle was used for cutting. The nozzle was replaced with one 5 inches in diameter for sweeping. According to Jourbert, the larger nozzle is more effective for driving and the smaller one better for cutting.

A second giant was set up at the lower end of the pit. By-wash water cascaded over the bank.

The first 200 feet of boxes was cleaned twice during the season, and the whole sluice was cleaned at the end of the season. As the sluice was cleaned it was repaired and worn riffle blocks were replaced. It required 2 months for one man to do this work.

Quicksilver was put in the first 4 or 5 boxes and renewed once a month. Four to five 75-pound flasks were used each season. One flask was put in at a time.

An average of 24 cubic yards per hour is handled during piping. The operating cost is about 11.5 cents per cubic yard.

Relief Hill.- The Relief Hill Mining Co. began operations near Comptonville in the autumn of 1931 and worked 4 months in 1932 before the water supply failed. An old mine was being rejuvenated; the gravel was 200 feet thick. About 500 miner's inches of water was used during the season. A total of 1,000 inches will be used when the mine is fully reopened.

The old pit was cleaned and virgin gravel reached in 1932 just as the water played out. Tailings were impounded behind dams in a dry canyon. The ditch line is 7 miles long. The pipe line is 14 to 22 inches in diameter, and the effective head is 210 feet. The sluice boxes are 48 inches wide; riffles are wooden blocks. A duty of 3 cubic yards per 24 hours per miner's inch is expected. A crew of 15 men worked 120 days in 1932.

Canyon Creek.- The United Placers, Ltd., was operating the Canyon Creek placers near Weaverville during the 1932 season. Four men were employed on each of 3 shifts per day. Two giants were in use in the pit. No operating data are available.

North Fork.- The North Fork placers on Trinity River at Helena were worked under a leasehold during the 1932 season by F. M. Reynolds, W. O. Kunman, and E. C. Mathews. A fourth man was employed. The mine was operated two 9-hour shifts with two men on a shift. The gravel deposit consisted of an old channel cutting through a ridge. The lower 15 feet of gravel was very tight and partly cemented. It was broken down by first cutting the bedrock from underneath it. After being broken down considerable piping was necessary to disintegrate the cemented fragments. The top gravel washed easily.

Water was brought to the mine from two sources in different flume lines. The lower flume emptied into a reservoir which supplied a giant with a 5-inch nozzle for about 5 hours' piping a day. A pipe line to the upper flume supplied one giant with a 7-inch nozzle steadily. Two bad breaks in the flumes during the season materially increased the cost per cubic yard washed. As the water supply decreased the diameters of the nozzles were reduced from 7 to 6 inches and finally to 5 inches.

Two sluices, consisting of seven 12-foot boxes 48 inches wide were used. One sluice emptied out of one end of the pit through a bedrock cut varying up to 30 feet in depth; the other box went out the opposite end. The riffles consisted of heavy rails placed crosswise in the boxes on top of 4- by 4-inch timber. An undercurrent was used at the end of the sluice that carried away most of the material. The undercurrent table was 12 by 20 feet and was decked with the type of Hungarian riffles used on dredges. Between 800 and 1,000 cubic yards was handled in 18 hours with a full head of water. The average daily yardage handled for the season was 770 cubic yards. As shown in table 12 150,000 cubic yards was washed during the season (Dec. 15 to June 30). The labor cost was 3 cents per cubic yard; supplies were estimated at 1 1/2 cents, making a total operating cost of 4 1/2 cents. The lessees had no supervision or general costs. The indicated costs do not include depreciation, interest on investment, or amortization.

Salyer.- The Salyer Consolidated Mines Co. began operations near Salyer in 1931.[25] Hydraulic operations were started in January and continued for about 3 months under difficulties on account of the unprecedented drought. The water was used largely for opening up pits preparatory to mining and running cuts for sluiceways. A total of 262,000 cubic yards was moved at a cost of 10.59 cents per cubic yard.

Hydraulicking began on February 10, 1932 and continued until May 22; 718,900 cubic yards of material were moved. The work was done in one pit near the river. About 40 feet of "blue" gravel overlain by 60 feet of recent gravels and clays was washed. The overburden was easy to mine, but the gravel was more difficult to cut, contained a large proportion of boulders, and had a clay capping.

Water was brought to the pit from Campbell Creek across the river through a series of flumes, ditches, and siphons costing about $300,000. The water system was capable of carrying 5,000 miner's inches of water. Two 16-inch pipe lines having a combined capacity of 2,800 miner's inches led to the pit from the main siphon under a head of 350 feet. Two giants were used most of the season with nozzles ranging from 5 to 8 inches, depending upon the

25 Information supplied by D. E. Carleton, manager, Salyer Consolidated Mines Co., Salyer, Calif.

quantity of water available which varied from time to time. The water system contained no reservoirs; this handicapped operation at the end of the season when the supply fell below 1,500 inches which was considered about the minimum requirement for the sluice in use at the property. The water duty was 4.33 cubic yards per miner's inch.

The sluice was 60 inches wide by 50 inches deep, 350 feet long, and set on a grade of 8 inches in 12 feet. Riffles consisted of 18-inch square wooden blocks 12 inches long. The length of the sluice was considered insufficient for the efficient recovery of the type of gold prevalent at Salyer. The wooden rifles were not satisfactory, as the large volume of material which moved rapidly due to the fast grade wore the blocks quickly and necessitated frequent renewals. Steel rails were to be installed.

Undercurrents and settling tanks were installed below the main sluice where about half the water and a large proportion of the fine material was taken from the boxes through a grizzly and run over small riffles on a 12- by 34-inch table and sixteen 4- by 12-foot boxes. About 90 percent of the gold recovered was saved in the main sluice and 10 percent on the undercurrent. Quicksilver was used in the main sluice and on the undercurrents.

Boulders in the pit were handled with a crane on a tractor. The mine operated 97 days with an average crew of 15 men. The daily yardage handled was 7,400 and the yardage per man-shift 500 while hydraulicking. Some preliminary fitting up was necessary, and the sluice was cleaned up after piping ceased. The direct mining cost was 2.63 cents per cubic yard. This figure included depreciation of machinery and mining equipment but not of the main water supply or other permanent installations. It did not include depletion of mine property nor general administrative expenses.

Oregon

Norton & Nelson.- Norton and Nelson operated a placer on Galice Creek near Galice during the 1932 season. The gravel averaged about 12 feet in depth and contained nearly 20 percent of boulders over 1 foot in diameter and very little clay. Bedrock consisted of slate of medium hardness; it was rough and had a grade of one half inch to the foot.

Water was brought through a 1 1/2-mile ditch, 500 feet of flume, and 300 feet of 15-inch pipe. The effective head was 90 feet. The maximum supply was 1,000 miner's inches and the average 600; a minimum of 500 inches was required to operate the mine. During the 1932 season the gravel was cut with a no. 2 giant with a 3-inch nozzle and driven by a no. 2 giant with a 4-inch nozzle. Only one giant was used at a time. Most of the water supply was used as a bywash. The sluice boxes were 20 inches wide, 20 inches deep, and 10 feet long. Ten boxes were used; the grade was 3/4 inch to the foot. Hungarian and pole riffles were employed. Boulders were handled with a 2-drum gasoline hoist.

The washing season in 1932 was 150 days; 12,000 cubic yards was washed. Two men operated the mine, and an average of 80 cubic yards was washed per day. The labor cost at $4 per shift would be 10 cents per cubic yard; supplies would amount to about 2 cents per yard, making a total operating cost of approximately 12 cents.

Salmon Creek.- An innovation in placer mining was being tried in July 1932 at the Salmon Creek mine near Baker by John M. Starr. The boulders were removed from the pit by a gasoline-driven shovel with a 1/2-cubic-yard dipper. The gravel contained a large proportion of boulders and clay which made it hard to cut and wash. Insufficient water was available to wash enough gravel per shift to make the mine pay unless other means of handling the boulders were provided.

A no. 2 giant with a 2 1/4-inch nozzle under a 150-foot head was used for cutting the gravel. Wash water coming over the bank assisted the water from the giant to transport the gravel to and through the sluice. As boulders were uncovered with the giant they were picked up in the dipper and cast to one side by the power shovel. When not otherwise occupied, the shovel was used in loosening the gravel. The power shovel had the ordinary type of dipper but was to be converted to a dragline with a clamshell or orange-peel bucket which would work to better advantage.

One foot of gravel, which contained most of the gold, was left on the bedrock and at the end of a month's run was taken up and washed separately. The boxes were 26 inches wide and had a grade of 1 1/4 inches to the foot for 80 feet, then a grade of 1 inch to the foot for 100 feet.

Riffles in the first 20 feet of boxes were iron rails placed lengthwise in the boxes. The next 60 feet were 4-inch pole riffles also set lengthwise. The function of the first 80 feet of riffles was to help break up the clay in the gravel. The lower 100 feet were Hungarian riffles made of 1 1/4- by 1-inch wooden cross strips iron-clad on top; the spacing was 1 1/2 inches between riffles.

The operating crew consisted of 1 piper, 1 shovel operator, 2 sluice tenders on each of two shifts, and a superintendent on day shift. During each shift one of the sluice tenders was detailed to keep small boulders moving down and out of the boxes. About 130 cubic yards was washed each shift. The gasoline consumption on the shovel was 12 1/2 gallons per shift. With labor at $4 per shift the labor cost per cubic yard would be 12 cents; at 20 cents per gallon the gasoline cost would be 2 cents per cubic yard; other supplies and repairs would cost an additional 3 cents and supervision 3 cents, making a total operating cost of 20 cents per cubic yard. This does not include rental on the shovel, interest, or amortization of the plant.

Blue Channel.- The Blue Channel mine on Coyote Creek near Wolf Creek was operated in 1932 by M. C. Davis. The deposit consisted of recent river gravels overlying an old blue channel. The channel, which contained most of the gold, was in a depression in the bedrock of the present stream bed; it consisted of cemented gravel, had a maximum depth of 10 feet, and was about 30 feet wide.

Water was brought to the mine under a 360-foot head through a 4-mile ditch and 1,100 feet of 32- to 18-inch pipe. A 4 1/2-inch nozzle was used for cutting, and 5-inch nozzles were used on the giants for sweeping the gravel to the sluice. A giant with a 5-inch nozzle was used also for stacking the tailings. Two giants were used at one time. Figure 11,B, shows the set-up of giants for working the mine.

The top gravel was first piped off, then the channel was worked by first blasting a trench lengthwise in the middle of a section of the channel. After one line of holes was blasted and the broken gravel piped out, a second row in the bottom of the trench was drilled and shot. The remaining gravel of the section was then plowed up with the giant, breaking to the trench made by blasting. The cemented gravel was disintegrated while being swept to the sluices. The sluice boxes were 36 inches wide and had a total length of 80 feet. A plank fence 12 feet high guided the gravel to the sluice at the end of the pit. (See fig. 11,B.) Cross riffles made of 2- by 4-inch timber, clad on top by 7/16-inch strap iron and spaced 2 1/2 inches apart, were used.

The operating crew consisted of 3 men, 2 in the pit and 1 at the reservoir. The reservoir held enough water for piping with two giants for 3 hours. The average piping time for two giants was 5 hours per day during the season, while at the end of the season only one giant could be used for 2 1/2 hours of piping. The crew worked on boulders while the reservoir was being refilled. The yardage handled and the days worked were not available.

Montana

Deep Creek.- L. E. Frank operated a new placer property on Deep Creek near Lozeau during the 1932 season. No other placer work had ever been done on Deep Creek or in the immediate vicinity. The gravel was tight in spots and contained about 15 percent of boulders. A pit 125 feet long, 30 feet wide, and averaging 15 feet deep had been washed from May 25 to July 9. A reservoir was formed by a board-and-log dam built across the creek bed. As the reservoir filled the gate was opened by hand. The water ran for 20 minutes, and the reservoir was refilled in 30 minutes. Water was conveyed to the pit through a 1,000-foot line of 10- and 8-inch pipe. The head was 135 feet. A no. 2 giant with a 3-inch nozzle was used for piping. About half of the available water was used through the giant and half as a bywash. Between piping periods the men removed boulders and put in boxes. The large boulders were left in the pit, and the bedrock was cleaned up around them.

Boxes were 20 inches wide; the grade was from 9 to 18 inches per 12 feet of box. Riffles were 3- to 4-inch poles in 6-foot sections, placed longitudinally in the boxes. Two boxes near the end of the sluice had wooden Hungarian riffles, but they were wearing out rapidly and were to be replaced with poles.

The crew consisted of 5 men - 1 piper, 3 men working on boulders and boxes, and 1 attendant at the dam. An average of 57 cubic yards per day was washed in July, or 11 cubic yards per man-shift. Wages were $4.50 per day, which made the labor cost 41 cents per cubic yard of gravel; supplies cost about 1 cent, making a total operating cost of 42 cents per cubic yard. Most of the workers were inexperienced; with experienced miners the labor cost should be less than half as much.

Yellowstone Cold.- The Yellowstone Cold Mining Co. had been operating on Emigrant Creek near Emigrant during the last 4 seasons (1929-32). The work consisted in running a cut and a pit to reach bedrock; the objective, however, had not been attained. A combination of hydraulicking and ground-sluicing methods was used. The cut was 500 feet long; 264 feet was timbered with tunnel sets, and boulders were piled on top and at the sides. The sluice ran through the tunnel. Gravel alongside the creek and in a bar 100 feet high was worked. The ground was easy to wash but had about 10 percent of boulders. Water taken directly from the creek was brought to the pit through 1,000 feet of 20-inch, 500 feet of 14-inch, 500 feet of 12-inch, and 400 feet of 10-inch pipe under a head of 175 feet. Piping was done with a no. 2 giant with a 4-inch nozzle. About one fourth of the time was used for cutting and three fourths for sweeping. Plenty of water was available, and all the bywash that could conveniently be used was run through the pit. The sluice usually was run full of water.

Boulders were handled by a derrick, which was run by a tractor engine. The hoist was attached to the front of the tractor. A 3-ton load could be raised.

Sluice boxes were 23 inches wide, 24 inches high, and 500 feet long, with a grade of 4 inches to 12 feet. Angle-iron riffles were used in the first 200 feet. These were 2 inches wide, 1 1/2 inches deep, and 1/4 inch thick, spaced 3/4 inch apart. The last 300 feet of sluice contained no riffles; the bottom of this section was protected with 2-inch plank that had to be replaced every 2 months.

The working seasons began about April 15 and extended into December when the water started to freeze. On July 12, 1932, the pit was about 150 feet long and 75 feet wide with a 50-foot face along one side. The first 2 weeks of the season three men were employed in repairing the tunnel and extending the sluiceway. One box per day was added during the first week of washing. Two men on each of two shifts worked most of the season. When a large quantity of boulders had been uncovered the full crew came out on day shift. One man handled the giant, and the other three worked on the boulders. The mine was worked on a fifty-fifty basis; the men and company each got half of the gold recovered. The company furnished all timber and other supplies. Timber cost $20 per M at Emigrant in 1932; other years it had cost $25 per M. At the current wage of $3.50 per shift the labor cost would be 11 cents per cubic yard; supplies cost about 2 cents, making a total operating cost of 13 cents per cubic yard.

Virginia City.- The Virginia City Mining Co. began working a 30-foot gravel bar at the head of Alder Gulch above Virginia City in 1932. An average of about 50 miner's inches of water was brought under a 65-foot head through 200 feet of 15-inch pipe from a side draw; in addition, about 15 inches came down the main gulch during the working season. A small reservoir had been constructed at the head of the pipe line. Cutting was done by means of a 2 1/2-inch nozzle on a 4-inch rubber firehose. A 2 1/2-inch hose with a 1 1/4-inch nozzle connected to the pipe line was laid flat in the sluice for washing the gravel through the boxes. Insufficient water was available to supply full pressure at the nozzles all the time; the large nozzle was closed part of the time to let the pipe and reservoir fill. The gravel was broken down very easily; most of the effort was expended in getting the gravel into and through the boxes. The head of the sluice line was kept very close to the bank being washed. There was considerable flat, angular rock in the gravel, which was extremely difficult to move with the quantity of water available under the relatively small head. Fragments too large to be moved by the water were loaded by hand into a 1-ton mine car on a 16-inch gage track and run to a dump; boulders too large to be loaded by hand were first bulldozed.

The sluice consisted of twelve 16-foot boxes 22 inches wide but with a false side which reduced the width to 14 1/2 inches. The full width was used in the spring when more water was available. Riffles consisted of four 6-foot sections of poles, one 12-foot length of 16-pound rails, and ten 4-foot sections of Hungarian riffles. The Hungarian riffles were made of 2- by 4-inch lumber beveled to give a pitch downstream and capped with 1/4-inch steel which extended about an inch over the wood on the downstream side. To July 5 an average of 140 cubic yards had been washed per day. Four men, including a working foreman, were employed on day shift and three on each of the other two shifts. A superintendent was also on the job.

Labor cost about 25 cents per cubic yard, supplies 2 cents, and supervision 10 cents, making a total operating cost of 37 cents per cubic yard.

Henderson.- The Henderson Mining Co. washed out two pits on a bar on Gold Creek during 1932. The second pit was worked after the water got low. The thickness of the gravel at no. 1 pit was 45 feet and that at no. 2 pit 15 feet. The gravel in no. 1 pit had a relatively high percentage of large boulders; this was reflected in the operating cost.

Water was brought to no. 1 pit through a ditch and an 18-inch pipe line from a reservoir that held about 1 acre-foot. The head was 270 feet. Two hours were used each shift for cutting and sweeping with a giant having a 5-inch nozzle; the remaining time was required to remove the boulders. Although more water was available during most of the run, two hours of piping uncovered as many boulders as could be handled during the shift. Boulders were handled by means of a power-driven derrick.

The sluice boxes were 22 inches wide and 24 inches deep, set at a grade of 8 inches in 12 feet. The riffles consisted of cast-iron bars 1 1/2 by 3 inches by 4 feet long set on 3- by 4-inch timber. The iron was fastened to the timber through lugs cast as a part of the riffle. The riffles were placed lengthwise, four to a box, with a 2-inch spacing. These same riffles were used crosswise in 48-inch boxes in early days of mining on Gold Creek.

The derrick had a 46-foot mast and a 46-foot boom, both made of 16-inch round timber. The lifting capacity was 6 tons. It was run by a 4-cylinder automobile engine belt-connected from a 5-inch pulley on the clutch to a 24-inch pulley, then to a double-geared 12-inch drum 18 inches long. Flexible wire rope five eighths inch in diameter was used for the working cable and guy wires. The drum was fitted with a hand brake.

Three men worked on day shift and two on night shift. About 9,000 cubic yards was washed in 45 working days. If the time for setting the boxes and cleaning up is included the labor cost was about 10 cents and supplies 2 cents, making a total of 12 cents per cubic yard.

After no. 1 pit was finished the gravel in no. 2 pit was washed with a no. 2 giant with a 2-inch nozzle, using an 80-foot head of water. Enough water was available for piping two 2-hour periods each shift. A 2-inch nozzle was used for cutting and a 4-inch nozzle for sweeping. Cutting was done for about 15 minutes each period, and the remaining time was used for sweeping. The gravel contained a relatively small quantity of boulders but considerable clay. Thirty-five hundred cubic yards was washed in 11 days. The same crew worked in both pits. A total of 55 shifts was worked at no. 2 pit, making a labor cost of 6 cents per cubic yard. If supplies cost the same as in the first pit, the total operating cost would be 8 cents. Although only about one seventh as much water was available and used in no. 2 as in no. 1 pit the labor cost was 6 cents in the former as against 10 cents in the latter. This difference in cost was due to the larger size and higher percentage of boulders in the gravel of pit no. 1.

Wisconsin Gulch.- The Wisconsin Placer Gold Corporation started operations at the mouth of Wisconsin Gulch near Sheridan in May 1932 and operated until July 6 when the water failed. The deposit lay in the bottom of the canyon where it joined the valley and consisted of a mass of hard boulders interspersed through the gravel. About 20 percent of the material was over 12 inches in diameter. A long cut starting in the valley was necessary to reach bedrock. Water from Wisconsin Gulch was conveyed to the mine in a 3/4-mile ditch and through a 2,700-foot pipe line. The diameter of the pipe at the intake was 46 inches and at a header box near the pit 24 inches. From the header box two 11-inch lines 400 feet long entered the pit. The head on the water at the pit was 220 feet. During the early run-off there was about 4,000 miner's inches of water; the average was about 2,800 inches. Operations continued as long as 1,600 inches was available.

A no. 4 giant with a 5-inch nozzle was used for cutting and another no. 4 giant with a 6-inch nozzle for driving during most of the season. As the water got low 4-inch nozzles were used on both giants. A large stream of by-wash water was used. At the end of the season a pit 150 feet long, 100 feet wide, and 27 feet deep had been washed.

The sluice boxes were 44 inches wide and 40 inches deep and set on a grade of 3 1/2 inches in 12 feet. The sides and bottoms of the boxes were constructed of tongue-and-groove Oregon fir costing $45 per M at the mine; 300 board feet of timber was required for each box. The main sluice was 1,900 feet long with two 500-foot wings at the discharge end. The branches allowed a wider distribution of the tailings and permitted the addition of boxes without shutting down the mine.

Forty-pound rails 30 feet long, set on 6-inch centers, were used as riffles. The rails rested on 4- by 6-inch cross-sills on 6-foot centers. The webs of the rails were far enough apart to allow fine material to drop through to the bottom of the box. The rails were set lengthwise in the boxes to offer a minimum of resistance to the coarse material handled. The large stream of water and the longitudinal rails permitted washing 18- or 20-inch boulders through the sluice despite the relatively flat grade.

As the tailing piled up the sluice was extended out into the valley. A crew of 5 men, working on each of the two shifts per day, built and put in the boxes at the ends of the sluice. As the tailing pile extends into the valley more space will be available and fewer new boxes will be necessary per season. Boulders too large to run through sluices were raised from the pit by means of a gasoline-driven derrick. The mast was 45 feet and the boom 50 feet long.

No quicksilver or undercurrents were used. The main sluice was cleaned up by 9 men working 4 days at the end of the season. After the rails were raised by means of a block and tackle the material beneath them was shoveled into a 12-inch box for separating the gold from the concentrate.

The workings were lighted by one hundred 60-watt electric lamps. The power was made by a direct-current generator direct-connected to a 16-inch Pelton wheel run by a 1-inch nozzle from a 4-inch line from the main pipe line. The lighting plant was about 1,000 feet up the creek from the pit, where a head of 180 feet was available.

Operations at the mine were carried on through three 8-hour shifts. The crew consisted of 2 pipers and 2 sluice tenders on each shift and 5 men making and putting in boxes on each of two shifts. Supervision was furnished by 1 man on day shift.

An average of 283 cubic yards was washed per day. If the 10 shifts each day spent in extending the lower end of the sluice are disregarded the labor cost per cubic yard would be 17 cents. If the men extending the boxes are included the labor cost per cubic yard would be 31 cents. The timber and other supplies cost about 3 1/2 cents and supervision 3 cents, making a total operating cost of 37 1/2 cents. To this should be added the proportionate share of the cost of the main sluiceway, pipe lines, and ditches to give the total mining cost.

Stemwinder.- I. H. Gildersleeve and brothers operated the Stemwinder mine on Snowshoe Gulch, an upper branch of Cedar Creek above Superior, during the 1932 season. The gravel consisted of a high bar with a 70-foot bank left by early miners. The washing season was short, as only water from melting snow was available. The water was collected in 3 miles of ditch and 3 small reservoirs and brought to the mine through a 10-inch pipe line. Enough water was available for only two 1- to 1 1/2-hour periods of piping on each of two shifts. A no. 2 giant with a 3-inch nozzle, working under a 225-foot head, was used. A separate small stream of bywash water came down in Snowshoe Gulch. Washing began May 15 and stopped June 15. Sawing timber for boxes, cleaning up, and other work in connection with the operation required 30 additional days. The crew consisted of 6 men. When washing, 3 worked on each of two shifts. Between piping periods the crew rolled the boulders out of the pit by hand. Those too large to roll were first blasted. A 50-pound box of 40-percent-strength gelatin dynamite was used during the season.

The top three fourths of the gravel bank was washed off during one season and the lower one fourth, which carried the greater part of the gold, the next. After the bottom gravel left from the season before was mined the top was stripped for the next year's operation. The disintegration of the lower gravel, by the freezing and thawing during the winter, materially assisted in washing it. The bedrock was soft, and 2 to 3 feet was piped off in cleaning up.

The sluice boxes were 24 inches wide and 18 inches deep. The grade was 7 inches in 12 feet. The gravel contained a relatively large proportion of black sand, and some of the gold was very fine. During the 1931 season a grade of 5 inches to the box was used. Gildersleeve considered that a grade of 7 inches to the box was better than one of 5 inches with the character of gravel handled and that a higher recovery of gold was obtained in 1932 than in 1931.

The riffles consisted of 4-inch lodge-pole pine poles cut in 5 1/2-foot lengths and nailed to 1-inch crosspieces on both ends. The sections were easy to make and handle. One set of riffles for the entire length of sluice was worn out and a second set about one half worn out during the 1932 season of 30 days in which 18,000 cubic yards was washed. Quicksilver was used in cleaning up the sluice boxes, but none was put in before.

The placer operators had their own sawmill. For the 1932 season 6,000 board-feet of timber was sawed.

The labor cost of mining was 7 cents per cubic yard. Supplies cost 2 cents more, making a total of 9 cents per cubic yard. There was no supervision or general expense.

Diamond City.- The Diamond City Mining Co. in 1932 was running a bedrock cut up Confederate Gulch near Townsend to open up a body of gravel farther up the stream. The cut was about 1/2 mile long, 30 feet wide at the top, and 10 feet deep. A bench 15 feet wide on which a power dragline operated was maintained along one side of the cut.

A combination of ground-sluicing and hydraulicking was followed. Water for the giant was brought under a 125-foot head from a side creek through a 4,500-foot pipe line of which 1,680 feet was of 18-inch and the remainder of 11-inch pipe. A no. 2 giant with a 2- or 3-inch nozzle was used for cutting and one with a 3- or 4-inch nozzle for driving. The water for ground-sluicing was stored in a small reservoir a short distance up the creek from the pit. The gate was opened by hand when the reservoir was full.

The boulders were removed from the cut by means of a full-revolving, caterpillar-tread, 150-hp., gasoline-powered dragline with an orange-peel bucket 15 cubic feet in capacity; the boom was 42 feet long. One man working a few hours daily easily handled all the boulders uncovered. The bucket could lift boulders from 40 feet below the bench. It could pick up a 5-ton boulder, and by using chains a 7-ton boulder could be removed. The dragline also was used for unloading made-up boxes from a truck on the road above and setting them in the cut below. It was very efficient, and the ease of handling boulders reduced the total cost of running the pit an appreciable degree. About 10 gallons of gasoline was used for 8 hours running time.

Each box was placed as soon as room was made for it, after first cleaning up that portion of the cut. One box was placed per day while washing. The boxes were 32 inches wide and 36 inches deep. Two 2- by 16-inch boards were used for the bottom and three 1- by 12-inch boards for the sides of the sluice. The 1-inch lumber cost $28 per M and the two 16-inch boards in the bottom cost $35 per M. The riffles consisted of poles 5 to 6 inches in diameter placed lengthwise in the boxes. They were made in sections 6 to 8 feet long. A 2-inch plank was nailed across the ends of the poles of each section.

A crew consisted of 3 men on day shift and 2 on night. The maximum handled daily during a week was 400 cubic yards; the average for the season up to July 13 was about 350 cubic yards per day. Labor was paid $2.50 per day and board, making a total of $3.50. The labor cost, then, was about 5 cents per cubic yard. The cost of supplies was about 2 cents and that of supervision 3 cents, making a total operating cost of 10 cents.

Superior.- The Superior mine is in a small basin at the head of Cedar Creek near Superior. It is alongside of an old dredge pond (see fig. 11,C); it was worked by a combination of hydraulicking and ground sluicing. The dredge tailings were well compacted and stood at a steep angle. The gravel contained a large percentage of boulders and black sand. A 5,000-foot bedrock cut in which the sluice was set had been run to the pit. The grade of the sluice was 2 3/4 inches in 12 feet. The bedrock was flat, and a minimum grade was used in the sluiceway.

A no. 2 giant with a 2 1/2-inch nozzle operating under a 90-foot head was used for cutting, sweeping, and cleaning up bedrock. A relatively large quantity of bywash water was used; a part of the bywash was used for booming. Each boom lasted 15 minutes and two occurred per hour. Figure 11,C shows the pit which had been washed during the 1931 and 1932 seasons. The cut being boomed out near the head of the sluices was for prospecting purposes. Work at the mine usually began about April 1 and lasted to about August 1.

The sluice boxes were 48 inches wide and 60 inches high. The riffles were square wooden blocks of random size, 7 inches thick. The whole length of the sluice was lined with blocks. A set of blocks lasted two seasons or for about 140,000 cubic yards. Three seasons were required to ground-sluice out the cut for the sluice and put in the boxes. Although the large quantity of bywash water used compensated partly for the flat grade the maximum size of boulders that could readily be run through the boxes was about 6 inches in diameter.

Boulders were handled on a stone boat by a 60-hp. steam logging winch. The empty stone boat was run back into the pit on an overhead line. Two smaller donkey engines were used for dragging boulders to one side in the pit whence they were removed later by the large winch.

A regular crew consisted of 8 men, 6 on day and 2 on night shift. A piper and a sluice tender worked on each shift. During the day shift the other 4 men used the boom water, removed the boulders, and did other necessary work in the pit. A superintendent also was employed. Although the mine was classed as hydraulic probably more gravel was removed by ground sluicing than by piping.

An average of about 560 cubic yards was washed daily. With labor at $3.50 per day the labor cost per cubic yard for the men working in the pit would be 5 cents a cubic yard. Supervision, supplies, and general expense amounted to about 7 cents per cubic yard, making a total operating cost of 12 cents. To the operating cost should be added interest, amortization, and a prorated cost of the mile-long bedrock cut and sluice box to give a total cost of washing the gravel.

Idaho

Hockensmith.- Charles J. Goff and brother had been operating the Hockensmith placer at Leesburg since 1917[26] The gravel ranged from 1 to 20 feet thick, the average being 12 feet. Water was brought to the mine through two ditch lines, 3 and 2 miles long, respectively. The ditches were 30 inches wide and 24 inches deep. The total capacity was 500 miner's inches; the average flow was 150 inches. The head at the mine from one ditch was 75 feet and from the other 150 feet. Piping in 1932 was done with a no. 1 giant using a 1 3/4- or 2-inch nozzle. By-wash water was used for booming. The reservoir had an automatic gate which opened when the reservoir filled. Boxes were 18 inches wide and 18 inches deep and had a grade of 4 1/2 to 8 inches per 12 feet of box. Pole riffles made up in 30-inch sections were used. Grass roots were used under some of the riffles to catch fine gold. Quicksilver was not used in the boxes. Boulders were piled by hand on cleared bedrock. Beginning at the giant, bedrock was cleaned up by piping a layer of bedrock into the sluice.

Two men, the owners, worked one shift per day. The season lasted about 125 days, and about 10,000 cubic yards of gravel was washed. At $3.50 per day the labor cost would be 9 cents per cubic yard and the total operating cost about 10 cents.

Golden Rule.- The Golden Rule mine near Warren was operated by L. E. Winkler and two partners during 1932. Water for piping under a 200-foot head was conveyed to the pit through a 1,400-foot pipe line 18 to 9 inches in diameter. One giant with a 4-inch or a 3 1/2-inch nozzle was used for cutting and driving. A pit 200 feet long and 60 to 100 feet wide was washed during the season. The head of the sluice was at the lower end of the pit. The gravel and water were directed into the boxes by means of a fence built of posts and plank placed across the head of the pit washed the previous year. At the finish of the run the gravel from the upper end of the pit had to be swept 200 feet to the sluice. Although water was still available for piping the maximum distance which the gravel could economically be driven had been reached. (See fig. 11,D.) All the bywash water needed was taken from the creek. This water cascaded down over the upper face of the cut.

The boxes were 30 inches wide and were set on a grade of 9 inches in 12 feet. A grade of 6 inches to the box was used formerly, but better results were obtained with the steeper grade. Riffles in the first five boxes consisted of 2- by 6-inch timber 12 feet long, placed lengthwise in the boxes, seven to a box. Pole riffles in 12-foot sections were used in the lower end of the sluice. A set of pole riffles lasted about 10 days. Most of the boulders too large to be piped were broken by blasting and then run through the sluice. Stumps were also blasted. A total of 250 pounds of explosives was used during the 1932 season.

Most of the gold was caught in the first two boxes. Quicksilver was used in the sluice. After the bedrock was piped off at the end of the run, a string of 12-inch boxes was set in the pit and the bedrock cleaned.

---

26 Information supplied by Charles J. Goff.

A total of 63 days of one 12-hour shift each was worked by three men. This included setting the boxes for the run and cleaning up. An average of 110 cubic yards was washed daily.

The labor cost, allowing $3.50 per shift, was 9 1/2 cents per cubic yard. The explosives cost $125 or nearly 2 cents per cubic yard. Other supplies cost about 1 cent per cubic yard, making a total operating cost of 12 1/2 cents. No overhead or incidental costs were incurred. The pipe line was built some years ago of salvaged material, and an old ditch was utilized. The operating cost in this instance can be considered as the total cost, except for depreciation of the value of the mine.

Colorado

Fortune.- Fortune Tarryall Gold Placers Co. began operations in July 1932 at old workings on Tarryall Creek, above Como. The placer was last worked about 20 years ago. The deposit consisted of an old channel constituting a bench along the present course of the stream. Bedrock consisted of sandstone and was easy to clean. There were very few boulders and none that three men could not roll out of the way. Any boulders under 18 inches in diameter went through the sluice. A 40-foot face of gravel was to be worked.

Water was brought to the pit through a 2,700-foot line of 24- to 12-inch pipe under a head of 200 feet. Washing was done with a no. 2 giant using a 3- or 4-inch nozzle. Boxes were 12 feet long, 26 inches wide, and 30 inches high, with a grade of 8 inches to the box. Riffles consisted of 16-pound rails 12 feet long; two sections of three rails each were placed side by side in a box. Spacers were used every 2 feet between the rails; the spacers of each section came half-way between those of the adjoining section. The rails rested on 2- by 4-inch timber placed crosswise in the boxes. Hooks were provided on the sides of the boxes for hanging up the riffle sections when cleaning up. Coco matting, over which 1-inch wire screen cloth was placed, was used on the bottoms of the boxes under the 2- by 4-inch supports for the rails.

The crew consisted of three men on each of two shifts and a superintendent. Wages were $3.50 for 10 hours. During the first part of the work 1,100 cubic yards was washed in 2 1/2 days. The indicated labor cost would therefore be 5 cents per cubic yard. Current supplies would cost about 2 cents and supervision 1 1/2 cents, making a total operating cost of 8 1/2 cents per cubic yard.

Dodman and Weston.- A 50-foot gravel bench was being mined in July 1932 by Dodman and Weston on the Blue River below Breckenridge by a combination of hydraulicking and ground-sluicing methods. The gravel was broken down easily and contained few large boulders. The ground-sluice water poured over the bank and cut down the upper part of the face. The bottom of the gravel bank was cut with a hydraulic giant using a 3-inch nozzle. The water for the giant came from a ditch at the top of the bank and was under about a 40-foot head. The giant was used also for washing the gravel free of clay. The gravel could be washed to the sluice boxes faster than it could run through. After a run of a few hours the boxes would clog and have to be cleaned out by hand. The boxes were 18 inches wide, 12 inches high, and 32 feet long. The sluice was at right angles to the river and cut through a high rim of bedrock. The tailings discharged into the river about 5 feet above the water level. The riffles were made of 2- by 4-inch lumber, clad on top with steel. They were made in sections 4 feet long with 2- by 4-inch lumber laid alternately lengthwise and crosswise in the boxes.

As ground-sluice water did most of the work the method used could be classified properly as a ground-sluicing one. However, as a giant was used it is considered with the hydraulic mines.

The crew consisted of two men on one shift. Up to July 18 about 8,500 cubic yards had been washed. A total of 164 shifts, including 32 on preparatory work, had been worked. The labor cost was 7 cents per cubic yard; supplies cost 1 cent, making a total of 8 cents. Disregarding preparatory work, which should be charged against the gravel washed up to July 18, the labor cost would be 5 cents per cubic yard.

Nevada

Round Mountain.- The mine of the Nevada Porphyry Gold Mines, Inc.,[27] in the Round Mountain district, Nye County, had been operating since 1914. The gravel ranged from a few feet to 50 feet in depth. It contained 15 to 20 percent of angular boulders. The bedrock was uneven, and considerable gold was reclaimed from crevices.

Water was brought to the mine by pipe lines from several creeks. The line from Jetts Canyon, which was the first to be built, had a total length of 45,300 feet. The first section, 14,000 feet long, consisted of riveted pipe ranging from 30 to 15 inches in diameter. The second section, 28,000 feet long, was an inverted siphon across a valley and consisted of welded pipe 1/4 to 5/16 inch thick. The floor of the valley was 1,142 feet below the intake of the siphon, hence the maximum static head was 495 pounds to the square inch. The third section consisted of 3,300 feet of riveted pipe 15 inches in diameter. Slip joints were used on 7,500 feet of the riveted pipe where the pressure was light; bolted forged-steel couplings were used on the remainder. The difference in elevation between the inlet of the pipe and the placer pits was 650 feet. The entire line was laid in a trench 4 feet deep to prevent freezing. The cost of the line was $150,000. In 1921 a dam was built to provide a reservoir about 1 mile from the workings in which water from the Jetts Canyon and Jefferson Canyon lines was impounded. This water permitted one giant to be operated 7 hours per day during periods of low water. The head from this dam was 350 feet. An average of about 400 miner's inches was available at the mine.

The sluice used from 1921 to 1932 was 3 feet wide by 3 feet high and 5,000 feet long, with a grade of 4 inches to 12 feet. It was branched at the lower end to permit placing boxes while the mine was in operation. Riffles were 3- by 4-inch crossties placed in the boxes at intervals of 4 to 6 feet over which were laid 7 parallel lines of 25-pound rails. The rails were set in wooden blocks which were even with the top of the rails. The rails were placed with the web down in the first 3,500 feet of the sluice and inverted in the lower 1,500 feet. It was considered that the inverted rails offer less resistance to the flow of material through the sluice.

The rails in the middle of the box were worn more rapidly than those at the side. The center rails showed appreciable wear after about 150,000 cubic yards of material had passed through the sluice. They were then moved to the outside and the outer rails placed in the center.

Piping was done with 2 small or 1 large giant in 1 of 2 pits. One pit was worked while the other was cleaned up. The giants were set at an angle to the bank to permit using the full force of the stream to sweep the angular rocks to the head of the sluice. Care was exercised to bring down the soil and other fine material at such a rate that on mixing with the rocks a maximum of the coarse material could be swept to the sluice.

Boulders were removed from the pit by two methods. In the first they were loaded on flat-topped cars running on 30-inch gage tracks in the pit, then pulled up a trestle by a hoist and dumped automatically. In the second method the boulders were raised from the pit

---

27 Smith, A. M., and Vanderburg, W. O., Placer Mining in Nevada: Univ. of Nevada Bull., vol. 26, no. 8, Dec. 15, 1932, pp. 70-79.

by a derrick. The derrick mast was set at a slight angle from the vertical so that the boom swung automatically out of the pit. Boulders too large to handle were block holed with a jackhammer and blasted.

A part of the gravel deposit was cemented and required blasting before it could be piped. Five-inch holes were drilled to bedrock about 50 feet apart and 25 feet from the edge of the bank by a Star drilling rig. One blast consisted of 4 holes with an average depth of 46 feet. Each hole was sprung with 15 pounds of gelatin dynamite, then shot with 175 pounds of Judson powder. This loosened thoroughly about 12,000 cubic yards of gravel. The total cost of drilling the holes and blasting was $582, or about 4.8 cents per cubic yard of gravel.

Quicksilver was added to the sluices as required. The upper 200 feet of the sluice was cleaned up semimonthly. The rails were removed, and while a small stream of water was running down the sluice the riffles, blocking, and lining boards were taken out and scrubbed with brooms to remove any gold or amalgam adhering to them. Additional quicksilver was added and the stream of water increased. As the material moved slowly down the sluice the coarse material was thrown out with an 8-tined fork. The gold and amalgam lagged behind the lighter material and were scooped up and placed in buckets. They were ground in a clean-up pan to separate the gold from quartz and other associated gangue minerals.

At the end of the season all the rails in the sluice were removed and several hundred feet of the upper part of the sluice cleaned up in the usual manner. The concentrate in the remaining part of the sluice which contained about $60 of gold per ton, was allowed to dry, then hauled to a quartz mill and treated in the same manner as ore.

After the bedrock had been swept by the giant it was allowed to dry and cleaned by hand. Specially constructed hand tools and whiskbrooms were used for removing gold from the crevices.

During one year when 128,000 cubic yards of gravel was washed, 73 percent of the gold was recovered in the semimonthly clean-ups of the upper 200 feet of the sluice, 11 percent from bedrock, 9 percent from the final clean-up of the section of sluice below the place where the semimonthly clean-ups were made, and 7 percent from the material milled. About 15 percent of the total operating cost was for cleaning up bedrock.

About 20 men were employed during the 1932 season. The quantity of gravel washed daily depended upon the proportion of boulders and cemented gravel handled. It ranged from 32 to 110 cubic yards per hour. No recent operating costs are available.

Mines with Ruble elevators

California

Redding Creek.- Placer operations on Redding Creek near Douglas City were begun in the spring of 1932; 56,600 cubic yards of gravel was washed by the time the water supply failed. The gravel bed, which was 9 feet deep and 120 feet wide, lay in a creek bottom. The fall of the creek was so slight (one tenth inch to the foot) that enough grade could not be obtained for sluice boxes. A Ruble elevator was used for elevating the gravel and boulders and sorting out everything over 2 inches in diameter.

Water under a 300-foot head was brought to the pit through a 24-inch pipe 3,000 feet long. The Y's in the pit were of 15-inch pipe. The gravel was cut and swept to near the entrance of the Ruble by a giant with a 5- or 6-inch nozzle, then the material was washed up the Ruble by means of a second giant with a 5-inch nozzle. A third giant with a 3-inch nozzle was used intermittently to level off the tailings piles.

The Ruble was 8 feet wide by 60 feet long and elevated the oversize 25 feet. (See fig. 9.) It was lined with sheet steel. The grizzlies were of 3- by 6-inch timber set on edge; the top edge was steel-clad. They were placed crosswise on 3- by 6-inch sills laid lengthwise on the sheet-iron bottom of the chute. The plus 2-inch material was washed up through the elevator by the giant; the undersize dropped through the grizzly and ran down the bottom of the chute to four 12-foot boxes, 48 inches wide, set at right angles to the elevator. As the gravel was only 9 feet deep, the Ruble had to be moved three times during the season. With 7 men and a caterpiller tractor a week was required to move the elevator to a new location. A second elevator was planned next season to allow continuous production. Boulders were bulldozed; 2,000 pounds of 40-percent-strength gelatin dynamite was used for this purpose during the 1932 season.

An average of 540 cubic yards per day was washed during the 1932 season. The operating cost of washing the gravel was 19 cents, of which three fourths was for labor. The cost did not include ditch work (other than the ditch tender), construction costs, interest, depreciation, or amortization.

Oregon

Browning.- The MacIntosh brothers operated the Browning mine near Leland under a lease during the 1932 season. The gravel deposit was along a small creek and averaged about 12 feet deep. Preparatory to mining, a 5,000-foot pipe line from the Columbia ditch was laid. The pipe diameter ranged from 32 to 16 inches. This work took 90 days with an average crew of 10 men. Further construction consisted of the erection of a Ruble elevator. Actual washing operations were carried on for 35 days. Ten days were required when water was available to make a new set-up of the elevator. Water was brought to the mine under a head of 300 feet. Four hundred feet of head was available, but the pressure was reduced 100 feet from the top. The mine was equipped with one no. 4, two no. 3, and three no. 2 giants; 4 1/2-inch nozzles were used on the cutting giant and the one at the Ruble elevator. A 5-inch nozzle was used for sweeping. The Ruble giant and one other were used at the same time. The Ruble elevator was 6 feet wide and 36 feet long and elevated the oversize 14 feet. The grizzly bars, which were set crosswise, consisted of 2- by 4-inch lumber clad on top with 1/2- by 2 1/2-inch strap iron. The spacing was three fourths inch. The gravel could be driven over the Ruble as fast as one of the other giants could get it to the elevator. A timber dam was built part way across the pit on either side of the Ruble to guide the gravel to the elevator. The sluice boxes were 4 feet wide and 22 feet long and had a grade of 7 inches to 12 feet. Standard dredge-type Hungarian riffles 1 1/4 inches wide by 1 inch deep were used.

Figure 12,A, shows the set-up of the giants as used for the final clean-up in the pit. A total of 30,000 cubic yards was washed during the season. The average daily yardage was 667. The operating crew consisted of 6 men on two 12-hour shifts; 4 men worked on day shift and 2 on night. At $5 per 12-hour shift the operating cost per cubic yard would be 4 1/2 cents. The supplies amounted to another 1 1/2 cents, making a total operating cost of about 6 cents. The labor cost of putting in the pipe line amounted to 12 cents per cubic yard of gravel moved during the season.

Mines with Hydraulic Elevators

Oregon

Llana de Oro.- The Llana de Oro mine (fig. 12,B) was in a flat river valley near Waldo. The deposit consisted of small-size gravel and clay overlain with soil. Apparently it contained too much clay for successful dredging. The mine was worked during the 1932 season by five men under a royalty agreement. The mine was well-equipped and contained a good stock of supplies at the beginning of the season. Timber for sluice boxes was cut and sawed on the premises at a cost of $8 per M.

Water was brought to the mine in three ditches. The upper had a capacity of 520 miner's inches and was used for bringing water to a hydraulic elevator. The effective head at the elevator was 360 feet. The middle ditch had a capacity of 1,800 inches and delivered water to the mine under an effective head of 125 feet. This water was used for cutting and sweeping in the pit and for stacking the coarse tailings. The combined average flow of the two upper ditches was 700 inches. The lower ditch delivered 10,000 inches which was used in a long tailrace for carrying away the clay and sand.

No. 3 giants with 3-, 3 3/4-, or 4 1/2-inch nozzles, depending upon the quantity of water available, were used for cutting and sweeping. A no. 2 giant with a 3-inch nozzle was used for stacking the tailings.

The gravel as cut in the pit was run through 180 feet of 30-inch boxes, set on a grade of 5/16 inch to the foot, to the hydraulic elevator. The lift of the elevator was 44 feet; the diameter of the standpipe was 20 inches and that of the nozzle of the high-pressure jet 3 3/4 inches. A second elevator was used as a water lift. From the elevator the gravel ran through 700 feet of 30-inch boxes, the bottoms of which were lined with sheet steel. Riffles consisting of steel rails were used in the first 300 feet of the sluice; no riffles were used in the last 400 feet. The grade of the main sluice where the elevator discharged into it was 2 inches to the 16-foot box. As the velocity from the elevator was lost the grade was increased to 3 1/2, 4, and 5 inches to the box. An undercurrent was taken out at the end of the riffle section. The undercurrent grizzly was 36 by 36 inches in plan. The grizzly bars had a spacing of 1/4 inch at the top and 3/8 inch at the bottom. They were set crosswise to the sluice; if set lengthwise, however, they would not have clogged as easily with trash or grass that came down the sluice. Black sand had a tendency to pack in the riffles of the sluice in the pit and thus reduced their ability to catch the fine gold. To overcome this difficulty the first two or three boxes were cleaned up daily. Two thirds of the gold saved was recovered in these boxes. Because of the fine size of the gold and the relatively large percentage of clay and black sand in the gravel it was estimated that only 60 percent of the total gold was saved.

Preparatory work for the 1932 season began November 6, 1931 and consisted of repairing flume lines and cleaning out the ditches. Washing started January 1, 1932 and continued to June 20, when the high-pressure water for the elevator failed. During the washing season piping was done on an average of 22 days per month. An average of 400 cubic yards per day was handled after washing began; the daily average for the season was 300 cubic yards, and as much as 500 cubic yards was handled in a day. With labor at $3.75 per shift the labor cost for the season would be 7 cents; supplies cost 1 cent per cubic yard, making a total of 8 cents. The costs do not include interest, depreciation, amortization, or general overhead expenses. The labor cost includes 3 1/2 percent for workmen's compensation.

Plataurica.- The Plataurica mine near O'Brien was operated during the 1932 season by Nelson and Harrison, lessees. The mine was fully equipped when they took it over. Water was brought through 11 miles of ditch and 4,000 feet of pipe line 22 to 15 inches in diameter. The branch lines at the mine were 11 inches in diameter. The head was 450 feet.

The gravel was about 35 feet thick. It was fairly easy to wash and contained few boulders. As no fall was available the washed gravel was removed from the pit through a hydraulic elevator with a lift of 54 feet. The throat diameter of the elevator was 18 inches; a 3 1/2-inch nozzle was used for the high-pressure water. One giant with a 3-inch nozzle was used for cutting the gravel and driving it to the foot of the hydraulic elevator.

Usually two or three 16-foot boxes with block riffles were used in the pit in front of the elevator. The tailings at the end of the main sluice were stacked by another giant with a 3-inch nozzle which was operated about 4 hours each day. When the tailings giant was in use the cutting giant was turned off.

The hydraulic elevator discharged into 256 feet of 30-inch boxes set on a grade of 6 inches to 16 feet. When the grade was less it was found that black sand packed tightly in the riffles. The riffles in the upper part of the sluice consisted of 7-inch wooden blocks and those in the lower end of 4-inch angle irons set across the boxes. An undercurrent was used but did not recover enough gold to pay for cleaning it. The main sluice was cleaned up four times during the season. Two flasks of quicksilver were used during the season. The lay-out of the mine is shown in figure 11,C.

Work for the 1932 season began November 10, 1931 and continued until June 25. The operating crew consisted of 1 piper and 1 sluice tender on each of three shifts and 1 pitman, 1 ditch tender, and 1 foreman on day shift. The daily labor charge was $40. As much as 600 cubic yards was washed in a day, the average being about 500.

The repairs on the elevator were excessive. The casting at the bottom, which cost $157, had to be replaced every 90 days. The lessees considered the ground too rocky for a hydraulic elevator. The cost for supplies, mainly in connection with the elevator, was about $1,000 for the season. The labor cost per cubic yard was about 7 cents, the cost of supplies 1 cent, and the total operating cost 8 cents. If each yard of gravel was charged with its proportionate share of the cost of ditch and pipe lines and of equipping the mine the total cost per cubic yard probably would be several times that shown.

Idaho

Davis.- J. J. Davis started operations on Grimes Creek at Centerville in June 1932. The gravel handled was 4 to 6 feet deep and was full of boulders and stumps. The adjoining ground had been worked by a dredge; this gravel had been left because of the shallow depth and the boulders. As no fall was available for a tailing dump a hydraulic elevator with a lift of 17 feet was necessary. Water was brought to the mine through a 1 1/4-mile ditch and a 1,200-foot pipe line. The diameter of the pipe ranged from 20 inches at the top to 11 at the bottom. The pipe line crossed a canyon by means of a siphon consisting of 20-inch pipe. Piping was done with a no. 1 giant with a 1 1/2-inch nozzle. The elevator had a 3 1/2-inch nozzle and a 10-inch pipe with an 8-inch throat. It used about four fifths of the total water. Boulders were moved by hand. Those too large to roll out of the way were blasted. A derrick was being installed, which should increase the efficiency of operations. The hydraulic elevator discharged into a sluice 32 inches wide and 72 feet long. Dredge-type Hungarian riffles were used.

Two men on a shift operated the mine, and two 12-hour shifts were worked per day. In July 1932 an average of 64 cubic yards was being handled in a day, or about 16 cubic yards per man-shift, which was a good record considering the amount of boulders and stumps moved by hand. Labor cost 25 cents and supplies about 2 cents, making a total operating cost of 27 cents per cubic yard.

Mines with both Ruble and hydraulic elevators

California

Gallia.- The Gallia Placer Mining Co. operated the Gallia mine on the North Fork of the Salmon River near Sawyer Bar during the 1932 season. The gravel was 30 to 35 feet thick and contained some large boulders. Water was brought to the mine under a 265-foot head through a 2,000-foot line of 36- to 15-inch pipe. Enough grade was not available for the disposal of tailings, and the gravel as mined contained too many boulders for the successful operation of a hydraulic elevator. The gravel was cut and swept to a Ruble elevator by a giant with a 3 1/2- or 4-inch nozzle. It was then put through the elevator by another giant with a 4 1/2-inch nozzle. The Ruble was 4 feet wide and elevated the oversize 25 feet. The grizzly consisted of 90-pound rails 2 1/2 inches apart set lengthwise on 10- by 10-inch stringers. The undersize from the grizzly went through a 24-inch sluice with riffles consisting of angle iron and rails placed crosswise in the boxes.

The sluice discharged into a hydraulic elevator with a 20-inch intake. A 4-inch nozzle was used in the high-pressure jet; the material was elevated 30 feet. The elevator discharged into a second sluice.

The capacity of the plant was limited by the quantity of material that could be run over the Ruble elevator. The cutting and driving giant was used for a few hours and then shut off until the accumulated material could be handled in the Ruble. The giant at the Ruble operated continuously.

About a week with the full crew was required to move the Ruble; it had to be moved every 3 weeks, as the dump room behind it was exhausted.

Boulders too large to go up the Ruble were moved back by a derrick. Large boulders uncovered in cutting were dragged from the pit by means of a donkey engine. Water was used not only for the giants in the pit and the jet of the hydraulic elevator but also for operating the derrick and running a dynamo for operating a sawmill and an air compressor.

The working crew consisted of 2 men in the pit and 1 man on the ditch line on each of two 12-hour shifts. Although 50 to 60 cubic yards per hour could be cut and swept to the Ruble by the cutting giant the average capacity, including the time for moving the Ruble, was 200 cubic yards per day. The labor cost, assuming $4 per man-shift, would be 11 cents per cubic yard. The cost of supplies would be about 2 cents and supervision 4 cents, making the operating cost 17 cents.

Figure 12,D, shows the lay-out of the mine.

Oregon

Lewis.- Harry Lewis had been operating a placer mine, working alone, for about 10 years on Rogue River, near Galice (fig. 12,E). The mine was situated at the edge of the river, and the fall was not sufficient for a tailrace. The water was brought to the mine through a pipe line 22 and 15 inches in diameter. The Y's in the pit were of 13- and 10-inch pipe. The gravel was cut and swept to the end of the pit by a no. 3 giant with a 3-inch nozzle. There it was driven over a steel plate with 5-inch holes. The undersize went to a hydraulic elevator with a 3 1/4-inch nozzle and a 9-foot lift and thence through a 30-inch sluice box 95 feet long set on a grade of three fourths inch to the foot. The oversize was swept by a giant with a 4-inch nozzle up a Ruble elevator made of round poles which raised it 11 feet onto a rock dump. The poles were set lengthwise and close together. A third giant was used about 1 hour per day in piping the washed gravel from the end of the sluice boxes. The hydraulic elevator and one giant were operated continuously during the shift.

The riffles in the sluice were made of sections of 40-pound rails 3 feet long, placed lengthwise in the boxes. At the end of each section of rails 2- by 4-inch lumber was placed crosswise in the box. The rails had been used for 15 seasons and were about worn out.

Three months were required to set up the equipment and do the necessary repair work on the pipe lines preparatory to washing. Water was available for washing for 66 days. An extra man was hired for 6 days while washing. About 7,000 cubic yards was washed. With wages at $4, the labor cost would be 8 1/3 cents per cubic yard; supplies amounted to about 1 1/2 cents per cubic yard, making a total operating cost of 10 cents, exclusive of interest, amortization, or new equipment.

Mines where water was pumped

Oregon

Conners.- During the 1931 and 1932 seasons J. C. Conners pumped water for washing a high gravel bar on Burnt River below Bridgeport. Some of the gravel which was cemented required drilling and blasting an occasional auger hole. A few boulders occurred in the gravel. The water was pumped through 500 feet of 5-inch pipe to a height of 225 feet. A working pressure of 65 pounds per square inch was maintained at the nozzle. Two geared pumps with 4-inch suction and 3 1/2-inch discharge, built for forest-fire fighting, were used. Each pump supplied 160 gallons of water per minute under the head used. One pump was driven by a 4-cylinder and the other by a 6-cylinder automobile engine. The engines together used 20 to 25 gallons of gasoline each shift. A 2 1/2-inch fire hose with nozzles from 5/8 to 3/4 inch in diameter was used for washing the gravel.

The main sluice consisted of a 12-inch box 90 feet long, set on a grade of three fourths inch to the foot; pole and Hungarian riffles were used. The Hungarian riffles were 1 1/4 inches wide, 2 inches deep, iron-clad on top, and spaced 1 1/2 inches apart. At the edge of the pit the gravel from the sluice box went over a grizzly with 3/4-inch spacing between bars. The oversize was dumped down the side of the mountain, and the undersize went through two 12-foot boxes containing riffles consisting of holes bored in a 2-inch plank.

Large boulders were rolled over the side of the hill. Others over 4 inches in size were either cast over the side or piled on cleaned-up bedrock. About half the time the hose was used for piping and the other half for washing the gravel through the sluice. A crew of 3 men operated the mine, 1 man with the nozzle, 1 working boulders, and 1 on the pumps. The pumpman also worked in the pit when his attention was not needed at the pumps. About 6 cubic yards was handled per man-shift. At $4 per day the labor cost was about 67 cents per cubic yard. The gasoline at 20 cents per gallon would have cost 22 cents per cubic yard; other supplies would have amounted to about 4 cents, making a total operating cost of 92 cents per cubic yard.

Montana

Eldorado.- The Eldorado Mining Co. during the 1931 season pumped water for mining the Eldorado bar on the Missouri River below York. A 250-hp. motor and a centrifugal pump were used for lifting 4,000 gallons of water per minute a vertical distance of 120 feet to the top of the bar. The suction pipe was 14 inches and the delivery pipe 12 inches in diameter. A 50-hp. motor was used for driving a booster pump at the top to supply a 40-pound pressure for cutting the gravel. A 6-inch fire hose with a 1 1/4- or 1 1/2-inch nozzle was used for cutting and the same hose with a 1 1/2- or 2-inch nozzle for sweeping. About four fifths of the water pumped was used as a bywash. The gravel was cut and washed easily. Boulders were removed by hand.

Sluice boxes were 400 feet long, 36 inches high, and 36 inches wide with a false side which reduced the width to 30 inches; the grade was 3 inches in 12 feet. Riffles in the first 200 feet of the sluice were 1/2- by 3-inch strap iron 16 feet long placed lengthwise in the boxes. The iron was fastened to 2- by 4-inch wooden cross strips spaced 4 feet apart. The space between the strips was 1 1/2 inches. The bottom of the lower 200 feet of box was lined with 4-inch wooden blocks 4 by 4 or 2 by 4 inches in cross section. The wooden blocks were used mainly to take the wear of the gravel.

An undercurrent was used at the end of the main sluice. The grizzly spacing was 1/4 to 3/16 inch. The undercurrent box was 12 inches wide and set at a grade of 1 inch to the foot. Hungarian riffles 1 inch wide by 1 inch deep, with a 1-inch spacing, were used over a blanket. Only a very small proportion of the gold was caught in the undercurrent, and panning indicated that no appreciable quantity went over with the tailing of either sluice or undercurrent. About 500 cubic yards was washed per day. The crew consisted of 4 men on each of 3 shifts and a superintendent. Wages were $3.50 for 8 hours. Power cost 0.9 cent per kilowatt-hour plus a flat rate of $1 per month per horsepower of connected load. The motors were run at an overload, making the power charge about $75 per day. Labor cost 9 cents, power 15 cents, other supplies and incidentals 2 cents, and supervision about 3 cents, making a total operating cost of about 29 cents per cubic yard.

Idaho

Old Garden.- In July 1932 John D. Smith of Centerville was operating a pit on the Old Garden placer on Grimes Creek as an experiment. Water was being pumped by a 75-hp. motor and a 3-stage, 8-inch pump to operate two giants under a 40-pound head. A second pump installation of the same kind was used for a hydraulic elevator.

The deposit consisted of stream gravel and sand at the side of the river. The gravel was easy to cut, and the small proportion of boulders made sweeping easy. The water was pumped from the river to the pit through 600 feet of 12-inch pipe to a Y consisting of two 150-foot lengths of 7-inch pipe. Water for the elevator was pumped through 700 feet of 12-inch pipe. A no 1 giant with a 2 1/2-inch nozzle was used on one branch and a no. 2 giant with a 3-inch nozzle on the other. The first giant was set 10 feet lower than the other. One giant was used for cutting and driving and one for driving only. The capacity of the elevator did not suffice, however, to handle all material that could be washed to it, and the stream of one giant had to be wasted part of the time. The elevator had a 19 1/2-foot lift and a 12-inch pipe with an 11-inch throat. The pressure nozzle was 3 1/2 inches in diameter.

The sluice boxes were 30 inches wide, and eight 12-foot boxes were used. The grade was 8 inches in 12 feet. The first box was lined with rails to take the wear from the discharge of the elevator and to offer a minimum resistance to the material in the boxes at the start. The riffles in the remaining boxes were 2-inch angle irons spaced 2 inches crosswise in the boxes. A derrick was set up near the elevator to remove stumps and the few boulders encountered from the pit.

The crew consisted of 2 men on each of 3 shifts. About 160 cubic yards was washed per 24 hours. Wages were $4 per shift. Labor therefore cost about 15 cents per cubic yard. Power cost 1 1/4 cents per kilowatt-hour. The cost per day was $33.60, or 21 cents per cubic yard. Other supplies and incidentals amounted to about 2 cents per cubic yard, making a total operating cost of 38 cents.

One half of the power was used for the elevator. The power cost, however, could be halved and the capacity of the two giants increased at least 50 percent if tailing room was available and the elevator was not necessary.

British Columbia

Boe.- B. Boe has been conducting a successful placer operation since 1924 on Cedar Creek in the Quesnel mining division in British Columbia. In this mine the water for hy-

Sluice Boxes

Construction

Sluice boxes are rectangular in section and are nearly always built of lumber; steel or iron sluices, however, were used at a few washing plants operated in 1932.

The construction of a wooden sluice box depends somewhat upon the size and service expected of the box; a number of types, however, may be used satisfactorily. Common types of construction for large and small boxes are illustrated in figure 13.

The important features in design are sturdiness and simplicity of construction. Large flumes may have to withstand severe battering and vibration from the passage of heavy boulders, hence they must be strongly constructed and well braced. In small flumes this feature is less important, but the use of lighter lumber increases the difficulties of maintenance and prevention of leaks.

The bottom of a narrow sluice should be a single plank if lumber of the desired width is obtainable; for wider boxes two or more bottom planks must be used. The bottom joints may be made tight by the use of soft-pine splines, by batten strips nailed on the outside, or by caulking with oakum or other material. Bowie[30] recommends half-seasoned lumber as most suitable for the construction of boxes. Where local timber is used it is common practice to cut the plank during the dry season or before snow is off the ground. It is not customary to use surfaced lumber for boxes, although a smooth bottom facilitates the clean-up. The lumber should be clear and of uniform size.

For any but small, temporary installations the sides of sluice boxes should be lined with a wearing surface of rough lumber or sheet iron. Otherwise the entire box must be replaced when the sides are worn out. Board lining is easier to place and replace than sheet iron. In early Californian practice some of the side linings were made of wide, thin blocks nailed on so as to present the endgrain to the wear. Worn iron or steel riffles are used for side lining at some places. Usually only the lower half or third of the side of the box needs this protection, and a single 2-inch board may serve not only for lining but as a cleat to hold down the riffles. False bottoms of planed or rough boards may be used to save wear on the box proper.

Each box should rest on three or four sills, equally spaced. The sills and upright members at the ends of the box serve as battens to prevent leakage at joints. The practice of tapering the box enough to permit a telescope joint is very convenient in small sluices, especially if the boxes must be moved occasionally. Small, three-board boxes may be braced with ties across the top, although this hampers shoveling and clean-up operations. Larger boxes should be braced externally from the ends of the sills, as illustrated in figure 13, A and B. Sills should be weighted with rocks to check any tendency of the sluice to rise. If the sluice is placed in a bedrock or other cut, water under it or at the sides has a strong lifting effect. Moreover, the vibration caused by boulders rolling through the sluice permits fine gravel to be washed under the sills placed on the ground.

The following table shows the price of lumber suitable for building sluices at various places in the summer of 1932:

---

30 Bowie, A. J., Hydraulic Mining in California: Van Nostrand Co., New York, 3d ed., 1889, p. 220.

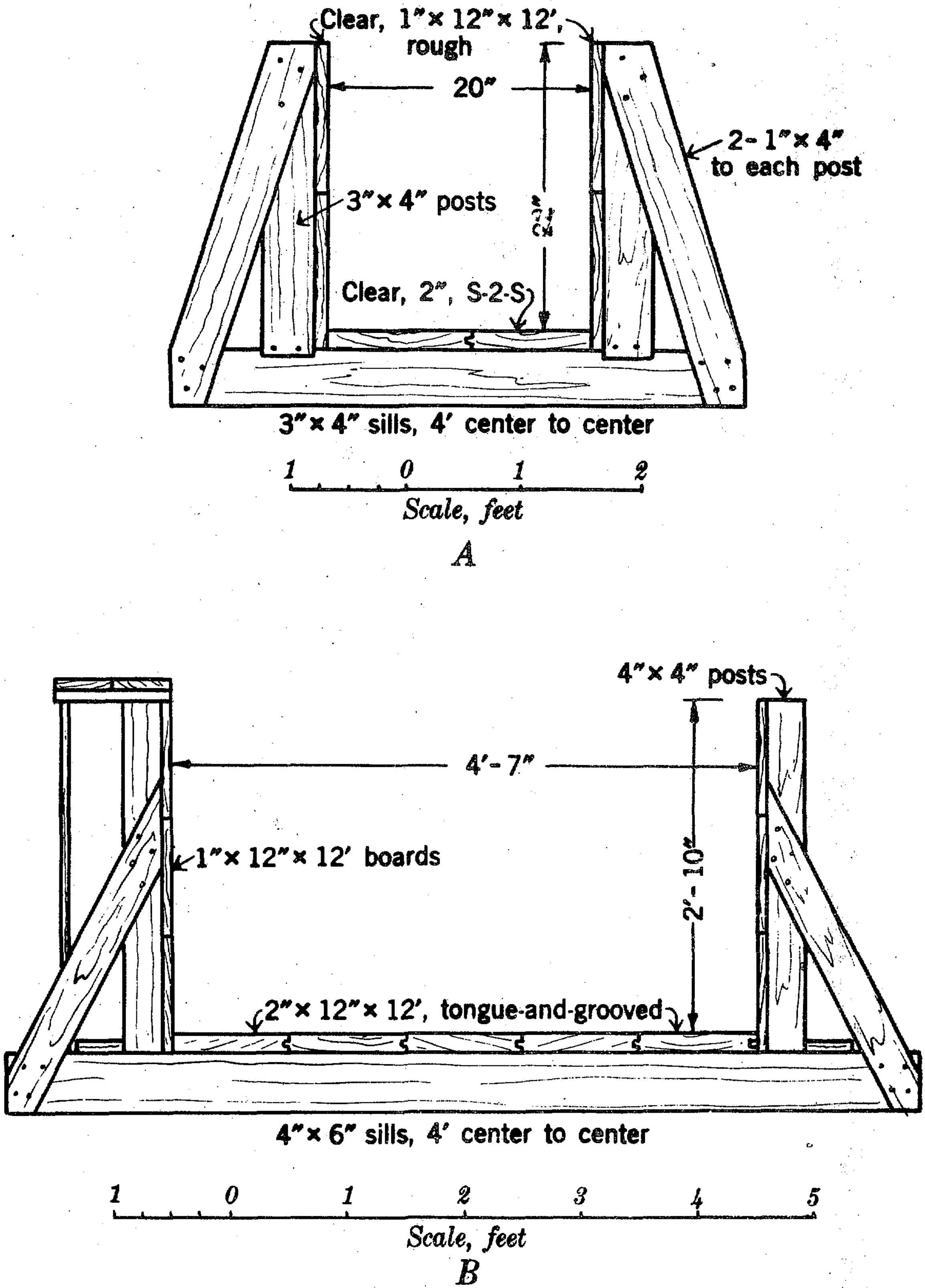


Figure 13.—Sluice-box construction: *A*, Twenty-inch box at Henderson mine, Gold Creek, Mont.; *B*, five-foot sluice box.

	Price per 1,000 board-feet
Oregon House, Calif.	$25.00
Sawyers Bar, Calif.	30.00
Waldo, Oreg. (cutting and sawing only)	8.00
Wenatchee, Wash.	20.00
Emigrant, Mont.	20.00
Townsend, Mont.:	
1- by 12-foot	28.00
2- by 16-foot	35.00
Therma, N. Mex.	22.00

As mentioned, the side lining plank may serve as a cleat under which the riffle sections can be wedged to the bottom of the sluice. Otherwise some other provision must be made as the riffles must be held securely. In small boxes it is customary to lay long, narrow boards on edge on top of the riffles and against the sides of the sluice. These boards are wedged down tightly under cleats nailed permanently to the sides of the box. The practice of nailing riffles to the bottom of the box, or using any device that requires driving nails in the bottom or sides, should be avoided as it results in leaks and eventually damages both sluice and riffles. Wooden blocks are the most difficult to secure in place but can be held by the method described in the following section. Rock pavement depends on its weight, on being packed tightly, and sometimes on the slight downstream tilt of the individual stones to resist the shifting action of the current.

Maintenance

Maintenance work on sluice boxes consists chiefly in alining and bringing to grade any boxes that have moved out of position, replacing linings, and plugging leaks. Attention to this work at clean-up time will be repaid by greater capacity and freedom from break-downs when the water again is turned into the sluice.

Size

As previously shown, sluice boxes seldom are built less than 10 inches wide for strictly mining purposes. Eight-inch boxes, however, may be used in sampling or cleaning up. The quantity of water, with its accompanying load of gravel, that will run through a sluice of a given size depends upon a number of factors. The practice at the majority of about 75 hydraulic and ground-sluice mines visited in the preparation of this paper indicates that the carrying capacities of sluices of various widths are within the following limits:

Width of box, inches	Miner's inches of water	
	From	To
12	25	100
18	100	300
24	200	600
36	500	1,300
48 to 60	1,000	3,000

These limits probably represent good practice.

More trouble is experienced from clogging of boxes that are too wide, because the depth and velocity of water are insufficient, than from failure of boxes to carry their load because they are too narrow.

The current velocities required to transport different sizes of material have been studied; works of various authorities are cited by Gilbert.[31] The following table is based chiefly on Dubuat's figures for competent velocity; the figures are adjusted to approximate mean velocity instead of bed velocity. The last three figures are taken from Van Wagenen.[32]

Size of material moved	Mean velocity, approximate feet per second
Sand:	
Fine	0.5
Coarse	1.0
Gravel:	
Fine	1.5
1-inch	2.5
Egg-size	4.0
Boulders:	
3- and 4-inch	5.3
6- to 8-inch	6.7
12- to 18-inch	10.0

Well-rounded pebbles are easier to move than angular ones, and rock of low specific gravity is appreciably easier to wash than heavy, dense rock such as greenstone or basalt.

Gold has a better opportunity to settle and be caught in riffles in a wide, shallow stream than in a deeper and narrower stream of the same volume; the wider sluice, however, usually must be set on a steeper grade.

Small- or medium-size boxes generally are roughly square in cross-section; large boxes usually are one half to two thirds as deep as they are wide. The water in a sluice should always be more than deep enough to cover the largest boulder that may be sent through. In practice, the depth of the stream in the main sluice at hydraulic mines usually is a fifth to a half the width of the box so as to prevent spills if the box is temporarily plugged by boulders or sand. Where screened gravel is being washed, as in undercurrents or on dredges, wide and shallow streams are necessary for the recovery of fine gold. In "booming" operations the boxes usually are run full in order to handle the relatively large volumes of water that flow for short periods only, and the sluices commonly are about as deep as they are wide. It would be desirable but impracticable to decrease the depth of water by using wider sluices, as flows of 5,000 to 10,000 miner's inches are not unusual when the gate of the reservoir suddenly is opened wide.

Grade

Usually the grade of the sluice depends upon the slope and contour of the bedrock. If the gradient of bedrock, however, is to low to permit sufficient fall for the sluice, cuts or tunnels may be run in the bedrock to overcome this difficulty. Very short sluices of only

31 Gilbert, G. K., The Transportation of Debris by Running Water: U.S. Geol. Survey, Prof. Paper 86, 1914, p. 216.

32 Van Wagenen, T. F., Manual of Hydraulic Mining: Van Nostrand Co., New York, 1880, p. 88.

1 or 2 boxes sometimes are set nearly flat where there is a drop at the end of the box, the gravel being forced through the sluice by the initial velocity and the head of water in the pit.

The opinion of most operators is that about 6 inches in 12 feet is the best grade for average conditions. As shown, grades as flat as 3 inches in 16 feet can be used but only at great loss of capacity. At the Depot Hill mine, where a grade of 3 inches in 14 feet is used, all rocks over 5 or 6 inches in diameter must be left in the pit. Because of the greater friction and the consequent lowering of velocity, steeper grades are needed for small sluices than for large ones; some operators favor grades of 12 inches to a 12-foot box. For maximum gold-saving efficiency, as well as for economy in dump room, grades should be as flat as possible without lowering the velocity to such an extent that the riffles pack with sand. Any increase in slope from that adjustment will increase the capacity of the sluice, increase the wear on the sluice, and decrease the efficiency of the riffles, resulting in gold losses if carried to extremes or if the gold is very fine. If water is scarce, gold recovery may well be sacrificed to capacity. Bowie[33] states that grades of 10 to 24 inches were used in some Forest Hill Divide (Calif.) mines for this reason. Increasing the proportion of water to solids decreases the tendency of riffles to pack with sand.

Sluice capacity increases with grade but more rapidly; that is, doubling the grade of sluice boxes will more than double the quantity of gravel that can be put through the boxes by a given flow of water. The absolute increase cannot be predicted closely as coarseness of gravel, velocity, and shape of the box appear to have some bearing on the relation of capacity to slope. For instance, Bowie[34] cites a mine at which changing the grade from 3 to 3 1/2 inches in 16 feet increased the quantity of gravel sluiced through the same boxes with the same flow of water by about one third.

The established grade should not be decreased anywhere along a sluice, otherwise gravel may accumulate where the current loses velocity. If the water and gravel, however, enter the first box with considerable speed, say, from the discharge of a hydraulic elevator, the first boxes may be placed on less than the regular grade. Bends or curves are undesirable as they complicate construction and induce clogging and running over. When a curve is unavoidable it should be as gradual as possible, the outside of the sluice should be elevated a fraction of an inch, and the grade should be increased perhaps an inch per box at and immediately below the curve. Similar rules apply to turn-outs or branches, and drops of 3 or 4 inches should be provided at junctions to check the deposition of gravel at these points. Such drops occasionally are inserted in straight sluices if the grade is available, particularly if the gravel is a difficult one to wash or if heavy sand tends to settle to the bottom. A drop of even a few inches from one box to the next has a disintegrating effect and mixes the material passing through the sluice, thus assisting gold recovery. At one place where drops were provided at intervals between different types of riffles, 25 percent of the gold recovered in the sluice was found at the drops.[35]

Riffles

Theory of gold-saving by riffles

The function of riffles is to hold back the gold particles that have settled to the bottom of a flowing stream of water and gravel. Any "dead" space in the bottom of a sluice

33 Bowie, A. J., A Practical Treatise on Hydraulic Mining in California: Van Nostrand Co., New York, 3d ed., 1889, p. 220.

34 Bowie, A. J., work cited, p. 266.

35 Theller, J. H., Hydraulicking on the Klamath River: Min. and Sci. Press, vol. 108, Mar. 28, 1914, pp. 523-526.

box, where there is no current, fills quickly with sand and thereupon loses most of its value as a gold saver, unless the sand remains loose enough to permit gold to settle into it; therefore, the shape of riffles is important, regardless of the fact that under some conditions, as with coarse gold and free-washing gravel, all forms of riffles are almost equally efficient. The riffle should be shaped so as to agitate the passing current and produce a moderately strong eddy or "boil" in the space behind or below it, thus preventing sand from settling there and at the same time holding the gold from sliding farther down the sluice. In other words, riffles, for maximum efficiency, should provide a rough bottom that will disturb the even flow of sand and gravel, will retain the gold, and will not become packed with sand. Where grade is lacking the riffles must be relatively smooth, so as not to retard the current unduly; under these conditions the sluice must be long enough to compensate for the loss in gold-saving efficiency of the individual riffles.

Natural stream beds act as gold-saving sluices, not because they are particularly efficient as such but because most gold is "hard to lose" and the streams are long.

Types of riffles

Riffles, of course, should be designed so as to save the gold under the existing conditions. They should also be cheap, durable, and easy to place and remove. Not all these qualities are found in any one type.

Sluice-box riffles may be classified roughly as transverse, longitudinal, block, blanket, and miscellaneous roughly surfaced ones, or, according to material, as wood block, pole, stone, cast iron, rail, angle iron, fabric, and miscellaneous. Usually more than one type of riffle is used, although in California very long sluices have been paved entirely with wood-block riffles, and on dredges the type illustrated in figure 14, A, is used almost exclusively.

Of about 80 hydraulic, ground-sluice, and mechanically worked placer mines visited in 1932 by the authors, approximately 25 percent used riffles of the transverse variety, loosely termed "Hungarian", consisting generally of wooden crossbars fixed in a frame and sometimes capped with iron straps. About 20 percent used the longitudinal pole type, 15 percent wooden blocks, and 15 percent rails, the last being placed crosswise or lengthwise. Angle-iron riffles, wire-mesh screen or expanded metal on carpet, blankets, or burlap, rock paving, and cast-iron sections together made up the remaining 25 percent. The only general rule observed was that the size of the riffles was roughly proportional to the size of the material to be handled and that for fine material, particularly the screened gravel washed in most of the mechanically operated plants, the dredge-type riffle found most favor.

For a small or medium-size sluice (if lumber is costly and a plentiful supply of small timber, such as the lodge-pole pine so common in many Western States, is available) peeled pole riffles (fig. 14 B and C) are perhaps the most economical and satisfactory of the various types. Their construction is evident from the drawing. Those of transverse variety may have a somewhat higher gold-saving efficiency, but undoubtedly they retard the current more and wear out faster. Poles 2 to 6 inches in diameter may be used, spaced 1 or 2 inches apart. Such riffles are cheap but wear out rapidly. The sections should be a third or half the box length for convenience and 1 or 2 inches narrower than the sluice. At the Golden Rule mine 6-inch pole riffles had to be replaced every 10 days or after each 1,200 cubic yards had been sluiced. The sluice was 30 inches wide and had a grade of 8 inches in 12 feet. At other mines poles last several times as long.

If sawed lumber can be obtained cheaply, riffles similar to the one described may be made of 1- by 2-, 2- by 2-, or 2- by 4-inch material, as shown in figure 14, D and E. The top surfaces of the riffles may be plated with strap iron (fig. 14, F and G). Transverse

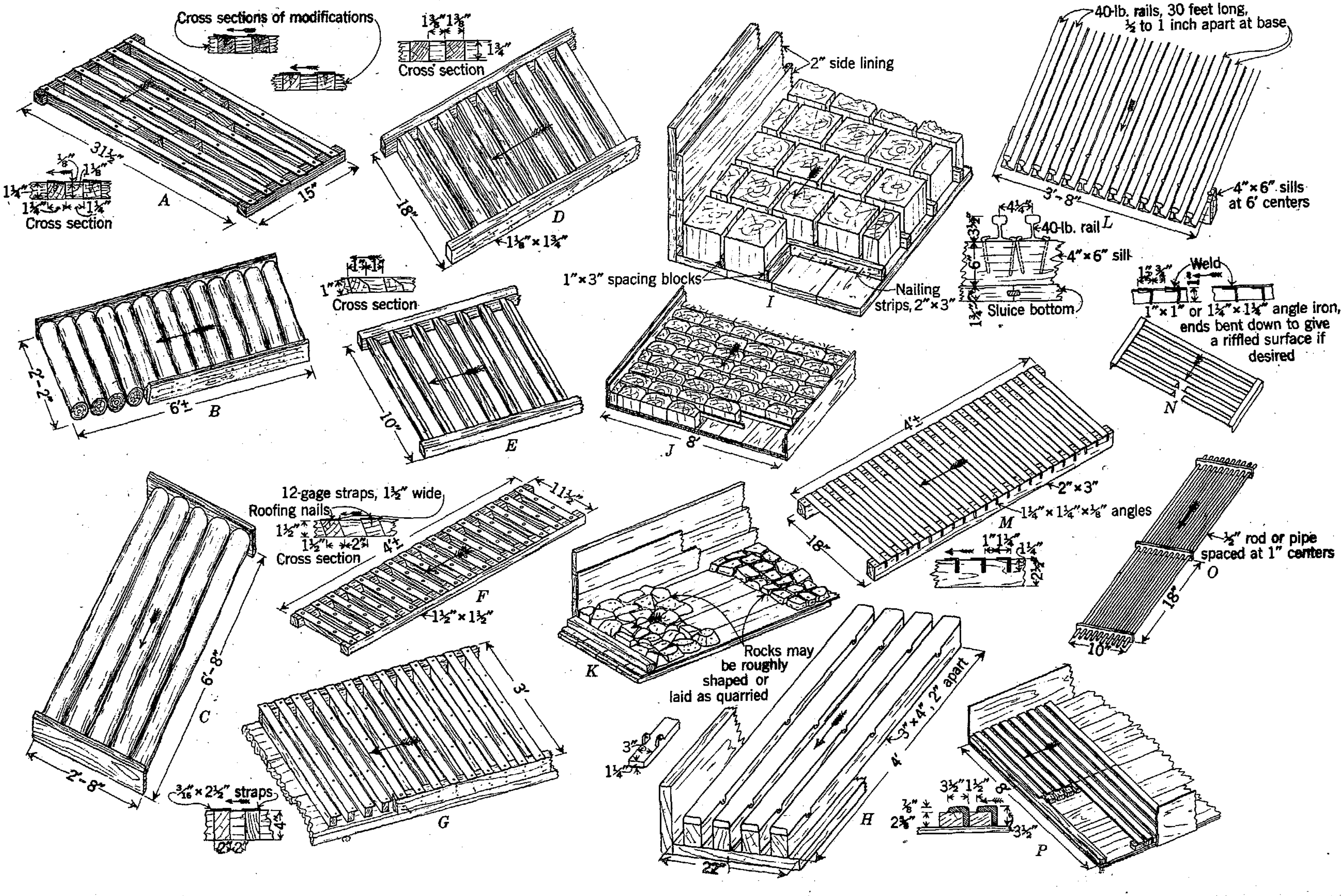


Figure 14.- Types of riffles: A, Transverse wooden, steel-capped riffles used on dredges; B, transverse pole riffles; C, longitudinal pole riffles; D, transverse wooden riffles, square section; E, transverse wooden riffles, beveled section; F, transverse wooden riffle, steel-capped, inclined section; G, transverse wooden riffles, steel-clad, with overhang; H, longitudinal wooden riffles capped with cast-iron plates; I, wooden-block riffles for large sluices; J, wooden-block riffles for undercurrents; K, stone riffles; L, longitudinal rail riffles on wooden sills; M, transverse angle-iron riffles; N, transverse angle-iron riffles with top tilted upward; O, longitudinal riffles made of iron pipe; P, transverse cast-iron riffles used in undercurrents.

riffles of this type may be slanted downstream, as shown in figure 14,F, and the top surfaces may be beveled to increase the "boiling" action, as with the dredge riffles. The effectiveness of this practice is not known, and the authors know of no conclusive tests having been made. Longitudinal riffles of 2- by 4-, 3- by 4-, or 2- by 6-inch material are used at some places. A longitudinal wooden riffle capped with cast iron is shown in figure 14,H.

Sluices in the Rock Creek sapphire mines were 12 inches wide and set on a grade not to exceed one half inch to the foot. A relatively flat grade is necessary to save the sapphires. Riffles were 2 by 4 inches in size set across the sluice 4 inches apart; they were tilted downward. The sluice was cleaned up each day. The sapphires were separated from the sands in a jig. They were then put through a set of seven screens, and other heavy minerals were picked out by hand. The black sand and other fine heavy minerals were drawn through the screen in the jig; the sapphires were taken off on top of the screen.

Wooden-block riffles (fig. 14, I and J) are held by Bowie[36] to be unexcelled in regions where the material is available cheap. The blocks are 4 to 12 inches thick and of corresponding diameters or widths. They may be round, partly squared, or cut from square timber. One- or two-inch wooden strips separate the rows of blocks, and they are held securely in place by nails driven in both directions. Wooden-block riffles are perhaps the hardest of all types to set because of their tendency to float away. They must be nailed to the spacing strips, as stated, and wedged securely at the sides. The spacing strips are held down at either end by the side lining of the sluice. Wooden-block riffles are durable, can be worn down to half their original thickness or less, and if made of long-grained wood (such as pitch pine, which "brooms" instead of wearing smooth) may catch some gold in the endgrain. When discarded, they are commonly burned and the ashes panned to recover any gold so caught. The life of 10- or 12-inch wooden-block riffles may be a few months to several seasons and, according to Bowie, ranges from 100,000 to 200,000 miner's inches of water; that is, with a flow of 1,000 inches one would last 100 to 200 days. The grade of the sluice apparently has much to do with the life of block riffles. At the Superior mine where the sluice was 48 inches wide and had a grade of 2 3/4 inches in 12 feet a set of blocks lasted two seasons, during which time 140,000 cubic yards was sluiced. At the Salmon River mine the grade was 7 inches in 12 feet and the width of the boxes 30 inches. Here block riffles lasted 60 to 70 days, during which time about 18,000 cubic yards was washed. On account of differences in the wearing rates only one variety of wood should be used in a section of a sluice. Douglas fir wears longer than other native western conifers.

Stone riffles are durable and fair gold catchers. Stones ranging from the size of cobbles to 8 or 10 inches in diameter are packed closely on the bottom of the sluice. (See fig. 14,K.) They may be held at intervals of a few feet by transverse wooden strips. In some instances the stones are roughly hand-shaped and set similarly to street paving. Stone riffles are difficult to set and generally are not used in portions of a sluice that are cleaned up frequently. Their main advantage is their long life. Because of their roughness, stone riffles require a steeper slope than wood blocks, a feature that sometimes would prohibit their use.

Where large quantities of gravel are put through sluices, iron or steel riffles generally are preferred. Their superior wearing quality as compared with that of wood permits longer runs without stopping to replace the riffles. Their durability may more than compensate for their higher cost.

Steel rails and angle iron are common riffle materials used in various ways. Old rails or angle iron can often be obtained cheaply in mining districts or near railroads. Various

---

36 Bowie, A. J., A Practical Treatise on Hydraulic Mining in California: Van Nostrand Co., New York, 3d ed., 1889, p. 225.

other steel products such as pipe and channels have been utilized for riffles. Cast iron is also used and has the advantage of a lower first cost than steel rail or angle iron.

Iron or steel riffles should not be used in units too long to be handled readily. Rope blocks on movable tripods have found favor at some places for lifting heavy riffle sections.

When used as transverse riffles lengths of steel rail usually are set upright, the flanges almost touching or not more than 1 or 2 inches apart. Where grade is lacking and gold saving is not particularly difficult, longitudinal rail riffles make excellent paving for a sluice as they provide a smooth-sliding bottom for the gravel and boulders. The rails ordinarily are bolted together by tierods passing through wood, pipe, or cast-iron spacing blocks, forming riffle sections the width of the sluice and any convenient length. At the La Grange mine in Trinity County, Calif., 40-pound rails costing about $125 per ton proved more satisfactory than wood riffles.[37] When 16- by 16- by 13-inch wood blocks were used the riffles tended to "sand up." Moreover, the blocks had to be replaced every 2 or 3 weeks. Lengthwise rails 8 inches apart lasted 2 months and rails 5 inches apart, 4 months. Strangely enough, transverse rails 5 inches apart lasted 6 months. The rails were spaced by cast-iron lugs and set right side up on timber sills. When the head of the rail was worn off the remainder was used for side lining. This sluice was handling a flow of about 4,000 inches of water and 1,000 cubic yards of material per hour, boulders as large as 7 tons being washed through. The eddies behind the rails were believed to be the cause of the improved recovery as compared with that using block riffles. The lower part of the branching sluice line was cleaned up every other season only.

The combination of steel rails and wooden sills used at the La Grange mine appears to make an excellent gold saver, and modifications have been used at many large mines. Figure 14,L, illustrates a combination of longitudinal rails and transverse timber sills.

At the Round Mountain mine 25-pound rails were placed longitudinally in a 36-inch sluice with a grade of 4 inches in 12 feet. After about 150,000 cubic yards had been run through the sluice the center rails showed considerable wear and were removed to the outside. At the Lewis mine on Rogue River a set of riffles made of 40-pound rails lasted 15 seasons. The sluice was 30 inches wide and had a grade of 8 inches in 12 feet. About 7,000 cubic yards was washed yearly. Only material under 5 inches in diameter was run through the sluices.

Angle iron is commonly used for making riffles, as illustrated in figure 14, M and N. Many methods of assembling the lengths of angle iron into riffle sections are in use, and no one method can be said to excel. The irons may be set with flat upper surfaces or inclined slightly to increase the riffling action. Usually the gap between the riffle bars is 1/2 to 1 inch. The effectiveness of this type of riffle is believed by some operators to depend largely on the vibration of the riffles under the impact of boulders which keeps the sand trapped under the angles in a loose condition favorable to gold saving.

Figure 14,O, illustrates an unusual all-metal riffle used at a Colorado drift mine, which was said to be giving satisfaction and appears to be simple to construct and convenient to use. The riffling effect could be increased, with some loss of velocity, by spacing the transverse bars closer.

Cast-iron riffles of all shapes and sizes have been used. If available at low cost they are very economical, as they wear slowly, can be quickly and securely placed, and are efficient gold savers if designed so as not to pack with sand. In an undercurrent at the Indian Hill mine, Calif., cast-iron riffles were in use that were 4 feet long, shaped like angle irons, and had equal 3 1/2-inch legs 7/8 inch thick. (See fig. 14,P.)

---

37 MacDonald, O. F., The Weaverville-Trinity Center Gold Gravels, Trinity County, Calif.: U.S. Geol. Survey Bull. 430, 1910, pp. 48-58.

One property in California was reported to be using old car wheels for sluice paving. They were laid close together, flange side up, in a box just wide enough to hold one row of wheels. The riffling action caused by the hubs, webbing, and spaces between adjacent wheels and under the flanges was said to have resulted in a satisfactory gold recovery. A gravel-washing plant in Arizona was provided with riffles made of standard 2-inch pipe and 2 1/2-inch angle iron welded into riffle sections resembling pole riffles. This riffle should be fast-running and as efficient as any longitudinal type of riffle, relatively light, and easy to handle. It would not be durable enough for very heavy gravel and would be relatively expensive unless salvaged material and welding equipment were available.

For shallow sluice streams carrying only fine material various gold-saving materials are used, including brussels carpet, coco matting, corduroy, and burlap. These may be held down by cleats or by wire screen. Fabrics often are used in combination with riffles to catch fine gold and hinder its being washed out of the riffles by eddies. A corduroy woven specially for a riffle surface is used by some large Canadian lode-gold mines to catch their "coarse" gold before flotation or cyanidation. As such gold would be considered fine by most placer miners it seems probable that such a fabric would be useful for treating finely screened placer sands. The corduroy in question has piles about 1/4 inch wide and 1/8 inch high, spaced about 1/4 inch apart. The piles are beveled slightly on one side. The cost in Canada is about $1.00 per square yard.

Heavy wire screen such as that used for screening gravel makes an excellent riffle for fine or medium-size gravel in fairly shallow sluice streams, and generally it is used with burlap or other fabric underneath.

Expanded metal lathing and woven metal matting are common types of riffles for fine material and are used with carpet or burlap. If the thin strands of metal slant considerably in one direction, the material should be placed with this direction downstream. Eddies in back of the strands will then form gold catchers, whereas if the recesses face upstream they will at once fill with a tight bed of sand and lose their effectiveness.

A matting woven of twigs or cane is recommended by Idriess[38] as an efficient gold catcher for a small, portable sluice box for shoveling-in operations or prospecting. Turf, as used at the Hockensmith placer in Idaho, is said to make an efficient trap for fine gold.

Solid-rubber riffles were noted at one washing plant. Sponge-rubber riffle material is on the market, but it was not observed in use and nothing is known by the authors of its merits or cost.

Another form of riffle often used as an auxiliary to other types is a mercury trap, consisting of a board the full width of the sluice with 1- or 1 1/2-inch auger holes in which mercury is placed. Instead of round holes, transverse grooves or half-moon-shaped depressions, 2 to 4 inches wide and with the rounded, deep side downstream, may be cut in a wide board and partly filled with mercury. These riffles have no apparent advantage over the ordinary transverse-bar type and are suitable only for fine gravel, as large pebbles would splash the mercury out of the traps.

Many ingenious and odd kinds of riffles are encountered in the field, some of which have been patented. It is very unlikely, however, that the advantage of any unusual or freakish design of riffle is sufficient to offset the cost of royalties on patented inventions.

Undercurrents

An undercurrent, as defined before, is a device for sluicing separately a finer part of the gravel passing through the main sluice. The fine material and a regulated quantity of

---

38 Idriess, I. L., Prospecting for Gold: Angus & Robertson, Sydney, 3d ed., 1932, pp. 64-65.

water pass through a stationary grizzly in the bottom and usually near the end of the sluice to one or more wide sluice boxes, commonly called tables, paved with suitable riffles. If the main sluice is in sections, with drops between, the water and sand may be returned from the undercurrent tables to the main stream, and several undercurrents may be installed at convenient points along a sluice.

The screen or grizzly in the main sluice may present the most difficult problem in building a satisfactory undercurrent. The screen should divert all the undersize yet not take so much water that it causes plugging of the main sluice below the undercurrent. The proper size of opening can be determined only by experiment. A screened or barred opening, the full width of the main sluice and a few inches to a foot or more long, will usually draw off as much water as can be spared. New water may be added to either the undercurrent or main sluice if the screen opening does not take out the right quantity for successful operation. Usually minus 1/4- to 1/2-inch material is desired for the undercurrent, and either punched-plate screen or iron-bar grizzlies may be used to make the separation. Grizzlies should be made of tapered bars or screens punched with tapered holes with the largest openings downward, otherwise they will plug and render the undercurrent ineffective.

Because undercurrents need a wide, shallow stream, grades of 12 to 18 inches per 12 feet must be used, depending largely on the type of riffle. Cobblestone, block, transverse or longitudinal wooden strips, rails, screens, or fabrics may be used for riffles. Often several types of riffles are used on successive parts of one undercurrent. Undercurrents may be a few to 25 or 30 feet wide and 10 to 50 feet long.

Most of the gold recovered by undercurrents is so fine that it does not settle in the relatively swift, deep current of the main sluice, but part consists of gold that is freed from its matrix of clay by dropping through the grizzly and rolling over the undercurrent riffles. All coarse gold is saved in the first few boxes of the main sluice unless conditions are radically wrong. Unless the undercurrent is installed at the end of the sluice, or at least below where gold is recovered, not all the saving in the undercurrent should be credited to its installation. In the early days when hydraulicking was at its height undercurrents were much favored, sometimes 5,000 to 10,000 square feet of undercurrent being used along a single sluice line. The gold saved in them occasionally exceeded 10 percent of the total clean-up but more often was less than 5 percent. As this recovery usually was effected by 5 or 10 large tables and as considerable would have been saved by the main sluice without the undercurrents, the economy resulting from their use was perhaps doubtful. Bowie[39] presents details of the use of undercurrents in early Californian practice and indicates that their particular field lay in the treatment of cement gravels. Of the several undercurrents observed by the authors in use in 1932 it is doubtful, as shown before, if many were justifying their installation. Table 13 gives data on undercurrents in use at mines operating in 1932.

Operation of Sluice Boxes

Under favorable conditions a properly designed and constructed sluice box requires little attention other than periodic clean-ups and minor repairs which are made at the same time. Unfortunately, such a combination rarely occurs, and an appreciable part of the miner's operating expense is chargeable to work along the sluice lines.

39 Bowie, A. J., A Practical Treatise on Hydraulic Mining in California: Van Nostrand Co., New York, 3d ed., 1889, pp. 252-262.

The best results are obtained when a steady flow of water and gravel passes through the sluice. An excessive flow of clear water through the sluice will bare the riffles, causing some gold to be lost. On the other hand, a continued overload of gravel will plug the sluice at some point so that sluicing must be stopped for the time needed to clear the obstruction; this time lost may be appreciable. If plugging cannot be prevented by increasing the grade or the flow of water or reducing the feed, one or more sluice tenders must work along the sluice with forks or shovels to keep it open. This added cost may be serious at small mines. All effort should be directed toward getting the gravel into the box and letting the water do the rest.

Large boulders are another cause of expense and lost time. When the maximum size of boulder that the sluice will carry is known, all boulders larger than this should be prevented from entering the boxes. Relatively little work directed to this end will save hours of delay in clearing plugged sluices and unnecessary wear and tear on the boxes and riffles.

An exception is found in the operation of "booming." A necessary condition of this work is a heavy head of water which usually fills the sluice to the brim. Sometimes little or no work can be done in the pit while the water is on, and the entire crew may profitably patrol the sluice with long-handled shovels to guard against stoppages which might be disastrous because of the large flow of water and gravel. Before each "boom" all oversize boulders should be moved out of the course of the water.

Cleaning Up

Clean-up time should be kept to a minimum. This can be done by cleaning up as seldom as practicable and by using efficient methods. Large hydraulic mines, particularly if the water season is short, clean up only once a season except perhaps the upper one or two boxes. Dredges clean up every 10 days or 2 weeks, because large amounts of gold are recovered in relatively short sluices with attendant possible loss when the upper riffles become heavily charged. This necessary delay is used for routine repairs on the dredge. In ground-sluicing the clean-up period ranges from weeks to months, while in shoveling-in operations the sluice may be partially cleaned up daily. The danger of theft from the upper, richer boxes can be lessened by filling them with gravel at the end of each day's work.

The general principle is the same in all clean-up operations, but practice differs widely. Clear water is run through the sluice until the riffles are bare, the stream being reduced enough to prevent washing out the gold. Then the water is turned off or reduced to a very small flow, and the riffles of the first box are lifted, washed carefully into the box, and set aside. Any burlap or other fabric used under the riffles likewise is taken up, rinsed into the box, or placed in a tub of water where it can be thoroughly scrubbed. Then the contents of the sluice are shoveled to the head of the box and "streamed down" with a light flow of water. The light sand is washed away, and rocks and pebbles are forked out by hand. This operation is repeated until the concentrates are reduced to the desired degree of richness. Gold or amalgam may be scooped up, as it lags behind the lightest material at this stage, or all the black sand with the gold, mercury, and amalgam may be removed and set aside for further treatment. Successive boxes are treated similarly, until the sluice is bare. The last step is to work over the whole sluice with brushes and scrapers to recover gold and amalgam caught in cracks, nail holes, or corners. At the Wisconsin mine a small box was set up in the main sluice and the concentrate from the riffles shoveled into it to reduce the bulk. At the Round Mountain mine the concentrate from the lower section of the sluice was treated in a quartz mill.

Use of Quicksilver in Sluicing

Quicksilver is used at nearly all placer mines. If it is not used to catch gold in the sluices, at least it is probably used in extracting the gold from the concentrates. The average market price for mercury in 1932 was about $58 per 75-pound flask, but quicksilver purchased in 5- or 10-pound lots from a chemical-supply house costs about $1 per pound. Except in districts where placer mining is particularly active, drug stores or other local retailers charge about double this amount. The price in January 1934 was $67.54 per flask.

The characteristics of quicksilver that make it of value to the miner are: (1) Its power of amalgamating with gold and silver; (2) its high specific gravity (13.5), which causes it to lie safely under a stream of water and gravel, floating off on its surface everything but the native metals; and (3) its relatively low boiling point (about 675° F.), far below red heat, which allows it to be driven off by heat from the gold with which it has amalgamated.

Amalgamation is a process in which mercury alloys with another metal. All metals but iron and platinum amalgamate more or less readily. Clean and coarse placer gold alloys readily, but if the gold is partly coated with iron oxide or other substances (for example, "rusty" gold) it amalgamates with difficulty. The mercury itself should be clean enough to present a smooth, shiny surface; the presence of some gold or silver in the quicksilver, however, is said to facilitate amalgamation, that is, to make it more "active."

Quicksilver is placed carefully in the sluice boxes, where it finds its way to the many recesses in the riffes and lies in scattered pools, ready to seize and hold any particle of gold that touches it. It is used in this manner in almost all important hydraulic operations, but some operators place it in the boxes only shortly before the clean-up, evidently believing that the added gold saved by its use during sluicing does not compensate for the loss of the mercury that passes through the sluice with the tailings or escapes through cracks or other leaks. In exceptional instances the conditions are such that the mercury "flours", that is, breaks into minute, dull-coated drops. Flouring is aggravated by agitation or exposure of the mercury to air. The common practice of "sprinkling" it into sluice boxes may be condemned on this ground, as well as for the reason given by Bowie[40] that the fine particles formed by careless sprinkling are more readily washed away and lost. Flouring is responsible for the most serious losses of quicksilver with the tailings.

Even under the best conditions, 5 to 10 percent of the mercury used is lost. If steep grades, heavy gravel with consequent severe pounding and vibration, old and leaky sluices, or other adverse conditions exist, the loss of mercury may be 20 or 25 percent.

Only clean mercury should be placed in a sluice; even this tends to become fouled or sluggish and to lose its effectiveness. The best cleansing process is retorting, which is discussed later. However, straining the mercury through chamois or tightly woven cloth removes some of the surface scum and foreign material, or the mercury may be treated with potassium cyanide or other chemicals to dissolve the impurities. It should be handled as little as possible and kept from contact with grease or other organic material.

Wilson[41] suggests a cow's horn, sawed off near the small end to leave a hole that can be stopped with the finger, as a useful implement for charging sluices. Most miners charge the sluice from stoneware or heavy glass bottles such as are used for champagne.

Mercury should be kept or carried only in iron, glass, or earthenware containers because of its tendency to amalgamate with zinc (galvanized iron), tin, or other metals.

---

40 Bowie, A. J., work cited, p. 244.

41 Wilson, E. B., Hydraulic and Placer Mining: John Wiley & Sons, New York, 3d ed., 1918, p. 230.

The quantity of quicksilver used differs according to conditions and custom. According to Bowie,[42] 200 or 300 feet of 6-foot sluice should receive about three flasks (225 pounds) as a first charge and a 24-foot square undercurrent, 80 or 90 pounds. At the Depot Hill mine one flask is placed in the first 4 or 5 boxes each month during the washing season. At the Plataurica two flasks were used in a season during which 100,000 cubic yards was washed. Dredge tables, with areas of 1,000 to 10,000 square feet, are charged with 150 to 3,000 pounds of mercury. According to Janin,[43] a 7-foot dredge with a table area of 2,800 square feet uses about 1,000 pounds on the sluices and in the traps. Probably in common practice the range is 1/10 to 1/4 pound per square foot of sluice area.

The sluice should be run long enough to plug all leaks before the mercury is added. Usually only the upper 2 or 3 boxes or a quarter or half of the sluice at most is charged with mercury, as otherwise considerable loss occurs. During a run more mercury is added periodically. Whenever the sluice is run down enough to expose the riffles the mercury can be examined. If it does not show here and there with clean surfaces nearly to the top of the riffles, more is added. As the quicksilver takes up gold near the head of the sluice it becomes pasty and finally quite hard, and more should be added to keep it in a fluid condition.

The use of mercury in recovering gold from sluice-box concentrates is discussed in the following section.

Amalgamating plates should be used only in treating fine material, generally well under one fourth inch in size and preferably not coarser than 10-mesh, as larger particles abrade the plates too rapidly and prevent building up of the amalgam. Consequently, the application of plates to placer mining is limited to the stamp milling of some drift-mine gravels and the treating of fine undercurrent or other screened sands. The use of plates in stamp milling is a phase of metallurgy beyond the scope of this paper, and reference is made to any standard text or handbook on gold milling.[44]

None of the other applications of amalgam plates to placer mining is of particular importance, probably because the recoveries seldom have justified the labor and expense. Plates may be set in undercurrents treating finely screened sands, such as beach sands or the Snake River gold-bearing sands. They usually are covered with burlap to assist in retaining the gold until it has come in contact with the amalgam. Many other amalgamating devices have been applied to such material, but none is known to the authors to have been of greater value than properly designed sluices and riffles.

SEPARATION OF GOLD AND PLATINUM-GROUP METALS FROM CONCENTRATES

General

No sluice box or other type of gold saver used in large-scale placer mining makes a clean separation of the valuable minerals. The concentrate obtained must be treated further to make a marketable product. Concentrate obtained in cleaning bedrock in some types of mining is treated similarly to sluice-box concentrates.

The concentrate may be cleaned by panning or rocking in auxiliary sluices or by blowing, or it may be amalgamated in a special type of apparatus. The treatment will depend mainly upon the scale of operations, the proportion of black sand in the concentrate, and the

42 Bowie, A. J., work cited, p. 244.

43 Janin, Charles, Gold Dredging in the United States: Bull. 127, Bureau of Mines, 1918, p. 143.

44 See also Chapman, T. G., Treating Gold Ores: Ariz. Bureau of Mines Bull. 133, Univ. Arizona, 1932; a brief, non-technical description of the methods of treating gold ores.

characteristics of the gold. The general methods of cleaning concentrate with pans, rockers, or small sluices are the same as those in small-scale mining, described in a previous paper,[45] except that more care is required and smaller quantities are treated at one time. In treating small quantities of concentrate, however, it should be remembered that colors of gold so fine as to present great difficulty in their separation by panning or rocking are probably of small value, and their loss would be inconsequential.

If precise results are desired for sampling or testing, the concentrates should be amalgamated.

Panning

Panning is the simplest method of separating the valuable constituents from the worthless material and generally is used in small-scale operation. The method, however is tedious if the gold is very fine and the concentrate contains much black sand. Mercury may then be used in the pan to collect the gold.

Rocking

Larger quantities of concentrate may be treated in a rocker and the resulting semifinal product cleaned further in a pan. A final or almost final product, however, can be made in a rocker, the flat, smooth bottom of which, set on a gentle grade with screen and canvas baffle removed, offe an ideal surface for the purpose.

The concentrates are placed at the upper end, and a small stream of water is poured over the sand while the rocker is swayed gently back and forth. The lighter material is washed down to the riffle at the lower end, and the coarser particles of gold are left behind. These are picked up with a scraper, and the operation is repeated, a portion of the concentrates presently being discarded with each washing until at length all gold of appreciable value has been recovered. This method is satisfactory with ordinary concentrates, but if the gold is very fine, slaky, or particularly light, porous, or angular, the separation is tedious and unsatisfactory, and amalgamation is to be preferred.

The same general method may be used in the mine sluice to recover the bulk of the gold amalgam.

Auxiliary Sluices

Sometimes an auxiliary sluice is used to reduce the volume of concentrate from the mine sluice or to treat concentrate after it is amalgamated. The small sluice in turn must be cleaned up. At one mine a 12-inch box was set up in the main sluice into which was shoveled the riffle concentrate from below.

Blowing

The grains of sand remaining in an almost final product may be removed from the gold by blowing. A flat metal or paper sheet, such as a piece of drawing paper or a large flat tin about 2 feet square with the edges bent up about one half inch, is best for the purpose. However, with care and skill the operation can be performed in a common gold pan, as is done by many prospectors, particularly when cleaning dry-washer concentrates. The material should

45 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part I. - General, Hand Mining, and Ground-Sluicing: Inf. Circ. 6786, Bureau of Mines, 1934, 73 pp.

be perfectly dry. Much effort is saved by using a magnet to take out any magnetite sand in the concentrates; often this mineral comprises as much as 90 percent of the material. A piece of paper folded around or held against the end of the magnet will keep the magnetite from sticking to the metal. When all the magnetite is removed, blowing gently on the remaining sand and gold will drive the former to the farther edge of the sheet, leaving the gold behind. In most instances the loss of a few fine colors is not serious.

Amalgamation

In Ordinary Gold Pans

A small quantity of quicksilver, ranging from an ounce to a quarter of a teaspoonful, will catch all the gold from a pan of sluice concentrates. The mercury is simply placed in the pan with about 5 pounds of concentrates and agitated under water until no more free gold can be observed. Then the sands are panned off, care being taken not to lose any of the amalgam or fine drops of mercury, which gradually will run together into a single mass. If the concentrates are nearly all black sand only a small quantity should be washed at a time, but if much light sand or rock is present larger quantities can be washed.

Copper-plated pans or pans with steel rims and copper bottoms are available and are useful for saving fine gold in concentrates. The copper is coated with mercury by first cleaning it with emery paper, then rubbing clean, bright mercury or amalgam on it until it presents a smooth, shiny surface. The gold in the material being treated is picked up quickly by the amalgam surface. Only fine sand can be treated to advantage as coarse sand or gravel will scour the amalgam off the copper. As fast as amalgam accumulates on the copper it is scraped off with a smooth, dull-edged, iron scraper such as a putty knife. More mercury may then be added to keep the surface bright and in a "receptive" condition.

Amalgamators

In nearly all large-scale operations most of the gold is amalgamated in the sluice boxes or on the riffle tables, and the amalgam is separated from the sands during clean-up operations or from the concentrates by rocking or panning. Tarnished or rusty gold or very fine gold, however, does not amalgamate readily because it is difficult to make contact between the gold and quicksilver. Such gold, generally included in a black-sand concentrate, requires agitation in the presence of quicksilver or, if rusty, grinding to remove the interfering coat for satisfactory amalgamation.

Mechanical amalgamators are used to treat such materials. Occasionally all of the concentrate from the sluice will be treated in an amalgamator, particularly if it contains rusty gold. The charges for the amalgamator should be kept clean; grease especially interferes with amalgamation.

A common type of amalgamator is the clean-up pan, which consists of a cast-iron, cylindrical, flat-bottomed barrel or tub 1 or 2 feet in diameter for small-scale work and 4 to 6 feet in diameter for mill service. The concentrate with 1 or 2 percent quicksilver by weight is placed in the pan with sufficient water to make the mass fluid and agitated by a revolving spider. The quantity of water added should be sufficient only to permit agitation without too great strain on the machine. The pulp should be thick enough to hold particles of mercury in suspension. Shoes on the lower end of the spider arms slide on a flat, circular race in the bottom of the barrel, thus adding some grinding to the agitation. After running for 1 or 2 hours the batch may be emptied through a drain plug in the bottom of the barrel and the mercury and amalgam separated from the sand by panning. Some pans are provided with side drain plugs at various elevations. The rotation may then be slowed from its usual speed of about 60 r.p.m., the shoes raised enough to stop the grinding, and water added. This will settle the quicksilver and amalgam; the waste sludge can then be flushed out through the upper drain plugs and almost complete cleaning of the amalgam and mercury made in the pan itself.

Another device, the so-called amalgam barrel, generally is used at large stamp mills and occasionally is employed in placer operations, particularly in dredging, to treat accumulated black sands, scrap metal, and other possible gold-bearing material from clean-up operations. It is merely a cast-iron or steel drum revolving on a horizontal axis like a ball mill and fitted with suitable drain plugs, handholes, manholes, or removable ends, depending on its size and use. The material to be treated is placed in the barrel with quicksilver, water, and a few iron balls, and the barrel is turned slowly for an hour or several hours. The barrel may then be flushed with water from a hose to wash away the lighter products of grinding, turned over, and emptied into a tub, the amalgam and mercury being recovered by panning. Potassium cyanide sometimes is added to brighten the gold, using only enough to make a very weak solution.

An amalgamator that occasionally is used, especially if a part of the gold is attached to particles of quartz, is the Berdan pan, which is relatively simple in construction and cheap to operate. The pan consists of a revolving cast-iron bowl, usually 3 to 5 feet in diameter, with a raised central hub for the drive shaft, giving it the form of a circular trough. The bowl is supported either by the drive shaft or by rollers and is set with a tilt of about 20 or 30° from the horizontal. It is driven at 10 to 30 r.p.m. either by a crown gear on the inclined shaft of the bowl or by a ring gear on the bottom of the bowl. One or two large cast-iron balls roll in the trough as the bowl revolves. Quicksilver is placed in the bowl with the charge, and as the device revolves a stream of water is directed into it and overflows at the lowest point of the rim. The material to be amalgamated may be added in batches or, if it is to be ground as well as amalgamated, by an automatic feeder, the slimes and fine material overflowing to waste; the bowl then acts as a classifier. For placer concentrates the batch process is used, 100 pounds or more being treated at a time. Too large a quantity of sand lessens the grinding effect of the balls.

A 1- or 2-cubic-foot, hand- or power-driven concrete mixer is a convenient amalgamating device for the small- to medium-scale placer miner, particularly if part of the gold is rusty. It costs only $20 to $30, excluding the small gasoline engine, and can be obtained from hardware stores or mail-order houses. The charge for such a machine is two or three pails of concentrates, 1 or 2 pounds of quicksilver, a few round cobblestones 3 or 4 inches in diameter, and water. About a 1-hour treatment will amalgamate practically all of the gold. The charge is emptied into a settling tub and then washed in a pan or small sluice box to recover the amalgam and mercury.

Regardless of the amalgamator used, too violent agitation of the mercury must be avoided otherwise excessive flouring hinders amalgamation and makes it difficult or impossible to recover the quicksilver.

Cleaning Amalgam

The mixture of quicksilver and amalgam from sluice-box clean-ups usually contains much more mercury than amalgam. It can be freed from sand, scraps of iron, and other solid impurities by careful panning and by washing with a jet of clean water. The amalgam can then be separated from the quicksilver by straining the mixture through buckskin, chamois skin, close-woven canvas, or other strong, tight cloth. This generally is done by hand, preferably under water to prevent scattering of the mercury. The quicksilver thus filtered off contains at the most only about one tenth percent of gold; this mercury is desirable for recharging the boxes as the small amount of gold makes it more active. The amalgam, after squeezing, still contains some mercury, part of which may drain off if the mass is suspended for several hours in a funnel or other similar container. With or without this last refinement, which one dredge operator used with success, the stiff, pasty amalgam is now ready for fire treatment to separate the gold. It contains 25 to 55 percent, commonly about a third by weight of gold and silver.

EXTRACTING GOLD FROM AMALGAM

Heating

Although retorting is the common method of separating the gold from the quicksilver in amalgam at dredges and other large-scale operations, the mercury in small quantities of amalgam may be volatilized by simple heating. A common method is to heat the amalgam on a clean iron surface over an open fire or forge, or in a furnace, until all the mercury is driven off. This is the usual expedient of the single miner or small operator who does not object to the loss of the small quantity of quicksilver involved. The mercury vapor may appear as heavy white fumes. Whether visible or not, mercury vapor is exceedingly poisonous, and the work must not be done except where a draft can be depended on to carry all the vapor away from the operator. As stated elsewhere, mercury boils at 675° F., a temperature about halfway between the boiling point of water and the first visible red heat of iron. However, it volatilizes at the boiling point of water enough to be dangerous to the health of persons exposed to it. Consequently, it should be handled carefully, particularly to avoid inhaling its vapors.

In another method of recovering the gold from small amounts of amalgam, a potato is used as a condenser. This is a device popular with prospectors because it is very simple, yet saves part of the mercury that would be lost by the method previously described. A large potato is cut smoothly in half, and in the flat surface of one half a recess is hollowed which should be considerably larger than the amount of amalgam to be treated. The amalgam is placed on a clean sheet-iron surface, the half potato is placed over it, and the whole is set over a hot fire. For convenience it may be done in a frying pan or the scrap of sheet iron put on a flat shovel so that it can be withdrawn readily from the fire. Some mercury vapor will escape under the edges of the potato, and, as before, these fumes must be avoided. After 15 or 20 minutes of strong heating the potato may be lifted off for inspection. If all the mercury is gone from the gold the potato may be crushed and panned, and a considerable part of the mercury will be recovered. It may be desirable to heat the gold further to anneal it; this can be done without removing it from the iron plate. Any tinned or galvanized metal intended for use in this process should be heated redhot and when scoured to remove all traces of the coating so that a clean iron surface will be presented.

A laboratory method of separating the gold is to put the amalgam in a small beaker and dissolve the mercury in a 1 to 1 solution of nitric acid and water. When all the mercury is dissolved, the gold may remain as a sponge, which can be washed gently in water and annealed in a small porcelain crucible. More frequently the gold will be recovered as a fine dust, which also can be washed and annealed but is less easy to handle.

Retorting

A very small amount of amalgam can be retorted quickly and easily in a laboratory in a glass tube 18 to 24 inches long, sealed at one end and bent 2 or 3 inches from that end to a slightly acute angle. A large tube three fourths inch in diameter is best. The amalgam is broken into pieces small enough to be dropped into the closed end where it is then heated, the fumes condensing in the long open end of the tube. The gold can be annealed by heating the tube to redness after all mercury is driven off.

A retort for treating a few ounces at a time can be made cheaply of 3/8-inch pipe, pipe connections, and a large grease cup. The lower and open end of the 3/8-inch pipe is inclosed in a larger pipe. Cooling water is poured through the space between the two pipes from an open connection in the top of the outer one. The charge of amalgam is placed in the grease-cup cover which is then screwed into place; graphite lubricant is placed on the threads to make a tight joint. Heat is applied to the grease cup, and the quicksilver is condensed in the lower end of the pipe. The method of using and the general arrangement of the device are similar to those of the next retort described.

The typical quicksilver retort for placer mines (fig. 15, C and D) is a cast-iron pot with a tight-fitting cover in which a hole is tapped to accommodate the condenser pipe. The capacities of such retorts range from a few to 200 pounds of amalgam, or about a quarter pint to 2 gallons. They are listed in chemical-supply catalogs at prices ranging from $4 to $30, not including the condensers. The condenser commonly used with this type of retort is an iron pipe 3 or 4 feet long leading from the hole in the retort cover at a downward angle of 20 to 30°; it is encased for most of its length in a considerably larger pipe through which cooling water is circulated. When heat is applied to the charged retort the mercury vapor enters the condenser pipe where it cools and condenses; it trickles down the pipe into a vessel placed under the open end of the pipe. In the treatment of a large amount of amalgam the temperature of the pipe might be raised to a point where some of the vapor would escape; therefore, a cooling device is necessary.

The retort may be heated over a large bunsen burner, by a gasoline blow torch, in a forge, or in one of several types of furnaces built for the purpose. Very high temperatures are unnecessary, and a wood fire is considered better than a coal fire. The flame should cover as much of the retort as possible.

A rigid, strong stand for the retort and condenser (fig. 15,A) should be constructed if the apparatus is to be used regularly.

The retort should be coated on the inside with chalk, or painted with a thin paste of chalk, clay, mill slimes, or a mixture of fire clay and graphite and thoroughly dried before putting in the charge. This prevents the gold from sticking to the iron, which sometimes causes trouble. A lining of paper serves the same purpose but tends to form an objectionable deposit in the condenser pipe.

The retort should not be filled over two thirds full of amalgam (a third or half full when retorting liquid mercury), otherwise there is danger of some of the contents boiling over into the condenser tube. The amalgam is broken into pieces and piled loosely. Then the cover is put on and clamped tightly with the wedge or thumbscrew provided, first making sure that the attached condenser pipe is clean and free of obstructions. The ground joint between the cover and body of the retort is seldom tight enough to prevent leakage and should be luted with clay or some sealing compound. One satisfactory cement is made readily by moistening a mixture of ground asbestos and litharge (red lead) with glycerin.

A low heat is applied at first, then after 10 or 15 minutes the temperature is increased just enough to start the mercury vaporizing and condensing. Too rapid heating harms the retort, and only enough heat should be used to maintain a steady trickle of quicksilver from the condenser. When no more mercury appears the temperature should be increased for a few minutes to red heat to drive the last of the quicksilver out of the retort; then the fire should be withdrawn from the retort and the latter allowed to cool. Some mercury vapor always remains in the retort, and the operator should take care not to breathe these fumes upon taking off the cover.

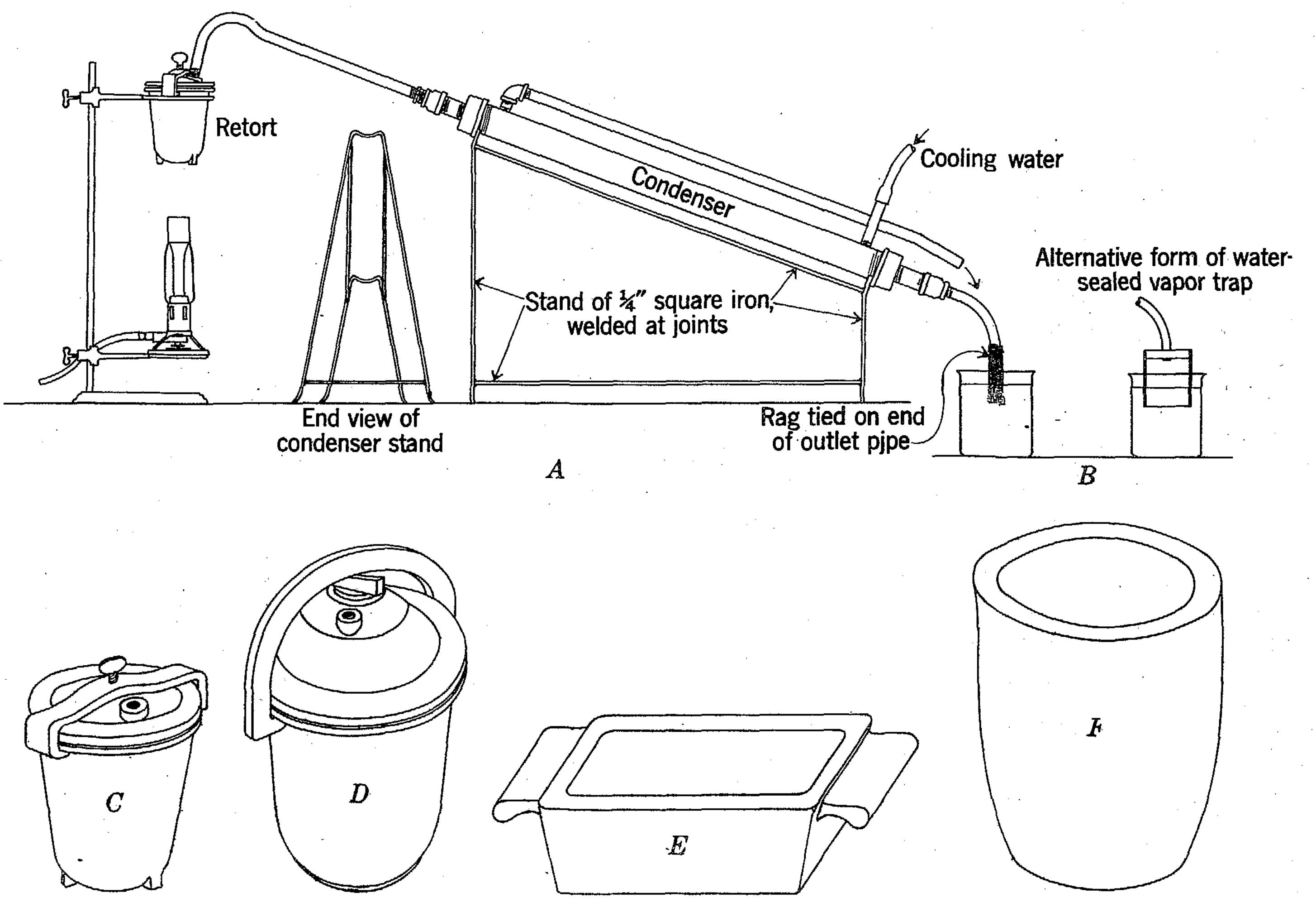


Figure 15.—Apparatus for retorting amalgam and quicksilver: *A*, Amalgam retort; *B*, Nevada-type retort; *C*, set-up of small retort; *D*, water-sealed vapor trap; *E*, graphite crucible; *F*, bullion mould.

The likelihood of dangerous amounts of mercury vapor passing through a long cold pipe without condensing is very small. However, if much amalgam is to be retorted, or if the operation is of daily or frequent occurrence, it usually is desirable to provide some form of water seal at the end of the condenser tube to prevent the escape of such fumes. Many miners have followed the dangerous practice of submerging the end of the condenser pipe in the bucket of water used to receive the condensed mercury. This should not be done, as a slight cooling of the retort would cause water to be sucked into the pipe, and if the water reached the retort an explosion would follow. Such an experience has taught more than one "oldtimer" the danger of this practice.

If the volume of the receptacle is very small compared with that of the condenser pipe and if the discharge pipe is barely submerged the danger is avoided, as any large rise of water in the pipe would lower the water surface enough to break the suction. At some properties the end of the condenser pipe is in a large sheet-iron cylinder, a few inches in diameter, open at the lower end, which may be placed 2 or 3 inches into the water in a receptacle of only slightly larger diameter, thus making a good water seal yet avoiding the danger of explosions. A laboratory adaptation of this device is shown in figure 15,B.

The simplest method is that recommended by Louis[46]; it consists merely of tying a piece of cloth such as canvas or burlap around the end of the condenser pipe and letting it dip in the water 2 or 3 inches below, forming a damp filter which will condense any escaping vapor yet not be tight enough to permit water to be sucked into the retort. This device is shown in figure 15,A.

Large gold mines use cylindrical retorts, usually set horizontally in specially built furnaces. Such installations probably would be needed in placer mining only by large dredging companies. The operation is similar to that of a pot retort, except that the amalgam usually is placed in several small iron trays, rather than on the floor of the retort proper, and charged through a door or removable cover at one end of the retort, while the condenser is attached at the opposite end.

SEPARATION OF PLATINUM-GROUP METALS FROM GOLD

In several localities in the Western States sluice concentrates from placer mining are likely to contain platinum or its associated metals in sufficient quantities to be of economic interest. The separation of these minerals from gold is difficult. Their specific gravity is too near that of gold to permit a separation by panning. Coarse platinum particles can be picked out of the gold by hand, but most placer platinum is exceedingly fine. Although platinum does not amalgamate, quicksilver can be made to coat and hold platinum particles by treatment with chemicals; thus it is possible to separate successively the gold and platinum from the concentrates.

One dredging company in California which recovers platinum metals uses the following clean-up procedure:[47]

> In cleaning up, the riffles are removed from the sluices, starting at the head end, carefully washing them off and washing the sluice down with water from a hose. This washes away the light sands and concentrates the amalgam and heavy sands, which are carefully scooped up into buckets and carried to a "long tom" for further treatment. In the long tom most of the mercury and amalgam and some

46 Louis, Henry, A Handbook of Gold Milling: London, 1894, p. 386.

47 Patman, C. G., Method and Costs of Dredging Auriferous Gravels at Lancha Plana, Amador County, Calif.: Inf. Circ. 6659, Bureau of Mines, 1932, pp. 12-13.

of the platinum-group metals are caught in the upper box. Most of the platinum, some rusty gold, scattered particles of mercury and amalgam, and the sand and refuse are washed out over riffles where the heavier components are caught. The sand finally passes through a screen at the end of the tom, into a sand box, and the gravel goes to waste. The mercury and amalgam from the upper box are transferred to a bucket, in which the gold amalgam settled to the bottom; the lead or other base-metal amalgams float on top. The latter is partially cleaned by panning, which separates some metallic platinum, then retorted. The gold amalgam is squeezed free of mercury and likewise retorted.

The gold amalgam, usually containing about 55 percent gold and silver, is retorted in a standard make of gasoline-fired retort. The mercury condenses in a water-jacketed pipe and drains into a bucket of water. The gold remaining in the retort is transferred to a crucible and fused in the same furnace. It is then poured into molds, producing bars which are shipped to the Selby smelter. The bullion averages 890 parts gold, 90 parts silver, and 20 parts impurities per 1,000.

The riffle concentrates and sand from the end of the long tom are placed in small batches in a steel barrel mill 4 feet long and 2 1/2 feet in diameter. Mercury is added and the batch ground for 1 or 2 hours. Then the amalgam is removed by panning and added to the other base amalgam for retorting. Further panning and rocking reduce the remaining sand and concentrates to a product containing about half black sand and half platinum, by volume. This is treated by the addition of water, mercury, zinc shavings, and sulphuric acid; this causes the platinum metals to be coated and held by the mercury, so that a final separation from the sand is possible. The final concentrate is then washed with water and drained to remove acid and excess mercury, after which treatment with nitric acid dissolves the mercury, leaving a final residue of platinum, iridium, and osmium.

The base amalgam, which includes shot, bullets, and small particles of copper and brass scrap, as well as some precious metals, is retorted to recover the mercury, melted, and poured into molds to form bars for shipment to the smelter. These bars range in value from $1 to $8 per troy ounce.

Zachert[48] states that platinum-group metals can be recovered on zinc-amalgam plates by using a solution of 0.05 percent copper sulphate and 0.05 percent sulphuric acid or by agitating with zinc amalgam in such a solution. At the Onverwacht mine in South Africa a process[49] similar to the above is used to treat a portion of the table concentrates:

The concentrates of the primary and secondary Wilfleys and of the James and corduroy tables are treated in lots of 1,000 lb. in a revolving amalgamating barrel, the amalgamation of the platinum being promoted by activating agents in the form of zinc amalgam, copper sulphate, and sulphuric acid. The barrel is revolved for 2 hours and then discharged via batea amalgamation plant and curvilinear table.

The dirty amalgam obtained is reamalgamated for half an hour with zinc amalgam, copper sulphate, and sulphuric acid. Thus cleaned it is now pressed and treated in earthenware jars with dilute sulphuric acid to remove zinc and iron.

---

48 Zachert, V. J., Process for Recovering Platinum: Min. and Sci. Press, vol. 117, Oct. 12, 1918, pp. 489-490.
49 Wagner, P. A., Platinum Deposits and Mines of South Africa: London, 1929, p. 274.

After this has been accomplished, it is retorted in small pot retorts. The retort sponge, after being subjected to further panning, sorting, and acid treatment, is washed and dried, giving a product assaying about 70 percent of platinum-group metals, which is shipped.

The recovery by amalgamation is about 98 percent and the all-over recovery of the plant ranges from 82 to 85.56 percent.

The United States Mint does not now buy platinum or pay for the platinum content of gold shipments, although it did about the time of the World War. The following buyers of crude platinum reported purchases in 1930:[50]

American Platinum Works, 225 New Jersey Railroad Avenue, Newark, N.J.
Baker & Co., Inc., 54 Austin Street, Newark, N.J.
J. Bishop & Co. Platinum Works, Malvern, Pa.
Sigmund Cohn, 44 Gold Street, New York, N.Y.
Thomas J. Dee & Co., 1010 Mallers Building, Chicago, Ill.
Kastenhuber & Lehrfeld, 24 John Street, New York, N.Y.
Pacific Platinum Works, Inc., 814 South Spring Street, Los Angeles, Calif.
Schwitter, Clover & Starkweather, Inc., 312 Passaic Avenue, Newark, N.J.
Wildberg Bros. Smelting & Refining Co., 742 Market Street, San Francisco, Calif.

Lots ranging from less than an ounce to hundreds of ounces ordinarily are marketable but preferably not less than 2 ounces. Settlement is based on assay, either by the buyer or, for large lots, by both parties. The price paid in 1930 for domestic crude platinum ranged from \$30 to \$40 per troy ounce[51]; the average quotation for the refined metal was \$45.

MELTING GOLD

The spongy mass of gold left after retorting can be sold to the mint or other agencies just as it comes from the retort, but generally it is melted and poured into molds to form bars or ingots for marketing.

The melting generally is done in graphite crucibles (fig. 15,F) placed in a special furnace. In small operations the crucible is usually heated in a blacksmith forge in which coke is used for fuel. The graphite crucible must be dried thoroughly before it is used by being warmed gradually for several hours.

Small quantities of gold frequently are melted without fluxes in makeshift devices such as dented frying pans; in most instances, however, some flux is desirable. If the gold is fairly pure, that is, has a bright yellow color, it may be melted with only a small quantity of borax glass for flux. If, however, it contains impurities and is grey or black in color, the melt requires larger quantities of flux to take up these impurities. Sometimes, niter, sodium carbonate, or silica is used to remove specific impurities. The flux is melted first, then the gold is placed in the crucible and likewise melted. Enough flux is used to form a covering about one half inch deep over the molten metal.

In large-scale operations the melted gold is poured from the crucible into cast-iron molds holding 50 to 1,000 ounces. (See fig. 15,E.) A mold should be larger at the top than at the bottom so that the bullion will drop out readily when it is inverted. A mold 3

50 Davis, H. W., Platinum and Allied Metals in 1930: Min. Res. of the U.S., 1930, pt. 1, 1931, p. 105.
51 Davis, H. W., work cited, p. 105.

inches by 12 inches at the top, 1 inch narrower and shorter at the bottom, and 3 inches deep holds about 1,000 ounces. The common practice is to smoke the mold over an oil flame, then to heat it before pouring the gold. Another practice is to coat the mold with graphite or oil or to pour a quarter inch of vegetable oil in the mold and heat it to boiling, then to pour the gold into the oil.

When the gold has just set in the mold and before the slag has hardened, the contents of the mold are tipped into water. This granulates most of the slag, and any particles still adhering to the gold usually can be brushed off. Tightly adhering slag can be loosened by washing the gold with nitric acid.

The bar of bullion may be stamped with identifying marks or names, or these may be cast in reverse in the bottom of the mold. The bar is then ready for market.

SAMPLING AND WEIGHING GOLD

Sampling

There are several methods of sampling gold bullion. The most accurate one is to dip a sample from the melted bullion before casting it. A graphite rod suitably shaped at one end to dip up the desired amount of gold, usually 1 to 5 grams, is heated redhot, stirred about in the melt, and lifted out with the sample. The sample is then poured into an oiled mold or into a shallow bath of heated oil. This method has been used by a few mining companies and is said to eliminate slight inaccuracies to which other methods are subject. It is impracticable for small amounts of gold and is inconvenient in that the sample is not obtained in a form convenient for assay; except for bullion containing large quantities of base metals, simpler methods generally are sufficiently accurate.

Other methods depend on taking samples from the solid cast bar of bullion. Chips can be cut with a cold chisel from the surface of the bar, at one or more places, hammered thin, and trimmed to the desired weight for assay, or holes can be drilled and the drill cuttings used for samples. The latter method is used most. Holes an eighth inch or less in diameter are drilled a quarter to a half inch into the bar, usually one on top and one on the bottom of the bar, on the center line a short distance from the opposite ends. Two diagonally opposite corners, one on top and one on the bottom, are sometimes preferred, although the difference probably is negligible. It has been found that in bullion containing base metals there is a strong tendency for the base elements to segregate at the bottom of the bar and for the top surface to be above the average fineness. Special methods of sampling then must be used. However, for most placer-mine bullion a sample of the desired weight, obtained in almost any convenient fashion, will be sufficiently accurate. Drill samples taken as described above usually check the mint or smelter return within 5 parts per thousand.

When gold is to be shipped to the mint, assaying the bullion is a needless expense as there is no recourse from the mint assay returns.

Weighing

Analytical balances suitable for weighing small amounts of gold with great accuracy cost from $150 to $300, and balances that will weigh large amounts of gold, such as the gold bars shipped to the mint by mining companies, with sufficient accuracy so that their value can be calculated to the nearest cent, are very costly. Balances capable of weighing a few ounces of gold to the nearest cent can be purchased for $20 to $30, and convenient pocket scales, either of the hand-balance type (see fig. 16,A) or arranged to be set up on the cover of their cases as mounted balances (see fig. 16,B) and capable of weighing 3 or 4 ounces to the nearest cent, are sold by most chemical-supply houses at prices ranging from $2.50 to $15, including weights. With little or no expenditure a set of hand balances can be made, similar to the manufactured set shown in figure 16,A, which will weigh 1 or 2 ounces of gold to within a grain or the nearest 5 cents. The balance beam may be of wood, 6 or 8 inches long, suspended by a pin, needle, or bent wire hook through a hole in the exact middle of the beam. The pans can be made of tin, cut 1 1/2 to 2 inches in diameter and hammered dish-shaped, or of the lids of small tin cans, each suspended by three threads from the ends of the beam by means of bent wire hooks. No pointer is necessary, as the beam can be leveled closely enough by eye. It is not necessary that all parts be of exact weight, as the balance beam or the pans can be trimmed to make the assembly balance. For even approximate accuracy, however, it is necessary that the pans be suspended from points on the beam the same distance from the center bearing, and for stability the end bearings must be slightly lower than the center one. The nicer the construction and the more nearly frictionless the method of suspension, the greater the accuracy; but even with very little attention to these points a sensitivity of less than a grain is obtainable when weighing as much as 2 ounces. Weights can be purchased in convenient sets for 50 cents or more or can be made or improvised from bits of wire or sheet metal cut to match any available standards. Coins may be used for weights in rough work. Table 14 gives the weight of United States coins in metric and troy units and the value of equal weights of gold of 4 degrees of fineness. The approximate weights of coins worn so as to be nearly smooth are included to show what allowances must be made when using coins for weights.

Table 15 shows the relations of metric and troy units of weight and the value of gold per unit in 4 degrees of fineness.

TABLE 14.- Weight of United States coins and values of equal weights of gold[1]

Coin	Weight				Value of equal weight of gold							
	Grams	Grains	Penny-weights	Troy ounces	1,000 fine At $20.67	1,000 fine At $35.00	900 fine At $20.67	900 fine At $35.00	800 fine At $20.67	800 fine At $35.00	700 fine At $20.67	700 fine At $35.00
Copper cent:												
New	3.1103	48.000	2.000	0.1000	$2.067	$3.500	$1.860	$3.150	$1.654	$2.800	$1.447	$2.450
Very worn	2.8	43	1.8	.09	1.85	3.15	1.65	2.85	1.50	2.50	1.30	2.20
Nickel:												
New	4.9999	77.160	3.215	.1608	3.323	5.626	2.991	5.064	2.658	4.501	2.326	3.939
Very worn	4.5	70	2.9	.15	3.00	5.05	2.70	4.55	2.40	4.05	2.10	3.55
Dime:												
New	2.5000	38.581	1.608	.0804	1.662	2.813	1.496	2.532	1.329	2.251	1.163	1.969
Very worn	2.3	35	1.5	.07	1.50	2.60	1.35	2.35	1.20	2.05	1.05	1.80
Quarter:												
New	6.2500	96.452	4.019	.201	4.154	7.033	3.738	6.330	3.323	5.626	2.908	4.923
Very worn	5.5	85	3.5	.18	3.65	6.20	3.30	5.55	2.90	4.95	2.55	4.35
Half dollar:												
New	12.500	192.904	8.038	.402	8.308	14.066	7.477	12.660	6.646	11.253	5.816	9.846
Very worn	11.7	180	7.5	.38	7.75	13.15	7.00	11.85	6.20	10.55	5.40	9.20
Silver dollar:												
New	26.730	412.500	17.188	.859	17.765	30.078	15.988	27.070	14.212	24.063	12.435	21.055
Very worn	25	380	16	.80	16.50	28.00	14.75	25.00	13.00	22.00	11.50	19.50

1 A gold double eagle ($20 gold piece) when new weighed 516.0 grains. The gold, like the gold and silver in all United States coins, was 900 fine, therefore the piece contained 464.40 grains of gold and 51.6 grains of copper. Hence, 1 grain of gold was valued at $20.00 ÷ 464.4, and 480 grains, or 1 troy ounce, was valued at $20.00 x 480/464.4 = $20.6718. The $10 and $5 gold pieces weighed exactly a half and a quarter as much, respectively, as the $20 piece.

TABLE 15.- Conversion table of metric and troy weights and equivalent values of gold

Weight				Value of equal weight of gold							
				1,000 fine		900 fine		800 fine		700 fine	
Grams	Grains	Penny-weights	Troy ounces[1]	At $20.67	At $35.00	At $20.67	At $35.00	At $20.67	At $35.00	At $20.67	At $35.00
1.000	15.4324	0.6429	0.03215	$0.6646	$1.1253	$0.5982	$1.0127	$0.5317	$0.9002	$0.4652	$0.7877
.06480	1.0000	.0417	.00208	.0431	.0729	.0388	.0656	.0345	.0583	.0301	.0510
1.5552	24.0000	1.0000	.05000	1.0336	1.7500	.9302	1.5750	.8269	1.4000	.7235	1.2250
31.1035	480.0000	20.0000	1.00000	20.6718	35.0000	18.6046	31.5000	16.5374	28.0000	14.4703	24.5000

[1]12 ounces troy = 1 pound troy = 0.823 pound avoirdupois;
16 ounces avoirdupois = 1 pound avoirdupois = 7,000 grains.

MARKETING PLACER GOLD

Five classes of buyers usually are available to the miner who wishes to sell gold dust, retort sponge, or bullion bar: (1) Individual gold buyers; (2) local stores; (3) local banks; (4) smelting companies; and (5) United States mints and assay offices. If the miner has base bullion or concentrates the smelter or custom mills are usually his only market.

Local stores are the principal buyers of small amounts of gold, ranging in value from a few cents to $50 or more. The merchant, who is often the chief retailer of supplies to the miners, finds it brings him trade to act as a commission buyer of gold, making it possible for the prospector and miner to convert their winnings promptly into cash. If his commission is fair, this is satisfactory, as it saves the miner much distasteful annoyance in preparing his gold for shipment, filling out various registration and report forms, and then waiting several days for his check. It likewise makes possible the sale of less than 2 ounces of fine gold at a time, which is the least amount of retort sponge, gold dust, or nuggets the mint will accept. The discount of the merchant ranges from $1 to $2 per fine ounce. The miner must remember that no placer gold is pure and that the merchant has only his judgment to tell him how much the mint will pay for his gold. Not all gold from a district assays the same degree of fineness, and the merchant is not to be blamed for staying on the safe side.

In most mining districts there are assayers, company officials, jewelers, metal brokers, or other individuals who for profit or for the convenience of employees, lessees, or customers make a practice of buying gold in small lots from prospectors and miners and paying cash for the value of the estimated weight of fine gold, less certain charges. Likewise banks in many districts receive gold, either purchasing it outright on the basis of their own or commercial assayers' analyses, or merely acting as shipping agents, receiving the gold, shipping it to the mint, and paying the miner upon receipt of mint returns. In the latter case a commission of about 1 percent usually is charged, for which the bank assumes all risk and trouble otherwise taken by the miner himself.

A few smelters or refineries buy gold or silver metals; the melting and refining charges probably will closely approximate those of the mint. Most smelter or refineries handling precious metal ores buy gold-bearing concentrate. Smelting charges on such material are variable, and an inquiry, accompanied by a close description or a sample of the material offered for sale, should be made in advance.

Gold can be shipped by express or by mail. If by express, the parcel can be insured with the express company for its full estimated value. United States mail shipments usually are sent as registered first-class mail and should be insured. The mail registry system provides insurance in graduated amounts from $5 to $1,000 at a cost of 15 cents to $1, including the registration fee but excluding postage. If regular mail shipments of considerable value are being made, it is possible to secure commercial insurance for them. However, for amounts greater than a few ounces the first-class postage rate of 3 cents per ounce becomes so costly that express shipments are advisable. All shipments must be prepaid.

The best container for shipping gold, either by mail or express, is a lead-sealed canvas sack, securely tagged with the addresses of the sender and addressee. Gold bars may be wrapped securely in canvas and packed in wooden boxes.

When a shipment of gold is sent to the United States Mint a letter should be sent separately containing the prescribed affidavits. Form TG-19, for a person shipping gold that he has mined himself, and form TG-21, for gold buyers, can be obtained by writing to a United States mint or assay office. Form TG-19 need not be sworn to if the amount of gold is 5 ounces or less. Since January 1934 the mints have paid $35 per troy ounce of fine gold, less one fourth of 1 percent, as compared with the former price of $20.67+ per ounce.

The mint charges $1 for melting any deposit of 1,000 ounces or less and 10 cents additional for each 100 ounces over this amount. An extra charge of $1 or more is made for melting gold dust or gold containing nonmetallics if the loss of weight in melting is more than 25 percent.

If the gold is 992 fine, or finer, no charge is made for parting and refining. If less fine, or if more than 50 parts base metals are present per 1,000, charges of 1 cent to 5 cents or more per ounce are imposed for parting and refining. Bullion less than 200 fine is not accepted.

Current market prices are paid for the full silver content; however, if the necessary forms are submitted for silver qualified under Executive Proclamation of December 21, 1933 the depositor will receive the number of silver dollars that can be coined from one half of the fine silver content. No other constituent in the bullion is paid for.

LAWS REGULATING ORE BUYERS

California and Colorado laws require ore buyers to take out licenses. The California Ore Buyer's License Act, passed in 1925, includes as buyers all persons sampling, treating, or buying gold dust, gold or silver bullion, gold or silver specimens or ores, or concentrates of these metals and gives the State mineralogist the duty of licensing such persons.[52] The license fee is set at $15 per year if the gold and silver treated or purchased in a year exceed $1,000 in value or at $2 if less. The licensee is required to keep on record the names of the sellers, the amount and description of each lot purchased, the stated source of each lot, and other data and to report all purchases monthly to the State mineralogist. Provision is made for the issuance of licenses, recovery of stolen metals, and penalties for violation of the act. The latter are severe. No regulation, of course, is placed upon the gold buyer as to terms of purchase, nor is this a matter of record under the act; however, one effect of the act, which primarily is intended to prevent the ready sale of stolen gold or silver, doubtless is to improve the chances of the small producer getting fair treatment from ore buyers by driving dishonest dealers out of business.

The Colorado law regulating the purchase of ores is similar to the California law. Nevada and New Mexico provide that persons buying ores must keep a record of purchases which shall be available to any one legitimately interested in tracing stolen ores but do not require the licensing of ore buyers. Other Western States have laws dealing with ore stealing and with "salting" or otherwise falsifying the value of mines, but none make such specific provision as do California and Colorado for the regulation of ore and metal buyers.

---

52 Ricketts, A. H., American Mining Law: Bull. 98, Calif. Bur. of Mines, 1931, pp. 640-646.

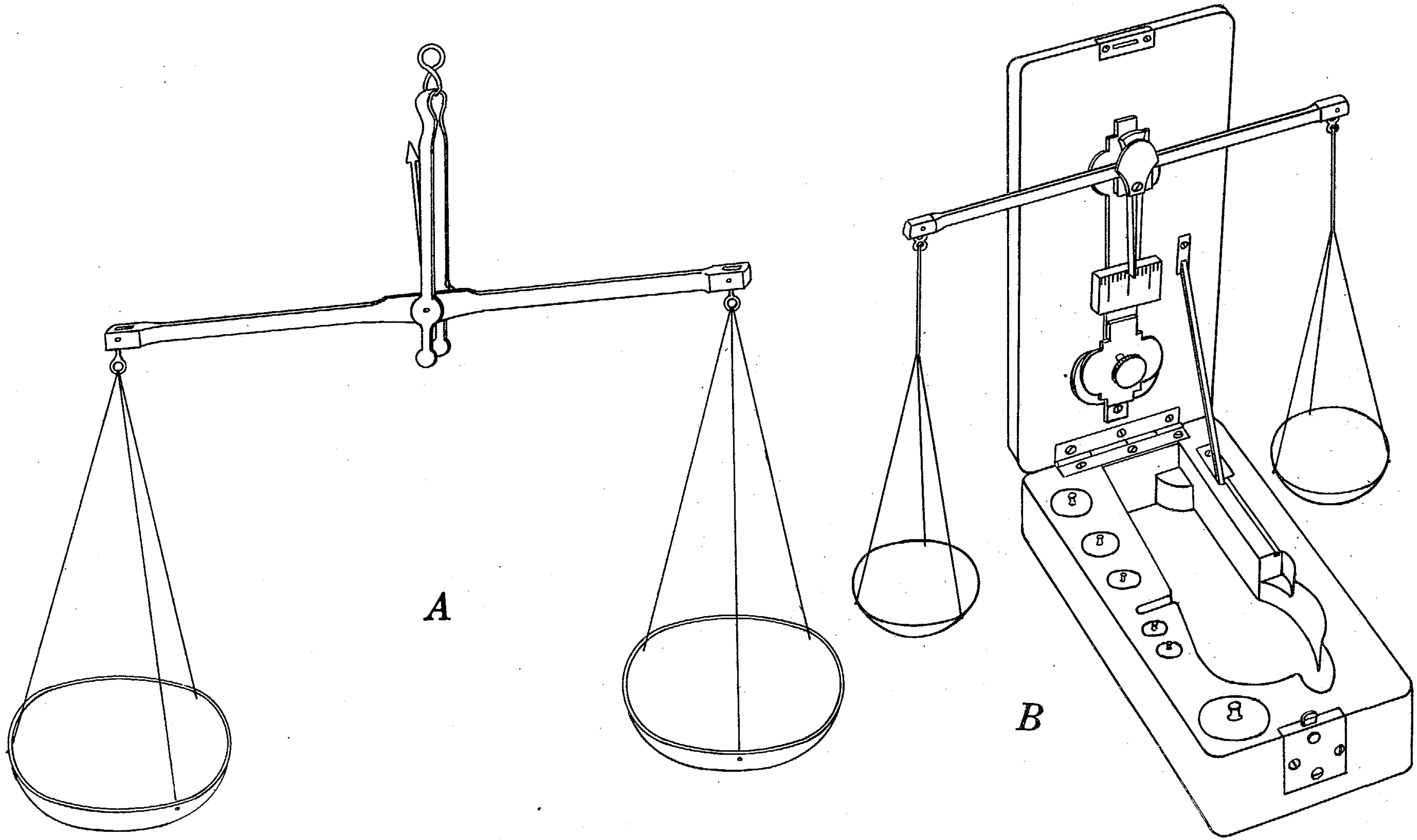


Figure 16.—Prospector's gold scales: *A*, Hand scales; *B*, folding pocket scales.

I. C. 6788

FEBRUARY 1935

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

PLACER MINING IN THE WESTERN UNITED STATES

PART III. DREDGING AND OTHER FORMS OF MECHANICAL HANDLING OF GRAVEL, AND DRIFT MINING

BY

E. D. GARDNER AND C. H. JOHNSON

I. C. 6788,
February 1935.

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

PLACER MINING IN THE WESTERN UNITED STATES[1]

Part III. - Dredging and Other Forms of Mechanical Handling of Gravel, and Drift Mining

By E. D. Gardner[2] and C. H. Johnson[3]

CONTENTS

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6788."

2 Supervising engineer, U.S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

3 Assistant mining engineer, U.S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

INTRODUCTION

This paper is the third of a series of three on placer mining in the western United States. The first paper[4] discusses the history of placer mining in the Western States and the production of placer gold, geology of placer deposits, location of placer claims on public lands, sampling and estimation of gold placers, and classification of placer-mining methods, together with hand-shoveling and ground-sluicing.

The second paper[5] deals with hydraulicking, sluice boxes and riffles, recovery of gold and platinum from placer concentrates, treatment of amalgam, and marketing of placer gold. The discussion of sluice boxes and subsequent subjects in the second paper applies to all forms of placer mining.

The present paper treats of dredging and other forms of mechanical handling of placer gravels, and drift mining.

ACKNOWLEDGMENTS

The authors wish to acknowledge their indebtedness to the operators of placers in the Western States who generously supplied information without which this paper could not have been written.

Descriptions of placer operations in Nevada were supplied by Alfred M. Smith of the Nevada State Bureau of Mines and Wm. O. Vanderburg of the United States Bureau of Mines, both of Reno, Nev., Francis C. Lincoln of the South Dakota School of Mines at Rapid City, S. Dak., supplied the description of placer operations in South Dakota.

The account of the dredging operations of the Fairbanks Exploration Co. at Fairbanks, Alaska, was prepared by C. G. Rice, vice president of the United States Smelting, Refining, & Mining Co., Boston, Mass.

Available literature upon placer mining, engineering, and allied subjects has been consulted; the authors have endeavored to make suitable reference throughout the text.

4 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part I. - General Information, Hand-Shoveling, and Ground-Sluicing: Inf. Circ. 6786, Bureau of Mines, 1934, 73 pp.

5 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part II. - Hydraulicking, Treatment of Placer Concentrates and Marketing of Gold: Inf. Circ. 6787, Bureau of Mines, 1934, 89 pp.

EXCAVATING BY TEAMS OR POWER EQUIPMENT

General Statement

Numerous gold-bearing deposits occur throughout the western placer districts that cannot be mined by the usual methods. Often there is insufficient water for hydraulicking or ground sluicing, or the deposits are too small to justify the building of ditches or pipe lines. Such deposits may not be amenable to dredging owing to lack of water or small size; also, the depth or character of the gravel or the topography and condition of the bedrock may make dredging impracticable. A large number of operations were begun in 1931 and 1932 in which mechanical equipment was used as the principal means of excavating the gravel prior to washing. This activity was due to two principal causes: (1) The increased general interest in placer mining and (2) the desire of excavating contractors and sand and gravel operators to use otherwise idle equipment and to keep organizations together. A third but less important cause was the endeavors of inventors and manufacturers of "trick" gold-saving machines to fine placers in which to install and test their equipment.

Although the early placer miners were as adept as the present generation and power shovels, scrapers, and other mechanical excavators have been tried for placer mining at many places during the last 40 years, the modern operator has at his command greatly superior excavating units and more efficient pumps and other mechanical equipment. Most of the present mechanical installations, however, have been built as cheaply as possible using second-hand or homemade equipment. Often the equipment used was not the best for the purpose but was employed because it was handy or cheap. For example, old automobile engines were used largely as power plants. At many mines much better fuel economy could have been obtained with a different type of engine of a horsepower more nearly corresponding to the work to be done.

In opencut copper and iron mining, as well as in large coal-stripping and quarrying operations, it is an axiom that all operations should be planned to serve the digging units and keep them working steadily and at full capacity. It is equally true in mechanical placer mining that both the excavator and washing plant must operate at capacity if the mine is to be worked at a profit. In the present stage of development of this form of placer mining the plants seldom work steadily at capacity. Standard power shovels or other forms of excavators can be obtained for digging the gravel. Standard set-ups, however, for washing the gravel and saving the gold have not been developed in this type of placer mining; nearly every plant has been built in accordance with a new design. Standard trommels of proved design are used for screening and washing gravels in the sand-and-gravel industry and on dredges; such equipment, however, seldom is purchased for the type of mining under discussion Delays due to breakdowns and remodeling are to be expected with newly designed or homemade equipment; this has been a contributing cause of failure at nearly all of the unsuccessful placer operations where mechanical excavating and washing equipment have been used. It is probable that eventually washing equipment like that for dredging will be perfected so that it can be operated steadily without breaking down.

Nearly all the excursions of excavating contractors and sand-and-gravel men into placer mining have proved unsuccessful. They failed principally because the gravel did not contain the expected amount of gold and because they did not consider the necessity of handling tailings. Sometimes, both of the foregoing reasons applied. In nearly all unsuccessful plants some innovation was tried.

During June and July 1932 the authors visited about 40 properties in the Western States at which teams or mechanical excavators were used. In addition, data were obtained by correspondence on 1 plant in Montana, 5 in North Dakota, and 3 in Nevada. Operations had progressed far enough so that approximate operating costs could be calculated at only 18 mines. These costs and other data on the mines are given in tables 1 to 5, inclusive. The other plants under consideration were being built, had not been run long enough, were run in such a manner that reliable cost figures could not be obtained, or had been shut down before enough data were available to calculate costs. With 4 or 5 possible exceptions, the gold recovered was not sufficient to meet the operating expenses if the workmen had been paid the prevailing wage in the district; at the majority of mines the labor was performed by men interested in the enterprise.

This method of mining has, for convenience, been divided into three classes, according to the method of excavating the gravel: (1) By teams or tractors, (2) by drag scrapers pulled by hoists, and (3) by power shovels or draglines. Teams and tractors are best adapted to relatively small-scale operations where the gravel is shallow and flexible operation is desired. Moreover, the first cost of the equipment is relatively low. The main disadvantage is the high labor cost per cubic yard handled. Lower operating costs may be attained by using scrapers and hoists, but there is less opportunity for selective mining. Under some conditions the scrapers may have advantages over power shovels or draglines for mining deposits within 400 or 500 feet of a stationary washing plant or where the shovel or dragline could not discharge directly into the plant. Scrapers on headlines have an advantage in that the material can be elevated to any desired height; moreover, they can be used for transporting gravel over rivers or other obstacles where other means of transportation would not be practicable.

Power shovels or draglines are adaptable to a wide range of conditions, but they have a high first cost and must be kept busy to be economical in use.

A large majority of the mines in this general group that were visited used power shovels or drag lines. Such mines are further subdivided into those (1) where stationary washing and gold-saving plants are used, (2) where movable washing plants on land are used, and (3) where floating washing plants are used. Operations of the last class are comparable to regular dredging operations except that separate excavators are used.

The great advantage of a movable washing plant is the elimination of trucking charges. Contract rates for trucking gravel at the properties visited in 1932 where movable plants were used ranged from 6 cents per yard for a haul of a few hundred feet to 16 cents for a haul of one half mile. Often the movable plant has the added advantage of easy disposal of tailings on the cleaned bedrock. Against these advantages are a number of disadvantages, which at times may be unimportant but have caused the failure of many projects.

Because the size and weight of movable washing plants must be held to a minimum most of them have been built either too cramped or too weak structurally to function well. Even if these faults are avoided the cost of designing and building such a plant is much higher than that for a similar permanent plant. Moreover, a wider range in design is possible in stationary plants.

Another result of the limited space in a mobile plant is lack of storage, which means that the feed to the gold saver may be very irregular even if an automatic feeder is used, as the plant must shut down whenever the shovel stops or a move is necessary. At a permanent plant, on the contrary, storage sufficient for several hours of operation may be provided if desired. Sluices that are too short are another result of striving for compactness in movable plants. The sluices, moreover, often are operated at a disadvantage because of the nuisance and delay in leveling the plant properly after each move. Most of the above disadvantages can be overcome, however, by building a constantly lengthening line of sluice boxes on trestles behind the plant as it moves upgrade, or by providing powerful jacks for leveling the plant. Water connections to a movable washer must be flexible, and they constitute a minor source of delay.

TABLE 1.- General data on placer mines where gravel is excavated by teams or power equipment, 1932

Mine			Gravel				Bedrock					
Name	Location	Operator	Depth, feet	Physical condition	Boulders over 8 inches in diameter, percent in gravel	Clay, percent	Kind	Physical condition	Kind of excavator used	Method of transportation	Kind of washing and gold-saving plant	Nominal daily capacity, cubic yards
Robbins	Vernal, Utah	F. W. Robbins.	4	Easy digging	0	0	Clay	Soft	Team and slip.	Team and slip	Screen and sluice.	6 1/2
Scott and Case	Douglas County, Nev.	Scott and Case	4	do.				do.	do.	Truck	Shaking box and sluice.	16
Roberts	Folsom, Calif.	W. H. Roberts.	(1/)	Loose	0	3	None		Tractor and scraper.	Tractor and scraper	Grizzly and sluice.	20
Delaney	Wenatche, Wash.	Delaney Bros.	6	Easy digging	5	0	Not reached.		Slip and hoist	Slip and hoist	do.	30
McElroy	Princeton, Br. Columbia.	T. E. McElroy.	6	Medium	10	0	do.		Scraper and hoist.	Scraper and hoist	do.	60
Mammoth Bar	Auburn, Calif.	F. W. Roumage.	30	Easy digging	3	0		Hard	Scraper on headline.	Headline and hoist	do.	200
Yellow Nugget	Hereford, Oreg.	S. A. Wells, et al.	8	Tight	5	0	Clay	Soft	Power shovel	Trucks	do.	160
Heine	Centerville, Idaho.	A. U. Heine, et al.	6	Medium	0		Granite	Easy	do.	do.	Sluice	1,000
Grant Rock Service 2/.	Fresno, Calif.	Grant Rock Service Co.	30	Easy digging	5	0	Volcanic ash	Soft, even.	Dragline.	Locomotive and dump cars	Trommel and sluice.	3,000
Skull Valley	Kirkland, Ariz.	Skull Valley Corporation.	8	Medium	1/2		Clay	Soft	Power shovel	Trucks	Trommel and Wilfley tables	90
Forbach and Easton.	do.	Forbach and Easton.	3	do.	0	15	do.	do.	do.	do.	Trommel and Deister tables	120
Mystic	Mystic, S. Dak.	Mineral leasing Co.	30				Slate		do.	do.	Shaking sluice.	3/300
Haag	Randsburg, Calif.	Haag Mining Co.		Very tight	15	10	Clay	Soft	do.	None	Movable plant with trommel and sluice.	100
La Cholla	Quartzsite, Ariz.	La Cholla Mining Co.	11	Tight	1		do.	do.	Dragline	do.	do.	400

1 Tailings pile.

2 Sand-and-gravel pit, gold a byproduct.

3 Reported.

TABLE 1.- General data on placer mines where gravel is excavated by teams or power equipment, 1932 - Continued

Mine			Gravel				Bedrock					
Name	Location	Operator	Depth, feet	Physical condition	Boulders over 8 inches in diameter, percent in gravel	Clay, per- cent	Kind	Physical condi- tion	Kind of excavator used	Method of transportation	Kind of washing and gold-saving plant	Nominal daily capacity, cubic yards
Bemrose	Breckinridge, Colo.	Buffalo Exploration Co.	15	Easy digging	5	0		do.	Power shovel	Elevator	Movable plant with trommel and centrifugal bowl.	200
Grand Hills	Custer, S. Dak.	Grand Hills Mining Co.	13	do.	1/2	0		do.	do.	None	do.	400
Kumle	Oregon House, Calif.	H. T. Kumle	15	do.	5	0	Porphyry	do.	Steam shovel	do.	Floating plant with grizzly and sluice.	250
Sumpter	Sumpter, Oreg.	Hofford and Jonnson.	13	Tight	10	3	Volcanic ash	do.	Dragline	do.	Floating plant with trommel and sluice.	640

TABLE 2.- Excavators and methods of transportation at placer mines where gravel is excavated by teams or power equipment, 1932

Mine	Excavator						Transportation	
	Type	Capacity of dipper, bucket, or scraper, cubic yards	Length of boom on shovel or drag line, feet	Diameter of drums on hoists, inches	Horsepower of engines	Kind of power or fuel	Method	Length of haul, feet
Robbins	Team and slip	1/7	0	0	0	Burros	Team and slip	25
Scott and Case	do.	......	0	0	0	Horses	One 1-cu.yd. truck	[1]3 1/2
Roberts	Tractor and scraper	1/4	0	0	0	Gasoline	Tractor and scraper	300
DeLaney	[2]Slip	1/5	0	10	......	do.	Slip scraper	20
McElroy	Scraper from hoist	1/4	0	14	......	do.	Bottomless arc scraper.	50
Mammoth Bar	Bucket on headline	1	0	24	95	do.	Headline	300
Yellow Nugget	Power shovel	5/8	15	......	......	do.	Two 4-cu.yd. trucks	2,000
Heine	do.	1 1/4	......	......	......	do.	Three 4-cu.yd. trucks	400
Grant Rock Service	Full-revolving dragline	5	100	......	......	Electric	20-cu.yd. dump cars and steam locomotive.	2,000
Skull Valley	Power shovel	3/8	......	......	18	Gasoline	Two 2-cu.yd. trucks	2,500
Forbach and Easton	do.	3/8	......	......	......	do.	One 2 3/4-cu.yd. and one 1 3/4-cu.yd. truck.	2,600
Mystic	do.	1/3	......	......	......	do.	Trucks	......
Haag	do.	7/8	18	......	......	do.	Dumped directly into hopper of washing plant.	......
La Cholla	Full-revolving dragline	2 1/2	85	......	......	do.	do.	......
Bemrose	Power shovel	5/8	......	......	60	do.	do.	......
Grand Hills	do.	1 1/4	......	......	110	do.	do.	......
Kumle	do.	3/4	......	......	......	Steam, coal	do.	......
Sumpter	Full-revolving dragline	1	......	......	75	Gasoline	do.	......

[1]Miles.
[2]Pulled from drum on transmission of old truck.

TABLE 3.- Washing and gold-saving plants at placer mines where gravel is excavated by teams or power equipment, 1932

Mine	Washing or disintegrating plant: Type	Spacing of grizzly bars, inches	Trommels: Length, feet	Trommels: Diameter, feet	Trommels: Diameter of holes, inches	Trommels: R.p.m.	Kind of power or fuel	Disposal of oversize
Robbins	Screen and sluice	1/16 by 1/4	None	...	...	...	None	Shoveled away by hand.
Scott and Case	Shaking box	...	None	...	...	...	Gasoline	Car to dump.
Roberts	Grizzly, bucket elevator, and sluice.	2	None	...	...	...	None	Pulled away by tractor.
DeLaney	Grizzly and sluice	3 by 3	None	...	...	...	None	Pulled away by scraper.
McElroy	do.	3 1/2	None	...	...	...	None	do.
Mammoth Bar	do.	4	None	...	...	...	None	Scraper from hoist.
Yellow Nugget	do.	6 1/2	None	...	...	...	None	Slide to dump.
Heine	Hopper and sluice	None	None	...	...	...	None	By hand.
Grant Rock Service	Sand and gravel plant	None	...	...	3/16	...	Electric	Elevator to crusher.
Skull Valley	Trommels and elevators	None	4 3	9 8	1/4 1/8	24	Gasoline	Boulders by hand; trommel oversize by belt stacker.
Forbach and Easton	do.	6	9 6	4 3 1/6	1/4 1/8	21 ...	do.	do.
Mystic	Shaking sluice	7 by 7	6	3 1/2	1/8 by 5/8	21	do.	Trucks to dump.
Haag	Movable plant with trommels and riffle tables.	8	22	4	5/16 and 3/4	...	do.	Boulders by hand; trommel oversize by belt stacker.
La Cholla	do.	9 by 9 3	8	5 1/2	3/4	...	do.	Belt stackers.
Bemrose	Movable plant with elevator and trommels.	None	14	3 4	1 1/4 1/8	9	do.	Boulders by hand; trommel oversize by belt stacker.
Grand Hills	do.	None	21	5	1/4 and 3/8	11	do.	do.
Kumle	Floating plant with grizzly and riffle tables.	1/2	None	...	...	...	do.	Belt stackers.
Sumpter	Floating plant, standard dredge equipment.	None	24	4	1/2	...	Electric	do.

TABLE 3.- Washing and gold-saving plants at placer mines where gravel is excavated by teams or power equipment, 1932 - Continued

Mine	Gold-saving plant						
	Sluice boxes				Riffles		
	Type	Width, inches	Length, feet	Grade, in. per ft.	Type	Size, inches	Center to center, inches
Robbins	Sluice	14	10	1 1/4	Wire screen over burlap		
		8	10	1 1/4			
Scott and Case	do.	10	48		Wire screen		
Roberts	do.	12	110	1 3/8	Hungarian over wire screen and burlap.	1 by 1	2 3/4
DeLaney	do.	20	128	1/2	Longitudinal wooden strips over screen and burlap.	1 1/4 by 2	2 1/2
McElroy	do.	18	48	1	Pole	3	4
					Hungarian	1 1/4 by 1	2 1/4
					Screen over cloth		
Mammoth Bar	do.	18	36	3/4	Angle iron	1 1/4 by 1 1/4	2 1/4
Yellow Nugget	do.	18	40	1	12-pound rails lengthwise	1 by 2	3
Heine	do.	32	125	3/4	20-pound rails	1 3/4 by 2 5/8	4 1/4
Grant Rock Service	do.	30	50	2/3	Dredge	1 1/4 by 1	2 1/2
Skull Valley	4 Wilfley tables	None			None		
Forbach and Easton	3 Deister and 1 Wilfley tables.	None			None		
Mystic	Shaking sluice	48	4 1/3	1 1/2	Transverse	2 by 2	6
	Sluice	25	9		do.	2 by 2	6
Haag	do.	[4]30	40		Rubber	3/8 by 1/2	7/8
		30	8				
La Cholla	do.	[6]14	40	5/6	Transverse pipe	2	3
Bemrose	4 centrifugal bowls	None			None		
Grand Hills	do.	None			None		
Kumle	Sluice	[6]27	30	1 1/2	Dredge	1 1/4 by 1	2 1/4
Sumpter	do.	30	(7/)		do.	1 1/4 by 1	2 1/4

[4]In parallel.
[6]Two sluices in parallel.
[7]Sluices in parallel, 900 square feet.

TABLE 3.- Washing and gold-saving plants at placer mines where gravel is excavated by teams or power equipment, 1932 - Continued

Mine	Water: Type and size of pumps	Vertical height pumped, feet	Pipe line: Length, feet	Pipe line: Diameter, inches	Pressure discharge, lb. per sq. in.	Miner's inches used
Robbins	3-inch centrifugal	12	25	3		
Scott and Case						
Roberts	3-inch centrifugal	20	20	4		
DeLaney	None					
McElroy	One 6-inch and one 3-inch centrifugal	6	20	7		
		10	25	7	5	
Mammoth Bar	6-inch centrifugal	35		8	5	82
Yellow Nugget	5-inch centrifugal	44	350	7	18	80
Heine	One 12-inch and one 7-inch centrifugal	37	700	14	5	350
		37	700	7	[1]20	30
Grant Rock Service						
Skull Valley	One 5 1/2-inch triplex and one 2 1/2-inch centrifugal.	240	7,800			[2]10
Forbach and Easton	One 5- by 6-inch triplex and one 2-inch centrifugal.	170	8,000	4		[3]22
Mystic						
Haag	(5)	1,800	6,000	4		[2]16
La Cholla	Trucked 5 miles				100	[2]9
Bemrose	None	Hydraulic head			150	17
Grand Hills	4-inch centrifugal					22
Kumle	7-inch centrifugal	15			15	
Sumpter	One 8-inch and two 5-inch centrifugals					2,000

[1]2-inch nozzle on pipe.
[2]New water.
[3]7 miner's inches of new water.
[5]Water purchased at $1 per 1,000 gallons.

TABLE 4.- Operating data at placer mines where gravel is excavated by teams or power equipment, 1932

Mine	Cubic yards treated per hour, average	Power and fuel: Excavator, Fuel consumed, Per hour, gal.	Excavator, Fuel consumed, Per cubic yard, gal.	Transport, Fuel consumed, Per hour, gal.	Transport, Fuel consumed, Per cubic yard, gal.	Washing and gold-saving plants, Motors, No.	Washing and gold-saving plants, Motors, Hp.	Washing and gold-saving plants, Fuel consumed, Per hour, gal.	Washing and gold-saving plants, Fuel consumed, Per cubic yard, gal.	Pumps, Motors, No.	Pumps, Motors, Hp.	Pumps, Fuel consumed, Per hour, gal.	Pumps, Fuel consumed, Per cubic yard, gal.	Cost per gallon of fuel
Robbins	0.8	0	0	0	0	0	0	...	...	1	1.5	0.2	0.2	$0.22
Scott and Case	2	0	0	0	0	1	...	...	...	...	...	...	...	...
Roberts	2.5	[1]2.5	[1]1.0	(2)	...	1	3	...	...	1	6	...	...	.15
DeLaney	3.8	1.25	.35	(2)	...	0	0	...	...	0	0	...	...	.20
McElroy	7.5	[3].8	.12	(2)	...	0	0	...	...	2	...	.3	.05	[3].33
Mammoth Bar	12	5	.6	(2)	...	0	0	...	...	1	20	1	.01	.15
Yellow Nugget	16	...	...	2.7	0.17	0	0	...	...	1	...	3	.19	.19
Heine	50	...	...	...	...	0	0	...	...	1	100	...	...	...
										1	10			
Grant Rock Service	330	...	...	...	...	...	...	...	...	...	...	...	...	...
Skull Valley	11	1.3	.12	2.0	.18	2	36	1.6	0.14	1	18	1.3	.11	[4].14
														[5].19
Forbach and Easton	15	1.9	.12	1.25	.083	1	50	3.7	[6].25	1	25	2	.13	[4].14
						1	9			1	...	2	.13	[5].19
														[6].17
Mystic	[7]30	...	...	...	...	...	...	...	...	...	...	...	...	...
Haag	6	3	.5	0	0	1	...	2	.3	...	...	...	...	.20
La Cholla	50	7	.14	0	0	1	6.5	7.5	.15	(8)	...	...	...	.17
Bemrose	20	1.5	.07	0	0	1	20	1.5	.07	0	0	...	...	.22
						1	8							
						1	4							
Grand Hills	40	6	.15	0	0	1	30	3	.08	1	12	1	.02	.16
						1	14							
						1	12							
Kumle	28	[10]18	[10].6	0	0	1	36	1	.04	1	35	(11)	(11)	.20
Sumpter	40	5	.12	0	0	1	25	[12]60	[12]1.5	1	25	(11)	(11)	[13].24
						1	10			1	30			

[1]Total for plant. [2] Same motor as used for digging. [3]Imperial. [4]Gasoline for washing plant. [5]Gasoline for trucks. [6]Distillate. [7]Reported. [8]Water trucked. [10]Pounds coal at $14 per ton. [11]Included with washing plant. [12]Kilowatt-hours. [13]Power, $0.03 per kilowatt-hour.

TABLE 4.- Operating data at placer mines where gravel is excavated by teams or power equipment, 1932 - Continued

Mine	Labor											
	Shifts		Excavator		Transport		Washing and gold-saving plant		Pumps		Total	
	Number per 24 hours	Length, hours	Men employed	Wages per hour	Men employed	Wages per hour	Men employed	Wages per hour	Men employed	Wages per hour	Per shift	Per 24 hours
Robbins	1	8	1	$0.37	0	0	1	$0.37	0	0	2	2
Scott and Case	1	8										
Roberts	1	8	1	.40			1	.40	0	0	2	2
DeLaney	1	8	2	.50	0	0	1	.50	0	0	3	3
McElroy	1	8	3	.28	0	0	2	.28	0	0	5	5
Mammoth Bar	2	8	1	.60	1	$0.60	1	.50	0	0	3	6
Yellow Nugget	1	10	2	.55	2	.50	2	.50	0	0	6	6
Heine	2	10	4		3		2	.40	1	$0.40	10	20
Grant Rock Service				1.00		.75		.50				
Skull Valley	1	8	1	.75	2	.40	2	.40	1	.40	6	6
Forbach and Easton	1	8	1	.75	2	.40	3	.40	1	.40	7	7
Mystic	1	10										
Haag	2	8	2	.75	0	0	4	.50	0	0	5	10
La Cholla	1	8	2	.60	0	0	3	.45	[8]1		6	6
Bemrose	1	10	1	.50	0	0	2	.50	0	0	3	3
Grand Hills	1	10	1	.60	0	0	3	.30	0	0	4	[9]4
Kumle	1	9	2	.50	0	0	1	.45	0	0	3	3
Sumpter	2	8	2	.50	0	0	2	.40	0	0	4	8

[8]Water trucked.
[9]Part-time superintendent at $0.60 per hour also employed.

TABLE 5.- Operating costs per cubic yard at placer mines where gravel is excavated by teams or power equipment, 1932

Mine	Excavation			Transportation			Washing and gold saving			Pumping			Miscellaneous			Super- vision	Total operating costs
	Labor	Sup- plies	Total	Labor	Sup- plies	Total	Labor	Sup- plies	Total	Labor	Sup- plies	Total	Labor	Sup- plies	Total		
Robbins	$0.46	[1]$0.02	$0.48	(2)	...	...	$0.46	$0.00	$0.46	(3)	$0.05	$0.05	$0.00	...	...	$0.00	$0.99
Scott and Case	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	1.50
Roberts	.16	.24	.40	(2)	...	...	.16	.01	.17	(3)	(3)	...	...	$0.05	$0.05	0	.62
DeLaney	.27	.12	.39	(2)	...	...	.13	.04	.17	$0.00	...	...	0	...	...	0	.56
McElroy	.11	.05	.16	(2)	...	...	.08	.03	.11	...	...	(3)	...	...	.03	0	.30
Mammoth Bar	.05	.11	.16	[4]$0.05	...	$0.05	.04	.005	.045	(3)	.015	.015	[5].015	.03	.045	.05	[6].365
Yellow Nugget	.07	.05	.12	...	...	[7].09	.06	.01	.07	...	.04	.04	...	...	.05	.02	.39
Heine	...	...	[7].20	...	...	(2)	.016	.004	.02	.008	[8].017	.025	.015	.02	.035	.01	.29
Grant Rock Service	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	[9].18
Skull Valley	.07	.03	.10	.07	$0.06	.13	.07	.04	.11	.04	.02	.06	...	.03	.03	.11	.54
Forbach and Easton	.05	.03	.08	.06	.06	.12	.08	.04	.12	.03	.05	.08	.03	.03	.06	.08	.54
Mystic	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	[10].25
Haag	.12	...	[11].55	0	...	...	.32	.10	.42	...	...	[12].18	...	...	.10	.05	1.30
La Cholla	.023	.044	.067	0	...	...	.027	.05	.077	...	...	[13].024	...	...	.05	.025	[14].243
Bemrose	.025	.03	.055	0	...	...	.05	.03	.08	0	...	...	...	...	.04	...	.175
Grand Hills	.015	.029	.044	0	...	...	.023	.037	.060	(3)	.004	.004	.002	[15].035	.037	.015	.16
Kumle	.03	.05	.08	0	...	...	.02	.01	.03	...	...	(2)	...	...	.05	.02	.18
Sumpter	.025	.04	.065	0	...	...	.02	.065	.085	...	...	(2)	...	...	.025	...	.175

[1]Burro feed.
[2]Included with excavation.
[3]Included with washing.
[4]General mechanic.
[5]Workman's compensation.
[6]Grand total, $0.455, including amortization.
[7]Contract price.
[8]Daily cost of electric power $17.00.
[9]Grand total, $0.20.
[10]Reported.
[11]Shovel rental including operator and supplies, $55 per two 8-hour shifts.
[12]New water purchased.
[13]Water trucked at cost of 0.1 cent per gallon at plant.
[14]Indicated from calculations.
[15]Includes workman's compensation, insurance, and miscellaneous expense.

Some physical characteristics of the deposit, often not considered beforehand, may alone be enough to cause failure. In very shallow ground moves will be so frequent as to reduce both digging and washing time to a point where profitable work may be impossible. In very deep ground, on the other hand, the stacker may not be long enough to dispose of the tailings. If the deposit is spotty, selective mining may be necessary, which is not practicable with most portable plants. If the gradient of bedrock is not favorable, provision must be made for drainage of the tailings water which otherwise may flood the pit or at least seriously hamper operations. Finally, steep or rolling bedrock may stop the advance of a heavy plant. One plant weighing nearly 100 tons, mounted on skids, was erected at a point where bedrock directly ahead of it rose on a grade of about 15 percent under a deceptively smooth cover of gravel.

At two properties visited the washing plants were floated. This arrangement eliminates most of the disadvantages of a movable plant but introduces another complication as the gravel must be excavated under water.

There is no apparent relation between the manner of excavating the gravel and the kind of washing plant used. The types of washing and disintegrating devices and the kind of gold savers selected should depend upon the quantity of water available and the physical characteristics of the gravel and contained gold.

Owing to lack of water or the cost of pumping the coarse material usually is screened out of the gravel before sending it through the gold saver. Moreover, some agitating or spraying device usually is needed to free the gravel of clay and thoroughly disintegrate it before extracting the gold. In some plants, however, only a grizzly is used ahead of a sluice box.

The revolving screen or trommel developed in gold-dredging and sand-and-gravel plants is an efficient and economical device for disintegrating and washing gravel. If the gravel is partly cemented and contains much clay, longer trommels are required than if the gravel is free-washing, and the first few feet of a trommel may be left blank; this permits full advantage to be taken of the disintegrating influence of the coarse material in a relatively large quantity of water. As an alternative to this method a disintegrator may be used ahead of the trommel; concrete mixers have been used for the purpose. A single trommel is preferable to one with concentric screens for treating clayey or cemented material; where only one screen is used a better opportunity is afforded for the coarse material to break up small lumps of cemented material than where the oversize is screened out in stages.

Riffled sluice boxes generally are used to save the gold. They are simple to build and operate and efficient if properly used. Many forms of riffles are employed. The design and operation of sluices is discussed in a previous paper[6]. The common practice of treating a screened product permits economy in the use of water and eliminates wear on the riffles from coarse material. Shaking or rocking sluices are used at a few places. This practice apparently increases the capacity of the sluice per unit of water used and may prevent sand from packing in the riffles.

Standard concentrating tables were used at a few plants visited and appear to have an advantage in treating screened gravel that contains a large proportion of black sand. In July 1932 a patented centrifugal gold washer was being used at three mines and had been or was to be used at a number of others. This machine was easy to clean and was said by its operators to save the gold satisfactorily. Its disadvantages were that it was costly and heavy and required power to operate. A carefully washed and screened product consisting of only a small part of the gravel excavated was treated in the machine. Apparently, an equally

---

6 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part II. - Hydraulicking, Treatment of Placer Concentrates, and Marketing of Gold: Inf. Circ. 6787, Bureau of Mines, 1934, 89 pp.

good recovery of gold could have been made with this prepared product in a properly designed sluice box with a corresponding quantity of water. Possibly under some conditions, however, a greater capacity per unit of water could be obtained with the machine than with a stationary sluice box. Because small machines of this type can be cleaned up quickly they appear to have merit for sampling and prospecting work. Other types of gold savers were seen but not in operation, and no operating data concerning them were obtained.

The descriptions of individual properties in the following sections illustrate practices in the Western States in 1932. A few references are made to plants that were operating in 1931; practices in Alaska, described in other publications, are cited to illustrate some conditions not met in western plants.

Teams or Tractors

Teams

Teams have an advantage in very small-scale work in that they represent a relatively small capital investment. They may have an operating advantage in very selective mining in shallow gravels and may be worked in water where a tractor could not run.

Robbins.- J. W. Robbins, with two young boys, was mining at Horseshoe Bend on Green River, below Vernal, Utah, during the summer of 1932. The gravel was loosened by hand picking and pulled about 25 feet to a sluice box by a span of burros hitched to a 1/7-cubic-yard slip scraper. A very small quantity of water was pumped from the nearby river by a 3-inch centrifugal pump run by a 1 1/2-hp. gasoline engine. The gravel was dumped through a trap into a small hopper, from whence it was drawn by a hoe to a screen with 1/16- by 1/4-inch holes. The water from the pump was discharged on the screen and together with the undersize dropped into the head of a sluice. The oversize was raked off the screen by a hoe and thrown to one side by hand shoveling. Because of insufficientwater more ground could be excavated in a shift than could be washed. The sluice consisted of two 10-foot boxes set on a grade of 1 1/4 inches to the foot. The first box was 14 inches wide and discharged into the second, which was 8 inches wide. Riffles consisted of burlap held down by wire screen. The gravel contained a high percentage of black sand, and the gold was extremely fine; however, a satisfactory saving of the gold apparently was made. Most of the black sand went through the sluice, but the tailing showed very little gold. Quicksilver was used in cleaning up. About 6 1/2 cubic yards was washed per day. The current daily cost of supplies was 33 cents for 1 1/2 gallons of gasoline and 10 cents for grain for the burros, which otherwise foraged for themselves, or less than 7 cents per cubic yard. Wages in the district were $3.00 for 8 hours. Allowing one man's wage for the two boys the labor cost per cubic yard would be 92 cents, or a total of 99 cents per cubic yard. (See table 5.) Amortization or rental for the complete outfit for the summer's work would be about $37.50, or 5 cents per cubic yard on 100 days' work, making a grand total of $1.04 per cubic yard.

Scott and Case.- C. F. Scott and S. C. Case were mining placer gravel with teams and scrapers in the Buckskin district in Douglas County, Nev., in the summer of 1932.[7] The gravel was 3 to 5 feet deep and lay on a decomposed bedrock. The soil and gravel were first plowed and scraped to one side by the team and scraper. Although it was reported that this material contained considerable gold it could not be treated at a profit. The bedrock was then plowed to a depth of 10 inches and scraped up a slide to a trap through which it was loaded into a small truck for transportation to the treatment plant 3 1/2 miles away at a

7 Smith, A. M., and Vanderburg, W. O., Placer Mining in Nevada: Univ. of Nevada Bull., vol. 26, no. 8, Dec. 15, 1932, pp. 39-40.

farm where water was available. The treatment plant consisted of a shoveling platform on which the gravel was unloaded from the truck, a shaking washing box to disintegrate the clayey material handled, and a sluice. The shaking box was 18 inches deep, 30 inches wide, and 5 feet long; it was made of sheet iron and was suspended from a rod by two strap-iron hangers. A screen consisting of 8-mesh woven-wire screen cloth on sheet iron with 3/4-inch holes was placed 6 inches above the bottom of the box. A horizontal shaking motion was imparted to the box by iron rods connected to 6-inch eccentrics. The eccentric shaft was belt-driven at 60 r.p.m. by a 60-hp. automobile engine.

Water was run onto the gravel as it was shoveled by hand into the box. Stones 5 and 6 inches in diameter were used to assist in the disintegration. The oversize from the screen was discharged into a 1-ton mine car and pushed to a dump. The minus 8-mesh product which contained the gold was run through a 10-inch sluice 48 feet long. Riffles were made of coarse-woven wire screen. The capacity of the plant was 16 cubic yards per day; operating costs were said to be $1.50 per cubic yard.

Arrowhead.- K. C. Nelson was operating the Arrowhead placer in the Lynn district, Eureka County, Nev., in the summer of 1932.[8] The gravel in the creek bed was loosened by a horse-drawn, spring-tooth harrow while water was flowing over it. A large part of the soil and clay was carried away by the water. The partly washed gravel was moved into a pile by a scraper drawn by a team. It was then shoveled by hand into a 12-inch rocking sluice, 10 inches high and 16 feet long, operated by a 1 1/2-hp. gasoline engine. Riffles consisted of a 6-foot plank with holes 1 inch in diameter drilled into it at the head of the box and 1- by 2-inch wooden cross-strips placed 2 inches apart in the lower 10 feet. About two thirds of the gold was recovered in the plank riffles. About 2 1/2 miner's inches of water was available; this was stored in a reservoir and an augmented flow used when sluicing. The sluice had a capacity of 4 cubic yards per hour. Early in the season when more water was available ordinary sluice boxes were used. When the water supply failed entirely dry-washing machines were used until autumn.

Horseshoe Bar.- The Utah Mining Co. was starting operations on the Horseshoe Bar on Green River near Vernal, Utah, in July 1932. The gravel was excavated by teams; after being plowed it was pulled in slip scrapers over a trap through which it went into the boot of a bucket elevator. The elevator discharged into a trommel with 18-mesh screen. The undersize was pulled out of a settling tank by a rake classifier and treated on Wilfley tables. The gold in the table concentrate was amalgamated in an ordinary copper-bottomed pan treated with quicksilver. Water was pumped from the river and settled before being used. The plant was homemade, second-hand materials being used. It was apparent that considerable remodeling would be necessary.

A crew of 20 men with three teams was employed. Five more teams were on their way to the mine. It was expected that 240 cubic yards of material could be handled per day. Not enough work, however, had been done to estimate the capacity of the plant or the costs per cubic yard.

Tractors

Roberts.- During 1931 and 1932 W. H. Roberts reworked the tailings from the Blue Channel drift mine near Folsom, Calif. The gravel as originally mined underground was tight, contained clay, and was partly cemented; apparently it did not disintegrate entirely during the first washing. In standing exposed to the rain and weather for a number of years, lumps of

8 Smith, A. M., and Vanderburg, W. O., Placer Mining in Nevada: Univ. of Nevada Bull., vol. 26, no. 8, Dec. 15, 1932, p. 50.

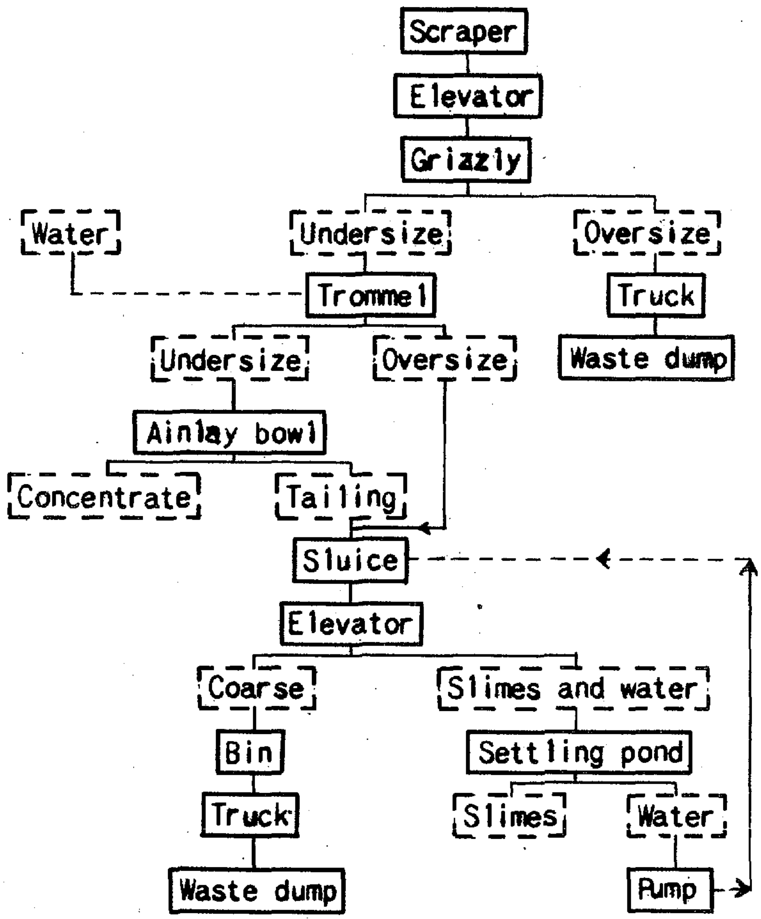


Figure 1.—Flow sheet of Queen placer, Manhattan, Nev.

the clay had been dissolved and particles of cemented sand and gravel had fallen apart. Roberts was reworking some of the dump that had been washed twice before. Most of the material being worked during June 1932 consisted of sand, small gravel, and clay. It was excavated by a tractor pulling a 1/4-cubic-yard scraper and dumped over a screen into a hopper whence it was elevated 26 feet by a bucket elevator to the head of a sluice box. The elevator was run by a 3-hp. gasoline engine. The sluice boxes were 12 inches wide and 110 feet long and were set on a grade of 1 3/8 inches to the foot. Wooden cross riffles of 1- by 1-inch material placed 1 3/4 inches apart were used over wire screen and burlap. Water was pumped from the old mine shaft by a 3-inch centrifugal pump run by a 6-hp. gasoline engine.

The gold was slightly rusty and did not amalgamate readily; no quicksilver was used in the riffles. In cleaning up, batches of the concentrates from the riffles, consisting chiefly of black sands, were placed in a 2-cubic-foot concrete mixer with about 25 pounds of rounded stones and 1/2 pound of quicksilver and run for about 1 hour. The batch was then dumped into a tub from which it was shoveled into a short sluice after removing the stones. Amalgam and quicksilver were recovered in the sluice. The concrete mixer was run by a 3/4-hp. motor. It had been purchased from a mail-order house for $26.75, was a very cheap and efficient amalgamator, and required no attention while running.

An average of 20 cubic yards was handled per 8-hour shift. One man ran the tractor and another operated the washing plant and did odd jobs. About 25 gallons of gasoline at 15 cents per gallon was used per shift for the tractor, elevator, and pump motor. The operating cost per cubic yard with labor at 40 cents per hour would be:

Labor	$0.32
Gasoline	.19
Other supplies	.06
Miscellaneous	.05
Total	$0.62

Queen.- A hillside deposit of recent wash gravel was being worked about 1 mile from Manhattan, Nye County, Nev., in 1932.[9] The gravel was about 22 feet deep and consisted largely of angular fragments of schist. The material was excavated and hauled to a washing plant by a Macmillan scraper drawn by a tractor. The average haul was about 75 feet.

At the plant the gravel was elevated 40 feet by an inclined belt-and-bucket elevator and discharged onto a grizzly with 2-inch openings between the rods. The oversize ran into a truck and then was hauled several hundred feet to a dump. The undersize passed by gravity to a double trommel; the inside screen had 1-inch round holes, and the outside one had 4-mesh openings. The oversize from both screens ran by gravity to a sluice with transverse riffles. The minus 4-mesh product passed through a pipe to the bottom of a 36-inch centrifugal bowl. The bowl was revolved at a speed of 100 r.p.m. by a 5-hp. motor and required 70 gallons per minute of water to operate. The tailings from the bowl joined the trommel oversize in the sluice where any nuggets in the oversize from the trommel were caught, as well as grains of gold not saved in the bowl. The sluice discharged into the boot of a bucket elevator. The coarse material was elevated 20 feet to a bin whence it was hauled by trucks to a waste dump. The slime and water overflowed from the elevator boot and ran to a settling pond from where about 25 percent of the water was reclaimed. A flow sheet of the plant is shown in figure 1. New water was bailed from a depth of 500 feet in a mine shaft and ran by gravity through a

9 Smith, A. M., and Vanderburg, W. O., Placer Mining in Nevada: Univ. of Nevada Bull., vol. 26, no. 8, Dec. 15, 1932, pp. 69-70.

3- and 2-inch pipe line 2 miles long to a 23,000-gallon storage tank at the treatment plant. The capacity of the plant was about 16 cubic yards per hour. A total of 20 hp. was required to operate the plant.

Burnt River.- Two material-handling contractors had just discontinued placer operations on Burnt River, Oreg., at the end of June 1932. Both used tractors and large Fresno slips. The operations of one had been unsuccessful because the mining costs were higher than the gold content of the gravel. At the other property the gravel was loose, and the permanent water level was near the surface. Neither the type of Fresno used nor the tractor operated satisfactorily in the water. Operations had been discontinued, and new plans were being formulated.

Scrapers and Hoists

Scrapers and hoists have been used for excavating and pulling placer gravels to washing plants. A scraper set-up with ground lines only consists of a hoist, usually with two drums, a scraper, and a cable. The scraper is pulled forward by the hoist over the gravel and picks up a load which is then pulled to the washing plant. The cable for pulling back the scraper goes through a sheave on the far side of the pit. To allow latitude of operation the sheave usually is attached to another cable stretched at right angles to the line of pull. The sheave sometimes can be shifted at right angles to the pull by means of a third drum on the hoist. The scraper is pulled on the ground both ways.

The set-up with an overhead cable is more elaborate; additional equipment consists principally of the overhead cable and a mast. After being filled the scraper is run to the plant and back on the cableway. The scraper or bucket is elevated by tightening the headline. Both bottomless and closed-bottom scrapers are used with ground lines, and only closed buckets, usually of the Page type, are used with cableways.

Boulders in the gravel and points of bedrock projecting up into the gravel cause the scrapers to jump. A bottomless scraper will loose its load on hitting a boulder, and a scraper of the closed type is difficult to fill in bouldery gravel. In easily dug gravel the bottomless scraper usually delivers a full load and can push considerable loose material ahead of it. The load is dropped by simply pulling the scraper backward, an advantage that scrapers with bottoms do not have. A closed-type bucket operating on a headline overcomes some of the difficulties of excavating with a drag; furthermore, it can be run at a greater speed once it is filled and the headline tightened. For long hauls the headline or cableway excavator has a further advantage in lower power and labor costs; moreover, the excavated ground can be elevated to the plant at any desired height with less trouble. However, this type lacks the mobility of the straight drag scraper, is more difficult to install, and because of the additional and heavier equipment has a higher first cost.

A scraper is not suitable for digging placer gravels under water. It follows the line of least resistance and leaves islands of bedrock untouched even where other conditions are favorable. The water is roiled by the digging, and the scraper works out of sight. Moreover, the stirring permits the gold to settle in the gravel being moved, and considerable gold may be left behind unless the pit can be pumped out for cleaning up.

For many years scrapers have been used successfully at sand and gravel pits. They have been tried at a number of placer mines in the Western States but generally have failed, usually because boulders were encountered in the gravel. In Alaska, however, scrapers have proved successful under favorable conditions and have been preferred to other types of excavators.[10]

10 Wimmler, Norman L., Placer-Mining Methods and Costs in Alaska: Bull. 259, Bureau of Mines, 1927, p. 94.

Scraper installations are less costly than power shovels or drag lines, but they have much lower capacities in most placer gravels so that the initial cost per cubic yard of daily output is roughly the same. The cost depends chiefly upon the type and size of power unit. Steam, electric, gasoline, and Diesel hoists are available. The usual installation ranges from 25 to 60 hp. Table 6 shows the approximate prices of several sizes of gasoline hoists as quoted by one manufacturer.[11]

TABLE 6.- Dimensions, capacities, speeds, and costs of hoists for scraping

Number of drums	Drum dimensions, inches: Length	Drum dimensions, inches: Diameter	Drum dimensions, inches: Flange diameter	Number of brakes	Rope capacities of drums, feet: 3/8-inch rope	Rope capacities of drums, feet: 1/2-inch rope	Rope capacities of drums, feet: 5/8-inch rope
1	14 1/4	11 1/2	17 3/4	None	850	500	...
1	16 1/2	16 1/4	26	None	2,200	1,300	850
1	16 1/2	16 1/4	26	None	2,200	1,300	850
2	6 5/8	11 1/2	18 1/2	None	480	275	...
2	5 3/4	11 1/2	17 1/2	1	350	200	...
2	4 3/8	16 1/4	24	2	550	325	215
2	7 1/2	16 1/4	26	None	1,100	650	425
2	4 3/8	16 1/4	24	2	550	325	215
2	7 1/2	16 1/4	26	None	1,100	650	425

TABLE 6.- Dimensions, capacities, speeds, and costs of hoists for scraping - Continued

Number of drums	Speed and pull of pull rope, (drum half full of rope)						Over-all dimensions						Engine hp.	Weight, pounds	Price, f.o.b. Chicago
	3/8-inch rope		1/2-inch rope		5/8-inch rope		Length		Width		Height				
	Speed, f.p.m.	Pull, pounds	Speed, f.p.m.	Pull, pounds	Speed, f.p.m.	Pull, pounds	Ft.	In.	Ft.	In.	Ft.	In.			
1	200	2,475	240	2,060	280	1,768	6	4	1	11	3	8	25	1,400	$1,100
1	230	3,600	315	2,600	400	2,000	8	6	2	3	4	1	35	3,300	1,570
1	230	5,000	315	3,700	400	2,850	9	4	2	3	5	2	45	3,600	1,885
2	200	2,475	240	2,060	280	1,768	6	4	2	1	3	8	25	1,560	1,275
2	200	2,475	240	2,060	...	...	6	4	2	1	3	8	25	1,585	1,275
2	230	3,600	315	2,600	400	2,000	8	6	2	3	4	1	35	3,380	1,850
2	230	3,600	315	2,600	400	2,000	8	6	2	3	4	1	35	3,330	1,765
2	230	5,000	315	3,700	400	2,850	9	4	2	3	5	2	45	3,690	2,160
2	230	5,000	315	3,700	400	2,850	9	4	2	3	5	2	45	3,640	2,080

The hoists listed are powered by well-known makes of 4-cylinder engines, connected by housed-in reduction gearing to the hoist drums, which are mounted in line with the engine center line. The drums are provided with external hand clutches and in some models with brakes. The drums of the 2-drum hoists are on a single shaft. Engine and drums are mounted on a channel-iron bedframe.

11 Sullivan Machinery Co., Chicago, Ill., February 1933.

Correctly designed head sheaves are important factors in the life of hoist ropes, which constitute the chief item of supply cost in scraper operation. The same manufacturer lists the following roller sheaves:

Size of rope sheave (outside diameter)........inches	8	10	12
Inside diameter of rope sheave........do.	6 1/4	8	9 3/4
Maximum size of rope to be used........do.	7/16	5/8	7/8
Maximum capacity........pounds	6,000	10,000	14,000
Weight........do.	30	50	85
List prices........	$37.50	$45.50	$57.50

These sheaves are of the snatch-block type and are provided with swivel hooks. The sheave wheels turn on roller bearings.

The following table concerning two sizes of gasoline-engine-powered scraper outfits was supplied by a maker of excavating equipment (Sauerman Bros., Inc., Chicago, Ill. Dec. 1932). The hoists comprise two tandem drums, connected by chain drive and reduction gears to a gasoline engine, all mounted on a channel-iron bed frame. The scraper is of the open-bottom type, crescent-shaped, and provided with digging teeth. The prices quoted include all accessory equipment needed for a set-up besides that listed, such as rope sockets, clips, and lashings. Larger and smaller sizes also are available. Both outfits are designed and furnished with sufficient rope for 300-foot spans but can be used for spans as long as 500 feet.

	Outfit A	Outfit B
Hoist:		
Horsepower of engine........	48	66
Inhaul speed........feet per minute	200	200
Backhaul speed........do.	400	400
Weight........pounds	5,415	5,830
Scraper:		
Capacity........cubic feet	20	26
Weight........pounds	420	685
Guide blocks: Number and size........inches	2 - 12, 1 - 14	2 - 14, 1 - 16
Cables:		
Load cable:		
Size........inches	5/8	5/8
Length........feet	350	350
Backhaul cable:		
Size........inches	1/2	1/2
Length........feet	675	675
Bridle cable:		
Size........inches	5/8	5/8
Length........feet	150	150
Tail guy cable:		
Size........inches	5/8	5/8
Length........feet	30	30

	Outfit A	Outfit B
Total weight, including accessory equipment.......pounds	6,665	7,630
Cost, f.o.b. Chicago, Ill., February 1933..........	$2,400	$2,935
Rated capacity, for continuous operation in easy-digging material, with 300-foot haul....cubic yards per hour	17	23

The following table from Bureau of Mines Bulletin 357[12] gives the strength, weight, and costs of nonrotating plow-steel hoisting rope of 18 strands with a hemp center and 7 wires to the strand.

Diameter of cable, inch	Breaking strength, short tons	Safe load, factor of safety of 8, short tons	Weight per foot, pounds	Cost per foot (December 1930 – Arizona)
1..............	33.8	4.25	1.73	$0.28
7/8...........	25.9	3.25	1.32	.225
3/4...........	19.0	2.50	.97	.18
5/8...........	13.3	1.75	.68	.135
9/16.........	10.8	1.50	.55	.120
1/2...........	8.7	1.00	.43	.105
7/16.........	6.7	.80	.33	.090
3/8...........	5.1	.60	.24	.085

The following table gives the weight and cost at Marion, Ohio, December 1932, of one type of Page bucket used on draglines or overhead cableways.[13] Lighter and heavier Page buckets are available as well as other types of dragline buckets.

Capacity, cubic yards	Shipping weight, pounds	Price
1/2...............	1,650	$285
3/4...............	2,425	370
1..................	3,460	522
1 1/2...........	4,540	698
2..................	5,630	868
2 1/2...........	7,430	1,315
3..................	8,200	1,475
4..................	10,000	1,765
5..................	11,775	2,160

Scrapers for drag-scraper service, whether of the Bagley or hoe type, are of lighter construction and cost less than corresponding sizes of drag-line buckets.

The following mines at which scrapers were used were operating in June or July 1932. The practices depicted are representative of this type of mining.

12 Gardner, E. D., and Johnson, J. Fred, Shaft-Sinking Practices and Costs: Bull 357, Bureau of Mines, 1932, p. 6.
13 Quotation by Marion Steam Shovel Co., Marion, Ohio.

Drag scrapers

DeLaney.- The mine of the three DeLaney brothers on Peshastin Creek near Wenatchee, Wash., is an example of a small placer mine operated mechanically with the minimum of equipment. The gravel was excavated at the side of a sluice box by a 1/5-cubic-yard slip scraper pulled by a drum fastened to the transmission on an old truck. The gravel was dumped through a 3- by 3-inch grizzly made of flat iron rods, placed crisscross, into a sluice box. The oversize was dragged over the sluice box to one side by the slip. The slip was pulled back and held by hand while being loaded. The truck engine was run in low speed while the slip was dragged forward and in reverse while the slip was pulled back. The gravel was dug readily by the slip, although in places some loosening of the material by hand picks was necessary. The average length of haul was 20 feet. The man who guided the scraper alternated with the one at the hoist controls. The third man tended the boxes and prevented the gravel being dumped from damming up the sluice.

The boxes were made of full 1-inch lumber and were 14 inches high and 20 inches wide at the upper end, the lower end being narrowed sufficiently for one box to telescope 8 inches into the next one below. Lumber cost $20 per thousand board-feet. Riffles consisted of 1 1/4- by 2-inch wooden strips 5 1/3 feet long, held 1 1/4 inches apart, by crosspieces at either end. Three sections of riffles were used to a box. Burlap held in place by wire screening was used on the bottom of the boxes under the wooden riffles. A piece of carpet fastened to a wooden frame, such as is used in hand rockers, was placed ahead of the riffles and caught about one half of the gold. This device could be removed quickly, the contents dumped into a pan, and the quantity of the gold caught determined readily. It was used to check continuously the value of the gravel being washed. Water was brought in a flume from a small diversion dam in the creek to the head of the sluice under the grizzly. As much water as could be used was available.

Under the best running conditions 30 cubic yards could be moved per 8-hour shift by three men. At 50 cents per hour the labor cost of excavation and transportation would be 40 cents per cubic yard. About 2 1/4 gallons of gasoline was used per hour by the truck engine. The cost per yard of the gasoline at 20 cents per gallon would be 12 cents and that of other supplies 4 cents, making a total operating cost of 56 cents per cubic yard of gravel removed.

McElroy.- An attempt was made by T. E. McElroy in June 1932 to mine a bar at the water's edge on the Similkameen River near Princeton, British Columbia, by means of a scraper and hoist. The gravel was fairly easy to dig but contained many boulders which seriously reduced the capacity of the scraper. The gravel was pulled up a steep slide onto a pole grizzly with 3 1/2-inch spacing. The oversize was pulled by the scraper over the grizzly and down another pole slide to a dump. (See fig. 2,A.) The undersize dropped through the grizzly into a flat-bottomed hopper from which it was fed by means of a regulating gate into a sluice box. The tailing from the sluice ran into the river. The scraper was operated by a 2-drum logging hoist run by a gasoline engine. The engine was set well back of the grizzly. The cables ran through snatch blocks set on a headframe at the proper height to pull the scraper up the slide and onto the grizzly which was about 10 feet above the surface of the ground. An arc-shaped bottomless scraper was used; when boulders were encountered it was replaced by a 3-pronged plow. The scraper was weighted on the rear to make it dig. Most of the digging was under water. When the gravel was dragged out of the pit by a bottomless scraper under water it was considered that an appreciable part of the gold had an opportunity to drop downward and be lost. Water for washing was pumped from the river. The stream from a 6-inch centrifugal pump emptied directly into the sluice box, and that from a 3-inch pump under a pressure corresponding to about a 10-foot head was used for washing the gravel from

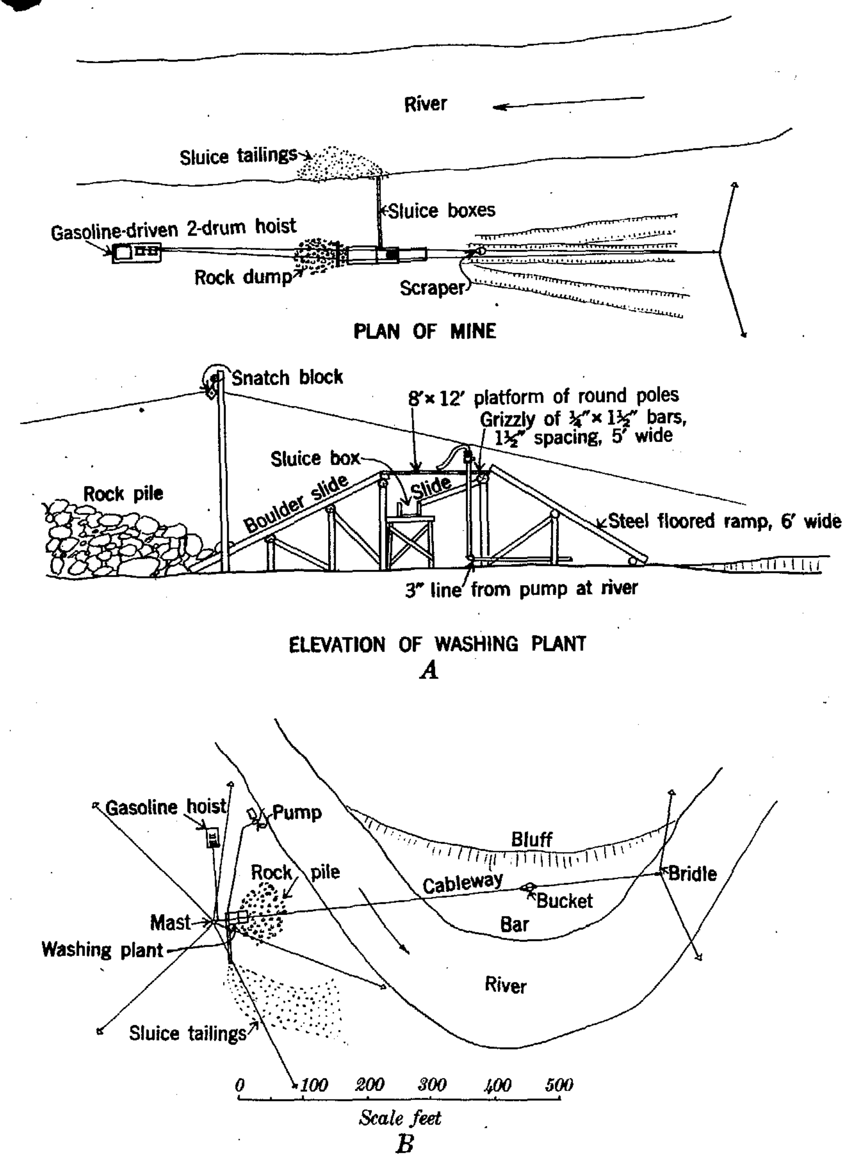


Figure 2.– Lay-outs of mines using scrapers: *A*, Drag scraper at McElroy mine, Princeton, B. C.; *B*, scraper on cableway, Mammoth Bar mine, Auburn, Calif.

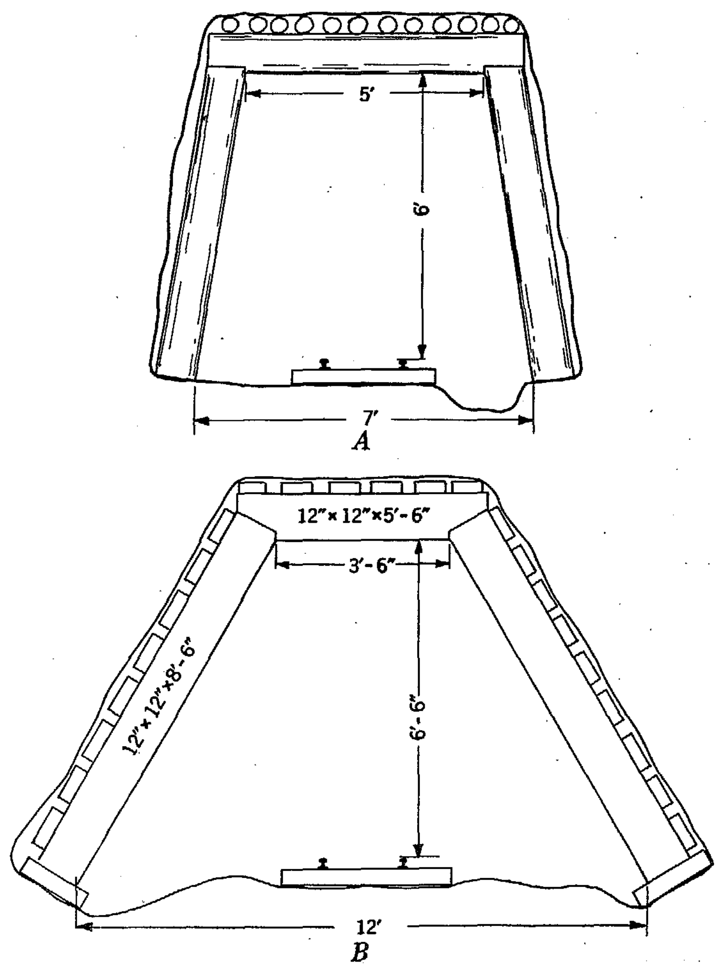


Figure 3.– Methods of timbering drifts: *A*, Drift set for normal ground; *B*, tunnel set used at Hidden Treasure mine, Placer County, Calif.

the hopper into the sluice. The sluice was 18 inches wide and set on a grade of 1 inch to the foot. The first 10 feet was lined with pole riffles in 5-foot sections; the next 5 feet had 1- by 1 1/2-inch Hungarian riffles; and the next 15 feet was lined with wire screen over carpet. The last 18 feet of the sluice contained no riffles.

The crew consisted of five men. About 1 1/2 hours of each 8-hour shift was spent in handling boulders. The plant had a capacity of 150 cubic yards per shift; the average daily tonnage, however, was only about 60 cubic yards. About 6 gallons of gasoline was used per shift for running the hoist and 2 1/2 gallons for the two pumps. The labor cost at 60 cubic yards per day and $2.25 per 8-hour shift was 19 cents. Gasoline and other supplies cost 8 cents per cubic yard. Miscellaneous costs were 3 cents. If supervision is disregarded the operating cost was 30 cents per cubic yard. (See table 5.)

Operations were unprofitable and were discontinued; digging under water was considered the reason for the low return in gold.

Nugget.- The Nugget Placer Mining Co. was mining an old channel on Libby Creek about 17 miles above Libby, Mont., in 1932.[14] The gravel was excavated by a 60-hp., gasoline, double-drum hoist on skids which pulled a Bagley 1-cubic-yard scraper. The tail block was about 250 feet from the hoist. The gravel was dragged up an incline and dumped on a shaking grizzly 8 feet long and 5 feet wide with 2 1/2-inch spacing. The lower end of the grizzly was pivoted, while the upper end was lifted about 2 inches and dropped on a bumper log by a stamp-mill cam mechanism. As the load was dumped it was sprayed with water. The fines were shaken through the grizzly, leaving nothing on it except clean boulders which were removed by hand and hauled away in a truck. The undersize was run through a sluice box 36 inches wide and 36 inches high with a grade of 4 inches to a 12-foot box. Riffles consisted of transverse 30-foot rails set 2 inches apart on 2- by 6-inch longitudinal chairs also set 2 inches apart. Concentrates from the main sluice were treated in a sluice 20 inches wide by 12 inches deep. the riffles consisted of 10-pound rails laid on 1- by 2-inch lumber like the rails in the main sluice. The concentrates from the small sluice were screened. The plus 16-mesh material was panned for coarse gold; the undersize was run over a small table covered with rubber matting held down by expanded metal. Assays of the tails from the table shewed $40 per ton in gold, indicating that a clean separation was not being made.

The set-up did not prove satisfactory. Water level was encountered about 15 feet above bedrock, and the scraper functioned poorly under water. Bedrock was uneven and extended up into the gravel; also, there was a layer of sticky clay with embedded boulders 8 feet below the surface. When the bottomless bucket encountered a boulder in the clay bed or projecting bedrock it jumped and lost its load. The grizzly mechanism gave trouble, and the capacity of the grizzly would have been insufficient if a more efficient digging unit had been employed.

In October 1932 plans were drawn to use a Page-type bucket on a headline similar to the installation on Manmoth Bar which is described later.

Molsom.- Unsuccessful attempts were made to use scrapers at two properties near Molsom, Wash., in 1932. At one the soil overburden was stripped off and the gravel was dragged up an incline to a grizzly with 4-inch spacing. Material over 1 inch in size was separated by a Hum-mer screen and the plus 1/4-inch mesh material by a second screen. The undersize went to a patented gold saver. The scraper could handle the gravel much faster than it could be washed. The enterprise failed owing to insufficient gold in the ground and to the low capacity of the gold saver.

At the other property the gold occurred in narrow streaks on bedrock. After the overburden was removed by a scraper the gold-bearing gravel was taken up by hand and washed in a rocker. The handwork was very slow, and the scraper could be used only a part of the time.

14 Information supplied by Sidney M. Logan, president, Nugget Placer Mining Co., Kalispell, Mont.

Minnick.- In July 1932 F. L. Minnick had just installed a scraper to mine gravel along Peshastin Creek, near Wenatchee, Wash. The gravel was dragged up a plank ramp onto a pole grizzly with about 4-inch spacings. The scraper pulled the oversize across the grizzly to a dump, and the undersize went through a 25-foot sluice box. Water was brought to the head of the sluice in a flume. The scraper was of the Bagley type and about 4 feet wide. The hoist was about 100 feet from the grizzly, and the tailblock was attached to a 20-foot gin pole 50 feet on the other side of the grizzly. The gravel was pulled downstream in and alongside of the creek. The ground was bouldery, and only a small capacity could be expected from the set-up. Not enough work had been done to allow costs to be calculated.

Alaskan practice.- Wimmler[15] has described scraper operation as practiced in Alaska. Operating costs at five plants using Bagley scrapers in the Fairbanks district ranged from 45 to 90 cents per cubic yard. An operator using a Bagley scraper on Willow Creek for a period of 5 years had an average cost of 30 cents per cubic yard. The cost at a smaller plant on Flat Creek was 68 cents per cubic yard. The cost of scraping 4 feet of gravel and bedrock with a slip scraper after the overburden had been ground sluiced was $1.55 per cubic yard in the Innoko district. The cost of a similar installation in the Hot Springs district was $1.32 per cubic yard.

Slackline cableways

Mammoth.- F. W. Roumage was mining the Mammoth Bar on the Middle Fork of the American River near Auburn, Calif., in June 1932. The deposit consisted of recent river gravel containing a relatively small proportion of boulders. The gravel was excavated by a 1-cubic-yard Page-type dragline bucket running on a 1 3/8-inch track cable across the river to the washing plant. (See fig. 2,B.) The bucket was operated by a 3-drum logging hoist run by a 95-hp. gasoline engine. The track cable ran from the top of a 64-foot spruce mast to an bridle cable across the river. The mast was 24 inches in diameter at the butt and 15 inches at the top and was held by 7/8-inch guy ropes. The in haul rope that pulled the bucket was 1 1/4 inches in diameter; 3/4-inch tension cable running through double 20-inch blocks was used to raise or lower the track cable. The bucket was hauled in at a speed of 300 feet per minute when loaded and returned by gravity at a speed of 1,200 feet per minute.

Most of the digging was under water. It was planned to pump out the pit at the end of the scraper operations, recover any "islands" missed by the bucket, and complete cleaning up of the bedrock by hand if necessary.

The gravel was dumped on a grizzly with 4-inch spacing between the bars. The undersize dropped into a hopper about 20 feet above the ground and thence was washed by sprays into a sluice box. The oversize, after the dumping room below the grizzly was used up, was to be dragged away by a scraper operated from one drum of the main hoist. The boxes were 18 inches wide and 36 feet long with a grade of 9 inches in 12 feet. Riffles consisted of 1 1/4- by 1 1/4-inch angle iron 3/16 inch thick, set flat in 2- by 3-inch wooden chairs on either side of the box. Every 4 feet a 2- by 3-inch wooden crosspiece was placed tightly on the bottom of the sluice. The lower edge of the vertical side of the angle-iron riffles was about 2 inches from the bottom of the box. Water was pumped from the river by a 6-inch centrifugal pump run by an old automobile engine.

The crew consisted of a man on the hoist, one at the washing plant, a mechanic on each of two 8-hour shifts, and a superintendent on day shift. The hoist used 5 gallons of gasoline and the pump 1 gallon per hour. Second-hand machines were used in equipping the mine,

15 Wimmler, Norman L., Placer-Mining Methods and Costs in Alaska: Bull 259, Bureau of Mines, 1927, 236 pp.

and considerable delay resulted from break-downs and repairs. Up to the middle of June as much as 240 cubic yards (but usually less than 200) had been handled in two 8-hour shifts. At 200 cubic yards per day the labor cost, including 1 1/2 cents per yard compensation insurance, would have been about 15 1/2 cents per cubic yard. Gasoline and other supplies cost 16 cents and supervision 5 cents, making a total operating cost of 36 1/2 cents. The cost of the plant was $7,500, and 100,000 cubic yards was expected to be washed. Amortization without interest would therefore have been 7 1/2 cents per cubic yard, making a total of 44 cents. If a larger average yardage could be handled the operating costs would be correspondingly less.

Power Shovels and Draglines

For many years power shovels and drag lines have been used for excavating earth and rock, consequently they have been improved so as to operate steadily and economically under severe conditions. They likewise have been tried often for placer mining. As early as 1897 a steam shovel was used on the placer gravels of Meadow Creek, below Warren, Idaho, for loading the gravel into tramcars which were then pulled up an incline and dumped into a sluice.[16]

Power shovels, as the term is used here, comprise machines that dig and load by means of a forward-thrusting dipper on a rigid arm or dipper stick. A dragline is a machine with a relatively long boom by means of which a scraper bucket, usually of the closed type such as is used for cableway excavators, is swung to any point within the range of the machine and filled by being pulled in by a separate hoist drum mounted on the machine.

Most power shovels can be converted to draglines by replacing the shovel boom, dipper stick, and dipper by a dragline boom and bucket, adding the fairlead for the drag cable, and sometimes adding an extra drum. Likewise, most machines can be equipped with booms of various lengths. If the boom length is increased the size of bucket must be correspondingly decreased.

Machines with dipper or bucket capacities of 1/4 to 20 cubic yards are available, but those most useful for placer-mining operations are the 2-yard or smaller machines. Two-yard and smaller power shovels ordinarily have digging radii of 12 to 25 feet, that is, can excavate cuts to grade 25 to 50 feet wide. They can dig, within a limited radius, to a depth of 5 to 10 feet below grade and can lift and dump their loads at heights of 15 to 25 feet above grade. Draglines designed to use buckets of corresponding sizes have much greater operating ranges both horizontally and vertically. One equipped with a 35-foot boom, for instance, and using a 3/4- or 1-yard bucket can usually dig within a radius of 35 to 45 feet (depending on conditions and the skill of the operator), can excavate to a depth of 15 feet below grade, and can lift its load and dump it at a height of 15 to 20 feet above grade. A dragline with a 90-foot boom, using a 2-yard bucket, normally has a digging radius of over 100 feet and can discharge at a height of about 45 feet; it can excavate to a depth of 35 feet below grade. The manufacturer's catalogs give operating ranges of particular models with given equipment and under given conditions.

Draglines and power shovels have respective advantages and disadvantages as applied to placer mining. Draglines have a greater excavating range and are used in placer mining chiefly with movable washing plants. Shovels, on the other hand, generally are used to load material into trucks or cars which transport it to the plant. With adequate truck service, a shovel thus used has greater capacity than a dragline with a bucket of equal size. When shovels load directly into the hoppers of washing plants the short digging and dumping radius

16 Lindgren, Waldemar, The Gold and Silver Veins of Silver City, DeLamar, and Other Mining Districts in Idaho: U.S. Geol. Survey, 20th Ann. Rept., 1898-99, pt. III, p. 234.

of the machine is a serious disadvantage, necessitating frequent move-ups with consequent lost time. The dragline, with its greater operating range, eliminates this disadvantage; in fact, it may be operated from the top of the bank, loading into a plant in the pit or on the side of the pit, whichever is most convenient. Its ability to dump at considerable height above grade may make an elevator on a movable washing plant unnecessary and thus simplify the flow sheet.

Much placer ground contains boulders, or is tight or even cemented. A power shovel moves boulders aside more readily than a dragline and digs better in tight or hard ground than a dragline of comparable size. Likewise the upper foot or so of bedrock is more easily dug with a shovel. For cleaning bedrock with a dragline the digging bucket may be replaced with one of the hoe type such as is used in digging trenches.

Neither shovel nor dragline is satisfactory for digging gold-bearing gravel under water. A shovel dipper usually leaks water badly, and gold may be lost in this way. Remedies are to use gaskets on the dipper, or to drain the dipper over the bank before swinging, hoping to pick up again any gold that is dropped. Although drag-line buckets may be made watertight they stir the gravel considerably before picking up a full load, and the gold tends to settle out of reach. Moreover, the exact location of the bucket when under water is uncertain, and the tendency is to leave untouched islands or ridges of gravel on bedrock.

The balance of these considerations decidedly favors the power shovel for the majority of placer deposits, although in some instances the use of a dragline may be very economical. Of 13 operating properties visited in 1932 and described in the following section, 10 were equipped with shovels and 3 with draglines. It is noteworthy that in spite of adverse digging conditions, such that dippers could seldom be filled without 2 or 3 thrusts, the capacity of the excavator was almost always greater than the capacity of the washing plant; this resulted in unusually high excavating costs.

Shovels and draglines used for placer mining are always caterpillar-mounted and full-revolving. They are very mobile, can be moved readily over poor and steep roadways, and can be used for a great variety of services such as moving plants, building roads and reservoirs, or making bedrock cuts.

Electrically powered excavators are more economical in power consumption and mechanical efficiency than steam, gasoline, or Diesel machines. When equipped with proper controls they have nearly the flexibility of steam shovels. However, as electric power usually is unavailable gasoline-engine power is used at most placer operations.

When using a shovel or dragline for mining placer gravel care should be taken not to drop oil or grease on the dirt that is to be washed or into water that will be used in washing, as the oil may cause some gold to float away or cause fouling of the quicksilver used in the sluices. Especial care must be taken when the machine is standing or digging in water.

Table 7, supplied by a manufacturer of power shovels,[17] shows the comparative possible outputs with both dipper and dragline types and operating data in a 15-foot gravel bank under ideal conditions. Such conditions, however, seldom if ever are encountered in placer mining, and much smaller yardages should be expected when estimating capacity. With the exception of the 3-cubic-yard draglines the machines are readily convertible from one type of excavator to the other. Moreover, the draglines can be purchased with various boom and bucket combinations. The tabulation shows the largest bucket with its proper boom length. The machines are available with either gasoline-engine, Diesel-engine, or electric-motor drives.

As shown in table 4, the shovel-operating crew consisted of 1 man at 8 properties where relatively small machines were used and 2 men at 4 properties where larger machines were employed.

17 Bucyrus-Erie Co., South Milwaukee, Wis.

TABLE 7.- Approximate outputs of various sizes of drag lines an shovels[1]

	Machine no. A	B	C	D	E	F	G
Drag line:							
Bucket capacity .... cubic yards	1/2	3/4	1	1 1/2	2	2 1/2	3
Average bucket load (material free from boulders) .... do.	0.4	0.6	0.8	1.2	1.6	2	2.4
Length of boom .... feet	35	35	45	45	45	50	85
Time of cycle .... seconds	25	28	30	30	30	32	34
Possible capacity .... cubic yards per hour	57	77	96	144	192	225	255
Capacity, at about 75-percent plant operation .... do.	43	58	72	108	144	160	180
Possible capacity per 8-hour shift, at 75-percent plant operation .... cubic yards	344	464	556	864	1,152	1,280	1,440
Shovel:							
Dipper capacity .... cubic yards	1/2	3/4	1	1 1/2	1 3/4	2 1/4	....
Average dipper load .... do.	0.4	0.6	0.8	1.2	1.4	1.8	....
Shovel cycle time .... seconds	15	18	20	22	22	22	....
Possible capacity .... cubic yards per hour	96	120	144	168	230	295	....
Capacity, at about 75-percent plant operation .... do.	70	90	108	127	170	220	....
Possible capacity per 8-hour shift, at 75-percent plant operation .... cubic yards	560	720	864	1,016	1,360	1,760	....

[1]Under very favorable conditions in readily excavated gravel; digging depth, 15 feet; 100° swing loading into hopper 12 feet high.

Table 8 is an estimate, supplied by the same manufacturer, of the fuel or electric-energy requirements of the various sizes of machines listed in table 7. The fuel consumption is calculated for sea-level operation; at higher altitudes the fuel requirements would be proportionately higher. The fuel consumption at a number of properties operating in 1932 is shown in table 4.

The manufacturers estimate that repairs, maintenance, and operating supplies for the machines listed in the foregoing table, exclusive of fuel and lubricants, range from 0.9 cent to 1.1 cents per cubic yard of material handled. This includes overhauling at proper intervals.

In steady operation under average conditions operating repairs, lubricating oils, renewals of parts, and depreciation of a power shovel may be figured as 20 percent of the first cost per year. In small friction-driven machines 35 percent of the first cost may be necessary. In very large machines, however, this charge has been as low as 5 percent of the first cost. For example, if a 2 1/2-cubic-yard machine cost $30,000 and 200,000 cubic yards of material was handled in a year, the digging cost, exclusive of labor and power or fuel, would be 20 percent of $30,000 divided by 200,000, or 3 cents per cubic yard.

The October 1932 prices, f.o.b. factory,[18] of the machines listed in table 7, equipped for dragline service, were as follows:

18 Bucyrus-Erie Co., South Milwaukee, Wis.

TABLE 8.- Fuel or electric-energy requirements of various sizes of power shovels and drag lines

	A		B		C		D		E		F	G
Machines with gas or Diesel engines:												
Bucket capacity of drag line......cubic yards	1/2		3/4		1		1 1/2		2		2 1/2	3
Dipper capacity of shovel........... do.	1/2		3/4		1		1 1/2		1 3/4		2 1/4	....
Gasoline consumption..................gal. per hr.	3-4		4-5		5.5-6.5		7-8		8-9		....	....
Lubricating oil........................ do.	.02		.02		.035		.045		.07		....	....
Fuel oil, Diesel........................ do.	....		2-3		3-4		3.5-4.5		5-7		6.5-7.5	8-10
Lubricating oil, Diesel engine do.	....		.045		.05		.05		.08		.10	.10
Machines with alternating-current motors:												
Driven by........................	Single motor		Single motor		Single motor		Single motor		Individual motors		Individual motors	Single motor
	Drag line	Shovel	Drag line	Shovel	Drag line	Shovel	Drag line	Shovel	Drag line	Shovel	[1]Drag line	Drag line
15-minute average input....................kw.	18	28	28	28	35	35	40	40	40	45	70	80
Momentary peaks.............................do.	50	65	65	65	80	85	95	95	100	105	170	175
Energy consumption............kw.-hrs. per cu.yd.	.3-.7		.3-.7		.3-.7		.3-.8		.3-.8		.3-.8	.4-.8

1 For a Ward-Leonard control (direct current motor under variable-voltage control) the input and energy consumption of class F drag line and shovel are:

	Drag line	Shovel
15-minute demand.............................kw.	50	50
Momentary peaks..............................do.	110	120
Energy consumption......Kw.-hrs. per cu.yd.	0.3-0.6	0.25-0.4

Machine no.	Capacity, cubic yards	Cost (including buckets)
A............	1/2	$6,200
B............	3/4	9,300
C............	1	11,000
D............	1 1/2	13,700
E............	2 or 1 3/4	17,250
F............	2 1/2 or 1 1/4	29,300
G............	3	48,400

Another Milwaukee manufacturer makes a series of shovels and draglines ranging in capacity from 1/4 to 4 cubic yards. Their drag lines are priced at $6,000 to $50,000.

Still another manufacturer[19] quoted the following approximate prices in October 1932 for four models of power shovels, f.o.b. Marion, Ohio.

Size of dipper, cubic yards	Hourly capacity, cubic yards	Cost, f.o.b. factory
3/4....................	40-120	$10,250
1..........................	45-140	12,000
1 1/4..................	50-160	13,500
1 1/2..................	60-175	16,000

A 3/8-cubic-yard machine, having three fourths swing and a 15-foot boom and powered by a tractor engine, was priced at $4,250 on the Pacific coast in October 1932.[20]

During 1932 used power shovels in good condition could be acquired at very substantial reductions or hired at reasonable rentals.

The following descriptions of plants illustrate the practices followed with different set-ups and washing plants where the gravel was mined by power shovels or draglines.

In addition, various unsuccessful efforts were made during 1931 and 1932 to mine gravel with power shovels, but no noteworthy features were observed at any of these operations.

Stationary washing plants

Yellow Nugget.- S. A. Wells and partners began working a high bar on Pine Creek near Hereford, Oreg., in 1932. The gravel averaged 8 feet deep, was tight, and was hard to dig. The bedrock was soft, and about 1 1/2 feet was taken up with the gravel. The gravel and bedrock were excavated with a 5/8-cubic-yard gasoline shovel and hauled in 4-cubic-yard trucks about 2,000 feet to a washing plant which consisted of a grizzly, pump, high-pressure sprays, and a sluice box.

All boulders over 6 or 8 inches in size that came to light while the trucks were being loaded were thrown out. An extra man at the pit worked with the drivers on the trucks to do this work. The gravel was dumped on a rail grizzly with 6 1/2-inch openings; the oversize went to a rock dump which was leveled off by hand as it was built up. The undersize was washed into the sluice from a flat, steel-lined hopper under the grizzly by water under 18 pounds per square inch pressure from five 2-inch sprays on goosenecks. The water was pumped

19 Osgood Co., Marion, Ohio.

20 Fordson-Tractor Sales, Ltd., Los Angeles, Calif.

44 feet vertically by a 5-inch centrifugal pump run by a 6-cylinder automobile engine; 900 gallons per minute was used. Three gallons of gasoline per hour was used for pumping.

The sluice was 18 inches wide and 40 feet long with a grade of 1 inch to the foot. Riffles consisted of 12-pound rails cut in 4-foot sections, laid lengthwise in the boxes, and 2- by 4-inch crosspieces between sections of rails. An undercurrent 10 feet long by 8 inches wide with 1-inch cross riffles of wood was used at the end of the sluice, mainly as an indicator. Quicksilver was used in all of the sluice except before the first riffle. Before quicksilver was used the loss of gold was appreciable.

One man operated the sprays and tended the upper end of the box. A second man tended the lower end of the sluice and kept the washed gravel from becoming dammed on the dump. The shovel and trucks could dig and haul the gravel faster than it could be washed. Little delay, however, was experienced in the operations. An average of 160 cubic yards was washed per 10-hour shift. The average gasoline consumption per shift was 30 gallons for the pump, 30 gallons for the shovel, and 27 gallons for the two trucks. The crew consisted of 6 men, 1 of whom acted as supervisor. Wages were 50 cents per hour for all except the shovel man, who was paid 60 cents per hour. The operating cost was 7 cents for labor and 5 cents for supplies for excavating, 9 cents per cubic yard (contract price) for hauling, 6 cents for labor and 1 cent for supplies for washing, 4 cents for supplies for pumping, 5 cents estimated miscellaneous, and 2 cents for supervision - a total of 39 cents per cubic yard.

Heine.- Dr. A. L. Heine and associates were mining a bar on Grimes Creek at Centerville, Idaho, in June and July 1932. The gravel averaged about 6 feet deep. It contained no boulders over 1 foot in diameter and few over 6 inches in size. The bedrock was decomposed granite; 1 foot of it was taken up with the gravel. The gravel was excavated by a 1 1/4-cubic-yard gasoline shovel and transported an average of 400 feet to the washing plant in three 4-ton trucks under contract. The contract price for excavation and transportation of the first 14,000 cubic yards was 25 cents per cubic yard and of the second 14,000 cubic yards 20 cents. The contractor's crew consisted of a shovel operator, oiler, mechanic, boss, and three truck drivers on each of two shifts.

The gravel was dumped into a hopper holding 6 cubic yards. No grizzly was used; one man at the hopper threw out all boulders over 6 inches in size. The gravel was fed into the sluice by means of a spray. Water for washing was supplied through a 14-inch pipe by a 12-inch centrifugal pump run by a 100-hp. electric motor. The head on the pump, including friction loss, was 37 feet. This water was discharged directly into the head of the sluice box. The water for the spray was pumped through a 7-inch pipe by a 10-hp. electric motor and discharged through a 2-inch nozzle. The electric-power bill for both pumps for 20 hours per day was $17. The sluice was 32 inches wide and 125 feet long and had a grade of 8 inches in 12 feet. Riffles consisted of 20-pound rails. In the first three boxes the rails were transverse; transverse and longitudinal sections alternated in the other boxes to the end of the sluice; 16 pounds of quicksilver was used in the riffles.

Three men were employed on each shift at the washing plant and pumps. At a wage rate of 40 cents per hour and 1,000 cubic yards per day the labor cost at the washing plant and pumps would be 2.4 cents. The estimated power cost would be 1.7 cents, supplies at the washing plant 0.4 cent, miscellaneous labor and supplies 3 1/2 cents, supervision 1 cent, and digging and trucking 20 cents - a total operating cost of 29 cents per cubic yard.

Great Bend.- A plant was built in the summer of 1931 to treat river gravels on the American River at Lotus, Calif.; operations were discontinued in November after the treatment of about 100,000 cubic yards. The gravel consisted mainly of old tailings and was dug by gasoline-power shovels and transported 500 feet in three 4-cubic-yard trucks to a washing plant. The trucking was done on contract at 6 cents per cubic yard. About 800 cubic yards was handled each 8-hour shift. The plant appeared to be well designed and was of substantial construction. The gravel was dumped from the trucks into a trap over a belt elevator which fed a 5- by 20-foot trommel with 9/16-inch holes. The oversize went to a belt stacker and the undersize over three 50-foot steel boxes 30 inches wide in parallel with 1/4-inch-square wooden cross riffles. The following electric motors were used: Belt elevator, 10 hp.; trommel, 15 hp.; stacker, 10 hp.; extension stacker, 15 hp.; feeder, 10 hp.; and pump, 50 hp.

Operations were discontinued, as the gravel did not carry the expected amount of gold. Cost data were not available.

Wallace.- The Gold Gravel Product Co. built a plant at Wallace, Calif., and washed about 30,000 cubic yards during the first half of 1931. The plant capacity was about 100 cubic yards per hour. The gravel was excavated by a 1 1/4-cubic-yard, full-revolving, caterpillar-tread, gasoline-driven shovel with a 30-foot boom and a 92-hp. engine. Haulage was by twelve 4-cubic-yard side-dump cars and two 8-ton gasoline locomotives running on a 36-inch gage track; 5 cars were run to a train. The distance from the pit to the washing plant was 1/2 mile. Gasoline consumption was 14 gallons per 8-hour shift.

The gravel was dumped through an 8-inch grizzly into a washer, consisting of a tube 5 feet in diameter and 16 1/2 feet long with a spiral vane 12 inches high on the inside and having 15 turns. Lifter bars also were used between the spiral blades. The washer which was revolved by means of a 30-hp. motor was set level; a particle would pass through in 2 1/2 minutes. The lower 4 feet of the tube outside of the spiral was flared outward and consisted of a screen with 1-inch round holes. The oversize from this screen went onto a belt stacker. An outside screen with 3/8-inch holes was used around the inner one. The plus 3/8-inch, minus 1-inch material was run through a 16-inch sluice 100 feet long and the minus 3/8-inch material through a 32-inch sluice also 100 feet long. The novel features of the operation were the use of cars and track for haulage and the spiral washer. About 3,000 gallons of water per minute was pumped for washin. the gravel, or 1,800 gallons per cubic yard of gravel excavated.

Cost data were not available.

Grant Rock-Service.- The Grant Rock-Service Co. at Fresno, Calif., produced gold as a byproduct to the sand, gravel, and crushed-stone business. The ground contained about 3 cents per cubic yard in gold. The deposit consisted of 30 feet of river gravel which was easily excavated; a cubic yard in place weighed 2,850 pounds. The bedrock was volcanic ash. The gravel was excavated with a 5-cubic-yard dragline with a 100- foot boom and transported to an inclined elevator in 20-cubic-yard dump cars pulled by a steam locomotive. The dragline could excavate 3,000 cubic yards in 9 hours. The gravel went through a standard sand-and-gravel screening and washing plant. The gold was extracted from the sand after it had passed through a 3/16-inch trommel screen. The oversize from the screen went to a sized gravel bin or a crushing plant. As the demand was greater for the rock and gravel than for the sand, most of the sand was washed back into the pit; however, all the sand was run through a 50-foot sluice 30 inches wide with dredge-type Hungarian riffles.

The work was on a large scale; the plant was of standard form, and the equipment used had all stood the test of time. A. H. Sienknecht, plant superintendent, made calculations which indicated that the cost of handling the gravel for the gold alone would be 18 cents per cubic yard. This included the cost of running all tailings back into the pit. By allowing amortization and general expense the total cost would be 20 cents per cubic yard. This cost should indicate what could be done with similar deposits on a large scale. It should be borne in mind, however, that the gravel was free from clay and large boulders, and almost ideal conditions existed for obtaining low costs. The digging was under water; with most types of bedrock this would have meant a relatively low recovery of gold in the pit.

Gold was obtained also as a byproduct from sand-and-gravel plants at Bakersfield, Calif., and at Denver, Colo. The gold was caught on riffles as at the Grant Rock-Service Co.

Skull Valley.- The Skull Valley Gold Corporation operated a placer mine in Skull Valley, Ariz., during 1932. Gravel was hauled to a central washing plant, and water was pumped from a distance. The gravel averaged about 8 feet deep, occurred on a clay bedrock, and contained a relatively large proportion of black sand. It was dug by a one-half-revolving, gasoline-driven shovel with a 3/8-cubic-yard dipper and hauled in two 2-ton trucks an average distance of 2,500 feet. The trucks dumped into a 110-cubic-yard bin. The gravel was pulled out through "Chinaman" chutes onto a 22-inch traveling belt and thence to a 4- by 9-foot trommel with 1/4-inch round holes. The oversize went to a dump via a stacker belt. The undersize went to a second trommel 3 by 8 feet in size with 1/8-inch square holes. Both trommels ran at 24 r.p.m. The launder between the trommels had traps to catch plus 1/8- to minus 1/4-inch nuggets. It was thought that about 70 percent of the material was rejected by the screens. The undersize from the second trommel went to four Wilfley tables in parallel. The tailings from the tables were elevated by a sand wheel onto the stacker belt. The concentrates from the tables were run over a 4- by 8-foot amalgamation plate. A quicksilver trap was used below the plate. The gravel contained cinnabar and native quicksilver; the latter was caught in the traps. About 10 pounds excess of quicksilver accumulated during the summer of 1932.

Water was pumped from a 12-foot well by a 2 1/2-inch centrifugal pump to a small tank, whence it was pumped 7,800 feet horizontally and 280 feet vertically by a 5 1/2- by 6-inch triplex pump to a 18,000-gallon tank at the plant. The same engine drove both pumps. About 100 gallons of water was used per minute; none was reclaimed.

The plant had a capacity of 120 cubic yards per day when it ran full time; however, the average daily yardage handled during the period of operation was about 90. Power was furnished by tractor engines; 1 was used on the shovel, 2 at the washing plant, and 1 on the pumps. Gasoline consumption per 8 hours was: Shovel, 10 gallons; trucks, 16; washing plant, 13, and pumps, 10. Gasoline cost 19 cents per gallon, but the State tax of 5 cents per gallon was refunded for all uses except the truck. The average daily cost for gasoline was $7.99.

Six men and a superintendent ran the plant. The wage scale was $0.75 per hour for the shovel men and an average of $0.40 per hour for the rest of the crew. The daily labor payroll was $22. Other daily operating expenses were estimated as follows: Trucking costs other than labor and gasoline, $2.70 (6 cents per mile); shovel, washing plant, and pump supplies and miscellaneous expense, $6.00; supervision, $10.00; making a total daily operating cost of $48.69. At the average daily capacity of 90 cubic yards the unit cost would be 54 cents; at full capacity the operating cost would be 40 cents per cubic yard.

Forbach and Easton.- Forbach and Easton worked a placer mine in Skull Valley, near Kirkland, Ariz., in 1931 and 1932. The gravel was 6 inches to 6 feet deep, the average depth being 3 feet. According to one preliminary test the gravel contained about 75 pounds of black sand per cubic yard, but in operation the table recovery indicated a content of about 35 pounds per yard. There were no boulders. The gold-bearing gravel was underlain by a false bedrock of yellow clay. The gravel was dug by a 3/8-cubic-yard, caterpillar-tread, gasoline-driven shovel and loaded into two trucks 1 3/4 and 2 3/4 cubic yards in capacity, respectively. The average haul to the washing plant was 1/2 mile. The gravel was dumped from the trucks over a 6-inch rail grizzly into a 55-cubic-yard bin. The oversize was thrown out by hand into a car and pushed to a dump. From the bin the gravel was taken by a belt feeder to a 4- by 9-foot trommel. The first 3 feet of the trommel was blank; the lower 6 feet had 1/4-inch round holes. This trommel was set on a slope of about three eighths inch per foot and revolved at a rate of 21 r.p.m. Sprays under approximately a 30-foot head played on the outside of the trommel and into both ends. The undersize went through a nugget trap, in which it was reported $10 to $50 worth of coarse gold was recovered each shift, and thence to a second trommel 38 inches in diameter by 6 feet long, made of double-crimped, 18-gage screen wire with 1/8-inch square openings. The oversize from both trommels was carried to a tailings dump by two 16-inch conveyor belts, the first 50 and the second 150 feet long. It was estimated that about 5 percent of the material was rejected at the grizzly and an additional 35 percent by the screens. The undersize from the second screen went to three Deister diagonal-deck sand tables arranged in parallel and run at 280 r.p.m. The table tailings went to a 4-foot sand wheel with 11 buckets bolted to the rim; it turned about 3 r.p.m. The sand from the wheel went to the second stacking belt which elevated the material about 30 feet to the dump. A hose was used periodically for washing the sand away from the end of the stacker. The Deister table concentrate went to a Wilfley table used as a cleaner. The tailings from the Wilfley table were carried back by hand to the head of the Deister tables. The concentrate in 2 1/2-ton batches was treated in a 5-foot amalgamator. The amalgamator was first run about one half hour to scour the gold; caustic soda was added to destroy any grease that might be present. Then 40 pounds of quicksilver and enough water to fill the tank were added, and the amalgamator was run 30 minutes more at 57 r.p.m., after which the top part of the charge was drawn off and put back over a table to catch any floured mercury. The rest of the contents were then drawn into a porcelain tub; the live quicksilver was decanted off and as much amalgam as could be collected taken out. The contents of the tub were run over the Wilfley table to catch the rest of the amalgam. The amalgam was squeezed in buckskin and then retorted over a wood fire; a condenser was used. The sponge gold was melted in a graphite crucible.

The trommels were driven by a 50-hp. gasoline engine that also operated a 10-kw. generator that supplied power to a motor to run the no. 2 stacking belt and the Wilfley table. A 9-hp., 1-cylinder gas engine drove the upper stacker, the Deister table, and the sand wheel.

The well from which water for operating the plant was obtained was 8,000 feet from the mine; it was cased with 8-inch pipe for a distance of 170 feet. A deep-well pump driven by a 25-hp. Diesel engine lifted the water 150 feet to a 2,000-gallon tank. A 5 by 6 triplex pump, run by the same engine, forced the water through a 4-inch pipe to 25,000-gallon tanks at the washing plant. The water from the washing plant was settled in a pond, and the clear water was returned to the tanks by a 2-inch centrifugal pump.

Ninety gallons was pumped per minute from the well over a period of 8 hours. About 250 gallons of water per minute was recirculated at the plant, and some was wasted. Clear water was used on the tables. About 175 gallons per minute was used for screening.

The mine was operated one 8-hour shift. The crew consisted of the following:

	Wages, rate per hour
1 shovel runner	$0.75
2 truck drivers	.40
3 plant men (1 at grizzly, 1 at feeder, and 1 at tables) each	.40
1 pumpman	.40
1 sampler (testin.. ground)	.40
1 superintendent	

The fuel consumption was as follows:

	Gasoline	
	Gallons per shift	Cost per gallon
Shovel	15	$0.14
Trucks (2)	10	.19
Washing plant	30	.14
Main pump	[1]16	.17
Return pump	15	.14

[1]Distillate.

The plant had a capacity of 200 cubic yards per day when it ran full time; however, delays due to break-downs and adjustments reduced the average to about 120 cubic yards per shift.

The daily cost was as follows:

Labor	$28.40
Gasoline and distillate	13.02
Trucking (oil, tires, and miscellaneous costs, excluding depreciation) at 8 cents per mile	4.80
Shovel, washing plant, and pump supplies and miscellaneous supplies	7.50
Supervision	10.00
Total	63.72

At 120 cubic yards per shift the unit operating cost was 54 cents. If the plant was operated at full capacity the cost would be 31 cents per cubic yard.

Morning Glory.- The Morning Glory Placers, Inc., operated on Upper Lynx Creek near Prescott, Ariz., in July 1932. The gravel, which was about 8 feet deep, was excavated by a 1-cubic-yard, gasoline, full-revolving, caterpillar-tread drag line and hauled in two 2-cubic-yard dump trucks about 1,600 feet to a washing plant. The capacity of the plant was expected to be 300 cubic yards in 8 hours. The contract price for excavating the gravel, delivering it to the washing plant, and cleaning up bedrock was 19 cents per cubic yard. The gravel was dumped through a 6-inch grizzly; about 5 percent of the material was discarded at this point. The gravel then went through a 4 1/4- by 14-foot trommel with 5/8-inch holes. Six hundred gallons of water per minute was used under a 50-pound-per-square-inch pressure in the trommel. About 50 percent of the material was discarded at the trommel and passed onto a stacker belt 15 feet long. The undersize went through two 12-inch boxes 45 feet long in parallel, having various types of riffles. The clean-up material was put through a pulsating jig to separate the gold. About 1,000 pounds of concentrate was obtained from 900 cubic yards of gravel. The jig concentrate was treated in a pan amalgamator.

The automobile engine to run the washing plant took 10 gallons of gasoline every 8 hours. The water was pumped with a 6-inch centrifugal pump run by another automobile engine. The water was reclaimed in a pond and recirculated. Insufficient work had been done to permit calculating the total operating costs.

Octave.- A plant was built in 1931 and run a short time at Octave, Ariz. The gravel was excavated by a power shovel with a 1-cubic-yard dipper and trucked to a 10-cubic-yard car on an inclined track. The car was pulled to a washing plant and dumped into a hopper.

Batches were fed into and washed in a large concrete mixer, then run through sluice boxes. The use of the concrete mixer for washing the gravel was novel.

Mystic[21] - The Mineral Leasing Co. of Rapid City, S. Dak., erected a washing plant with side-shaking sluices on Castle Creek at Mystic in the central Black Hills of South Dakota in 1932. The gravel which occurred in a side gulch was 30 feet thick and covered with a few feet of soil. It was excavated by a power shovel with a 1/3-cubic-yard dipper; the pay dirt was transported by motor truck to a treatment plant a short distance away on Castle Creek.

The gravel was dumped upon a screen with openings 7 inches square constructed of old boiler tubes. The material passing through this coarse screen dropped into a bin with a capacity of 5 cubic yards, from which it ran by gravity to a 22- by 34-inch reciprocating pan feeder and thence to a trommel. The latter was 6 feet in length, set at a slope of 1 inch to 1 foot, and ran at 21 r.p.m. It was composed of two concentric wire-mesh screens; the inner one was 2 feet in diameter with 1-inch square openings and the outer one was 42 inches in diameter and was made of screen cloth with 1/8- by 5/8-inch rectangular openings. Spray water for the screen was provided by a centrifugal pump. A 50-foot stacker belt raised the oversize from the screen to an elevated hopper whence it was drawn into a motor truck for transportation to a dump. The undersize from the revolving screen ran into a shaking sluice 48 inches wide, 52 inches long, and hung on a grade of 1 1/2 inches to the foot. The sluice was shaken sideways by an eccentric at the rate of 100 strokes per minute; the throw was 1 inch. Transverse riffles were used. The tailings from the shaking sluice ran through a fixed sluice 25 inches wide with a grade of 1 inch to 1 foot. The shaking sluice had not proved satisfactory at this plant due to the large proportion of black sand in the gravel and the high rate of feed, and it was proposed to discontinue its use and substitute fixed sluices and Wilfley tables. The material from the sluice ran into a tailings ditch beside Castle Creek.

The treatment plant was run by a tractor engine and the stacker belt by a separate gasoline engine. The plant had a reported capacity of 300 cubic yards per 10-hour day, and the cost per cubic yard was said to be about 25 cents.

Britten and Murphy.- Britten and Murphy mined gravel from a side gulch of Goler Gulch near Randsburg, Calif., in the summer of 1932.

The gravel was dug by a gasoline-power shovel and trucked uphill about 1,500 feet to a mill built on the hillside. The gravel was dumped over a 4 1/2-inch grizzly. The oversize of the grizzly fell down the steep hillside. The undersize passed through a 5-foot concrete-mixer shell run as a pebble mill to wash the gravel. It then ran through a 3 1/2- by 12-foot trommel which made two screen products - minus 1/16 inch and plus 1/16 minus 1/2 inch. The coarser material ran through an iron sluice box with riffles and the finer product over a Deister-Overstrom table.

At the start water was purchased for $1 per 1,000 gallons. Tests indicated that a cubic yard of gravel after washing would retain 60 gallons of water. The novel feature in the plant was the use of a concrete-mixer shell as a washer ahead of the trommel.

Movable washing plants

Haag.- In June 1932 the Haag Mining Co. was mining placer gravel with a power shovel and washing it in a portable plant on Benson Gulch west of Randsburg, Calif., under very adverse conditions. The gravel was tight and clay-bound and contained a large proportion of boulders A part of the gravel was cemented so that it could not be dug with the shovel in use. The shovel was full-revolving and had a caterpillar tread and a 7/8-cubic-yard dipper. In the

21 Reported by F. C. Lincoln, South Dakota School of Mines, Rapid City, S. Dak.

part of the channel being worked the shovel could dig 200 to 300 cubic yards in two 8-hour shifts, which was far in excess of what the washing plant could handle. The shovel, including labor and supplies, was rented for $36 per 8-hour or $55 per 16-hour day. It used 40 gallons of gasoline in 8 hours when operating steadily.

The gravel was dumped on a grizzly with bars spaced 6 inches apart. The oversize was thrown to one side by hand. The undersize went into a hopper from which it was fed into a double trommel; the inner screen had 3/4-inch and the outside one 5/16-inch holes. All plus 3/4-inch material was taken from the trommel by a belt stacker to a dump. The minus 3/4- and plus 5/16-inch material passed separately through an 18-foot sluice 30 inches wide to catch nuggets that might be contained in the gravel. The tailings from the end of the sluice were discharged onto a main belt stacker by means of an auxiliary bucket elevator. The minus 5/16-inch material ran through a 40-foot sluice 30 inches wide on the other side of the machine. Both sluices were floored with solid-rubber matting with transverse riffles 3/8 inch wide and 1/2 inch deep, spaced 1/2 inch apart. The riffles were kept about one half full of quicksilver. On the coarse-gold side one 2-foot section had every other riffle cut out to accommodate any large particles of gold. The washing plant was self-contained. It was propelled by means of a tractor over which it was built, and the trommel and stacker were run from the tractor engine. Water was purchased from wells in the valley near Goler at $1 per 1,000 gallons and delivered through a pipe line into a tank above the workings. The water, after going through the plant, was impounded and pumped back for re-use. About 180 gallons of new water was used per cubic yard of material washed. This high consumption was due mainly to losses from the impounding pond and evaporation; probably not more than 60 gallons per cubic yard was held in the tailings. The washing plant had a capacity of about 100 cubic yards per 8 hours, but due to delays caused by break-downs and moving, the yardage handled per shift was reduced to less than one half of this quantity. The trommel did not disintegrate the gravel thoroughly, and 15 to 20 percent of the oversize discharge was cemented material that contained gold. Although the plant probably would have proved satisfactory in material easier to handle, it was not strong enough for the conditions at this place and had insufficient disintegrating capacity to free the gold thoroughly from the gravel.

One man was emloyed at the hopper, one at the pumps at the impounding dam, and two on the washer each shift. One man operated the shovel. The total operating cost of handling 100 cubic yards of gravel per day was $1.30 per cubic yard. (See table 5.) The segregated costs are as follows:

	Daily operating cost	Cost per cubic yard
Shovel hire (2 shifts)	$55.00	$0.55
Washing plant labor (8 shifts at $4)	32.00	.32
Gasoline for washer circulating pumps, 30 gallons at 20 cents	6.00	.06
Water, 180 gallons per cubic yard at $0.001	18.00	.18
Other supplies (estimated)	4.00	.04
Miscellaneous	10.00	.10
Supervision	5.00	.05
Total	130.00	1.30

Triangle.- The Triangle Gold Mining Co. was making a placer cut at Therma, N. Mex., in July 1932. The gravel was dug by a 1 1/4-cubic-yard, Diesel-driven power shovel with a 28-foot boom. The washing plant was mounted on skids and consisted of a trommel 6 feet long with 3/4-inch holes and a line of sluice boxes on a trestle. The boxes had a grade of 8 inches to 12 feet. As the trommel was moved up section lengths of boxes were put in. Six hundred gallons per minute or 53 miner's inches of water under a 30-pound head was used for washing the gravel. The oversize from the trommel was hauled away in two 2-ton trucks. As the cut was extended not enough grade was available to carry the sluice out of the cut for the disposal of tailings; difficulty was encountered in keeping the roadway open for hauling away the oversize material and providing drainage in the cut. The plant had not been operating long enough for operating costs to be estimated.

Holland.- In May and June 1932 the Gold Mining Co. used a drag line to mine a placer deposit near Holland, Oreg. The gravel was 25 to 30 feet deep, and the pay channel was 40 feet wide along a present stream course. The dragline, which had a 40-foot boom and a 1 3/4-yard bucket, stood on the edge of the bank at the face of the workings and loaded into a patented, movable washing plant on one side of the pit and on the original ground surface.

The gravel was dumped into a hopper on the washer. From the hopper it was fed onto a traveling chain grizzly with 3- by 5-inch holes. The oversize discharged onto a 30-inch, rubber belt conveyor that discharged onto a second belt conveyor which elevated the material 50 feet to a dump. A wooden beam placed over the first conveyor diverted all boulders over 10 inches in diameter through a chute back into the pit. The undersize from the first grizzly dropped to a second traveling grizzly with 7/8- by 1 1/2-inch openings. The oversize joined that from the first grizzly on the belt conveyor. The sands were run through sluice boxes. An area 5 by 10 feet under the grizzlies and 33 feet of sluice boxes was lined with brussels carpets held down by heavy wire screen with 3/4- by 3-inch openings. Most of the gold from the excavated material was caught in the space under the grizzly. The sluice consisted of two 30-inch boxes in parallel for 12 feet, joined for the next 8 feet in one box 60 inches wide and separated again for an additional 10 feet.

In cleaning up, the carpets first were shaken in a tub which removed about 75 percent of the gold. Then they were placed in a regular cylindrical laundry machine where the remainder of the gold was washed out.

The plant was on skids and was moved up about 20 feet every other day by the dragline. The gravel was easily dug and readily washed. The dragline was said to have a capacity of 400 cubic yards per 8-hour shift, but the washing plant could handle only a part of this yardage. The gravel was cleaned readily by the sprays while on the traveling grizzly. Where the gravel, however, was tight or contained appreciable clay, as at many places, this type of washer would not be satisfactory.

The plant was run by two motors of 8 and 20 hp., respectively. One and four tenths gallons of gasoline was used per hour. Three men worked each shift on the plant and in the pit; one man operated the dragline.

The bedrock was hard and could not be cleaned by means of the dragline bucket. After all material that could be excavated by the drag line had been removed from a section of bedrock it was cleaned with a giant. A line of sluice boxes was maintained in the pit for washing the clean-up material. Previously the mine had been worked by giants; the equipment included two hydraulic elevators. Then the material from the sluice boxes in the pit was discharged into the intake of one of the hydraulic elevators and thence elevated 31 feet into a second sluice. The head of water on giant and elevator was 160 feet. The necessity for maintaining the sluice in the pit prevented dumping the gravel washed in the plant directly back into the excavation. Along one side of the pit a dry wall was built of the boulders chuted into the pit from the washer. Operations were said to be unprofitable, mainly because it was difficult to dispose of the tailings and because it was necessary to keep the pit open and drained.

Fontana.- The Fontana Washington Gulch Mining Co. mined a cut 350 feet long and 40 feet wide on Washington Gulch near Deerlodge, Mont., in 1931. The gold was contained in 2 1/2 to 3 feet of gravel on a clay bedrock. The gravel was overlain by 2 feet of loam and 6 feet of old tailings from workings farther up the creek.

The gravel was first stripped by a 75-hp., gasoline-driven dragline with a 90-foot boom and a 1 1/4-cubic-yard Page bucket. The overburden was first piled on one side of the cut; then the dragline was taken into the pit and the pay gravel dug and loaded into a movable washing plant on the other side of the cut. The gravel after going through a 12-inch grizzly was fed into a 4- by 16-foot trommel with 3/4- by 1 7/8-inch rectangular holes. The oversize from the trommel was discharged into the pit by a 28-inch belt conveyor 25 feet long. The undersize was divided into two streams and run through two 24-inch steel boxes 30 feet long with Hungarian riffles. The sluices also discharged into the pit. A drainage trench was dug in the middle of the pit by the dragline and lagged over with slabs. The washing plant used 1,000 gallons per minute, or 88 miner's inches, of water. The pumping plant consisted of a 6-inch centrifugal pump and a 45-hp. gasoline engine. The water was pumped through an 8-inch pipe line, from which connection was made to the washing plant by a 6-inch hose. The plant could handle 350 cubic yards in 8 hours when running full time. The dragline required 38 gallons, the pump 18, and the trommel and stacker engine 3 gallons of gasoline per 8 hours. Seven men were employed. The dragline was brought out of the pit to move the washing plant. Cost data are not available.

Driskill.[22] - A portable washing plant was operated on Bear Creek in the northern Black Hills of South Dakota during 1931 and 1932 by the Driskill Mining Co. of Spearfish. A 1 1/4-cubic-yard gasoline shovel stripped the overburden, mined the pay dirt, and moved the treatment plant. The pay dirt was delivered by the shovel to a flat grizzly 6 feet square with 8-inch square openings consisting of mine rails placed across pieces of drill steel. The oversize was thrown off the grizzly by hand; the undersize fell into a hopper from which it was fed upon an 18-inch belt conveyor 30 feet long and inclined at an angle of 30°. The elevator was driven by a 4-cylinder automobile engine which also turned a trommel 30 inches in diameter and 8 feet long into which the elevator dropped its load. The trommel had 5/8-inch square openings and was fitted with a 2-inch central spray pipe. The oversize from the revolving screen ran through a chute to the waste dump; the undersize was delivered to a sluice 24 inches wide and 50 feet long which zigzagged back and forth beneath the screen. The sluice was divided through its center into two equal compartments and provided with a two-way gate at the top so that one compartment could be used while the other was being cleaned. Each side of the sluice was fitted with transverse riffles made of 2-inch iron bars held at 8-inch intervals by strap iron. For convenience in cleaning, each compartment had a side chute at the lower end through which the concentrates could be drawn into a barrel for further concentration in a Richards pulsator jig with 4- by 14-inch screen. The sluice tender changed compartments and cleaned up every half hour or so, depending upon the richness of the ground. The concentrates contained values not only in gold, which ranged in size from fine colors to nuggets, but also in cassiterite, tantalite-columbite, and scheelite.

The tailings from the sluice were collected in a settling box from which the sand was removed to a tailings stacker by means of a flight conveyor composed of a belt with 2-inch angle-iron bars fastened across it at 1-foot intervals. The tailings stacker was driven by a 3-hp. gasoline engine.

22 Reported by F. C. Lincoln, South Dakota School of Mines, Rapid City, S. Dak.

The water supply for the washing plant was obtained from a sump in the creek below the plant into which the tailings water and the creek water ran. This water was pumped through a 3-inch pipe to the spray pipe by a rotary oil pump at the rate of 200 gallons per minute (18 miner's inches); the pump was belted to the power take-off of a 15-hp. tractor.

The plant was reported to have a capacity of about 300 cubic yards per 10-hour day. The operating cost was about $25 for wages, $10 for fuel, $5 for miscellaneous, and $20 for overhead - a total of $60 per day. The average daily yardage treated was not available for publication.

Overpeck.[23] - In June 1932 mining was begun by the Mines Royalty Co. on Spring Creek above Sheridan, S. Dak., where a movable washing plant had been installed. Operations ceased September 17. The gravel was 8 to 25 feet thick and was covered with 1 to 7 feet of soil. The gold occurred mainly as coarse, flat colors and small, flat nuggets concentrated upon and in bedrock. From 2 to 8 feet of the gravel above bedrock contained fine colors, and higher layers of gravel occasionally were auriferous. Bedrock was a decomposed schist which was easily dug. The overburden was removed by a gasoline-driven drag line with a 3/4-cubic-yard bucket. The machine was then converted into a shovel with a 1-cubic-yard dipper for mining the gold-bearing gravel and bedrock. The shovel was used to move the washing plant which was mounted on two log skids 27 feet long and 11 feet apart. In moving the plant a cable fastened to the back of the shovel was run back through a snatch block on the plant, thence to the head of the shovel where it was attached to the dipper by means of a loop. The dipper was moved forward to pull up the plant.

The gravel was dumped at the top of the plant on a grizzly with 6-inch spacings between the bars set on an inclination of 15°. The oversize was thrown from the grizzly by hand; the undersize was fed by means of a 20- by 36-inch reciprocating pan feeder to a 24-inch inclined belt 12 feet long set at an angle of 23°, which delivered the material to an 8-foot revolving screen. The trommel was set at an inclination of 1 inch to 1 foot and made 21 r.p.m. It consisted of two concentric wire-mesh screens, the inner one having a diameter of 3 feet and openings 1 inch square and the outer one a diameter of 52 inches and 1/8- by 5/8-inch openings. A spray pipe supplied water to the interior of the revolving screen, the water being pumped by a 3-inch centrifugal pump. The oversize from the screen dropped to an 18-inch stacker belt 14 feet long. The belt, however, was too short; experience at this plant indicated that the belt should have been at least 25 feet long to dispose properly of the oversize. The undersize from the screen went to a shaking sluice which was supported by four hangers and given a sidewise reciprocating motion by an eccentric. The sluice was inclined 1 1/2 inches to 1 foot and given a side shake of 1 inch 100 times per minute. It was 6 feet long and 56 inches wide and had 1-foot sides sloping outward at 30°. Upon the bottom of the sluice were six transverse wooden riffles 1 inch high and 9 inches apart. A satisfactory saving was not made in the sluice, probably because of the difficulty of keeping the plant level.

A 4-inch centrifugal pump removed seepage water from the bedrock; the tailings were pumped away by a 4-inch sand pump.

The crew consisted of 6 men, of whom 1 acted as boss. From the time operations began in June, to September 1, an average of about 40 cubic yards was handled per 10-hour shift. During 2 weeks in September about 300 cubic yards was handled each shift. The small yardage handled most of the time was due to delays in making alterations and repairs to the washing plant. The shovel had a possible capacity of 500 cubic yards per shift.

La Cholla.- The La Cholla Mining Co. began placer-mining operations in the summer of 1932 about 5 miles from Quartzsite, Ariz. The gravel was dug with a full-revolving, gasoline-

23 Reported by F. C. Lincoln, South Dakota School of Mines, Rapid City, S. Dak.

powered drag line equipped with caterpillar treads, an 85-foot boom, and a 2 1/2-cubic-yard bucket. The gravel was washed in a self-contained movable machine; the oversize was screened out, and the tailings were dewatered in a tank on the machine. The tailings with the oversize were sent to a dump by a belt stacker. Water was hauled by truck from Quartzsite and cost, at the mine, $1 per 1,000 gallons.

The gravel was excavated to a depth of 10 to 12 feet and was said to be gold-bearing for a width of 300 feet. Most of the wash was angular in shape. The gravel was dumped into a 15-cubic-yard hopper through a grizzly with 10-inch square openings. Very few oversize boulders had to be thrown off the grizzly. The dry gravel from the hopper was fed over a rail grizzly with 3-inch spacing by means of a shaking feeder. The oversize went onto a belt conveyor to a belt stacker, where it was elevated to the tailings pile. The minus 3-inch material was elevated 30 feet by a bucket elevator to the front end of the trommel. There the gravel was picked up by a jet of water issuing from a 5-inch nozzle under a 100-pound head and thrown violently against a baffle plate at the far end of the trommel to disintegrate all friable material. The trommel was 8 feet long and 5 1/2 feet in diameter, with 3/4-inch round, punched holes. The oversize from this screen joined the plus 3-inch material on the stacker belt. The undersize and water went over a sluice consisting of two parallel, 14-inch boxes 40 feet long. Riffles were made in sections, each consisting of four 2-inch pipes, 36 inches long, spaced about an inch apart and welded at the ends to 14-inch lengths of 2 1/2- by 2 1/2-inch angle iron. From the sluice the tailings went into a tank holding about 40 tons of water and 35 tons of gravel and sand. The tailings were removed from the tank by a drag, such as used in drag classifiers. The drag ran at a very low speed and was held a few inches off the bottom of the tank by rails to let any gold coming through the sluice settle on the bottom. The tailings were discharged on the stacker. The stacker belt was 24 inches wide and 85 feet long. It operated on a boom that could be swung through an arc of 180°. Up to the first of August 1932 the plant had not been run on regular production; 3,600 cubic yards (bank measure) had been treated in a trial run with a consumption of 23 1/2 gallons of water per cubic yard of gravel. The dewatered tailing (25 percent of the total material moved) contained 15 to 40 percentof moisture by weight, depending upon the quantity of sand and clay present.

The machine was moved on skids with the dragline acting as a tractor. About 1 1/2 hours was required to move it 100 feet and re-level it by means of wedges placed under the rollers. An average of 47 cubic yards per hour was handled during the trial run. Improvements were under way to increase the capacity.

If the installation could have operated full time at the rate of 50 cubic yards per hour the following operating costs would be indicated:

Excavating:		
Labor (1 man at $0.75 per hour and 1 man at $0.45 per hour)	$0.023	
Gasoline (7 gallons per hour at $0.17 per gallon)	.024	
Other supplies	.020	$0.067
Washing plant:		
Labor (3 men at $0.45 per hour)	.027	
Gasoline (7 1/2 gallons per hour at $0.17 per gallon)	.026	
Other supplies	.024	.077
Water at $0.001 per gallon (24 gallons per cubic yard)		.024
Miscellaneous (including major repairs)		.050
Supervision		.025
Total		0.243

The above cost does not include interest on the investment, amortization, and costs of remodeling plant. High remodeling costs would be expected on any new design of such a large and elaborate installation.

Bemrose.- The Buffalo Exploration Co. was washing gravel in a patented gold saver near Breckenridge, Colo., in June and July 1932. The gravel was about 14 feet deep, contained relatively few boulders, not much clay, and only a very small quantity of black sand; it was easily dug. Most of the gold was relatively coarse, although some fine gold was said to be recovered.

The gravel was excavated by a 60-hp., caterpillar-tread, gasoline shovel with a 5/8-cubic-yard dipper. It was dumped into a hopper in front of the washing machine, where boulders were picked out, then elevated by a 36-inch belt conveyor to a double trommel 14 feet long. The inside trommel was 36 inches in diameter; the first 4 feet was blank, and the lower 10 feet had 1 1/4-inch round holes. The gravel was disintegrated and washed in the blank section of this trommel by high-pressure sprays (160 pounds per square inch). The outside trommel was 48 inches in diameter and had 1/8-inch holes, except for the last foot which had slots 1/4 by 3/4 inch in size. The trommel oversize was stacked by a 36-inch belt conveyor. The material going through the slots went over riffles in a special 4-inch box to catch any nuggets that it might contain. The balance of the undersize from the trommel went to four bowl-shaped, centrifugal concentrators. The feed was delivered through pipes to the bottom of the bowls; the tailings were discharged over the rims. A series of corrugations or riffles on the inside of the bowls caught the gold. The bowls weighed 1,200 pounds each and were driven at 100 r.p.m. The four bowls were cleaned up at the end of each shift in about one half hour. First they were run a few minutes with the feed cut off; then, after the power was turned off, a plug at the bottom was pulled, and the contents of the riffles were washed into a tub placed underneath. The tailings from the plant were run to waste through a flume when dump room was available at the machine; otherwise they were removed farther by means of a no. 5 sand pump. The washing plant ran on a track and was moved forward by the shovel. About 200 gallons of water was used per minute. It was brought to the machine through a pipe line under a 400-foot head.

The shovel had a capacity of about 40 cubic yards per hour and could deliver gravel faster than the plant could wash it. About 200 cubic yards was washed per day when the plant was operating; delays, however, were numerous; the machine had been practically redesigned and partly rebuilt after being placed on the ground. One man ran the shovel and two men, of whom one was the superintendent, ran the washing plant. On the basis of a 200-cubic-yard daily production, the labor cost, including superintendence, would be 7.5 cents per cubic yard. The shovel and washing plant each used 15 gallons of gasoline per 10-hour shift; at 22 cents per gallon the cost of fuel would be 3.3 cents per cubic yard. Other supplies would amount to 2.7 cents and current replacements to about 4 cents, making a total operating cost of 17.5 cents per cubic yard. If a rental of $36 per 10-hour shift was paid for the shovel the cost of excavation would be 18 cents per cubic yard; if the shovel was operating at full capacity, however, the cost would be 9 cents. Remodeling and construction costs, which are charged to amortization, apparently would be very high. The washing plant was said to have cost $25,000 to install.

Grand Hills.[24] - In June 1932 the Grand Hills Mining Co. erected a washing plant equipped with four centrifugal-bowl gold separators on French Creek 2 miles west of Custer, S. Dak. The gravel where the plant was erected was 5 to 8 feet deep and was covered with 5 to 8 feet of soil overburden. The gold particles were small but not greatly flattened, so they concentrated readily.

24 Reported by F. C. Lincoln, South Dakota School of Mines, Rapid City, S. Dak.

The gravel was excavated with a 1 1/4-cubic-yard, 110-hp., gasoline shovel with caterpillar tread, which also stripped the overburden and moved the treatment plant. The plant was mounted on two 4-wheel trucks running on a 7-foot gage track. The gravel was dumped onto a grizzly with a 45° slope and 10-inch spaces between bars, which was set above a 5- by 5-foot steel hopper; from there the gravel was fed by a 2- by 5-foot reciprocating pan feeder onto a belt elevator 24 inches wide and 40 feet long. The elevator transported the material to a semicircular hopper at the top of the treatment plant.

From the hopper the gravel went into a trommel 60 inches in diameter and 21 feet long. A spray pipe which extended through the center of the revolving screen for its entire length supplied all the water required by the treatment plant. The trommel had a grade of 1 1/2 inches in 1 foot and made 11 r.p.m. It consisted of a blank scrubbing section 4 feet long, an 8-foot section perforated with 1/4-inch round holes, a 4-foot section with 3/8-inch round holes, and a 2-foot section with 1/2- by 1-inch slots. The oversize material was delivered to a 30-inch stacker belt 45 feet long which piled it upon bedrock behind the plant. The undersize from the screen, except that from the slotted section, was divided into four parts and fed to four 36-inch, centrifugal-bowl gold separators like those used at the Queen and Bemrose placers previously described. The coarse gravel which passed the 1/2-inch slots in the revolving screen was concentrated in a nugget sluice 5 feet long, 6 inches wide, and 4 inches deep. This sluice was set at a grade of 2 inches to 1 foot and provided with transverse riffles 1 inch in height set 1 1/2 inches apart. The tailings from the bowls and the nugget sluice, flowed into a 15-foot settling tank 28 inches wide and 24 inches deep with its rear end inclined upwards. A flight conveyor with 18- by 6-inch steel blades attached to chains at 20-inch intervals scraped the settled gravel up the incline of the settling tank and delivered it to the stacker belt. Sand, soil, and excess water were pumped by a 4-inch centrifugal pump from the settling tank into settling ponds behind the tailings piles.

The water supply for the treatment plant, consisting of 250 gallons per minute (22 miner's inches), was obtained from a sump in the creek bed. It was pumped into the spray pipe of the trommel by a 4-inch centrifugal pump driven by a 4-cylinder, 14-hp. gasoline engine. The pump and engine were mounted upon a steel frame provided with wheels. Of this supply only about 50 gallons per minute was fresh water from the creek; the other 200 gallons was returned to the sump through ditches connected with the tailings ponds.

The following engines were used on the washing plant:

	Hp.
1. Driving screen, sand drag, tailings pump, and four bowls	30
2. Driving tailings stacker	12
3. Driving belt feeder and pan feeder	14

The bowls were cleaned once a day and the nugget sluice twice a month. The concentrates were panned to recover the gold. Practically no gold was found when the settling tank was cleaned, and no gold had been found on panning the tailings, which indicated that the recovery was very high. The entire equipment cost about $40,000.

The water supply was sufficient for running the plant only one 10-hour shift per day. An average of 400 cubic yards was handled daily.[25]

The labor was as follows: 3 men on the washing plant at 30 cents per hour, 1 shovel operator at 60 cents per hour, and 1 superintendent at 60 cents per hour. The gasoline

25 Cost and operation data were supplied by C. E. Gish, manager, Grand Hills Mining Co., Denver, Colo.

consumption was 100 gallons per shift. The total operating cost per cubic yard was as follows:

	Cents
Labor (including supervision)	5 1/2
Gasoline	4
Miscellaneous repairs	3
Compensation insurance, depreciation, and miscellaneous expense	3 1/2
Total	16

Two months after starting the plant just described, the company installed a second on French Creek about 1 mile above the first. The second plant was essentially the same as the first, except that a 3/4-cubic-yard gasoline shovel was used in place of the one with a 1 1/4-cubic-yard dipper. There was little overburden at the second location, but the gravel contained a large proportion of boulders over 10 inches in diameter. The plant had a capacity of 50 to 75 cubic yards per hour with a gasoline consumption of about 10 gallons per hour. It was operated by four men with a part-time superintendent.

No operating data are available for this plant.

Gold Dust.- The Placer Syndicate operated the Gold Dust placer near Hillsboro, N. Mex., during part of 1932. The gravel was excavated by a 5/8-cubic-yard, gasoline-driven shovel with a 60-hp. engine and treated in a portable washing plant containing four 36-inch centrifugal-bowl gold separators. The plant was constructed of steel and mounted on four wheels that ran on wooden stringers. It was moved ahead by jacks. The gravel was dumped into a hopper from which it was carried by an inclined feeder belt to a trommel with three concentric screens. The inside screen was 2 feet 4 inches in diameter and had 2-inch round holes; the next screen was 3 feet 5 inches in diameter and had 3/4-inch round holes; the outer screen was 4 feet 4 inches in diameter and had slots 1/4 by 1 inch in size. A housing around the outer screen acted as a further scrubber. Water under 70 pounds pressure was used in the sprays. The trommel was 12 feet long and had a slope of 3/4 inch to 1 foot. The oversize from the trommel went onto a belt stacker. The undersize was divided and fed to the four centrifugal separators. The tailings from the bowls were elevated by a sand pump to a settling pond which also was used as a reservoir. Plans were made for putting a dewatering tank on the machine and discharging the sands on the stacker belt. The machine was driven by a 60-hp. gasoline engine. Water was pumped from a well by a 7-inch centrifugal pump run by a 50-hp. gasoline engine; the pump capacity was 400 gallons per minute.

At the time the authors visited the plant it was shut down and had not run long enough for operating costs to be indicated.

The crew consisted of 1 man on the power shovel, 1 man panning and testing the gravel, and 4 men on the washing plant. The engines on the shovel and plant each used 25 gallons and the one on the pump 21 gallons of gasoline per 8 hours. The capacity was expected to be about 1,000 cubic yards per three 8-hour shifts.

The following table gives the approximate cost of the machinery and of equipping the mine.[26]

26 Data supplied by H. S. Coulter, general manager of the Placer Syndicate Mining Co., Hillsboro, N. Mex.

Item	Cost
Four bowl washing plant, f.o.b. Denver, Colo.	$15,180.00
Dewatering equipment, f.o.b. Denver, Colo.	1,550.00
Freight and trucking, washing and dewatering plants	700.00
Rebuilt 5/8-cubic-yard power shovel	6,500.00
Freight on shovel	300.00
Seven-inch water well pump, 400 gallons per minute capacity	700.00
Fifty-horsepower engine to run pump installed	750.00
Freight, trucking, and installing pump	150.00
Pipe, 6,000 feet of 4-inch, used	1,500.00
Hose, 100 feet of 4-inch, new	75.00
Drilling well	400.00
General camp buildings	1,000.00
Miscellaneous expenses and equipment	2,500.00
Total	31,305.00

Floating washing plants

Kumle.- During the summer of 1932 H. T. Kumle was operating a placer property near Oregon House, Calif. The gravel was about 15 feet deep along the course of a present stream. It was excavated by a standard 3/4-cubic-yard, full-revolving steam shovel with caterpillar treads. Coal at $14 per ton and a daily cost of $11.40 was used for fuel. The shovel worked in 3 feet of water and dumped the gravel directly into a washing plant floating in a pond maintained for the purpose. The draft of the plant was 2 feet. A rise of 6 inches in the water level would drown the shovel, and operations would have to cease until the water level went down. The shovel discharged the gravel onto the upper ends of a grizzly 12 feet long with one half inch between bars. The grizzly had a slope of 6 3/4 inches in 12 feet, which was to be increased. High-pressure sprays washed the oversize to the bottom of the grizzly and onto a 30-inch rubber belt 40 feet long which stacked it in tailings piles to 12 feet above water level. The undersize was divided into two parts and ran through boxes 27 inches wide and 30 feet long on either side of the float. Standard dredge-type Hungarian riffles were used in the boxes. Quicksilver was used in the riffles; a quicksilver trap was installed 6 feet from the end of each sluice. The method of operation was patterned after dredging practice. The washing plant floated on two wooden barges with a well between. It was moved by two headlines and a tail line by means of hand capstans. The sands were discharged into the pond at the rear of the float. The backwash would occasionally ground the plant, but it was easily pulled clear. Water for the sprays was supplied by a 7-inch centrifugal pump run at slow speed. The pump and conveyor belt were run by a 36-hp. truck engine. About 9 gallons of gasoline and 1 quart of oil were used per shift.

One 9-hour shift was worked per day. The average running time was 7 1/2 hours. The crew consisted of 3 men - 1 engineer at $4.50 per day and 1 fireman at $4.00 on the steam shovel and 1 man on the washing plant at $4.00 per day. The plant operator spent most of his time at the grizzly. About 250 cubic yards was being washed per day.

The steam-shovel expense per cubic yard was 3 cents for labor and 5 cents for fuel. The cost of washing was 2 cents for labor and 1 cent for fuel. The general expense, including replacements on the shovel and repairs on the washing plant, would be about 5 cents per cubic yard. With supervision at 2 cents, the total operating cost would be about 18 cents per cubic yard.

The washing plant was built at a cost of $1,200; with the exception of the elevator belt, second-hand material was used. Built of new material the cost would have been about $2,500. New timber cost $25 per 1,000 feet.

An unusual feature of the installation was the fact that the gravel was dug from under water with an ordinary steam-shovel dipper. Usually where this practice had been tried too much gold had been lost to make the enterprise profitable. At this plant, the dipper after being loaded was held or swung along the bank to be mined next until all the water had drained out. The bedrock was soft, and from 6 to 18 inches was taken up by the shovel with each cut. It was considered that very little gold was lost in the pit.

Sumpter.- Hofford and Johnson began placer operations in 1932 alongside an old dredged area on the Powder River near Sumpter, Oreg. The gravel had not been dredged originally because bedrock was too high. An unsuccessful attempt had been made later to work the ground with a light-draft dredge using standard washing and gold-saving equipment but substituting a suction pipe with a special agitating device for a digging ladder. The enterprise failed because the capacity of the plant was too low and because the gravel was too tight to be dredged successfully with the equipment used. This second dredge had been acquired by Hofford and Johnson for a washing plant; the digging device had been removed. The gravel was excavated for 5 feet below and 8 feet above water level with a 1-cubic-yard, full-revolving, caterpillar-tread, 75-hp. dragline working on top of the bank. The washing plant had a draft of 3 feet. A standard Page dipper was used for digging and a hoe-type trench digger for cleaning bedrock, which consisted of a layer of clay on volcanic ash. The plant was moved and held in place by head and tail lines operating from power winchs. The gravel was dumped into a hopper built on the plant at the end of the trommel. The trommel was 4 by 24 feet and had 1/2-inch round holes. A standard belt stacker was used for disposing of the oversize. Standard dredging practice was followed in saving the gold. The tables were 30 inches wide and had a total area of 900 square feet. Riffles were 1 1/4 by 1 inch in size and spaced 1 inch apart. Water was supplied by one 8-inch, low-pressure and two 4-inch, high-pressure pumps. A total of 2,000 gallons per minute was used. Electric power for operating the washing plant was generated by a 200-hp. Diesel engine. An excess of electric power was available as the principal demand for power on the suction dredge had been for the suction excavator. The present boat required 80 hp. which cost over 3 cents per kilowatt-hour. The shovel used 40 gallons of gasoline, costing 24 cents per gallon, per 8 hours.

Two 8-hour shifts were worked. Two men were employed on the shovel and two on the boat each shift. Supervision was furnished by the shovel operator on day shift. The plant had a capacity of 400 cubic yards per 8 hours, but due to time lost in cleaning up and making repairs an average of about 40 cubic yards per hour was handled during most of the first season.

Labor on the shovel cost 2 1/2 cents, gasoline 3 cents, and other supplies 1 cent, making a digging cost of 6 1/2 cents per cubic yard. The labor cost for two men on the boat was 2 cents per cubic yard for washing. About 1 1/2 kw.-hr. was used for each cubic yard of gravel washed; at 3 cents per kilowatt-hour the power cost would be 5 1/2 cents. Other supplies cost about 1 cent, making a total washing cost of 8 1/2 cents. Miscellaneous costs, such as trucking and insurance, were estimated at 2 1/2 cents per cubic yard, making a total operating cost of 17 1/2 cents.

Summary of Operations at Mines Using Teams or Power Equipment

At the time of writing none of the strictly placer operations in the Western States using mechanical excavators had operated long enough to get all the "kinks" straightened out of the lay-outs. Usually lower costs can be expected with improved practice. Operating costs range from $0.16 to $1.50 per cubic yard and on the whole are higher for this type of mining than for hydraulicking, ground sluicing, or dredging. The average combined yardage handled daily by all of these plants is less than that of one dredge; moreover, the total quantity handled is insignificant compared to the total yardage dredged.

It should be borne in mind that the operating costs shown in table 5 and indicated in the foregoing descriptions are not the total costs of mining the gravel. Alterations to plants and major replacements of machinery have been charged to capital account and are not included herein. Amortization in some of the operations may equal or even exceed the strictly operating expense. For example, an operating cost for excavation of 4.4 to 10 cents per cubic yard is shown at some of the plants where the gravel is dug by power shovels or draglines. If depreciation of the machine, interest on the investment, and major repairs, or, in lieu, fair rental of the excavators, are included the total excavating cost would sometimes be more than double that indicated.

Ordinary sluices with riffles were used satisfactorily at most of the mines using mechanical methods for excavating the gravels. At three plants, side-shaking boxes were used to increase the capacity of the plant per unit of water available. At one, the side-shaking sluice apparently did not afford a satisfactory recovery of the gold, probably due to the difficulty of keeping the movable plant level on which it was used.

Standard concentrating tables were used at four places for treating screened products containing large proportions of black sands.

A patented, bowl-shaped, centrifugal gold saver was used at five plants described. A number of other plants had been equipped with this type of gold saver, but no operating data concerning these were available. Other types of patented gold savers were seen, but either operations had not been started or had ceased.

DREDGING

General

This paper does not propose to discuss gold dredging in detail, since other publications of the Bureau of Mines[27] have covered the subject adequately and are still up-to-date enough to serve as valuable references. However, an attempt will be made to show the present status of gold dredging in the United States, to discuss any noteworthy trends in design or operation, and to show the cost of dredging under present conditions. (1932)

As stated before, only tracts of placer ground large enough to justify the necessary investment are suitable for dredging. Moreover, the physical conditions of the deposit and the character of the gravel and bedrock must be favorable for successful dredging. Enough water must be available to float the dredge and wash the gravel. The material dredged must be tight enough to hold water in the dredge pond. The most successful dredging operations have been applied to valley deposits.

Status of the Dredging Industry

The production of gold by dredging in the United States and Alaska, by States, and the number of dredges in operation from 1896 to 1932 are shown in the first paper of this series.[28] In 1932 about 27 dredges were in operation in the United States (excluding Alaska).

---

27 Jennings, Hennen, The History and Development of Gold Dredging in Montana: Bull. 121, Bureau of Mines, 1916, 62 pp.
Janin Charles, Gold Dredging in the United States: Bull. 127, Bureau of Mines, 1918, 226 pp.
Ash, S. H. Safety Practices in California Gold Dredging: Bull. 352, Bureau of Mines, 1932, 31 pp.

28 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part I. - General, Hand Mining, and Ground Sluicing: Inf. Circ. 6786 Bureau of Mines, 1934, table 2, p. 10.

Their total capacity was approximately 4,250,000 cubic yards per month; about 90 percent was dredged in California. Since 1922 the average annual recovery by dredges in California has ranged from 8 to 10.7 cents per cubic yard.

The present low cost of labor and supplies has prolonged dredging operations in some fields and encouraged the opening of new ones. Several new dredges have started in relatively small fields in recent years; for example, in the Rogue River area near Grants Pass, Oreg.; the Warren district in central Idaho; and the Steel district near Silver City, Idaho. However, modern dredging technique does not differ enough from that of 15 to 20 years ago to hold much hope for a revival of dredging in lower-grade gravels or under more difficult conditions; with the exhaustion of the Yuba and American River fields in California a decided falling off in total dredge production is likely. In 1929 it was reported that the Natomas Co., principal operator on the American River, had 10 years of life ahead of it in that field.[29] The Marysville field, on the Yuba River, now has only three boats, which probably will last less than 5 years.[30]

Tables 9 to 13 give the essential data on all dredges active in the United States in 1932.

Discussion of Modern Dredges

The average depth of gravel worked is 35 feet. The deepest dredging is done in the Yuba River fields above Marysville where one dredge in 1931 averaged 74 feet including the height of the bank above water level. One of the boats there can dig 82 feet below water.[31]

The bucket-ladder dredge is the only type that has been successful for placer mining in this country and until recently at least, anywhere. The suction dredge, with or without cutter attachments, has failed repeatedly, although for river and harbor work it has proved of value, as evidenced by the recent construction of a huge hydraulic dredge for Panama Canal service. Which has a capacity of 500 to 1,000 cubic yards per hour and can dredge "the hardest materials" to a depth of 60 feet.[32] Notwithstanding past failures, attempts to use the suction-type dredge for placer mining are still made occasionally. In 1928 several such boats were being built for service in the Malaysian tin fields. Some of the objectionable features of the hydraulic dredge are its relatively low power efficiency as a result of the volume of water that must be raised, the problem of moving oversize boulders or sunken logs, the variation in the table feed which is extremely erratic even when compared to that of a bucket dredge, and the often serious effects of pumping the heavily mud-laden water from the bottom of the pond over the tables.

About half of the boats now in commission, comprising mostly the smaller ones, have wooden hulls; the others have steel hulls. The life of a wooden hull is usually stated as 10 years. That this figure can be exceeded is shown by the story of the Feather River no. 1 dredge.[33] This boat was built in 1906 and operated practically without a shut-down until 1929, when the machinery was transferred to a steel hull, now in use south of Oroville by the Shasta Butte Gold Dredging Co.

---

29 Engineering and Mining Journal, vol. 127, Mar. 3, 1929, p. 458.

30 Since the price of gold has been changed from $20.67 to $35 per ounce low-grade gravels in the dredging fields that could not hitherto be handled can be worked at a profit. The change in the price of gold probably will extend the life of the dredging areas.

31 A new dredge being built in February 1934 in the Yuba River field (The Mining Journal, Arizona, Feb. 28, 1934, p. 19) is reported to have a digging depth of 110 feet and an estimated capacity of 15,000 cubic yards per day.

32 Engineering and Contracting, Twenty-four-inch Diesel Electric Hydraulic Dredge, Las Cruces: Vol. 68, January 1929, pp. 37-38.

33 Young, G. J., California Gold Dredge in Operation over Twenty Years: Eng. and Min. Jour., vol. 123, June 25, 1927, pp. 1042-1046.

TABLE 9.- Dredges in operation in United States in 1932 and depth and nature of gravel dredged

Dredges					Nature of gravel dredged			
Owner	Location	Dredge no.	Type	Bucket size cu.ft.	Depth dredged, feet Minimum	Maximum	Average	Remarks
Capital Dredging Co.	Folsom, Calif.	1	Stacker	8 1/2	15	30		
Do.	do.	2	do.	8 1/2	15	30		
Do.	do.	3	do.	18				
La Grange Gold Dredging Co.	La Grange, Calif.	2	do.	10	12	42	30	Loose; 5 percent boulders over 1 foot; almost no clay.
Lancha Plana Gold Dredging Co.	Camanche, Calif.		do.	6	20		40	Medium tight; 20 percent clay and loam.
Natomas Co.	Natoma, Calif.	2	do.	8 1/2	28	34	30	
Do.	do.	4	do.	15	23	35	28	
Do.	do.	5	do.	11	41	55	50	
Do.	do.	7	do.	9	30	46	36	
Do.	do.	8	do.	15	48	62	58	
Do.	do.	10	do.	15	46	57	51	
Placer Development, Ltd.[1]	Lewiston, Calif.		do.	7	10	45	35	Loose; few boulders over 1 foot; little clay.
Shasta Butte Gold Dredging Co.	Oroville, Calif.		do.	7 1/2				Fair digging; some clay and some hard gravel.
Snelling Gold Dredging Co.	Snelling, Calif.		do.	6 1/2	9	26	15	Easy-washing gravel; few boulders over 1 foot.
Trinity Dredging Co.	Lewiston, Calif.		Flume	11		40	25	
Yuba Consolidated Gold Fields	Hammonton, Calif.	14	Stacker	14			54	
Do.	do.	15	do.	14			58	
Do.	do.	19	do.	14			74	
Do.	La Grange, Calif.	Merced unit.	do.	9	12	35	22	
Continental Dredging Co.	Breckenridge, Colo.		do.	7				Some boulders; some clay.
American Gold Dredging Co.	Murphy, Idaho		do.	2 1/2			15	No boulders or clay.
Crooked River Mining Co.	Idaho City, Idaho		do.	3		26	20	Some boulders over 1 foot.
Idaho Gold Dredging Co.	Warren, Idaho		do.	3 1/2			20	
Empire Gold Dredging & Mining Co.	Prairie City, Oreg.		do.	5		36	24	
Rogue River Gold Co.	Rogue River, Oreg.		do.	7	5	50		Hard digging; many boulders.
Superior Dredging Co.	Bridgeport, Oreg.		do.	7 1/2	20	40		Loose; few boulders; very little clay.

1 Formerly Lewiston Dredging Co.

TABLE 10.- Data on hulls, spuds, and digging ladders of dredges operating in the United States, 1932

Dredge	Hull					Number of spuds	Digging ladder		
	Material	Length, feet	Width, feet	Depth, feet	Draft, feet		Maximum digging depth below water, feet	Size of buckets, cu. ft.	Number of buckets in line
Capitol no. 3	Steel						70	18	
La Grange no. 2		104	44 1/2	9 1/2		2	30	10	60
Lancha Plana	Wood	98 1/2	41	7	5	2	54	6	82
Natomas no. 2	do.						35	8 1/2	78
Natomas no. 4	Steel						40	15	67
Natomas no. 5	Wood						60	11	101
Natomas no. 7	Steel						60	9	98
Natomas no. 8	do.						60	15	83
Natomas no. 10	do.						60	15	85
Placer Development	Wood	100	43	9	8	2	38	7	72
Shasta Butte	Steel					[1]2	41 1/2	7 1/2	73
Snelling	do.					[2]1		6 1/2	
Trinity	Wood	107	48	7 1/3	6	2		11	[3]42
Yuba no. 14							79	14	
Yuba no. 15							82	14	
Yuba no. 19							69	14	
Continental	Wood	133	40	10	8		65	7	95
American	do.	80	40		3 1/2 to 4	[4]1	18	2 1/2	46
Crooked River	do.	70	30		5	None	35	3	65
Idaho	do.			7	5	2	32	5 1/2	75
Empire	do.		40		5	[5]1	36	6	77
Rogue River	do.	100	50	9		[2]1	30	7	70

1 Spuds not used; digging on headlines.

2 Single spud on center line of hull.

3 Open-connected bucket line.

4 One of original two removed to reduce weight.

5 Built with two spuds, but only one used.

TABLE 11.- Washing and gold-saving plants on dredges in operation in 1932

Dredge	Screens						Gold-saving plants					
	Length, feet	Diameter, feet	Pitch, in. per ft.	Speed, r.p.m.	Diameter of holes, inches: Upper part	Diameter of holes, inches: Lower part	Approximate total table area, sq. ft.	Slope of tables, in. per ft.	Size of riffles, inches: Depth	Size of riffles, inches: Width	Size of riffles, inches: Distance apart	Water pumped on dredge, gal. per min.
La Grange	36 1/2	6	1 1/2	11	1/2	5/8	2,000	1 1/4	1 1/4	1 1/4	1 1/4	7,500
Lancha Plana	33	6	1 3/8	9	7/16	1	2,000	1 3/8	1 1/4	1	1	3,500
Natomas no. 8							5,000		1 1/4	1 1/4	1 1/4	
Placer Development	30 1/2	6	2		3/8	1/2		1 1/2	1 1/4	1 1/4	1 1/4	6,200
Shasta Butte	30	6			3/8	3/8	1,300		1 1/4	1 1/4	[1]1	
Snelling	23	6			3/8	1/2			1	1 1/4	1	
Trinity	15	5			6 to 10		[2]360	3/4	3	2	2	7,500
Yuba no. 14	35	9			3/8	3/8	9,000					
Continental	40	6		15	3/8	1/2	1,300	1 1/2	1 1/4	1 1/4	1 1/8	4,700
American	30	4 2/3			3/8	3/8	1,000	1	1 1/4	1 1/4		
Crooked River	25	4 1/2		13	3/8	3/8	870					4,000
Idaho	18	5		11	3/8	3/8	[3]510	1 1/2				
Empire	36	6			1/4	5/8	1,300		1 1/4	1 1/4	1 1/4	
Rogue River	40	6			3/8	[4]1 1/4 by 2 1/2		Various	1 1/4	1 1/4		

[1]Being increased to 1 1/4-inch spacing.
[2]90 feet of 4-foot-sluice, lined with angle-iron riffles.
[3]Being increased to 2,000.
[4]Also two sections with intermediate size openings.

TABLE 12.- Monthly yardages, power, labor, wage scales, and miscellaneous operating data on dredges operating in 1932

Dredges	Average monthly yardage[1]	Average daily dredging time: Hrs.	Average daily dredging time: Min.	Number of man-shifts per day	Man-hours per cubic yard	Wage scales: Winchmen	Wage scales: Oilers	Wage scales: Shoremen	Total connected power on dredge, hp.	Kilowatt-hours per cubic yard dredged
La Grange	217,000	20	40	...	...	$5.20	$4.20	$3.60	[2]500	0.80
Lancha Plana	63,500	18	45	11	0.042	6.00	5.00	5.00	420	1.88
Natomas:										
No. 2	152,000	20	40	...	...	...	...	...	...	...
No. 4	271,000	20	29	...	...	...	...	...	...	...
No. 5	147,000	18	32	...	...	...	...	...	...	...
No. 7	177,000	20	32	...	...	...	...	...	...	...
No. 8	237,000	20	25	...	...	...	...	...	...	...
No. 10	259,000	20	27	...	...	...	...	...	...	...
Total	1,244,000	20	11	...	...	...	...	...	...	...
Placer Development	100,000	20	40	15	.036	5.50	4.25	...	370	...
Shasta Butte	130,000	18	00	12	.024	6.00	5.50	5.50	[3]440	...
Trinity	100,000	20	30	13	.031	5.00	4.50	4.00	400	1.5
Yuba nos. 14, 15, and 19	[4]900,000	21	25	...	...	...	...	...	850	...
Yuba-Merced	200,000	21	40	18	.021	...	...	...	...	1.5
Continental	122,000	20	00	14	.027	[5]4.00	[5]3.60	[5]3.20	400	1.1
American	50,000	20	00	9	.039	5.00	4.00	...	[6]130	.7
Crooked River	60,000	20	00	10	.040	5.00	4.00	2.50	[7]170	...
Idaho	75,000	22	00	12	.038	5.00	3.75	...	[8]180	...
Empire	75,000	20	30	13	.042	...	...	...	295	...
Rogue River	120,000	22	00	16	.032	...	...	...	...	...

The following are the average monthly yardages (estimated in part) of the known operating dredges not included in this table:

Capitol, nos. 1, 2, and 3 650,000
Snelling 150,000
Superior 90,000

[2]Usual demand charge 425 hp.
[3]Usual demand charge 400 hp.
[4]Total monthly yardage of 3 boats, 1931.
[5]Excluding bonus.
[6]Usual demand charge 100 hp.
[7]Diesel engine.
[8]Steam engines.

TABLE 13.- Dredging costs per cubic yard of dredges operating in 1932

Dredges	Labor			Power	Supplies			Supervision	General	Taxes and insurance	Prospecting and sampling	Deferred capital charges	Total	Remarks
	Operation	Repair	Total		Operation	Repair	Total							
La Grange	...	...	$0.0162	$0.0071	...	...	$0.0099	$0.0022	$0.0019	$0.0012	Excluded	Excluded	$0.0385	Fiscal year 1931-32.
La Plana	$0.0286	$0.0052	.0338	.0199	$0.0058	$0.0359	.0417	.0172	.0114	.0045	do.	$0.0181	.1466	Costs given are for dredging 732,000 yards in 1930. In 1931 762,000 yards was dredged at a cost of $0.0994 per yard.
Natomas no. 2	.0121	.0038	.0159	.0131	.0036	.0136	.0172	(1)	.0065	Excluded	do.	Excluded	.0527	All Natomas costs are for year 1931; approximately half of the operating supply cost is chargeable to the cost of water.
Natomas no. 4	.0079	.0026	.0105	.0103	.0030	.0102	.0132	(1)	.0037	do.	do.	do.	.0377	
Natomas no. 5	.0133	.0063	.0196	.0154	.0066	.0249	.0315	(1)	.0067	do.	do.	do.	.0732	
Natomas no. 7	.0114	.0042	.0156	.0139	.0044	.0137	.0181	(1)	.0054	do.	do.	do.	.0530	
Natomas no. 8	.0103	.0033	.0136	.0144	.0051	.0142	.0193	(1)	.0046	do.	do.	do.	.0519	
Natomas no. 10	.0085	.0025	.0110	.0132	.0031	.0136	.0167	(1)	.0040	do.	do.	do.	.0449	
Average	.0101	.0035	.0136	.0132	.0041	.0143	.0184	(1)	.0049	do.	do.	do.	.0501	
Placer Development.	...	...	.0255	.0137	...	...	.0325	.0033	.0003	.0044	do.	do.	.0797	Costs given for 1929. In 1931 the cost from Apr. 1 to Oct. 1 was $0.0652 per yard.
Shasta Butte	.014	.005	.019	.010	...	...	.008	(1)	.009	(1)	$0.008	.008	.062	Approximate cost, first half of 1932.
Trinity	...	...	...	...	...	...	...	...	...	...	...	...	.0604	Cost for 1931, excluding only depreciation and depletion.
Yuba nos. 14, 15, and 19, average.	...	...	...	...	...	...	...	...	...	...	...	...	.0408	For year ending Feb. 29, 1932, excluding taxes, insurance, and administrative expense.
Continental	...	...	.0195	.0189	.0039	.0145	.0184	.0026	.0043	.0024	.0013	Excluded	.0673	Cost for 1,340,000 yards in 11 months of 1930, excluding interest, depreciation, and royalty.
American	...	...	.04	.01	...	...	...	...	...	...	...	...	.07	Approximate cost in 1932.
Idaho	...	...	.045	.015	...	...	.040	(2)	(2)	Excluded	Excluded	Excluded	.100	Estimated cost in 1932. Power cost represents cost of wood fuel.
Empire	...	...	...	...	...	...	...	...	...	...	...	...	.0625	Total cost in 1932, including approximately $0.01 for depreciation. See Eng. and Min. Jour., vol. 128, Nov. 9, 1929, p. 737, for 1929 cost.

1 Included in General.

2 Included in Labor.

One feature of modern design and operation is a tendency to use only one spud or to work entirely on the lines. The new steel boat of the Snelling Gold Dredging Co., on the Merced River, has only one spud which is placed on the center line at the stern. This necessitates placing the stacker off center and using a rock chute to transfer the screen oversize to one side of the spud onto the belt; however, advantages in the design of the stern of the hull, as well as in operation, led to the adoption of the single spud. Many other boats are using only one of their two spuds and moving up on their lines. Considerable time is thus saved, but one objection is the possible difficulty of keeping the boat precisely on line and of securing just the desired advance while moving forward by using the shore lines. This might be a particular handicap when the bank is under water. The dredge of the Crooked River Mining Co., Idaho, has no spuds but moves and digs entirely on its shore lines.

Estabrook dredge

Although the Estabrook dredge in Trinity County, Calif., was dismantled recently, the following data on it are given because this dredge was the largest built in the country, and few large new dredges have been built since its time for which details are available.[34] It was built on the Trinity River, Calif., some 40 or 50 miles over mountain road from the railroad. It was 152 feet long, 68 feet wide plus 6 feet of overhanging deck on either side, and 13 feet deep, requiring about 1,000,000 board-feet of lumber, of which most was cut locally. The digging ladder, 125 feet long, carried eighty-three 20-foot buckets, weighing 5,650 pounds each. This line was driven through double gears by a 500-hp. motor. The screen was 9 by 54 feet and was driven by a single roller. Single-deck tables had an area of 4,400 square feet. The stacker carried a 44-inch belt 140 feet long, was arranged to swing 15° to either side of the center line, and was balanced by ballast tanks alongside the hull. The two 70-foot steel spuds were 3 feet 4 inches by 5 feet 4 inches in section. The dredge was designed to dig 400,000 to 450,000 cubic yards per month and to dig to a depth of 40 feet. After operating intermittently from 1920 to 1927, the boat was shut down and dismantled.

Fairbanks Exploration dredges

The Fairbanks Exploration Co., Fairbanks, Alaska, is working several modern dredges successfully under adverse conditions. As this operation has not been described previously, a brief description of it by H. W. Rice, vice president of the United States Smelting, Refining & Mining Co., is given here, although Alaskan placer-mining practice in general is not included in this paper.

The Fairbanks Exploration Co., a subsidiary of the Unite States Smelting, Refining & Mining Co., operates 5 dredges in the Fairbanks district of Alaska. Of these 3 are on Goldstream Creek about 14 miles from Fairbanks and 2 are on Cleary Creek about 25 miles from Fairbanks.

Gold was first discovered in the Fairbanks area in 1903. The greatest concentration of values was found in ancient stream beds covered with considerable gravel and muck. The gold in the gravel is principally on or near bedrock and may penetrate the latter to a depth of 5 feet where it consists of blocky schist. The gravel contains few boulders over 1 foot in diameter and very little clay. The gold-bearing areas were worked by drift-mining methods; later some areas were worked by opencut methods. The dredges handle ground that was too low in grade to be worked profitably by the earlier methods and some previously worked areas.

34 Peake, H. G., Largest Capacity Gold-Mining Dredge in the World: Eng. and Min. Jour., vol. 109, May 15, 1920, pp. 1106-1109.

The gold-bearing gravel not only is frozen but also is covered with a layer of frozen overburden which in places is 120 feet thick. As this overburden must be removed and the gravel artificially thawed before dredging can be done, it is necessary to start operations years in advance of the actual dredging. The areas were prospected thoroughly, using Keystone drills, and open holes were drilled except in locally thawed spots where it was necessary to use casing. The dredge areas were laid out and the starting point for each dredge was determined from the prospect data.

Active work on the project was started in 1924; actual construction began in 1925. The construction included a steam power plant, a transmission line, a 90-mile ditch, dredges, and the necessary shops, bunkhouses, and miscellaneous items. The offices, power plant, and shops are at Fairbanks, the northern terminus of the Alaska Railway. The power-plant equipment includes two 1,000-hp. Stirling boilers with chain-grate stokers, two 3,750-kilovolt-ampere, 4,000-volt, General Electric turbo-alternators, one 625-kilovolt-ampere, 4,000-volt General Electric turbo-alternator, and necessary accessories. Lignite obtained from the Healy River district about 115 miles by rail from Fairbanks is used for fuel. Current is transmitted at 33,000 volts to the two dredge areas and carried onto the dredges at 2,200 volts.

The first step in ground preparation is the removal of the overburden which consists largely of frozen silt with a top covering of moss or tundra. This frozen silt, or "muck" as it is termed locally, usually contains 60 to 70 percent of ice by volume and considerable organic matter. Shallow muck is thawed in place and handled by the dredges. Deep muck is stripped off as completely as practicable, depending largely upon drainage conditions. The first step in stripping is to remove the covering blanket of moss by means of hydraulic giants and high-pressure water, thereby exposing the muck to the sun. Thawing of the muck then proceeds rapidly so long as a blanket of thawed material is not allowed to accumulate on the surface; to prevent such accumulation the surface is washed off from time to time with high-pressure water. As the hydraulic gradient of the local creeks is low, considerable water is necessary to transport the solids in the muck from the dredging area. This water could be obtained only at a distance.

The Davidson ditch, which furnishes water for the stripping operation, has a capacity of about 5,000 miner's inches and is about 90 miles long. The ditch line includes 15 steel siphons 46 to 54 inches in diameter aggregating about 6 miles in length and a 4,000-foot tunnel. Water is taken from the Chatanika River just below the junction of Faith and McManus Creeks and is delivered to the dredging areas on Cleary and Goldstream Creeks at a working pressure of 80 to 160 pounds per square inch, depending upon the location. Practically all of this water is used for stripping.

Frozen gravel is thawed by the Miles cold-water process which has been described by Janin[35] and Wimmler.[36] The practice at Fairbanks follows the general procedure outlined in these bulletins, although the equipment and procedure have been greatly improved through experience and research. The thawing season is from about May 10 until about September 20. Gravel less than about 35 feet in depth is thawed with driven points on 16-foot centers and at greater depths with points set in holes drilled on 32-foot centers. Thawing on 16-foot centers is completed in about one half a season; thawing on 32-foot centers requires from one to two seasons. Most of the water used for thawing is circulated by pumps. So far as possible a reserve of thawed ground equal to at least one season's dredging is kept ahead of each dredge. No difficulty has been experienced with freezing of artificially thawed ground

35 Janin, Charles, Recent Progress in the Thawing of Frozen Gravel in Placer Mining: Tech. Paper 309, Bureau of Mines, 1922, 34 pp.

36 Wimmler, Norman L., Placer-Mining Methods and Costs in Alaska: Bull. 259, Bureau of Mines, 1927, 236 pp.

except for a seasonal freeze-back at the surface of not more than 7 or 8 feet which thaws naturally early in the summer season.

The dredges are of all-steel construction; they were designed by company engineers, fabricated by a ship-building concern of San Francisco, and erected in the field by the company. The design follows conventional lines but embodies certain special features to meet conditions existing in northern latitudes. The dredges are housed and heated; both the stacker and ladder are heated.

The more important data concerning these dredges are given in table 14. The dredging season starts early in April, the winter ice being cut and removed from the pond to facilitate an early start. The average operating season is about 210 days, and the maximum to date for any one dredge has been 270 days during which time the minimum temperature was minus 37° F. Excessive ice formation in the pond is the principal cause of the shut-down at the end of the season; a continuous temperature of 10° to 15° F. below zero, or several days of 30° to 35° below zero, causes ice to form too rapidly for operation.

Hungarian riffles are used on the gold-saving tables with quicksilver traps at the head ends and coco matting and expanded metal at the lower ends of the transverse tables. Practically all of the gold will pass 8-mesh, but a few nuggets up to an ounce in weight have been found. The dredges are cleaned up at about 2-week intervals; the amalgam and black-sand concentrates are taken to the gold room at Fairbanks where the amalgam is cleaned and the black-sand concentrates are treated to recover fine and rusty gold that did not amalgamate. These sands contain no platinum metals nor, except for gold, other metals or minerals of commercial value. The clean amalgam is retorted and melted in a single operation using graphite crucibles and a specially designed combination retort and melting furnace. The resulting bars after assay are shipped to the mint.

The following figures for 1931 give an idea of the magnitude of this operation. Stripping for the season totaled 7,011,000 cubic yards, averaging 52,000 cubic yards per working day. Thawing totaled 8,133,000 cubic yards, or 64,000 cubic yards per working day. The dredges handled 6,916,000 cubic yards of material, or 30,800 cubic yards per working day, part of which was muck that was too shallow to strip or remained after stripping.

General Operating Practice

No marked changes in operating practice have been made in the United States in recent years, and the general statements made by Janin as regards depth of cut, speed of swing, bucket speed, and distance stepped ahead still hold true. It is common practice to take a light cut and to swing fast. Small boats normally are stepped ahead 4 or 5 feet and large boats 6 or 7 feet. The average width of a single cut is about 125 feet, and usually two or three cuts are carried abreast.

Dredging time per 24 hours is the chief measure of operating efficiency, indicating both the success of the boat as a dredging machine and the competency of its operators. This factor has not been improved much recently, apparently having reached its maximum in the first 10 or 15 years of dredging in this country. Winston and Janin[37] present figures to show that the average operating time of the dredges in California about 1908 was only slightly more than 18 hours per day. By 1914 the average of Californian boats was about 20 hours.[38] In 1931 the average operating time of all dredges in California was about 20 1/2 hours; the same average also applies to all dredges in other States during the year.

37 Winston, W. B., and Janin, Charles, Gold Dredging in California: Bull. 57, Calif. State Min. Bur., 1910, pp. 100-103.

38 Janin, Charles, Gold Dredging in the United States: Bull. 127, Bureau of Mines, 1918, p. 128.

TABLE 14.- Data on dredges of Fairbanks Exploration Co., Fairbanks, Alaska

Dredge no.	2	3	5	6	8
Location	Goldstream Creek	Cleary Creek	Cleary Creek	Goldstream Creek	Goldstream Creek.
Type	Stacker	Stacker	Stacker	Stacker	Stacker.
Gravel:					
Average depth material dredged[1]	45.2	46.5	33.5	22.3	18.9.
Condition	Frozen	Frozen	Frozen	Frozen	Frozen.
Hull:					
Length ... feet	128	148	108	108	99.
Width ... do.	60	60	60	60	50.
Height ... do.	12	12	9	9	10 1/2.
Draft ... do.	9 1/12	8 11/12	6 5/6	6 5/12	7 3/4.
Spuds:					
Number	2	2	2	2	2.
Location	Stern	Stern	Stern	Stern	Stern.
Digging ladder:					
Depth dug, total ... feet	64	78	53	53	44.
Depth dug under water ... do.	48	60	36	36	28.
Buckets:					
Number in line	93	104	78	78	68.
Size ... cubic feet	10	10	6	6	6.
How connected	Close	Close	Close	Close	Close.
Trommel:					
Length ... feet, inches	44, 7 1/2	44, 7 1/2	43, 1/2	43, 1/2	36, 2 1/2.
Diameter ... feet	8	8	6	6	6.
Pitch ... inches per foot	1 5/8	1 5/8	1 5/8	1 5/8	1 5/8.
Speed ... r.p.m.	6.74	6.74	8.9	8.9	8.9.
Diameter of holes:					
Upper part (round) ... inch	3/8, 1/2, and 5/8	3/8, 1/2, and 5/8	3/8, 1/2, and 5/8	3/8, 1/2, and 5/8	3/8, 1/2, and 5/8.
Lower part (slotted) ... do.	7/8 by 1 1/2 and 1 1/8 by 1 3/4.	7/8 by 1 1/2 and 1 1/8 by 1 3/4.	7/8 by 1 1/2 and 1 1/8 by 1 3/4.	7/8 by 1 1/2 and 1 1/8 by 1 3/4.	7/8 by 1 1/2 and 1 1/8 by 1 3/4.
Tables:					
Total area ... sq. ft.	4,535	4,535	2,125	2,125	1,460.
Slope:					
Transverse tables ... in. per ft.	1 1/4	1 1/4	1 1/4	1 1/4	1 1/4.
Longitudinal tables ... do.	1 1/8	1 1/8	1 1/8	1 1/8	1 1/8.
Riffles:					
Type	Hungarian	Hungarian	Hungarian	Hungarian	Hungarian.
Depth ... inches	1 to 1 1/4	1 to 1 1/4	1 to 1 1/4	1 to 1 1/4	1 to 1 1/4.
Width ... do.	1	1	1	1	1.
Spacing ... do.	2 1/4	2 1/4	2 1/4	2 1/4	2 1/4.

[1] Figures for 1931 season.

TABLE 14.- Data on dredges of Fairbanks Exploration Co., Fairbanks, Alaska - Continued

Dredge no.	2	3	5	6	8
Pumps:					
Number	3	3	2	2	2.
High pressure:					
Size...inches	14	14	12	12	10.
Pressure...lb. per sq. in.	27	27	25	25	22.
Water pumped...gal. per min.	5,500	5,500	4,000	4,000	4,000.
Low pressure:					
Size...inches	14	14	12	12	10.
Pressure...lb. per sq. in.	14	14	14	14	11.
Water pumped...gal. per min.	5,500	5,500	4,000	4,000	3,000.
Hopper:					
Size...inches	6	6			
Pressure...lb. per sq. in.	27	27			
Water pumped...gal. per min.	1,000	1,000			
Power:					
Kind	Electric	Electric	Electric	Electric	Electric.
Motors:					
Digging...hp.	250	250	150	150	150.
Screen...do.	75	5	60	60	40.
Swing winch...do.	40	40	25	25	25.
Stacker...do.	50	50	25	25	25.
High-pressure pump...do.	150	150	100	100	75.
Low-pressure pump...do.	75	75	60	60	40.
Hopper pump...do.	40	40			
Fire pump...do.	25	25	25	25	25.
Miscellaneous motors (3)...do.	44	44	44	44	44.
Total...do.	749	749	489	489	424.
Average load...do.	640	640	360	260	260.
Maximum 3 minute peak[1]...do.	925	925	455	360	470.
Cuts:					
Width of each cut...feet	120 to 190	150 to 230	100 to 180	90 to 170	70 to 100.
Number of cuts	2 to 4	2 to 3	2 to 7	2 to 5	2 to 3.
Average total width of advance...feet	471	349	445	395	310.
Total distance stepped up	1,482	3,061	2,302	3,634	5,057.
Method of moving up	Spud	Spud	Spud	Spud	Spud.
Number of operating days[1]	204	224	226	225	243.
Number of shifts per day	3	3	3	3	3.
Average yards per day[1]	5,800	8,770	5,990	5,600	5,040.

1 Figures for 1931 season.

Screening

The feature of dredge design and operation that is of most interest in other types of placer mining where the gravel is washed mechanically is the screening, washing, and gold-saving plant. An attempt has therefore been made to gather detailed information on the subject.

Although horizontal shaking screens formerly were used to some extent in this country and still are used by some foreign dredge builders, cylindrical revolving trommels are used universally in the United States for washing and screening. These range from 4 1/2 to 9 feet in diameter and from 20 to 40 feet in length. The screen plates are of rolled steel or cast-manganese steel, punched or drilled with holes 3/8 to 5/8 inch in diameter. It is common practice to use two sizes with the smaller holes in the upper portion of the trommel to distribute the load to the tables better. One dredge in Oregon is using a trommel having four sizes of holes with satisfactory results. The uppermost section is punched with 3/8-inch round holes. The second section is punched with oblong holes or slots 3/8 inch wide and 1 1/8 inches long. The third and fourth sections are punched similarly with oblong holes 3/4 inch by 1 1/2 inches and 1 1/4 by 2 1/2 inches, respectively. As shown in table 14, holes of five different sizes are used on the dredges of the Fairbanks Exploration Co. The slots are arranged so as to parallel roughly the movement of the gravel over the screen and are tapered to an outside diameter one eighth inch greater than the inside one. The last feature is practically universal practice in dredging and is effective in reducing the blinding of screens.

The function of a dredge trommel is as much to disintegrate as to screen, except in rare instances of very free-washing, loose gravel. Hence, high-pressure internal sprays are used with water under 25 to 50 pounds pressure per square inch. Lifter bars are used to raise and agitate the bed of gravel; the lifter bars, however, are too small and the speed too low to cause appreciable cascading.

Gold saving

Riffled tables are used to save gold on the dredges in the Western States; at the Natomas Co.'s operations Neill jigs are used as auxiliary traps for the fine and rusty particles of gold that do not amalgamate readily. Excepting the Natomas practice and the two flume-type boats in Trinity County, Calif., the standard and only gold saver is a single or double bank of tables 1,000 to 9,000 square feet in area and covered with transverse riffles. In addition to riffles, an amalgam trap is used at the head ends and a section of expanded metal over coco matting at the lower ends of the transverse tables of the boats of the Fairbanks Exploration Co. The tables on dredges are set on grades of 1 inch to 1 1/2 inches per foot. The so-called Yuba-type riffle, or modifications of it, is used on most stacker-type dredges. The riffles consist of wooden cross strips 1 1/4 by 1 1/4 inches in cross section, capped by strap iron and spaced 1 1/4 inches apart. One modification consists of the substitution of angle irons welded into sections of convenient size and approximating the dimensions of the wooden riffles. Dredge riffles were discussed in more detail under Sluice Boxes and Riffles in a previous paper.[39]

The sluices of flume-type boats commonly are set on grades of 1/2 or 3/4 inch per foot and paved with riffles of 2- to 3-inch angle iron.

39 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part II. - Hydraulicking, Treatment of Placer Concentrates, and Marketing of Gold: Inf. Circ. 6787, Bureau of Mines, 1934, 89 pp.

Quicksilver is always used on the tables and in the sluices. Clean-ups are made at regular periods of a week or 10 days; dredging then ceases, and advantage is taken of the opportunity to perform routine repair work.

Dredging recovery

The question of the percentage of recovery of the total gold content in the gravel dredged is still unsettled. Estimates by experienced dredge operators range from 60 to 90 percent. The first point of loss is under water where depressions in bedrock may be missed, gold may be retained in crevices in hard bedrock, or gold-bearing gravel may be left by unskillful operation. Moreover, the action of the buckets and occasional caving of the bank may throw some gravel and gold back beyond reach of the bucket line. The solids in the water near the bottom of a dredge pond may increase the density of the water sufficiently to hinder settling greatly, thus permitting gold to be carried away from the digging face in suspension. Some material drops from the buckets, not all of which is caught by the save-all sluice. Some gold is not washed free in the trommel but adheres to clay lumps or boulders which are discharged onto the stackers. Moreover, occasionally a nugget too large to go through the screen is carried through to the tailings pile. Some further loss occurs on the tables, as not all of the fine gold, especially if rusty, is caught in the riffles. All of these losses, except those of nuggets, can virtually be eliminated by suitable regulation of operations, but only in the sluices can any tests be made that will indicate what the losses are and show how successfully they are reduced by changed conditions.

Thurman[40] states that the short sluices of the early dredges were responsible for much loss of gold, as tests by the Natomas Co. showed that at least 90 feet of sluice length was necessary for a nearly perfect recovery. His statement that the modern dredge is capable of making a 99 percent saving, however, must be doubted.

Smith[41] presents an excellent summary of the many factors responsible for dredging losses. Among the unfavorable physical characteristics of the placer deposit itself he lists clay; heavy soil overburden; hard or rough bedrock; and fine, flaky, or rusty gold. The clay strata, according to Smith, seldom contain gold but often are overlain by rich streaks so closely associated that the digging action may knead the clay and gold-bearing strata into a mixture which is not broken apart in the screen or on the tables. A thick clay or soil overburden may muddy the water so as to lower the efficiency of the gold-saving sluices. He also states that a number of dredge operating features likewise may cause loss of gold, such as failure to clean bottom properly between adjacent cuts, thus leaving unseen submerged ridges of the best "pay dirt" over 5 to 10 percent of the total bedrock area; an uneven feed, varying grade, quickly fouled or floured quicksilver, and careless clean-up of the tables; insufficient water supply due to the lowered efficiency of the pumps after long service; and various human factors such as the lack of attention to the tables and screen by a crew primarily concerned with yardage. Tests were made on a number of dredges in the Oroville and Natomas fields, apparently by taking large samples of the tail stream. Assuming that the gravel that passed over the tables was 50 percent of that dug and that 1 cubic yard loose was equivalent to 3/4 cubic yard in place, 2 dredges lost 15 percent of the gold in the table feed and 1 lost 30 percent. Another dredge, in the Oroville field, lost 6.8 cents per cubic yard. Samples were taken of the sand pile behind a sunken dredge in the Natomas field which

40 Thurman, C. H., Possibilities of Dredging in the Oroville District, Calif.: Min. and Sci. Press, vol. 118, Feb. 22, 1919, pp. 257-258.

41 Smith, R. G., The Discrepancy Between Drilling and Dredging Results: Eng. and Min. Jour., vol. 112, Nov. 19, 1921, pp. 812-815.

indicated a loss of 3.9 cents per cubic yard, or 31 percent. One dredge digging entirely in old dredge tailings recovered 2 1/2 to 4 cents per cubic yard (recalculated to original bank measure) due to an estimated loss by the first dredge of 18 to 25 percent. In view of the above data Smith concludes gold saving on dredges holds more hope for improvement by study than the technique of sampling and valuing dredging ground.

Bellinger[42] states that by cleaning up a sluice line from distributor to tail sluice, section by section, and plotting the amounts of gold recovered along a horizontal axis a close estimate can be made of the total gold fed to the sluice and the amount lost. The additional work in this "sampling" process devolves chiefly on the retort house where it is necessary to clean up and retort many small batches of concentrates. There should be little delay in actual dredge operation. The results of a series of such tests should be of great value to any dredge operator.

Dredge sluices unquestionably are the most efficient gold-saving devices known in placer mining, partly because they treat a screened product but chiefly because they are closely adapted to conditions. They work under severe space limitations imposed by the size of the boat, and the grades of the sluices vary 1 or 2 percent because of raising or lowering the heavy digging ladder. As a rule all of the tables of a dredge are set on a uniform grade. An interesting exception to the rule is on the boat previously referred to where the trommel produces four sizes of sluice feed. The grade of each sluice is set to give the best results on its particular size of gravel, all grades being relatively steep as compared with usual dredge practice.

Water Consumption

The amount of water and the spray pressure used in dredging are of interest but seldom are known from actual measurement. The total water used on stacker dredges ranges from 3,000 to 15,000 gallons per minute, depending on the size of the boat and the easy-washing, tight, or clayey nature of the gravel. If high-pressure streams are needed to wash the buckets at the upper tumbler 25 to 50 percent more water is used than otherwise. Pressures corresponding to heads of 60 to 90 feet usually are maintained on the spray nozzles. In more than a dozen dredges, including several cited by Janin,[43] the water consumption on the boats ranged from 1,400 to 2,800 gallons per cubic yard dredged and averaged 1,900, equivalent to a duty of about 9 cubic yards per miner's inch. This excepts two flume-type boats on which the pumps furnished 3,600 and 4,800 gallons, respectively, per cubic yard dredged. According to Janin, a stacker dredge usually discards over half of the volume of material dug, hence the quantity of water used per unit of gravel passed over the tables is much greater than these figures imply.

The quantity of water required to maintain a dredge pond situated above the natural water level depends upon the porosity of the ground. At the Lancha Plana dredge it was noted that 2,000 gallons per minute, or roughly 180 miner's inches of water, was needed for this purpose. This quantity lies well within the range of 100 to 300 inches given by Janin. Likewise the Trinity Dredging Co. near Lewiston, Calif., has to pump about 2,000 gallons per minute to maintain the dredge pond at the desired level.

Power

Only 2 of the 25 or more dredges operating in the Western States in 1932 used other than electric power. (See table 12.) One of these was the steam dredge of the Idaho Gold Dredg-

42 Bellinger, B. W., Dredge-Sluice Efficiency: Eng. and Min. Jour., vol. 132, Nov. 9, 1931, pp. 403-4.

43 Janin, Charles, Gold Dredging in the United States: Bull. 127, Bureau of Mines, 1918, pp. 130-132.

ing Co., at Warren, Idaho. It was powered by two 90-hp. boilers. One 90-hp. engine drove the digging ladder and winch. Another operated the screen, stacker, and all pumps. Electric lights and small auxiliary electrical equipment were provided for by a 15-kilowatt generator. The owners expressed satisfaction with their power plant. Its choice doubtless was indicated by the remote locality and plentiful supply of firewood. The dredge used an average of 3 1/2 cords of dry lodgepole pine per day which cost $5 per cord delivered to the bank. This included a year-round daily average of one half cord used in the auxiliary heating boiler. A liberal estimate of the cost of power for this boat was 1.5 cents per cubic yard. That the power plant functions well was indicated by a reported average dredging time of 22 hours per day.

The dredge of the Crooked River Mining Co. near Idaho City, Idaho, was equipped with a Diesel power plant. This was reported to have two serious disadvantages: (1) Excessive vibration, which led to high repair costs on the dredge, and (2) fouling of the dredge pond with oil, which was believed to have an adverse effect on gold recovery.

Cost of Dredging

Cost figures collected by the authors for 1931 represent about 75 percent of California's yardage and 50 percent of that of other States, or slightly more than 70 percent of the total. The average cost, on a yardage basis, excluding taxes, insurance, and capital charges, for the sake of uniformity, was approximately 5.1 cents per cubic yard for all States. The cost for the Californian dredges was 4.9 cents. The range was from 4 to 12 cents per cubic yard. (See table 13.)

The above figures, based on an annual yardage of about 36,000,000, may be compared with 1914 costs as given by Janin.[44] Costs in that year, based on a yardage of about 70,000,000 for which data were available and excluding taxes, insurance, and capital charges, ranged from 2.65 to 5.3 cents and averaged 4.2 cents. Wages in 1914 were lower than the 1931 Californian wage scale, winchmen receiving $4 to $5, oilers $3 to $3.50, and shoremen and helpers $2.50 to $3.50 per 8-hour shift. In 1932 wages were being reduced by some of the large companies.

Cost of dredges

Few entirely new dredges have been built in the United States in recent years. There appears to be a large reserve of old but serviceable dredge machinery not in use as a result of the decline in the number of boats in operation. The following communication from L. D. Hopfield[45] is nevertheless of value in stating a range of costs for new boats:

> Dredges are built in various sizes from 2 1/2 to 18 cubic feet of bucket capacity and designed to dig from 12 to 80 feet below water level. The cost ranges from $50,000 to $500,000, depending on the capacity and design.
>
> A dredge with 6 1/2-cubic-foot bucket capacity recently built in California, with steel hull and wooden superstructure, designed to dig 25 feet below water level, cost approximately $140,000. A 10-cubic-foot capacity dredge designed to dig 40 feet below water level costs from $250,000 to $275,000. A 15- or 18-cubic-foot capacity dredge that will dig 60 to 80 feet below water level is estimated to cost between $450,000 and $500,000. These figures do not include the cost of

44 Janin, Charles, work cited, pp. 157-179.

45 Department manager, Natomas Co., Natoma, Calif.

land, camp buildings, roads, or freight. Smaller dredges than the ones mentioned will cost less, according to their size.

A 200-ton steel hull for a 7 1/2-cubic-foot capacity dredge cost $19,000 in about 1929. The 9-cubic-foot, flume-type Madrona dredge, in Trinity County, Calif., was reported to cost $150,000, being made partly from salvaged material.[46] In 1928 the dredge of the Empire Gold Dredging & Mining Co., after working 12 years on the John Day River, in Oregon, was dismantled and rebuilt at a new site 14 miles upstream.[47] It was a stacker-type electric dredge, with seventy-seven 6-cubic-foot buckets and capable of digging 30 feet below water. The machinery was hauled in 5-ton trucks at a cost of $2.25 per ton, or a total cost of $1,200. A new hull, sheathed with 4-inch fir, cost $12,000. The total cost of dismantling, moving, and rebuilding was $32,000.

Resoiling

So far as is known no dredge is resoiling its ground in the United State at present, although much money has been spent in experimentation involving the resoiling of considerable areas. Von Bernewitz[48] states that in 1919 Natomas boats had dredged and resoiled about 250 acres each and that the land appeared to be left in good condition. The equipment necessary was said to involve 2 cobble stackers, 2 pebble stackers, and 2 long, double tail sluices. The extra cost of operation, in addition to the considerable capital cost, comprised extra power for the stackers and the pay of one extra attendant.

It is noted that a large dredge built in this country and recently operating in Japan[49] was equipped to resoil its ground, obviously a necessity when dredging agricultural land in such a thickly populated country. The extra labor, if any, was no serious drawback there, as the dredge company employed 120 men for the operation of this boat, the average wage being 40 cents per day; 30 men were employed on the dredge alone in three shifts of 10 men, and 10 bankmen were employed on day shift. The resoiling equipment on this boat included two stackers with double chutes at their ends, sand wheels, and sand pumps. The resoiled area was said to lie at first about 3 feet above its former level, but soon it subsided to nearly its original level. The resoiling operation was said to be largely automatic.

Accident Prevention

The work of a dredge crew, involving the handling of large boulders, heavy cables, buckets, blocks, and other mechanical parts of the dredge, is done in an environment that requires constant alertness on the part of the men if accidents are to be avoided. However, Ash states:[50]

> As in many other industries of the country, considerable safety work has been done, and at present the gold dredges of this State are among the most adequately guarded types of mechanical equipment.

The provision and maintenance of the best-known physical safeguards against injury to the workmen and the strict enforcement of suitable safety rules will eliminate most accidents

---

46 Engineering and Mining Journal, vol. 124, July 9, 1927, p. 62.

47 Engineering and Mining Journal, vol. 128, Nov. 9, 1929, pp. 736-737.

48 von Bernewitz, M. W., Dredging and Resoiling: Min. and Sci. Press, vol. 118, Apr. 5, 1919, p. 471.

49 Little, H. S., Japanese Gold-Dredging Enterprise: Eng. and Min. Jour., vol. 130, Nov. 24, 1930, pp. 513-514.

50 Ash, S. H., Safety Practices in California Gold Dredging: Bull. 352, Bureau of Mines, 1932, p. 1.

even in this naturally hazardous occupation. The bulletin cited gives details of accident-prevention methods in the Californian industry.

DRIFT MINING

General

Drift mining in the United States has been applied chiefly to the exploitation of buried Tertiary river channels in the foothills of the Sierra Nevada in California. It has also been applied extensively, although on a smaller scale, to the mining of rich streaks on or near bedrock in more recent gravels where pay dirt is covered with a thick mantle of unproductive material. Ground may also be drifted where there is insufficient grade or water for hydraulicking or where conditions are unsatisfactory for dredging. Bedrock under rivers has also been drifted where it was impracticable to divert the stream; however, loose gravel containing a large quantity of water cannot be mined successfully by drifting. Usually the method is one of last resort and can be applied only to rich gravel. Even under favorable conditions 6 feet of gravel on bedrock generally must average at least $2.50 per ton to be mined profitably by drifting. Ground that has been drifted by the oldtimers with limited capital has been worked by other methods later; in these instances the overburden carried enough gold to pay for mining on a large scale.

In the latter part of the nineteeth century many large and productive drift mines were operated in California; according to Hill,[51] 11 million dollars in gold was produced in California by this method from 1900 to 1928, inclusive. In the summer of 1932, however, there were no large-scale operations in the United States, and the production of gold by this method was relatively unimportant. Two well-equipped properties, Vallecito Western and Calaveras Central, were doing development work but no regular breasting.[52] The washing plants were used when enough gravel had accumulated to run the plant most of a shift. A few men were employed at a number of old properties in an endeavor to find new deposits of gravel. At a few other old mines lessees were taking out a very limited tonnage from around old workings. Throughout the western placer districts small operations were under way, but relatively little systematic breasting was being done.

Most of the present drift mines are operated through shafts, although in the past some large and productive mines were worked by adits. In many districts large quantities of water must be pumped.

In mining, the gravel is either drilled and blasted or picked by hand to break it down, then it is shoveled into cars and trammed to the surface or to the hoisting shaft. At the surface the gravel is sluiced or put through a washing plant to recover the gold. The gravel from most drift mines requires mechanical methods of washing to disintegrate it and free the gold.

Milling practices bear no direct relation to mining methods at drift mines and are treated separately in this paper.

---

51 Hill, J. M., Historial Summary of Gold, Silver, Copper, Lead, and Zinc Produced in Caliornia, 1848 to 1926: Econ. Paper 3, Bureau of Mines, 1929, 22 pp.

52 "Breasting" is the term used in drift mining to designate the mining of the gravel; it corresponds to "stoping" as used in lode mining.

Development

General development

The general development plan of a drift mine usually resembles that of a lode mine where similar flat-lying deposits are exploited. Lateral development and the blocking out of the pay gravel are modified to fit local conditions.

Bench deposits or old channels exposed by later erosion or covered by only moderate depths of overburden may be opened and mined through adits. Ventilation shafts, however, may be required in extensive workings.

Deeply buried deposits must, of course, be mined through shafts. This form of entry also is used for mining relatively shallow deposits where adits are not practicable. Occasionally long drain tunnels will be run and the gravel mined through a series of shafts sunk along the course of the pay gravel. Moreover, shafts may prove more economical for mining shallow deposits where their use obviates long underground trams. Conversely, adits may be run for drainage and to work gravels which have been developed through shafts.

Some of the ancient channels are buried as much as 500 feet deep by later gravel and lava flows or beds of volcanic ash. The gravel is hoisted through a central shaft; one or more auxiliary shafts usually are required for ventilation. A buried gravel deposit generally is prospected by a drift along the course of the channel and crosscuts from the drift to either rim. Raises also are occasionally put up to prospect for possible rich strata above. As stated elsewhere, the buried Tertiary channels of the Sierra Nevada are not related to the present stream system; competent geological advice is needed to plot their probable course and aid in their development.

Adits should be run at such a horizon or shafts sunk deep enough to insure drainage in the workings. Drifts generally are run upstream on bedrock to allow drainage to the shaft or out of the entrance adit. Where water is not a serious item drifts may be run both ways from a crosscut or a shaft; any water from the downstream branch is pumped into the drainage system. The breasting is done upgrade by retreating toward the shaft or crosscut. At drift mines in the frozen gravels of Alaska the common practice is to drift in both directions from a shaft.

Ideal conditions, of course, would be an even bedrock and a grade sufficient to allow drainage but not too steep for easy tramming; such conditions, however, seldom exist. A prospecting drift may be run partly in bedrock to avoid swinging it from trough to rim and back again so as to keep a practical grade for tramming. With a rapid rise of bedrock, however, as where a waterfall or rapids existed in the original stream, the drift has to be run entirely in bedrock with raises put up to prospect the gravel above or the drift continued on a higher level with a transfer point at the break. This, of course, increases the cost of handling the material. If the size of the deposit as shown by the development work justifies the initial expense, tramming drifts may be run on an even grade in bedrock and the gravel from breasting operations above dropped into raises from which it can be drawn into cars. Then the development drifts and crosscuts are used for extracting the gravel.

Sometimes drifts at different levels are run from the shafts to mine deposits at these orizons. More than one channel may be worked from the same shaft.

In shallow deposits little or no mechanical equipment may be used except for hoisting; in small-scale work hoisting also may be done by hand. The development and mining of deeply buried channels require expensive installations and usually must be done on a moderately large scale. Hoisting and pumping equipment and air compressors such as those used for lode mining are required for mining this type of deposit, as well as air drills and mechanical haulage equipment.

Shafts.- Shafts seldom have over three compartments; in small-scale work one compartment usually suffices. Untimbered shallow shafts may be as small as 2 by 5 feet, the minimum section in which a man can dig.

Sinking practices are similar to those at lode mines except that blasting is seldom done; the gravel is loosened by picking or moiling. The shaft lining usually consists of lagging back of standard framed-timber sets.[54]

Considerable water may have to be handled in sinking deep shafts in gravel, in which case ample pumping capacity is needed. Ordinary sinking pumps usually are employed. Steffa[55] has described the sinking of a 2-compartment shaft at Vallecito, Calif., in which a novel method of handling the water was used; other sinking practices at this mine, however, conformed to the general practice. He states:

> The shaft of the Vallecito Western was located at a point 50 feet north of the actual channel in order that the shaft station, at a depth of 153 feet below the collar, might be in the solid slate bedrock. At the point selected the shaft passed through 143 feet of volcanic cobble, ash, and sand and gravel before reaching the slate. It was sunk a total depth of 167 feet, providing a 14-foot sump below the station.
>
> The shaft is 4 feet by 7 1/2 feet in the clear and has one 4- by 4 1/2-foot skip compartment and a 2 1/2- by 4-foot manway. It is timbered with 8- by 8-inch Douglas fir, excepting that 6- by 8-inch material was used for dividers, and is lined with 1- by 12-inch boards.
>
> The shaft was sunk to bedrock without blasting, picks and gads being sufficient to loosen the material for shoveling. The 24 feet through rock was sunk by hand drilling, using 10 to 12 holes per round, light charges of powder, and electric delay detonators.
>
> A 12-inch churn-drill hole was sunk first at one end of the shaft to handle the flow of water which was struck at a depth of 8 feet and amounted to about 35 gallons per minute throughout the work. The hole was sunk to a depth of 187 feet and cased with perforated 7-inch inside diameter stove-pipe casing. A deep-well type of turbine pump was installed which was powered with a 20-hp. vertical electric motor, the motor resting on staging about 4 feet above the shaft collar. Three-foot lengths of pump column were used, and as the shaft deepened from day to day enough blocking was removed from under the motor support to keep the pump intake at the level of the bottom of the shaft. When blasting, during the latter part of the work, the casing and pump column, exposed in one end of the shaft, were protected from damage by a heavy plank hung from the bottom end plate directly in front of the drill hole.
>
> Numerous strata of sand and volcanic ash were encountered, one such bed at a depth of 70 feet being 7 feet thick. A large part of this fine material was carried to the surface by the pump. A test showed that at one time the pump discharge was one third sand by volume. The pump impellers wore rapidly, three sets being used. Moreover, the drill hole rapidly filled with sand to the level of the pump, after which the pump could not be lowered farther. Twice the pump was removed and the hole cleaned with a sand pump. Finally, at a depth of 75 feet, this difficulty was remedied by cutting a slot in the casing pump. As the shaft

---

54 Gardner, E. D., and Johnson, J. F., Shaft-Sinking Practices and Costs: Bull. 357, Bureau of Mines, 1932, pp. 48-60.

55 Steffa, Don, Gold Mining and Milling Methods and Costs at the Vallecito Western Drift Mine, Angels Camp, Calif.: Inf. Circ. 6612, Bureau of Mines, 1932, p. 7.

deepened the slot was likewise cut down. To secure suction with a shallow sump, such as could be dug out easily by hand in this manner, a 4-inch strainer was substituted for the original 3-foot one. The pump was run continuously and regulated by the gate valve on the discharge pipe to the exact amount of water flowing into the small sump.

It required 90 days to complete the shaft. The average progress in sinking, including timbering, was slightly less than 1 foot per shift, working two 8-hour shifts per day. The cost was $39.50 per foot. Shaftmen and the foreman received $6 per day and engineers $5. Timber and lumber laid down at the shaft cost $42 per thousand board-feet.

Drifts and crosscuts.- As used in this paper, a "drift" designates a development working parallel to the major axis of the deposit; a "crosscut" is a transverse working. This distinction is not observed strictly in the terminology of the mining districts.

Drifts may be run as small as 3 1/2 by 5 1/2 feet in section where the handling of a minimum of material is desirable. In pay dirt they may be run up to 7 by 9 feet in size or as large as they can safely be held. The size of crosscuts depends upon the service required of them.

The gravel in the ancient channels generally is compact enough to stand without timbering; blasting usually is required. The number of holes required to the round depends upon the compactness of the gravel. A simple toe-cut round - that is, one with the cut holes pointing downward - usually suffices for breaking the ground. It is desirable when blasting pay dirt in both development work and breasting to pulverize it as much as possible to facilitate washing operations. Heavy blasting, however, should be avoided so as not to scatter the gold-bearing gravel. In loose gravels the main difficulty in driving may be to prevent caves until the timbering is in place; the gravel is excavated by picks and shovels.

Wheelbarrows may be used in short drifts or buckets on trucks in small-scale work where the broken material is hoisted. In more elaborate workings, however, cars running on rails are employed.

For drifting in pay dirt, a wide drift may be run and the boulders piled at the side to form dry walls. Where timber is brought from a distance regular drift sets of square timber generally are used for supporting the drifts, but if round timber is available locally sets usually are made of it. The posts of the sets generally are stood with a batter so that the drift may be given a section more nearly approaching an arch. (See fig. 3.)

In loose or running ground spiling or forepoling must be used. The first step in spiling is to place bridging over the foremost standing set. Bridging usually consists of a 4- by 8- or 4- by 10-inch lagging laid parallel to the cap on top of 6-inch blocks at either end. This lagging is blocked solidly to the ground above, leaving a space 6 inches high above the cap through which the spiling is driven. If side spiling is necessary bridging is placed on the outside of the posts. Spiling usually consists of 2- to 5-inch timber 4 to 10 inches wide and as much as 9 1/2 feet long, depending upon the weight to be borne and ease of driving; one end of the spiling is sawed as shown in figure 4. The top spiling is driven at an upward angle into the caved or loose ground. In mines having compressed air a drilling machine with a special tool may be used for driving the spiling. The spiling extends over the cap far enough to provide room for placing a complete set. The upward angle is sufficient to allow bridging to be placed over the new set. The first spiling usually is driven at one side of the bridging close to the bridging block at such an angle that the forward end when in place will be 6 or 8 inches beyond and above the cap and close to the wall. The remaining ones are driven at such angles that they "fan" and form a complete covering for the set of timber to be put in place. As each spiling is driven ahead some of

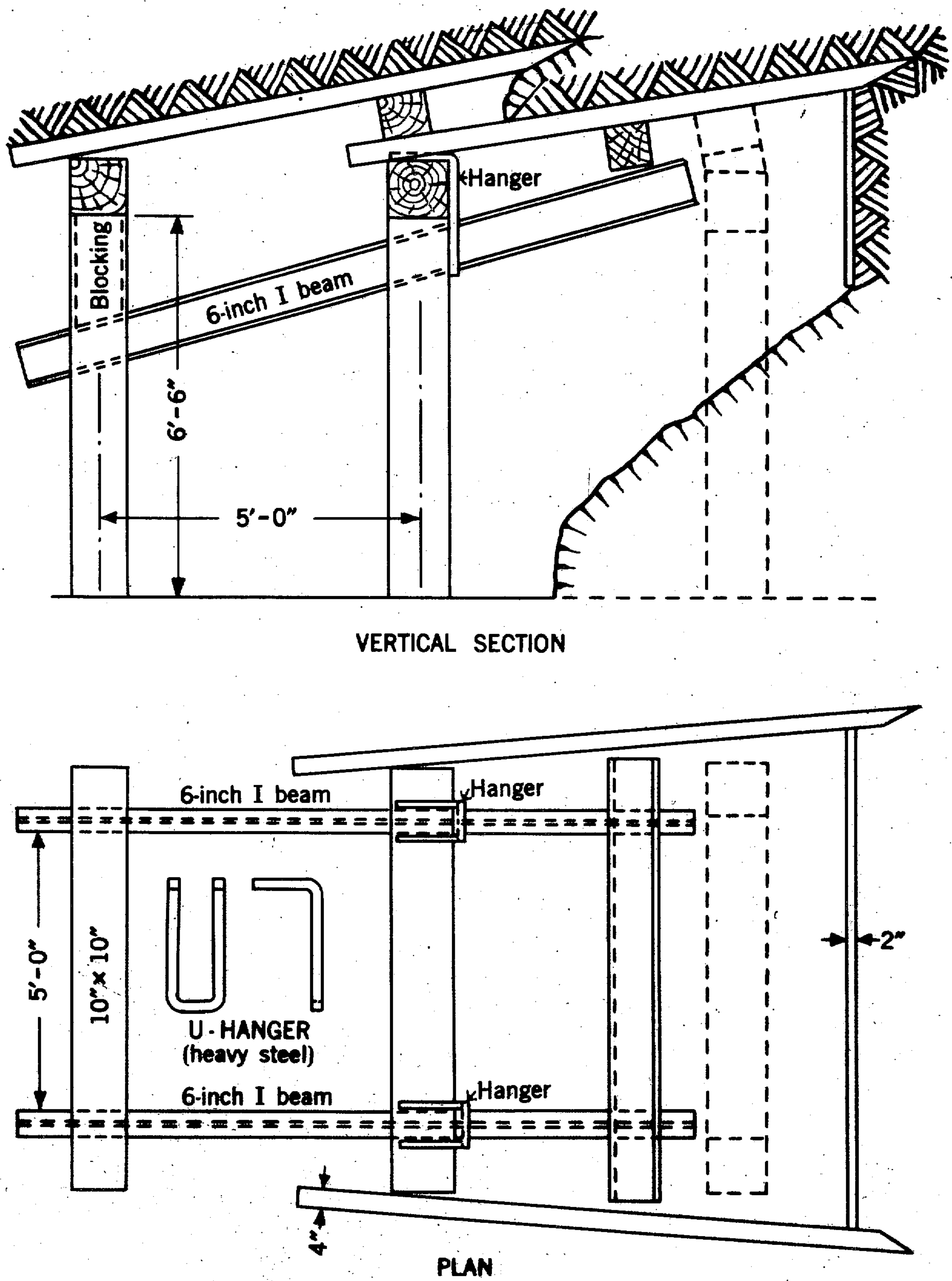


Figure 4.—Method of spiling in loose ground.

the gravel is cleared away from underneath it so that if any large boulders are encountered ahead of the spiling they can be barred out of the way or taken down. After the top spiling is in place side spiling, if necessary, is driven in the same manner, beginning at the top. Two 6-inch I-beams, or heavy timbers, are then hooked on the last cap (fig. 4) by heavy steel hangers. The ends of these beams are extended forward to just back of where the cap of the next set will be when in position. A crosspiece is then placed across their forward ends and brought up snugly against the spiling; the back ends of the beams are blocked down under the second cap back. When the gravel is removed the next set is put in. The beams support the top spiling while the set is placed. Posts and caps of ordinary drift sets are used.

The same method of top spiling is used for breasting in running ground. The I-beams or timbers with overhead lagging may be used in firmer ground to protect men working ahead of the last set in position from falling material, both in drifting and breasting operations.

Steffa[55] gives the drifting practice at the Vallecito drift mine as follows:

> Both gangways and crosscuts are generally 7 by 7 feet in section. The usual drill round consists of six holes drilled 5 or 6 feet deep and breaking an average of 4 feet per round. The gravel drills easily, 2 1/2 hours generally being sufficient to drill the round. Drill steel is of 7/8-inch hollow-hexagonal material, sharpened with cross bits. Slightly more than 9 pounds of 25-percent strength powder is used per round, with 4 sticks in each of three lifters, 3 sticks each in the two cut holes, and 2 in the single back hole. Caps are treated with a standard waterproofing compound.
>
> The broken gravel is shoveled by hand into 18-cubic-foot, end-dump, roller-bearing cars holding 1 ton each. Track consists of 16-pound rails laid to 18-inch gage. The grade of the channel has proved uniform over considerable distances and averages 75 feet to the mile. Track has been laid therefore on a grade of 1 1/2 percent upstream. It has seldom been necessary to take up bedrock to maintain the grade; wherever a dip in the floor has been found the track has been kept on grade, and bedrock has always been found at the expected elevation when reaching the opposite side of the dip.
>
> In the opening of new areas by drifts or crosscuts, samples are taken from the skip at the collar of the shaft, a sample consisting of one full pan or about 20 pounds of gravel. Samples taken at this point have the advantage, as compared with samples taken from the solid face, of being representative of a larger volume of ground and of being mixed thoroughly by the blasting and by the handling of the gravel from muck pile to car and to skip. Thus an experienced panner is able to make fairly accurate estimates of the value of the gravel developed.
>
> Drifts and crosscuts are driven by crews of three or sometimes four men, making an average advance of 4 feet per shift. The cost of driving main headings averages $16 to $17 per foot. In a pay area 65 feet wide, where gravel can be breasted 10 feet high, each foot of heading developed 45 tons of gravel. (It is estimated that the gravel expands one quarter on being broken, and a ton of broken gravel has a volume of 18 cubic feet.)

The cost of running a drift under average conditions at a small-scale mine where no other work was being done was shown by the Golden Belt Gold Mining Co. which was developing a drift mine on Magpie Gulch near Helena, Mont., in the summer of 1932. An 80-foot shaft

55 Steffa, Don, Gold Mining and Milling Methods and Costs at the Vallecito Western Drift Mine, Angels Camp, Calif.: Inf. Circ. 6612, Bureau of Mines, 1932, pp. 8-9.

had been sunk, and a drift was being run up the channel; the drift was 160 feet long and 5 by 6 feet in section and was timbered with 8-inch round timber sets placed 4 feet apart. The top and sides of the drift were lined with split lagging. The timber was cut and sawed on the ground. The gravel was picked by hand and trammed in a 6-cubic-foot car. It was hoisted in the body of the car, which at the surface was placed on a truck and trammed to the washing plant. The cost of running the drift was $6 per foot, excluding supervision. The surface equipment at the shaft consisted of a headframe and a hoist run by a 15-hp. electric motor. Power cost 1.07 cents per kilowatt-hour.

An example of the cost of running a drift under adverse conditions in small-scale operations was illustrated at the Lucky Charles Mining & Milling Co. small drift mine on North Clear Creek, Blackhawk, Colo., which in July 1932 was being developed through a 40-foot 2-compartment shaft; 50 feet of drift had been run but no breasting done. The property was well equipped with an electric hoist, a deep-well pump, a substantial headframe, and an ore bin. About 20 gallons of water was being pumped per minute. A 10-hp. motor operated both the pump and a hoist which had an 18-inch drum. The gravel was hoisted in a 7-cubic-foot bucket attached to a 1/2-inch cable.

The gravel was 3 to 5 feet thick and was overlain with 5 feet of quicksand which required both top and side spiling. The drift was 6 feet high, 5 feet wide at the top, and 6 feet wide at the bottom. Sets of 6-inch round timber were placed 2 feet apart. Top spiling was 3 by 6 inches by 5 1/2 feet; side spiling was 1-inch boards.

An advance of 1 foot per day was being made by 2 men underground and 1 man on the surface. The cost per foot of drifting was as follows:

Labor (3 men at $4)	$12.00
Power (hoisting)	1.00
Timber	1.80
Other supplies	1.00
Total	15.80

Breasting

A number of different methods of breasting are employed at drift mines, depending mainly upon the nature of the deposit. Drift-mining methods were evolved in the early Californian diggings; present methods do not differ materially from those of the early days.

In narrow channels the gravel may be mined on either side of the drift as it is advanced or the drift advanced the full width of the pay streak. In wider deposits the drift may first be run to the limit of the deposit and then the gravel mined, retreating toward the shaft. In extensive deposits the gravel usually is divided into blocks preparatory to mining. The blocks generally are mined by retreating. Pillars usually are employed only to protect haulageways. A modified room-and-pillar system, however, has been used at some mines in which the pillars, if in rich gravel, were later removed.

Breasting may be done from crosscuts or from drifts run parallel to the haulageway. Breasting from the crosscuts may parallel the haulage drift on the retreat toward it. When working from drifts the line of retreat usually is parallel to the drift although sometimes toward it. The spacing of crosscuts or drifts at different places ranges from 40 to 200 feet, depending mainly on the system of breasting. Crosscuts generally are turned off at such an angle as to give the proper gradient for tramming.

Cuts or slices range from 2 1/2 to 8 feet wide. If cars are used in long faces the tracks are shifted after each cut. Usually all of the gravel rich enough to mine and enough of the overlying gravel to provide headroom is taken out. Rooms generally are 6 or 7 feet high; the minimum height in large operations is 5 feet. At the Vallecito mine, described later, the thickness of the pay dirt varied up to 14 feet, although at most mines it was less than 6 feet. The rooms may be broken to a strong strata of ground where such strata occur. In some California drift mines volcanic ash makes a strong roof.

In compact or cemented ground the breasts are broken by blasting drill round; holes may be 2 1/2 to 6 feet apart. At most places, however, breasting is done with picks. At many places the gravel is undercut, usually in the upper and softer part of the bedrock; the remaining gravel in the face is then broken to the undercut. Usually 1 or 2 feet of bedrock is taken up. Often, bedrock with deep crevices containing gold can be picked. Hard bedrock is cleaned carefully by hand, as in surface mining. Boulders and gravel too low in grade to take out are piled back of the working face.

Low-built cars usually are preferred for the sake of easier shoveling and tramming in the low workings. Scrapers in drift mines have not proved successful, but with the recent improvements in equipment and technique this method of moving gravel offers possibilities.

Some timbering usually is required in breasting, if only an occasional stull which may be recovered later. Regular timbering consisting of stulls with headboards is used at most mines. If the bedrock is soft, footboards also are used; in soft ground lagging is required overhead. Heavy ground generally is supported by lines of sets. In narrow channels tunnel sets with long caps may be used.

The following descriptions of individual mines illustrate current breasting practices, beginning with the simplest form and progressing to more elaborate operations. Two mines worked in the early days in California and one in Alaska are included to show typical methods not illustrated by modern operations.

Representative Mining Practices

Recent gravels

Greaterville.- Simple and more or less haphazard methods of drift mining have been followed in the Greaterville district in southern Arizona for many years. Along the poorly defined channels or pay streaks untimbered shafts 6 to 15 feet deep are sunk at random to bedrock, which often is a clay stratum. At the bottom of the shaft, galleries are run about 3 1/2 or 4 feet high in the middle and tapered down to the thickness (usually less than a foot) of the pay streak on either side. The pay streak of gravel is then gouged out as far as it can be reached. The gravel is fairly compact, and no timber is used in the workings. The pay gravel is scooped into pails and generally hoisted hand over hand; in the deeper shafts a hand windlass is used. The galleries are extended as far as pay streak can be followed economically by such primitive methods or until the working becomes unsafe, then a new shaft is sunk. The dirt too low in grade to wash is piled in old workings. The ground is dry, and no running water is available except after storms. The gravel usually is washed in rockers, for which water is packed on burros to the workings.

Bear Creek.- Two men were drift mining in a bench deposit on Bear Creek above Bearmouth, Mont., in July 1932. Short adits were run across the bar on bedrock which was there about 40 feet above the creek. About 2 feet of gravel on bedrock was then gouged out by hand for 6 feet on either side of the drift. Boulders removed in this work were used as rock packs to help support the back. No timber was used. When all the gravel that could be mined safely was removed from one drift another parallel working would be run.

The gravel was taken in a wheelbarrow to the surface where it was dumped over a 2-inch bar grizzly. The undersize dropped into a 2-cubic-yard bin whence it was taken in a wheelbarrow to a sluice box on the creek and washed. About 3 tons per day or 1 1/2 tons per man-shift was mined and washed.

Magpie.- In deeper gravel more systematic methods are followed, as shown at the Magpie Gulch, south of Helena, Mont. Here the gravel was 40 to 50 feet deep. A drain tunnel had been extended upstream about a mile until bedrock was reached; it was then continued on bedrock. Working shafts were sunk 100 to 200 feet apart, and the gravel was hoisted to a washing plant. As the drift was extended upstream from the working shaft the gravel was breasted out; when ventilation became a problem a new shaft was sunk or raised and the surface plant moved. The water continued to flow out the drain tunnel. Below the point where the drain tunnel reached bedrock shafts were sunk in the same manner, but pumping was necessary.

Dakota mine.- As stated previously, a narrow deposit can be mined by advancing an adit the full width of the channel. This practice was followed by Anton Gustafson and three partners at the Dakota mine on the head of Quartz Creek in the Cedar Creek mining district near Rivulet, Mont. (See fig. 5,A.) The drift started in the face of old hydraulic diggings under a cover of about 80 feet. In July 1932 it was 450 feet long. Ventilation at the face was poor, and an air raise to the surface would soon be required. The channel was 6 to 20 feet wide. Seven feet of gravel was mined; it contained a large proportion of boulders. The tops of clay streaks in the gravel contained relatively high gold values.

The method of mining was as follows: Sets 4 feet between centers were placed as room was made. Top lagging 4 1/2 feet long, consisting of split poles, was driven ahead as the ground was picked out. A false set held up the top lagging until the regular set was placed; no side lagging was necessary. Round timber cut on the ground was used for sets; the caps were 12 to 15 inches, posts 9 to 12 inches, and girts 6 inches in diameter. Caps were 10 to 14 feet long, depending on the width of the channel; a minimum width of 10 feet was required to provide room to stack boulders. All rock over the size of a man's fist was left behind, except occasional large boulders for which room was not available. Dry walls were built up on either side of the 18-inch gage track, leaving barely room enough to push out an 8-cubic-foot car 3 feet wide. Some boulders too large to move by hand were drilled and blasted; about 150 pounds of 40-percent-strength gelatin dynamite was used per year. A derrick at the portal of the tunnel was used to dispose of large boulders brought to the surface.

Two men worked on each of two shifts and brought out 7 or 8 cars per shift. A set was put in every 2 days, including the time for cutting the timber. A set of ground averaged 13 feet wide, 7 feet high, and 4 feet long and contained 13 cubic yards or 20 tons of gravel. An average of 2.5 tons of gravel was mined per man-shift; including the time for washing the gravel. At $4 per shift the labor cost per ton of gravel mined at the face amounted to $1.60 per ton. Thirty cubic feet or 1.7 tons of gravel was brought to the surface each man-shift. The total labor cost, therefore, was $2.35 per ton of material trammed. No hoisting or pumping was necessary, and there was no overhead nor cost of supervision. The total cost for supplies was about 25 cents per ton, making a mining cost of $2.70 per ton of material trammed. Seven months of preliminary work was necessary before any gold was produced. Considerable excavation was required in the face of the old hydraulic workings to get down to bedrock, and the adit was driven through 150 feet of previously drifted ground to reach virgin gravel.

Townsend and Hornbrook.- Relatively small-scale drift mining by a retreating method with inexpensive equipment is illustrated by the work of Townsend and Hornbrook on the Klamath River a few miles south of Hornbrook, Calif., in 1932. The mine was worked through a 46-foot, 2-compartment shaft, situated about 50 feet from the river. The first 20 feet of the shaft was concreted to hold out surface water; the lower 26 feet was cribbed with 3-inch plank. Although the lower 12 feet was said to be gold-bearing, only 6 feet, which carried most of the values, was mined. Drifts stood without timber, except that an occasional stull set was necessary. In preparation for breasting, a drift was run to the limit of the area to be worked, then the ground was taken out on either side 10 feet from the center of the drift. (See fig. 5,B) retreating toward the shaft. Two rows of stulls 4 feet apart with

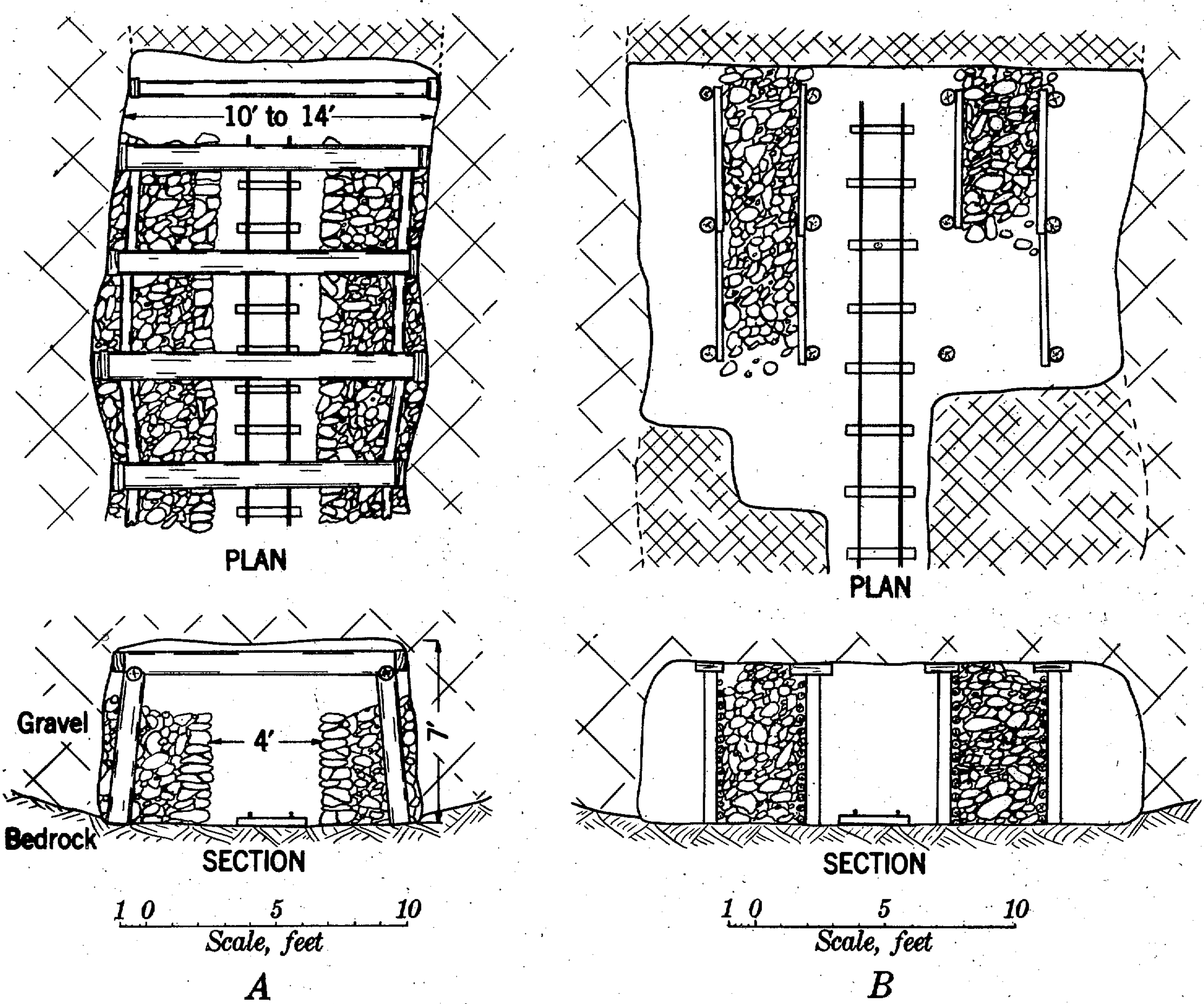


Figure 5—Breasting methods in narrow channels: *A*, Advancing from shaft, Dakota mine, Rivulet, Mont.; *B*, retreating to shaft, Townsend and Hornbrook mine, Hornbrook, Calif.

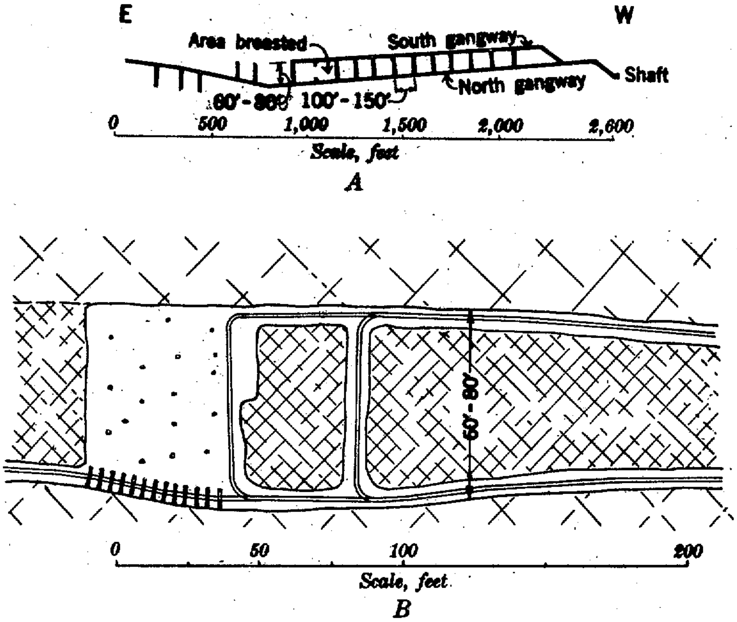


Figure 6.—Mining methods at Vallecito Western mine, Vallecito, Calif.: *A*, Plan of principal workings; *B*, breasting method.

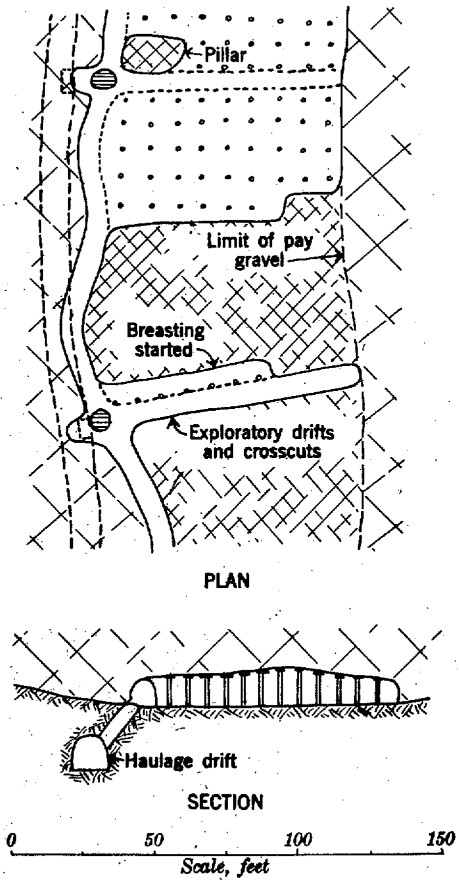


Figure 7.—Proposed method of breasting, Calaveras Central mine, Angels Camp, Calif.

5 feet between the rows were placed on either side of the drift. Boulders were piled between the two rows of stulls on either side to form a solid pack to the back. No drilling or blasting was necessary.

The surface equipment consisted of a 15-foot headframe made of round timber cut on the ground, a 7-hp. gas engine, a small hoist, a pump, a 500-gallon tank, and a sluice box. The hoist and pump were run by the same engine. About 30 gallons of water per minute was pumped from the mine. The gravel was hoisted in a 1/4-cubic-yard bucket on a 3/8-inch cable.

The crew consisted of 2 men underground, 1 hoist engineer, and 1 man who attended to the washing. Five gallons of gasoline was used per 8-hour shift for pumping and hoisting. The average production was about 6 cubic yards or 9 tons per day.

The daily operating cost of mining was as follows:

Three men at $4	$12.00
Gasoline, 5 gal. at 16 cents	.80
Timber	1.20
Miscellaneous	2.00
Total	16.00

The daily washing cost was:

One man	$4.00
Supplies	.50
Total	4.50

Then mining costs would be $1.78 and milling costs $0.50 per ton, or a total operating cost of $2.28 per ton. The cost of sinking the shaft and of the surface equipment must be prorated to each ton mined to obtain the total mining cost.

Ancient gravels

As stated before, relatively large-scale operations are necessary for operations to be profitable in mines in the ancient channels in California.

Vallecito Western.- The method of developing a single channel is illustrated at the Vallecito Western drift mine at Angels Camp, Calif. Up to June 1932 the mine had not been put on regular production. The channel gravel, buried by volcanic ash and late sediments, was well compacted, and explosives were required to break it. Drifts stood without timbering. The mine was worked through a 2-compartment timbered shaft 153 feet deep. Gravel was hoisted in skips from a single level. Surface equipment consisted of a hoist, air compressor, blacksmith shop, and washing plant. The trommel in the mill was driven by a 10-hp. electric motor; a 15-hp. motor was required for pumping.

The general plan of development, breasting methods, drainage, and costs of mining and milling are described by Steffa as follows:[56]

> A diagonal crosscut was run southeast from the shaft to the north or near side of the channel and a drift started upgrade (east) along the north rim (as shown in figure 6,A). About 600 feet east of the shaft a crosscut was driven through the pay gravels to the south rim. Bedrock dropped away southward, and a

56 Steffa, Don, Gold Mining and Milling Methods and Costs at the Vallecito Western Drift Mine, Angels Camp, Calif.: Inf. Circ. 6612, Bureau of Mines, 1932, pp. 8, 10, and 13.

winze at the south rim showed it to be 5 1/2 feet lower on this side than on the north. This was due probably to the channel here being entirely in slate, which had permitted the cutting of a deep trough next to the abruptly rising south rim. To the eastward a harder granite floor gradually encroached from the northward upon the channel until it covered its entire width, whereupon the bedrock assumed equal elevations on both sides. However, it was necessary to reach grade on the south side, and therefore a crosscut was started, as shown in the diagram, about 250 feet from the shaft. At this point the site of an ancient waterfall had been encountered with a rise of about 5 feet, and the grade of the north gangway had been raised correspondingly by the installation of a transfer platform. Loads coming down were dumped at this point through a hole in the platform into cars, which were then trammed to the shaft. The crosscut was therefore extended along the west or low side of the falls and at the south rim was turned east and driven in slate on a grade to intercept the bottom of the deep trough discovered at the first crosscut.

Crosscuts are run at intervals ranging from 100 to 150 feet. A total of 44 have been driven to date (August 1931), 18 of which were in the first or westernmost of the pay areas developed. Of these, several connected to the north and south drifts or gangways, serving both to improve ventilation and to speed up the work of breasting. The other crosscuts were projected away from the pay areas onto benches and were extended short distances up the rims to prospect for potential concentrations. The total footage of drift and crosscut to date is 6,300 feet.***

Figure 6,A indicates the location of the one area breasted so far. This averaged 65 feet wide, ranging from 60 to 80, and was 240 feet long. Near its center the pay gravels extended to a height of 14 feet and were extracted to that distance above the floor.

Breasting began at the upper end, the gravel being broken down along the side of a crosscut (fig. 6,B). Holes 6 feet deep spaced 4 feet apart in two rows, one at the top and the other at the bottom, were drilled across the face. Light explosive charges sufficed to make a clean 6-foot break and to loosen a foot or two more of ground to be picked down by hand. Heavy blasting is avoided because of the scattering effect on the fine gravel and its gold content.

The gravel is compacted so strongly that it stands without scaling over great widths with only occasional light stulls for support. The stulls are 8-inch round timbers set about 10 feet from the face, topped by headboards or caps, and wedged tight to withstand blasting. In the entire area breasted only 48 stulls were used. The roof is arched from a height of as much as 14 feet in the center to 7 to 10 feet at the sides, which increases its strength and tends to prevent sloughing.

As soon as the first slice is broken down along the crosscut muckng begins. The gravel is shoveled by hand into the cars. Large boulders, constituting about 30 percent of the whole mass, are rolled back from the face and sometimes stacked up to the roof to furnish additional support. Very heavy boulders, weighing from a few hundred pounds to several tons, are rare. Fully 80 percent of the total weigh less than 100 pounds.

The top of the pay gravel is defined by a capping of coarse sand. Horizontally, the extent of breasting is controlled by pan sampling underground, the number of colors in a single pan indicating to an experienced gravel miner the approximate value of the ground. In places at this mine the pay lead is heavily concentrated and narrows to a width of 20 feet with barren ground on both sides. At others, as noted, the width of pay gravel is 80 feet. The width of face is varied accordingly.

The gravel is trammed by hand in 1-ton (18-cubic-foot) cars to the shaft and dumped directly into a 1 1/2-ton skip. The skip raises the gravel to the top of the 80-foot headframe where it is dumped into a 75-ton gravel bin.

The breasting operation was conducted to discover by a mill test the actual value of a given large mass of gravel, as well as to learn the north-south limits of the pay streak. Only one face was attacked, whereas in regular operation each crosscut would give a starting point for two faces. In full-scale operation, moreover, mechanical loading at the breasts and motor haulage should lower the cost of operation.

Breasting operations extended over a 10-month period, during which development work also was being pushed. The tonnage from breasting was segregated and treated separately, totaling 9,500 tons. Five men breasted and trammed 1,300 feet to the shaft, an average of 5 tons per man shift. Powder consumption averaged 1/2 pound per ton, and the timber cost was 1/2 cent per ton.***

The gravel is treated in a plant near the collar of the shaft having a capacity of about 15 tons per hour.***

The gravels are wet when first opened but drain rapidly, and at present the flow of approximately 48,000 gallons per day is confined to bedrock, flowing between rails in the drifts. Drips from walls or roof are found only occasionally. At the shaft a vertical centrifugal pump, driven by a 10-hp. motor, is mounted in the manway about 10 feet above the station level. This pump has a capacity of 100 gallons per minute. It is controlled by a float switch and handles the regular mine drainage with about 8 hours of pumping per day. A second turbine of double that horsepower and capacity is installed at the opposite side of the shaft ready for use in emergency. Power is taken from a Pacific Gas & Electric Co. line which passes 600 feet south of the shaft.

The following costs are for combined extraction and milling of the 9,500 tons taken from the area breasted as described above. Prevailing wages during the 10 months in question were $4.50 a day for muckers and trammers and $5 for miners. The costs, apportioned to the mining and treatment of the breasted gravels, excluding development but including all other operating costs of mining and milling, were as follows:

Cost of mining and milling

Labor	$2.02
Supervision and insurance	.40
Explosives	.25
Timber	.01
Power	.30
Other supplies	.12
Total, per ton	3.10

Milling at this mine is discussed later.

Calaveras Central.- The Calaveras Central mine at Angels Camp, Calif., was reopened in 1931 by the Calaveras Central Gold Mining Co. Up to June 1932 underground work had been confined to developing new deposits; regular breasting operations had not begun. Twenty men were employed.

The gravel was similar to that in the Vallecito Western mine in the same district. It varied up to 21 feet thick, averaging 7 feet, and was overlain by an average of 350 feet of volcanic ash and later sediments. The mine was well equipped and had a complete surface plant. Air was furnished by an electric-driven, 350-cubic-foot-per-minute air compressor; the hoist and pumps were also electrically driven, and the mine was lighted by electricity.

The mine was operated through a 350-foot, 3-compartment shaft. The gravel was trammed by two 4-ton battery locomotives in 2-ton side-dump cars and hoisted in two 2 3/4-ton skips. A total of 250,000 gallons of water was pumped per day.

Two levels 100 feet apart had been opened up. Development and exploration drifts were run on bedrock and extraction drifts in the bedrock, which was slate; drifts were 7 by 7 feet in section. In the gravel a 10-hole round 5 1/2 feet deep could be drilled and blasted and the broken gravel loaded out in a shift. Ten to fourteen holes were required in slate, and an average of 1 1/4 shifts was taken to a round. Drift rounds usually were loaded by a mechanical shovel. Drifts or crosscuts were not timbered. Crosscuts were run from the exploration drifts preparatory to breasting.

In breasting, slabbing rounds 2 to 6 feet wide will be blasted from the sides of the crosscuts and the gravel dragged by scrapers into raises from footwall drifts. (See fig. 7.) It will then be drawn into cars for tramming to the shaft. Boulders will be piled to one side. Pillars will be left where necessary and weak places in the back held up by stulls. In rich ground the pillars will be robbed upon retreating. The normal capacity of the mine would be about 75 tons of gravel per 8-hour shift. Development costs are estimated at 50 cents per ton and mining and milling costs at $2.50 - a total of $3.00 per ton. The wage scale in July 1932 was $5 for shovelers and $5.50 for miners.

<u>Hidden Treasure</u>.- The Hidden Treasure in Placer County, Calif., is reported by Powers[57] to have been the largest deep-drift mine in the world. He states that the channel gravel was not cemented and therefore required little or no blasting. The slate bedrock had a tendency to swell when exposed to the air. The channel system was mined for a distance of about 4 miles and for a width ranging from 200 to 800 feet. In the early days of the mine 1 1/2 to 2 million board-feet of timber was used yearly. Timber was plentiful and was cut on the company's holdings. The mine was operated through adits. The timbering of the main tunnel, on a swelling bedrock, is shown in figure 3,B.

In preparation for beasting, gangways were driven to the rim rock at right angles to the haulage-way adit and at intervals of about 200 feet. Drifts were then run 110 feet upstream and 90 feet downstream from the ends of the gangways. The gangways were timbered with sets using 10- by 10-inch posts 6 feet long and caps of the same material 5 feet long; the lagging was 1 1/2 by 6 inches by 5 feet long. The drifts on the rim were timbered with 8 by 8-inch sets with both the posts and caps 5 feet long; the same lagging as that above was used.

Breasting began on the inside of the rim rock drifts and proceeded toward the main tunnel. (See fig. 8,A.) The breasts were timbered in the same manner as the drifts with the sets 5 feet apart. If the ground was heavy, as it usually was, a line of posts and caps of the same size as those used in the sets was placed under the lagging midway between the row of sets. In addition, it was often necessary to place a center post under the caps of the sets; thus the lagged back was held up by posts on 2 1/2-foot centers. Moreover, if the ground was wet foot blocks were required. Lagging was driven forward ahead of the set nearest the face; it was held up by a false set until the regular set was placed. All waste and boulders excavated in breasting were piled into packs for further support of the roof. The roof was maintained only at the working faces.

---

57 Powers, Harold T., Timbering in Deep Placer Mining: Min. and Sci. Press, vol. 115, Aug. 11, 1917, p. 19.

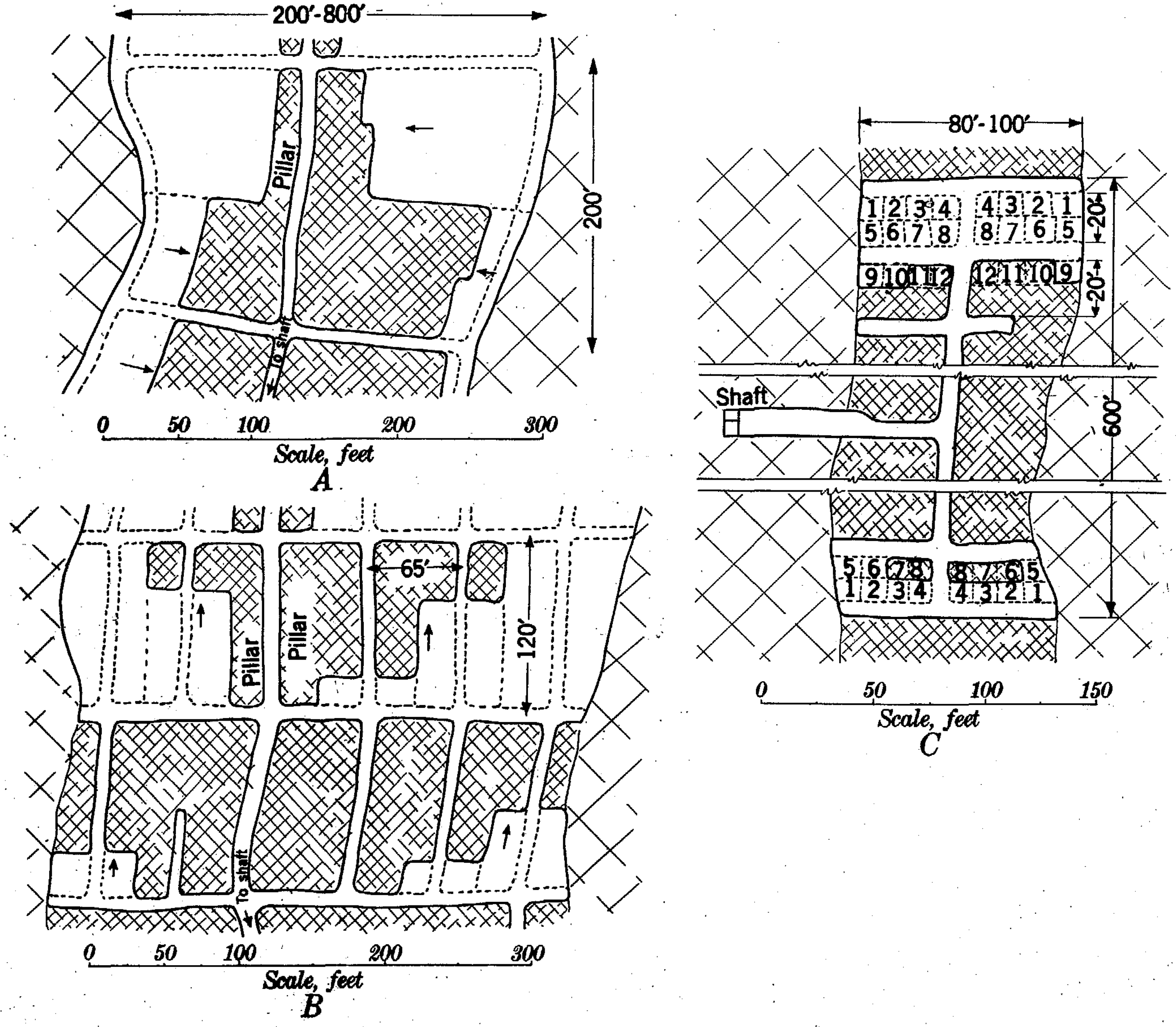


Figure 8.—Breasting methods used in former drift mines: *A*, Hidden Treasure mine, Placer County, Calif.; *B*, Red Hill mine, Placer County, Calif.; *C*, Wild Goose mine, near Nome, Alaska.

Red Point.- The method of blocking out the ground, used at the Red Point mine in Placer County, Calif., is given by Dunn.[58] A main haulage drift, which was kept as straight as possible, was run in the center or lowest depression of the channel. Gangways were then run on about 120-foot centers at right angles to the rims of the channel or the limits of the pay lead. (See fig. 8,B.) The gangways were connected with drifts parallel to the haulage-way on 65-foot centers, thus cutting the gravel in blocks 65 by 120 feet in size. The ground was hard and compact, and except in the breasts openings required no timber.

In the Bald Mountain mine at Forest City, Calif., the practice was to run both the gangways and drifts 80 feet apart, leaving a pillar of 40 feet to protect the main tunnel.

Wild Goose.- The mine of the Wild Goose Mining & Trading Co. was situated in the Nome district of Alaska. It was being worked in 1905. The gravel was not frozen and was very difficult to hold up. The bedrock was a micaceous schist; from 1 1/2 to 2 feet of bedrock and 2 feet of gravel overlying bedrock were run through sluice boxes to save the gold. Timber was costly ($60 per thousand feet plus the freight from Nome) and difficult to secure. A method in which a minimum of timber was required was devised to mine this gravel. According to Fleming[59] the method of working was as follows:

> The channel, which ran nearly straight, making it easy to follow, was divided into sections 600 ft. long, and a shaft was sunk midway between the end lines of each section, care being taken to sink the shafts on the lower side of the channel and not nearer than 25 ft. from the lower rim so that caving would not affect the alinement of the shafts when the channel was worked out, and always permitted our shaft houses, bunkers, and strings of sluice boxes to be on solid ground. The shafts were 6 by 8 ft.; they had a single compartment, with a manway for ladders and pipes. They were from 69 to 140 ft. deep; 140 ft. being the depth to bedrock on the crest of the divide.
>
> A drift was run from the bottom of each shaft at right angles to, and toward, the middle of the channel. From the shaft this drift was wide enough for a double track for a distance (usually about 40 ft.) sufficient to give a sidetrack for holding empty cars. When the middle of the channel was reached the drift was split and continued lengthwise with the channel a distance of 300 ft. each way, thus covering the section of 600 ft. into which the channel had been divided.
>
> At each end of the main gangway the drift was teed by a crosscut to each rim. The channel was from 80 to 110 ft. wide. At first we tried breasting as described by Mr. Power (Hidden Treasure mine), using 10- by 10-in. and 12- by 12-in. posts, placing them on 3-ft. centers, but we soon abandoned this method as we could not recover any timber when once placed and it proved dangerous. One breast or stope, after it had been opened clear across the channel, caved in a rush to the surface, and we not only lost the breast and all the timber but nearly lost the men who were working in the face.
>
> We then devised the method of block-stoping. This proved successful, permitting us to save all our timber, using it over and over again, only now and then losing a stick. To show to what extent we succeeded, we worked out the entire channel in one 600-ft. section, losing only 5,000 ft. B. M., for we waited until fall, when the frost set the gravel walls of the shaft. We even pulled the

58 Dunn, R. L., Eighth Annual Report of the State Mineralogist of California. 1888, quoted by Haley, C. S., Gold Placers of California: Calif. State Min. Bur., Bull. 92, 1923, p. 58.

59 Fleming, E. E., Block Stoping and Timbering in Deep Placer Mining: Min. and Sci. Press, vol. 115, Sept. 15, 1917, p. 378.

shaft timbers. This method may have been used elsewhere before, but it was original with us at the time and was adopted afterward throughout the Nome district in all the deep mining in thawed ground. Frozen ground did not present any difficulties, as no timber was required.

Our greatest difficulty was caused from swelling bedrock in the main gangways. To overcome this trouble we used the methods described by Mr. Power (Hidden Treasure mine).

In block stoping we commenced at both ends of each T and took out a block of ground (each marked no. 1 in fig. 8,C), never more than 10 ft. square, using 8- by 8-in. posts set on 3-ft. centers, with 3- by 12-in. caps 1 1/2 ft. long, never permitting the caps to cross two posts. When the gravel was so loose that it ran, we filled the spaces with false lagging. When this 10-ft. square block was out and the bedrock cleaned up, we pulled every post (using double or triple block and tackle) except the row against the two solid walls of unworked ground on two sides of the block. Behind this row we lagged solid with 2- by 6-in. lagging against the unworked walls. Usually the roof caved as fast as we pulled the posts and the block was immediately filled. Then we took out all blocks marked no. 2 and worked back toward block no. 1 (which we had just left) until we struck the wall of lagging, cleaned up the bedrock, and pulled as before, repeating until the main gangway was reached.

In the meantime new crosscuts had been cut from the main gangway to the rims, leaving a pillar 20 ft. wide of unworked ground to make a new T. The blocks were worked out as before. We kept these crosscuts just ahead of the stopers. This method permitted us to work in four places in the mine, besides the crosscuts, and was sufficient to keep the hoist on a single-compartment shaft busy.

We gradually worked back toward the shaft and, as before stated, recovered practically all our timber and never lost a man, the secret being that we always worked under solid ground and moved ground so quickly that it had no chance to get heavy and take weight.

Milling

As stated before, nearly all gravel mined by underground methods requires some mechanical or hand method of washing to disintegrate it and free the gold from the clay. In recent stream gravels little preliminary washing may be necessary, while in some of the Tertiary channels the gravel must be crushed by machinery to free the gold. The simplest washing device is a box at the head of a sluice in which the gravel is puddled by hand with hoes, shovels, or rakes or washed by water from a nozzle. Washing is performed in trommels at a majority of mines. The screening out of the coarse material assists in the gold-saving operation.

Gold-saving devices other than sluices or amalgamation plates have been used or have been proposed for treating gravels from drift mines, but none was in actual use in June or July 1932, and no first-hand information as to their efficiency could be obtained. Amalgamation plates were used successfully for treating stamped material in one mill, but they are not suitable for anything except screened material as gravel or coarse pebbles scour off the amalgam.

Milling methods are illustrated at the following representative plants visited by the authors in 1932. It is the authors' opinion that these plants represent the best practices under the conditions given.

Representative milling practices

Dakota.- At the Dakota mine, previously described, gravel was brought to the surface from the adit and dumped from an 8-cubic-foot car directly into the head of a sluice box. When much clay was in evidence the gravel would be partly puddled by hand. The first 50 feet of the sluice was built inside the adit for protection against snow and frost in winter. The sluice line was 250 feet long; the first two boxes were 20 inches wide and the rest 10 inches. The grade was 5 inches in 12 feet. Riffles were used only in the first 96 feet of the sluice. Concentrates were panned; no quicksilver was used. Water was brought into the sluice by a pipe line from the creek. An ample supply was available.

Ralston.- Two men were working a drift mine on Magpie Gulch near Canyon Ferry, Mont. The gravel was dumped in a puddling box where it was disintegrated and freed of clay by means of a hoe, rake, or shovel. After it was washed a gate was opened and the material run into a sluice box 36 feet long.

Lucky Charles.- The gravel from the Lucky Charles mine at Blackhawk, Colo., was pulled from an ore bin into an iron box 8 feet long, where it was puddled by hand to dissolve the clay. All gravel over 1 inch in diameter was then forked out and the remaining material run through a 10-inch box 24 feet long, set on a grade of one half inch to the foot. Riffles consisted of 1/2-inch round iron bars set one half inch apart lengthwise in the box, held at intervals by 1/2-inch iron crosspieces that fitted tightly on the bottom of the sluice. By the time a batch of gravel was puddled the box was full of water; the gate was then opened and the stored water assisted the regular stream of 20 gallons per minute in carrying the material through the sluice. The hoist engineer did the sluicing.

Townsend and Hornbrook.- At the Townsend and Hornbrook mine near Hornbrook, Calif., the gravel was dumped from 1/4-cubic-yard buckets onto an inclined grizzly of 18-pound rails laid upside down with 1 1/4-inch spacing between. The gravel was washed on the grizzly by a spray of water under about a 10-foot pressure from a tank. The oversize dropped into a car which was pushed by hand to the rock dump. The undersize went through three 16-foot sluice boxes 12 inches wide, set on a grade of 1 1/4 inches to the foot. Steel matting was used for riffles. No quicksilver was used. Water pumped from the mine shaft (30 gallons per minute) was used for washing. It was stored in a tank and only turned into the sluice when gravel was being dumped. The top crew consisted of a hoist engineer and sluice tender.

Milling costs for a production of 9 tons per day were $4.00 for labor and $0.50 for supplies, making a total daily cost of $4.50, or $0.50 per ton. The pumping of water was charged to mining.

Baker Divide.- The Baker Divide drift mine is near Michigan Bluff, Calif. The washing plant consisted of a washing bin, where the gravel was disintegrated with water from a high-pressure hose, and a line of sluice boxes. (See fig. 9.) Boulders over 6 or 8 inches in size were sorted out underground. The gravel was dumped from mine cars along one side of the bin, which was 10 feet deep, 11 feet wide, and about 12 feet long with an open end. The bottom sloped from both sides to a 10-inch sluice box, 3 1/2 feet from one side.

The hose used for disintegrating the gravel had a 2 1/2-inch nozzle; the water pressure was 45 pounds per square inch. The gravel was washed back and forth on the bottom of the tank until it was clean. The high sides and end prevented the fine material from splashing out of the tank. Boulders were thrown out by hand on a rock pile. The riffles in the box in the washing tank were covered with a wire screen with 1/4-inch openings. For 8 feet below the tank the riffles were covered with wire screen with 1-inch square openings. Below this was 200 feet of sluice with alternating sections of transverse and longitudinal riffles. The riffles were 1 by 2 1/2 inches in section, spaced 8 inches apart and topped with strap iron.

Van Patten, Nensiis, and McKim.- These men were developing a claim on Burnt River near Bridgeport, Oreg., in June 1932. The gravel was 16 feet deep and occurred in a relatively narrow channel under the present course of the stream. A shaft had been sunk 18 feet to bedrock and drifting begun. Steel caissons had been used in sinking the shaft through loose surface gravel to keep out the water from the river. The ground on bedrock was tight, and relatively little water was coming into the workings.

A well-built, portable, labor-saving surface plant had been installed. The washing plant consisted of a 1 1/2- by 4-foot trommel with 1/4-inch holes, three steeply inclined steel boxes 6 inches wide with wooden cross riffles, and a wooden tail box with riffles. Water from the mine was discharged into a small settling basin whence, with an added supply from the river, it was forced by a high-pressure pump through sprays in the trommel. One man ran the hoist and pumps and shoveled into the washing plant all the gravel two men below could send up. A 4-cylinder automobile engine drove a hoist drum, a deep-well pump, the trommel of the washing plant, and the high-pressure pump. On hoisting, the bucket was swung by hand on a crane and dumped on a platform, whence the gravel was shoveled into the trommel. It was expected that 5 cubic yards per shift could be handled with 2 men underground and 1 on top.

Golden Belt.- The Golden Belt mill of the Belt Gold Mining Co. was on Magpie Gulch near Canyon Ferry, Mont. The gravel consisted of angular wash containing a high percentage of clay. It was dumped from a 6-cubic-foot car on a platform, whence it was shoveled in 3-cubic-foot batches into a concrete mixer and washed for 2 1/2 minutes. The concrete mixer was then dumped over a grizzly made of 1-inch pipe spaced 1 inch apart. About two thirds of the material hoisted went through the grizzly and thence into a sluice. The oversize was shoveled by hand into a car and trammed to a rock dump. The sluice consisted of three 12-inch boxes 12 feet long. The grade was 8 inches to 12 feet; riffles consisted of 2- by 4-inch lumber cut diagonally and placed flat side up in the box.

Rising Hope.- The Rising Hope mill was near Auburn, Calif. The ore was drawn from a bin into a 4- by 5-foot blank trommel where it was disintegrated and washed. It then ran over a 3- by 4-foot screen with 1-inch square holes. The oversize dropped into a car and was trammed to a dump. The undersize went through 150 feet of 16 inch steel boxes with Hungarian and cast-iron riffles. The trommel was run by a water wheel which was operated by 4 miner's inches of water under a 200-foot head. The water cost 25 cents per miner's inch per day. Quicksilver was kept in the first 2 1/2 feet of the sluice and a little sprinkled occasionally along the first two boxes. In cleaning up, the concentrates were first panned, then put through a barrel amalgamator with quicksilver.

Vallecito Western.- According to Steffa:[60]

> The gravel is treated in a plant near the collar of the shaft having a capacity of 15 tons per hour. (See fig. 10.) From the shaft bin the gravel is washed by water from a 2-inch line through the bin gate, into and through a 11-foot Hungarian-riffled sluice. From the lower end of this sluice the gravel discharges into the hopper of a 3- by 18-foot trommel, set at right angles to the line of the sluice. This trommel has two compartments, one for washing and disintegrating, the second for screening and further washing. The first, 8 feet long, is of unpierced steel, lined on the inside with 4-inch angle irons. As the trommel revolves these fins lift the gravel and cascade it to the bottom again, producing a crushing and disintegrating action similar to that in a ball mill.

60 Steffa, Don, Gold Mining and Milling Methods and Costs at the Vallecito Western Drift Mine, Angels Camp, Calif.: Inf. Circ. 6612, Bureau of Mines, 1932, 14 pp.

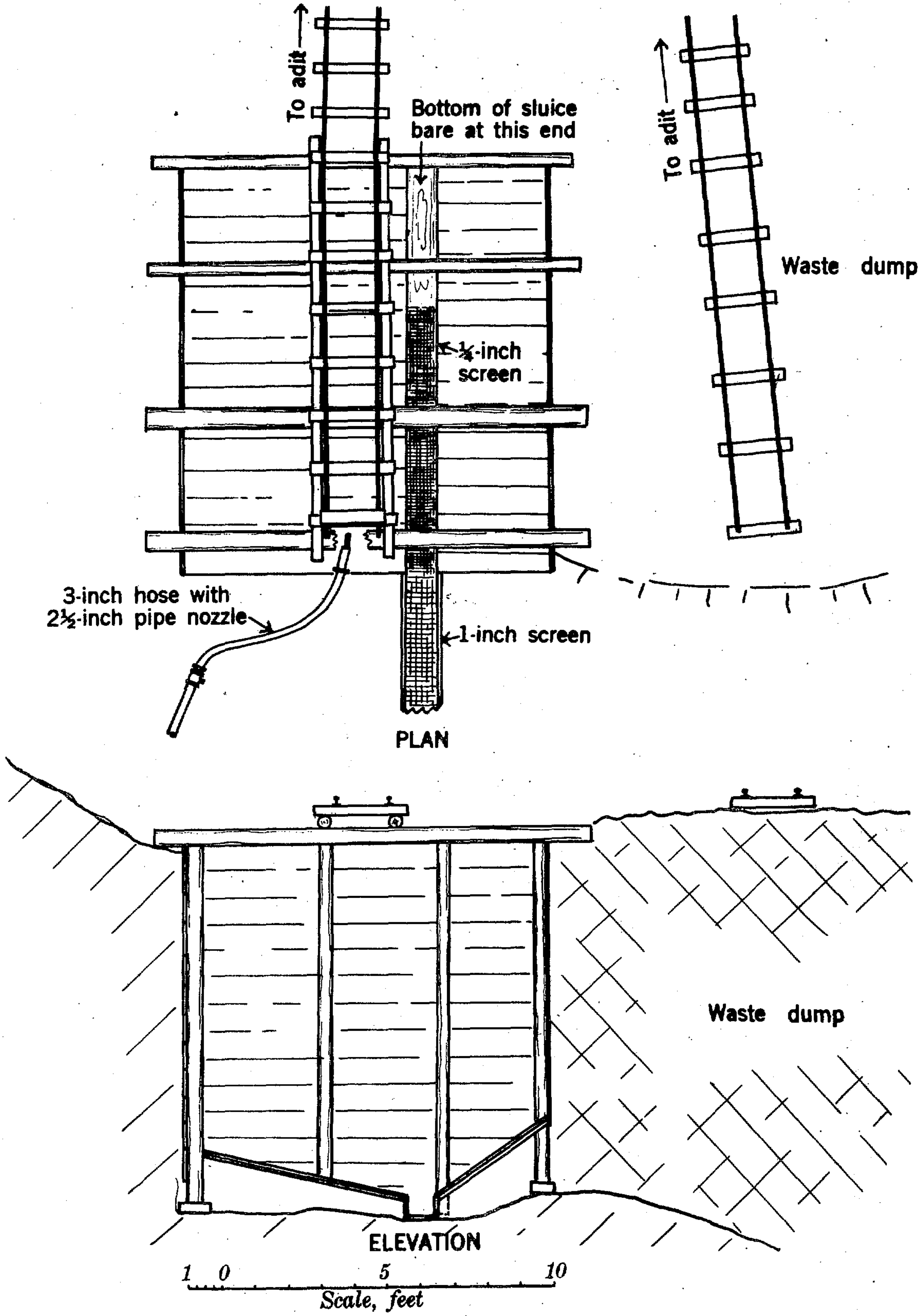


Figure 9.—Washing plant, Baker Divide mine, Michigan Bluff, Calif.

The trommel is set on a slope of 1/2 inch per foot and is revolved at a speed of 28 r.p.m. by a 10-hp. motor.

The lower 4 feet of the trommel consists of two concentric screens, the inner perforated with 1 1/2- and the outer with 3/8-inch holes. All the material from the disintegrating section of the trommel passes by gravity onto the 1 1/2-inch screen. The oversize is discharged into a steel-lined sluice 60 feet long. It is forced through this by a stream of water from a 6-inch line and passes first over 6 feet of Hungarian riffles, then over 100 feet of pole riffles, and then to the waste dump. The pole riffles are constructed of longitudinally placed 8-pound steel rails.

The undersize of the 1 1/2-inch trommel drops onto the 3/8-inch outside screen, where it is washed by a stream of water from a 1-inch line. The washed oversize of this screen joins the discharge of the coarse screen. The minus 3/8-inch material drops to a 4-foot Hungarian riffle and passes over this onto a 3 1/2- by 6 1/2-foot "screen table." The latter is simply a 1/4-inch mesh heavy-wire screen laid flat in a widened sluice box of the dimensions noted. Spreading the fines over this table permits most of the fine gold which has escaped the riffles above to settle in the interstices of the screen. No quicksilver is used. It has been found that under the conditions at this property the mercury "flours" and gradually migrates under the action of swiftly flowing water. Fine gold which would otherwise remain in the riffles is then caught up and carried to the dump.

From the screen table the fines pass through a 20-foot section of Hungarian riffles, then over two 12-inch by 6-inch baffle plates, then over a final 12-foot Hungarian riffle. After this they join the coarse discharge of the trommel at the head of the pole riffles and pass eventually to the waste dump.

Above the baffle plates sluice boxes are 16 inches wide at the bottom and below the baffles are 12 inches wide. The function of the baffle plates is to smooth out the flow of sand and water, giving a more or less even feed to the last riffles. The first sluice box, between the bin and the trommel, is set at a slope of 2 inches per foot. The 4-foot segment between the trommel and screen table, the screen table itself, and the succeeding 20-foot section are all set on a 1 1/2-inch slope. Below the baffle plates the grade of the last section of Hungarians and of the pole riffles is 1 1/4 inches per foot. Any lower grades in the upper section of the plant result in crowding of the riffles with sand and overburden and the escape of fine gold due to the obstruction of effective water action.

Practically all of the coarse gold and 60 percent of the total is recovered in the first 11-foot riffle immediately below the gravel bin. Following the washing and screening in the trommel, a very large part of the remainder is caught in the 4-foot Hungarian above the screen table. Occasional very large nuggets will accompany the oversize of the trommel and be caught in the first few feet of pole riffles.

When gold-bearing gravels are being milled, the upper two sluices are cleaned up twice a week, regardless of the tonnage, and daily if the gravels are unusually rich. The screen table is cleaned about once a month and the lower riffles only every 6 or 8 months.

Clean-ups are started in the uppermost sluices. The riffles are lifted from the box, maintaining a reduced head of water just sufficient to wash the sand and lighter material through the box as the mass is agitated slowly by hand and to precipitate the gold to the bottom. The last riffle remains in place and serves as a dam until the contents of the box have been reduced to a pan or less of concentrates. Coarse gold, if any, is screened out, and the fines are recovered on a small concentrating table. The percentage of flour gold is negligible.

The plant requires about 500 gallons of water per ton of gravel treated or in cubic measure about 3.7 cubic feet of water per cubic foot of gravel. The mine drainage water is discharged into a 400,000-gallon reservoir close to the shaft. From the reservoir the water is forced into a 1,500-gallon, steel pressure tank by a 15-hp. automatically controlled centrifugal pump. When the air pressure in the top of the tank drops below 15 pounds per square inch the pump is started; when it reached 40 pounds the pump is stopped. A pressure of about 25 pounds is maintained when operating the plant, and the pump runs continuously.

A second and higher reservoir of 4,000,000 gallons capacity has been constructed in a gulch 1,200 feet northeast. In the rainy season this serves to store run-off water which can be drawn off as needed. It is dry during the summer months.

The centrifugal pump has a capacity of 500 gallons per minute, but only about 300 of this is needed for sluicing operations - 100 in the upper line to carry the gravel through the trommel and 200 to carry away the oversize to the dump. The mill capacity varies from the 15 tons per hour noted above, with the character of the gravel, which ranges from well-rounded easily washed material to clayey or sandy gravel containing angular pieces of bedrock and flat pebbles.

The effectiveness of the gold recovery has been shown by a mill test of the tailing. Two hundred tons of these, rehandled, yielded $0.175 per ton. As the original gravel had averaged $8 per ton the losses were only slightly over 2 percent.

Calaveras Central.- The Calaveras Central mill at Angels Camp, Calif., has a capacity of 75 tons per 8 hours. The gravel was passed over an 8-inch grizzly into a 3 1/2- by 8-foot trommel with 1 1/2-inch holes. The oversize from the grizzly and trommel, which comprised 60 to 75 percent of the feed, was taken to a dump by a belt stacker run by a 3-hp. motor. The undersize ran through 32 feet of 12-inch, steel-lined sluice boxes with a grade of 1 inch to the foot. Riffles were cut on a bias with the top horizontal; they were 1 3/4 inches high and 1 1/2 inches wide, topped with 1/16-inch strap iron and set 2 inches apart. A trap at the head of the first box caught about 90 percent of the gold saved in the 36-foot sluice. The gravel contained considerable black sand; whenever the riffles had a tendency to pack with the sand more water was used. At the end of the first sluice the gravel was run through a second trommel with 5/8-inch openings. The oversize was taken to the dump by a belt conveyor. The undersize went to the dewatering box of a secondary washing plant, whence the sands were taken by a drag classifier to two 34-inch by 6-foot rod mills, each run by a 15-hp. motor. The sands were ground to pass an 8-mesh screen and were run over a steel-lined table 7 feet long and 30 inches wide with a divider in the middle. Wooden dredge-type riffles were used on the first 2 feet of the table. Gold passing over the riffles was caught in two traps containing quicksilver. The secondary mill was used only part of the time gravel was being washed; about 90 percent of the gold recovered was caught in the first sluice. About 250,000 gallons of water per day was pumped from the mine, and a plentiful supply was available for the mill.

Baltimore.- The Baltimore mill of the Mayflower Gravel Mining Co. is at Foresthills, Calif. Gravel from several adjoining drift mines had been treated in this plant. The gravel handled was tight and partly cemented. Boulders were coated with clay which contained placer gold. The flow sheet of the plant is shown in figure 11. The gravel as brought from the mines was dumped into a 150-ton bin from which it was drawn dry over a 5- by 12-foot grizzly with 1 1/2-inch spaces between the bars. The oversize went to a platform where fragments of cemented sand or gravel were sorted out and fed by hand into a 12-inch jaw crusher. Boulders and rock went to a trommel washer where the clay and sand were freed from the coarse material. The trommel had rows of 1/4-inch holes, 8 inches apart, and lugs to cause cascading of the material and thus facilitate washing. The oversize went to a belt conveyor and thence to the rock pile. The undersize from the trommel went through 48 feet of 12-inch sluice boxes set on a grade of 1 1/2 inches to the foot. Riffles consisted of 1- by 3-inch wooden strips, 2 inches apart, inclined downstream; the tops of the riffles were parallel with the bottom of the box. Riffles in the first box were covered with a 1/4-inch screen. Material from this sluice emptied into the tail sluice. The gravel from the crusher and the undersize from the grizzly went to three batteries of three 1,200-pound stamps each. The batteries had triple discharges; the material was crushed to pass through a 1/4-inch screen. The stamps dropped 107 times per minute and had a drop of 7 inches. Quicksilver was used in the battery box, where 85 percent of the gold was recovered. The stamped material went over three standard amalgamation plates 6 by 12 feet in size. A 6- by 1 1/2- by 1-foot trap was used at the bottom of each plate to catch nuggets or balls of amalgam. The material in the traps was agitated by water under pressure to prevent packing. Standard quartz mill practice was followed in cleaning the plates and treating the amalgam.

The mill had a capacity of 180 tons per 24 hours but has been run only periodically of late years. Electric power for running the mill was made below the mine by a Pelton wheel. The boulder washer was run by a 15-hp. motor, the crusher by a 30-hp. motor, and the stamps by a 25-hp. motor - a total of 70 hp. Plenty of water was available, and no measurements had been made of the quantity used. A saving of 95 percent of the gold was reported to have been made.

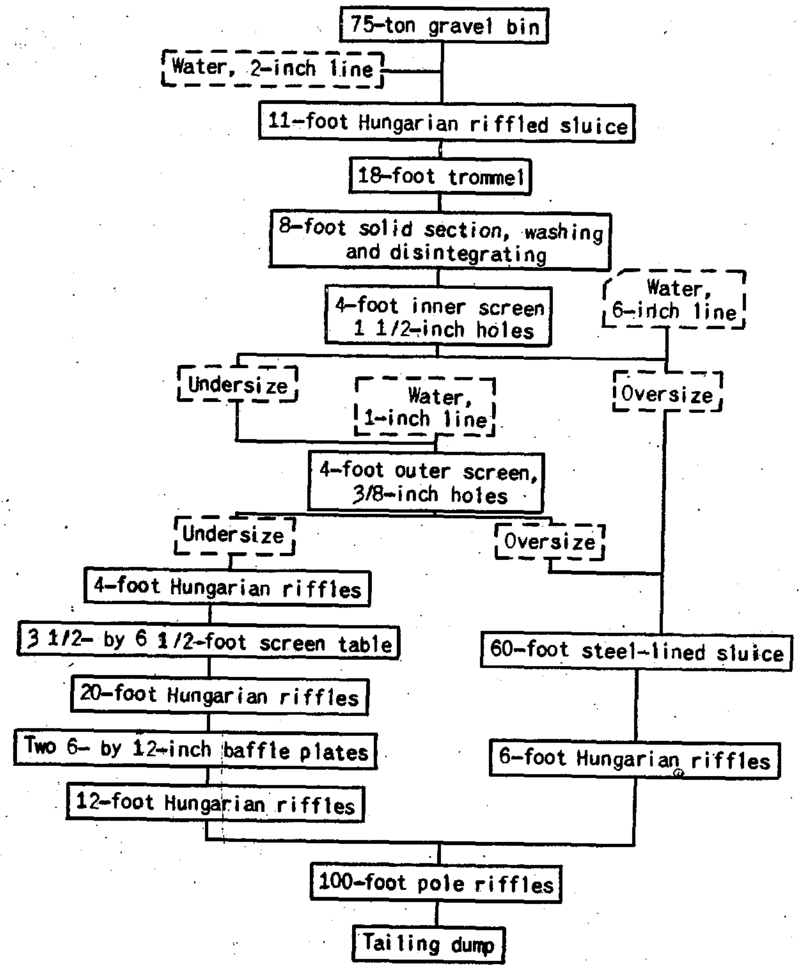


Figure 10.—Flow sheet, Vallecito Western mill.

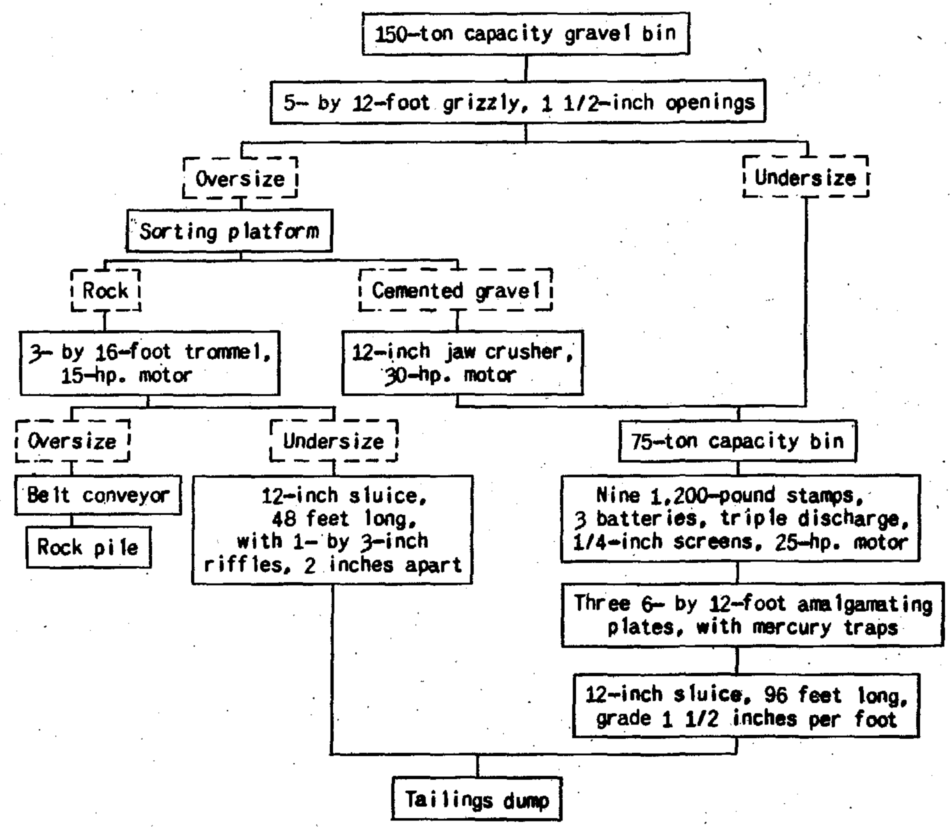


Figure 11.—Flow sheet, Baltimore mill, Foresthill, Calif.

I.C. 6843
Reissued
April 1936

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PROSPECTING FOR LODE GOLD[1]

By E. D. Gardner[2]

CONTENTS

INTRODUCTION

The Bureau of Mines receives many inquiries concerning favorable areas in which to prospect for gold, procedure to be followed, equipment required, and allied subjects. This circular has been prepared for use in reply to these inquiries and is a preprint of part of a bulletin to be issued later on "Equipping, Developing, and Operating Small Gold Mines."

The increased interest in gold mining manifested during the past few years (1932-1935) has stimulated prospecting. Many adventurers have taken to the field to search for new gold deposits. A large percentage of them have had no previous experience in prospecting for lode gold; this paper has been written with the hope that it might assist these newcomers.

PROSPECTING

A majority of the metal mines in the United States have been discovered by qualified prospectors who were searching for valuable minerals at the time. Chance, however, always

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6843."

2 Supervising engineer, Southwest Experiment Station, U.S. Bureau of Mines, Tucson, Ariz.

has played a large part in finding mineral deposits. Some of the discoveries of the past were made by men on other errands, such as rounding up burros or hunting game. Accidental discoveries of ore bodies have been made in building roads and trails and in excavating for mine structures. Evidence of ore has been brought to the surface by burrowing animals and by ants; gold found in the craws of fowl has led to discoveries of deposits. Important discoveries have been made by men who had no knowledge of rocks or minerals; on the other hand, many ore bodies have been found by experienced prospectors, sometimes after hundreds of untrained men had already passed over the ground.

The prospector who carries on his work diligently and intelligently is of course more likely to be rewarded for his efforts than the lazy or unintelligent worker; nevertheless, it is obvious that if valuable deposits do not exist at the place being prospected none will be found. Conscientious and painstaking efforts to trace gold to its source usually disclose nothing more valuable than some narrow, unworkable seams; however, many deposits that were developed into profitable mines were found by this method of prospecting. Although some prospectors have made several lucky strikes, many others have spent their working lives searching for mineral without finding anything worth while. Probably only one prospector out of several thousand ever finds anything worth developing. Moreover, only 1 out of every 300 or 400 properties developed becomes a profitable mine.

Prospecting began in the Western States in the fifties as the miners looked for the source of gold found in placers. The search for gold has been continuous since that time; the number of prospectors in the field at any one time, however, has varied greatly. Except in some desert regions, practically all of the placer fields now being worked were discovered by old timers; most of the important gold districts also were found by early prospectors. Important discoveries of lode mines, however, have been made from time to time. Most of the area in the mining regions of the West has been gone over many times by prospectors, and nearly all of the easily found deposits have been located, but it is reasonable to expect that new gold mines will continue to be found. Most of the future discoveries undoubtedly will be of deposits that do not outcrop. Prospecting for such deposits requires considerable digging.

Several important discoveries were made in 1934. One of these, the Rogers-Gentry gold mine at the edge of Antelope Valley in Los Angeles County, Calif., was found by an experienced prospector on an old patented homestead a number of miles from the nearest producing mine. The initial discovery at this mine was an iron-stained, decomposed, siliceous limestone outcrop, with no vein structure evident at the surface, near a small, barren quartz outcrop and a water seep. Another discovery, the Silver Queen gold mine, in the same general region and near Mojave, Calif., was found by an experienced miner on an open fraction 400 by 1,400 feet in size between two old properties which were thought to have been worked out years ago. The Silver Queen discovery was made as the result of finding a single piece of float unlike any ore in the region. The vein did not outcrop; the discovery point was under 6 feet of cover.

To prospect for lode gold one should know first of all how to take care of himself in the hills or on the desert. He should be physically able to stand hard work and know how to use a pick and shovel. Most prospectors also have occasion to drill holes by hand and know how to use explosives. To prospect for lode gold intelligently one should be able to identify gold and the minerals usually associated with it, besides being able to distinguish one general class of rock from another.

Most prospectors work alone and are accustomed to solitude. As discoveries that can be sold for cash are few and far between prospectors must have some other resources for subsistence. Many prospectors work in the mines in the winter and prospect in the summer. Others do the assessment on claims for owners to earn enough money to buy supplies for prospecting. In the old days many prospectors were grubstaked by merchants, individuals, groups of individuals, or companies, usually on a 50-50 basis. The practice now is followed less than formerly, but a professional prospector of good repute usually can get a backer.

Favorable Areas

Although the old saying that "Gold is where you find it" is quite true, the probability of finding gold in paying quantities will be increased greatly if prospecting is done in areas geologically favorable for the occurrence of gold. Regions in which gold is known to occur naturally are more favorable for prospecting than those where no gold has ever been produced.

The important known gold deposits in the United States occur in regions where intense igneous activity has occurred at some time. The most promising fields for finding new deposits of gold, therefore, should be in or near igneous rocks.[3] Not all igneous formations, however, are favorable for the deposition of gold. It probably would be a waste of time to prospect in dark lava flows. Large masses of granites or related coarsely grained crystalline igneous rocks are unlikely to contain gold deposits unless cut by dikes or other intrusions of finer-grained and usually light-colored igneous rocks, such as porphyry, rhyolite, or andesite.

Areas are favorable for prospecting where the principal rocks are granites (as suggested above), schists, slates, greenstones, or related rocks cut by later intrusives. Areas in which the principal rocks are the light-colored, finer-grained igneous rocks, especially if of several varieties, are also favorable.

One of the most favorable areas for the occurrence of gold is where the country rock is made up of surface flows, sills, dikes, and other intrusions of these light-colored igneous rocks.

Profitable gold deposits sometimes are found around the borders of great masses of granitic rocks, both in the granites and in the surrounding rocks but more often in the latter.

Large areas of sedimentary rocks,[4] such as shale, sandstone, and limestone, are unfavorable for prospecting unless the sediments are cut by the light-colored intrusions previously mentioned, and even where so cut the sedimentary areas seldom contain workable quantities of gold unless they have been metamorphosed (changed by pressure and heat) to slate, quartzite or marble.

Gold Lodes and Ore Shoots

Gold in paying quantities does not exist indiscriminately in country rock but where it has been deposited in definite zones usually termed "lodes." Solutions containing the gold have arisen from great depths and have been deposited by relief of pressure, cooling of the solutions, or other causes. For a lode deposit to have been formed there must therefore be some form of opening or zone of weakness through the rocks along which the solutions may rise. Earth movement or faulting commonly causes zones of weakness. Therefore, in prospecting it is well to keep a look-out for fracturing.

---

3 According to Dana's Manual of Mineralogy, 14th ed., issued in 1929, p. 345, "Igneous rocks, as the name indicates, are those which have been formed by the cooling and consequent solidification of a once hot and fluid mass of rock material."

4 According to Dana, p. 350, "Sedimentary rocks are secondary in their origin, the materials of which they are composed having been derived from the decay and disintegration of some previously existing rock mass"

Parts of the lodes that contain gold in sufficient quantity to be ore (that is, material that can be mined at a profit) are called "ore shoots." Shoots seldom extend between walls but may be confined to a relatively narrow streak or streaks. Gold-ore shoots usually are relatively small. After a gold-bearing lode is located considerable work may be necessary to find an ore shoot; frequently the gold will not occur in sufficient quantity for any part of the lode to be worked at profit.

The simplest and most common form of gold lode is what is termed a "true fissure vein" by the miners. Fracturing, with or without faulting, has occurred in a relatively narrow zone with well-defined walls. The ore minerals have been deposited in this zone and may fill the space between walls completely. Usually, however, fractured country rock and, if the movement has been great, gangue or slickensides occupy part of the space.

Another type of lode is the shear zone. Here the walls usually are not well-defined. The ore-bearing solutions have deposited the gold and associated minerals in the cracks made by the fracturing. If present, ore shoots may occur anywhere in the fractured zone. Usually they overlap and occasionally may be parallel.

The contact between two different kind of rocks, especially an igneous rock and something else as schist, is generally a line of weakness. Ore-bearing solutions may have been able to rise along the places of weakness and form ore bodies. Such a lode is called a contact vein. Gold ore also may occur in bedded sedimentaries where a fissure cuts a contact, particularly between limestone and quartzite, where conditions are otherwise favorable geologically for the deposition of ore.

Searching for Gold Deposits

In looking for gold deposits vein or lode outcrops are sought and when found are examined for gold-bearing material. Portions of the veins have been eroded away; on steep hills part of the outcrops may have broken off and rolled down the hillside. Mineral-bearing fragments of vein material are called "float." Many deposits have been found by tracing float to its source. In prospecting, a lookout always is kept for such material. Float in the gravels of large streams may have come from many miles distant. In such instances the presence of the float indicates only that the gold-bearing material exists in the watershed above. Where float is found on a hillside the fragments are sought upward until no more are found. If the surface is covered with overburden trenching will be necessary to disclose the lead.

Lodes also may be located by panning loose material below for free gold. Placers are formed from disintegration of rock containing gold. During the ages gold lodes are eroded away at the surface, the gold-bearing rock is ground to powder, and the gold is concentrated in stream beds or desert deposits. The gold of rich placers, however, may have come from a multitude of narrow or low-grade streaks that could not be worked at a profit. The presence of placer gold in a stream bed indicates that the region above contains or has contained lode gold. In seeking for lodes in such a region the gravel of stream beds or debris of dry washes is panned to trace the gold to its source. If the gold suddenly plays out in the main watercourse attention then is directed to the side gulches, which in turn are followed up until no more placer gold is found. The debris on the mountainsides is then panned and the gold traced to its source. At this stage of prospecting float in the overburden may help in the search or be the key to the source of the gold.

Occasionally rich accumulations, called pockets, of free gold are found in the hillside debris. Especially in California pocket hunters have made a living by searching out these accumulations. The same procedure is followed whether the search is for a lode or a pocket. Valuable deposits in place have been found by pocket hunters.

As placer gold travels from its source it becomes flattened or rounded. Angular or jagged gold usually has not traveled far. The same is true of float. Well-rounded fragments of vein quartz may have traveled far, while angular pieces are not likely to have been transported a great distance. In flat, glaciated country float or free gold may have come from hundreds of miles away and may signify nothing as far as the immediate region is concerned.

Quartz, which usually is a constituent of gold ores, is hard and resistant to weathering. Furthermore, frequently the mineralization of a vein is accompanied by silicification of the vein filling and the immediate wall rock, which increases the resistance to weathering and erosion. Hence, a majority of veins containing gold ores outcrop above the surrounding surface. In flat regions, however, the outcrops may be covered with overburden brought down by floods. In some instances the vein may be badly fractured; any quartz present may be in narrow seams in a gangue of shattered country rock. When this is the case the vein at the surface may be softer than the adjoining wall rock and cause a depression. Trenching, therefore, is necessary to disclose the lode in place.

In searching for a hidden vein the following features which may be caused by the existence of a lode should be noted:[5]

1. A natural trench or ditch that does not run directly down the slope of the hill or mountain.
2. A sudden change of slope.
3. A sharp notch that crosses a ridge that has a rather uniform altitude on both sides of the notch.
4. Several springs in a line.
5. A sudden change in the kind or quantity of vegetation (may indicate a contact or, of the change in vegetation is found over a narrow strip of ground, a lode may be beneath).

Although many other possible causes may be responsible for these structural features some trenching would be justified if float was found immediately below and not above any particular one of them.

Iron sulphides, which frequently are associated with gold, oxidize to red or yellow oxides when exposed to the surface elements. The presence of a lode very often is disclosed by the stain of these iron minerals.

Present-day prospectors examine old cuts or other workings on abandoned claims. It is possible that with the increased price of gold and the improvements in metallurgy since the original work was done material passed up by the oldtimers may now be valuable.

With a few notable exceptions, the gold in lode deposits occurs as the native metal. At Cripple Creek and some of the other Colorado districts the gold is a constituent of telluride minerals; in general appearance these minerals resemble the iron sulphide minerals.

Gold and Associated Minerals

Gold can be identified readily by sight. It is the only soft, yellow substance with a metallic luster occurring in nature. It can be flattened easily without breaking and be cut or scratched readily with a knife. It is sometimes confused with pyrite, chalcopyrite, or other sulphide or with plates of yellow mica. Pyrite is too hard to be scratched with a knife, and sulphides that resemble gold crush into black powder. Yellow mica yields a white

---

5 Butler, G. M., Some Hints on Prospecting for Gold; Arizona Lode Gold Mines and Gold Mining: Bull. Univ. of Arizona, Tucson, Ariz., vol. 6, p. 251.

powder when scratched with the point of a knife. When any doubt exists the suspected substance ordinarily is not gold.

The principal gangue mineral in gold deposits usually is quartz. This mineral is distributed widely in mineralized areas, but a very small percentage of it will be found to contain gold. Glassy or what is called "bull quartz" by the miners seldom if ever is gold-bearing. Massive quartz leads may be very persistent but generally are barren, except in some cases where secondary quartz with more of a porcelain appearance has been deposited. In the Mother Lode region of California the gold usually is associated with this secondary quartz.

With a few exceptions gold below the zone of oxidization generally is associated with or accompanied by sulphides. The principal sulphide ordinarily is pyrite; in some cases, however, chalcopyrite, arsenopyrite, or galena may be the important gold carrier. Gold may occur, however, in quartz without the associated sulphides or their oxidization products or in veins where quartz is not important. For example, in the Oatman (Ariz.) district all the gold is free, and the principal gangue mined is calcite.

Iron streaks or vugs (cavities, usually lined with a crystalline incrustation) in quartz-lead matter are promising places for native gold to occur. Frequently if present it can be seen by the naked eye or with a glass, therefore the prospector is on the look-out for iron-stained or honeycombed quartz. Outcrops, consisting mainly of iron oxides or lead matter heavily impregnated with iron (called gossan), when found in a mineralized region always should be tested for gold; the gossan may be at the top of copper or lead ore bodies with the latter two metals leached out. Sometimes the gossan carries paying amounts of gold.

Any outcrop or float of iron-stained, fractured, light-colored, igneous rock recemented with silica or showing evidence of silicification and banding should be investigated. The ore of the Silver Queen mine near Mojave, Calif., is of this latter type.

In glaciated regions and occasionally elsewhere sulphides occur at the surface. Moreover, float containing sulphides occasionally is found. Both outcrops and float usually are tested for gold by the prospector. Frequently the outcrop of a lead[6] is shown by green or blue copper stain. Should the original copper sulphide have been associated with gold the possibility of a deposit of gold ore exists.

Prospectors usually do not confine their efforts to the search for gold but will locate any deposit that promises to be of value, irrespective of the kind of contained mineral. To be present in sufficient quantities for the material to have value as an ore the base metals must occur in amounts readily discernible by the eye; the base metals, however, may have been removed from outcrops by leaching.

Sampling and Panning

As mentioned before, gold occasionally is visible in vugs or high-grade seams, but usually the gold in its ores is not visible either to the naked eye or with a glass. Rock suspected of containing gold may be tested by assaying or panning. Of course, the former method is to be preferred, but the cost ($1 to $1.50 per determination) precludes its general use by most prospectors. Some of the large mining and smelting companies, however, will assay free a reasonable number of samples sent in by bona-fide prospectors. In this way they may be the first to learn of new discoveries.

---

6 Commonly used as a synonym for ledge or lode. Many mining-location notices describe the locator's claim as extending a certain number of feet along and so many feet on each side of the "lode, lead, vein, or ledge." The word is pronounced "leed" and should not be confused with the metal lead.

Gold prospectors make a practice of panning (or horning) likely looking rock. The sample is first ground in a mortar or otherwise pounded into powder. A small frying pan from the 10-cent store appears to be preferred by most prospectors for panning rock samples. Although the greater reliability of relatively large samples is realized most prospectors when grinding the rock by hand and panning in the field use 1- or 2-ounce samples. In prospecting, the best-looking material is panned. After the rock has been shown to contain gold the value per ton should be ascertained by assaying. Samples for assaying should be cut over a definite width of the exposed vein.

A competent panner can estimate fairly closely the gold content of ore with which he is familiar. An expert panner with a 1-ounce sample can detect gold in rock that will assay only about 0.02 ounce of free gold per ton. A milligram of gold in an assay ton (29.168 grams) indicates 1 ounce of gold per ton of 2,000 pounds. An ounce avoirdupois is 28.350 grams.

Not all gold-bearing rock pans. Where the gold is associated with or contained in sulphides, grinding in a mortar may not liberate enough of the gold to be detected in the pan. In the United States, however, the gold in outcrops usually has been liberated by oxidization to such an extent that it can be panned.

In searching for gold most professional prospectors carry a mortar and a canteen of water. Likely looking rock is panned as found. By this procedure a load of rock is not accumulated, and many samples are tested that would not be carried to camp. Furthermore, there is no confusion regarding the location of the gold-bearing material, as often is the case when samples are accumulated.

Although an experienced man may identify gold telluride in the ore or in the pan the sample should be assayed when their presence is suspected. Assays also are of course necessary to tell whether sulphides contain gold.

All major exposures of veins or other structures that appear favorable for the occurrence of gold should be sampled and assayed. No ore should be shipped without being assayed; almost invariably when this is done the shipper is disappointed. As mentioned above, samples should be cut across definite widths of the vein. Hand and grab samples of ore to be shipped are almost always high.

Surface Weathering

Weathering and leaching by surface solutions may remove the base metals from surface outcrops. Gold, however, is very resistant to leaching, and the weathering of the iron and associated minerals may increase the value per ton of surface ore; hence, it cannot be expected that the value of gold deposits will increase with depth; usually the contrary is true.

Prospecting on Patented Ground

Prospectors are reluctant to prospect on patented ground, as anything found would belong to the owners of the land. The author believes, however, that opportunities occur for finding new deposits on some of the thousands of idle patented mining claims held throughout the West. Many of these claims are held by estates. Even where the owners of idle claims would be willing to draw up papers to the effect that a discoverer of new ore would benefit from his findings, the average prospector would decline to go to this trouble and conduct his searches elsewhere.

Prospecting Outfits and Provisions

The outfit to be taken on a prospecting trip depends upon the mode of transportation, work contemplated, and the funds available. Enough equipment should be taken, but unnecessary articles make extra work. When a more or less permanent camp is established added equipment for personal comfort and efficiency can be obtained. Usually a cabin is built for a permanent camp.[7]

Transportation

An automobile is to be preferred for transportation if the region is one where it can be used. It has the advantage that a complete outfit can be carried and trips can be made out for supplies with relative ease. Most present-day prospectors use automobiles, especially in the desert regions. In mountainous regions away from roads the old stand-by, the burro, still finds favor. A prospector working alone generally uses two burros; occasionally, however, one animal suffices. A string of six or more burros will be used by a party; in this case a packer will be needed to look after the stock. About 150 pounds can be packed on each animal. A burro can live off the country and can go almost any place a man can get afoot. The principal objection to the use of burros is that they must be rounded up each day to keep them from wandering off beyond reach; a prospector will spend about a fourth of his working time chasing his burros. Mules or horses are used under some conditions but on the whole are less satisfactory for prospecting than burros. A horse cannot live off the country, and both mules and horses must be hobbled to keep them within reach.

Some prospectors prefer to get as near as possible to the area they wish to prospect by car and then carry supplies in on their backs to "spike" camps rather than bother with animals. Each year as more roads are built new country becomes accessible by car.

Camp Outfits

A roll of 3 or 4 blankets (the number depending upon the climate) in a canvas cover, a tent, a tight wooden box with a lid for storing food away from insects and rodents, and a canvas "war bag" for clothes usually make up the minimum camp outfit. A stove will be needed if prospecting is done in cold weather. A folding cot is desirable; in a permanent camp a bunk usually is built, as well as other needed furniture.

Tools

A full-sized ax and a good pocketknife are the first requirements for camping. A saw and a hammer with 2 or 3 pounds of assorted nails will be needed for fixing up a camp. A 50-foot length of 1/2-inch manila rope usually comes in handy. A miner's acetylene lamp provides a good light; a 5-pound can of carbide will last a camp all summer. A flashlight and a supply of batteries are conveniences that may be well worth their cost. A 2-quart canteen with a shoulder strap usually is needed for carrying drinking water or water for panning. A 2-gallon canteen and a 5- or 10-gallon water keg are necessary in some districts.

A pick, a long-handled, round-pointed shovel, a gold pan, and a prospector's pick are indispensable. If claims are to be staked a compass will be needed for running out the lines. A hand magnifying glass is a great help in identifying minerals. A mortar and pestle,

---

7 Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States: Part I - Prospecting Outfits and Provisions; Inf. Circ. 6786, Bureau of Mines, 1934, 73 pp.

a horn spoon, or a small pan will be needed for testing rock for free gold or other havy minerals. A blowpipe outfit and determinative tables are of service to those who know how to use them. Bags for taking out samples usually are needed. Double paper bags with rubber rings cut from old automobile tubes for closing them permit large numbers of samples to be collected with little expense for bags.

A single-jack hammer with 2 or 3 moils will come in handy for taking samples and for loosening rock found in making cuts.

Some prospectors carry 1 or 2 sets of hand steel and several pounds of powder. A few rounds may be drilled and blasted before the steel has to be resharpened. If any extensive rockwork is to be done a forge and a set of blacksmith tools are necessary; usually these are brought in later.

Cooking Equipment

For a 1- or 2-man party a frying pan, a coffee pot, a large and a small stew pan or pot, and a Dutch oven are needed. A knife, fork, spoon, cup, and plate are required for each man. A few extra plates come in handy. A good butcher knife, a water pail, a can opener, and a few tea towels complete the outfit. Other dishes can be taken according to personal preferences.

Provisions

The variety of food taken on prospecting trips depends upon the method of transportation and the prospector's pocketbook. If the supplies are to be packed on animals bulky foods, such as potatoes and canned articles, are omitted. If there is need for economy in making purchases the list will consist mostly of dried staples and vegetables, if available locally.

Bacon, flour, beans, oatmeal, dried or canned fruit, coffee, syrup for hot cakes, and sugar and canned milk for the coffee are the stand-bys in prospectors' camps. As funds get low more beans and less bacon are eaten, and canned fruit is omitted. Canned tomatoes are in common use; they are cheap and supply needed food elements not contained in dry staples (for example, they help to prevent scurvy). Where available locally Irish potatoes, onions, and other vegetables are eaten. Fresh meat is not used much in camps in summer on account of the difficulty of keeping it.

A proper balance should be made in compiling a "grub" list so that needed items will not run short. Everyone prefers certain articles of food, and these likes should be followed as much as practicable. It has been found by experience, however, that fancy groceries are the ones left over, and the first supplies to be used are bacon, potatoes, and flour. Plain, wholesome fare seems to be preferred in camp, especially where hard work is done.

The following weekly allowance of food for one person to give a balanced diet is condensed from suggestions made by Doctor Smith:[8] Three 1-pound cans evaporated milk; 2 pounds potatoes; 4 pounds onions, cabbage, beets, or other vegetables; 3 pounds citrus fruits, 6 pounds fresh apples, or equivalent dried prunes, apricots, etc.; 3 pounds dried beans; 6 to 8 pounds cereals, whole-wheat flour or bread, rolled oats, shredded wheat, etc.; 2 1/2 pounds dried meat, bacon, ham, or cheese (fresh meat or eggs may be substituted if available); 3 pounds sugar; 1 pound coffee; 1/4 pound salt; 1/2 pound butter; and baking powder.

The cost of provisions for prospecting in the West will average about 50 cents per day per man. Many prospectors live in the hills when short of funds on as little as 25 cents per day each; they are not well-nourished, however, and do not have a balanced diet.

---

8 Smith, Margaret Cammack, Food Suggestions for Prospectors; Arizona Gold Placers and Placering: Bull. Univ. of Arizona, vol. 3, no. 1, Jan. 1, 1932, pp. 96-98.

In many districts of the Southwest water must be carried. The quantity required depends upon the time of year and the amount of work done by the miner. Men working where temperatures range from 100 to 110° drink 2 gallons or more of water per day; under such conditions a 10-gallon tank would last one man 3 days, allowing for cooking but not for the radiator of a car. In cooler weather a 10-gallon tank should last one man a week or 10 days.

Clothing

The most important item of clothing is a pair of stout, thick-soled shoes of good quality, preferably hobnailed. If an extensive trip is planned a second pair may be needed. Other clothing can be patched, but when a prospector's shoes go to pieces his trip is ended. A pair of rubber boots will prove a comfort if much placering is done.

Woolen socks to wear under the heavy shoes help to prevent blisters; several pairs may be worn out in a season.

Other clothes are chosen for the climate and service. A leather jacket is very serviceable and comfortable in cool weather, or a sheepskin coat may be needed when it gets colder. Many prospectors in mountainous regions wear flannel shirts and woolen underwear. Overalls are a common garb.

A compete change of clothing should be taken on all but the shortest trips to permit changing into dry clothing after being caught out in the rain or working in water all day.

First-Aid Supplies

As prospectors are likely to be away from medical aid, some medical and first-aid supplies should be taken along. These should consist of a laxative (castor oil or salts), iodine or merchurochrome to disinfect cuts or bruises, and a first-aid kit. A snake-bite kit may also prove invaluable.

LOCATING CLAIMS ON THE PUBLIC DOMAIN

By Fred W. Johnson[9]

CONTENTS

INTRODUCTION

In addition to a knowledge of ore minerals and their identification and occurrence, methods of sampling and prospecting, and supplies and equipment required, the prospector must have a general knowledge of how and where mining claims can be located on the public domain. The following information about mining laws has been compiled to meet this need, and every reasonable care has been taken to make it accurate and reliable at the time it was written (January 1936). The reader is warned, however, that changes are made from time to time in the laws and regulations covering the location and patenting of mining claims and that the Bureau of Mines is not the authority on the subject. <u>If specific advice is desired as to any provision of the United States mining laws, inquiry should be directed to the Commissioner of the General Land Office, Washington, D.C. Inquiries concerning the status of any tract of land should be made to the Register of the U.S. Land Office of the district in which the land is situated.</u>

Lands to Which Mining Laws Apply

The laws (act of May 10, 1872, and amendments) pertaining to acquiring mining claims on vacant public lands apply to Arizona, Arkansas, California, Colorado, Florida, Idaho,

9 Commissioner of the General Land Office.

Louisiana, Montana, Nebraska, Nevada, New Mexico, North Dakota, Oregon, South Dakota, Utah, Washington, and Wyoming, and to Alaska. Lands containing vein or lode deposits can be located and patented where vacant and unappropriated in the public domain, in National forests in the States named, in patented and unpatented stock-raising homesteads, in other agricultural entries not perfected or patented where prospecting can be done peaceably, and in railroad grants that have not been patented. Public lands temporarily withdrawn from settlement, location, sale, or entry and reserved for classification or other public purposes are, as a rule, open at all times to exploration for metalliferous minerals and to location and purchase under the mining laws. However, lands withdrawn for reservoir sites and certain other purposes are not subject to mining location; neither are those included in power sites unless restored under the Federal water power act. One desiring to prospect any particular lands should ascertain from the Register of the U.S. Land Office whether or not they are withdrawn or reserved from mining location. Placer claims generally can be located on lands having the same status as lands subject to location if containing vein or lode deposits, except that deposits of coal, oil, gas, oil shale, sodium, phosphate, potash, and sulphur (in Louisiana and New Mexico) belonging to the United States can be acquired only under the mineral leasing laws and are not subject to location under the United States mining laws.

Mining claims cannot be filed upon patented land except where the minerals have been reserved to the United States, on military reservations, or in National parks or monuments (except Mt. McKinley National Park in Alaska and Death Valley National Monument in California). Lands below high tide or the beds of navigable lakes and rivers are not subject to mineral location.

New lode locations can be made over abandoned earlier locations.

Public lands valuable for minerals are not subject to entry or patent under homestead or other nonmineral laws, except under certain laws that provide for patent with reservation of the minerals. Stock-raising homestead entries may include mineral lands not embraced in valid mining claims and may be patented with reservation of all minerals in the land to the United States. Lands entered or patented under this law may be prospected for minerals and, upon discovery, located as mining claims; but the miner's rights are restricted to the minerals in the land and the use of so much of the surface as necessary to mine and remove the minerals.

A lode or vein known to exist in a placer claim prior to the date of the filing of the application for placer patent can be located in the same manner as on vacant public land, but such location is limited to 25 feet on each side of the vein or lode at the surface. A lode deposit cannot be held under a placer location, but once a placer claim is patented the owner owns and may mine all lodes not known to exist at the time the application for placer patent is filed.

Mineral Discovery

The first requirement for locating a lode claim is to make a mineral discovery. This should consist of a "vein", "lode", "ledge", or "crevice" containing valuable "mineral" in place. It is not required that the mineral showing be of sufficient size or grade to be mined at a profit. The finding of float on a lode claim, even if present in sufficient quantity that it may be collected at a profit, does not constitute a mineral discovery. Although technically a valid lode claim cannot be located until actual valuable mineral has been found in place, it is common custom to post a location notice on open ground where hidden leads are being sought by excavating. If valuable mineral is found by this work, the prospector is afforded some protection by having the location posted should an effort be made to "jump" the ground.

Location of Lode Claims

Mining locations may be made by citizens of the United States, by those who have declared their intention to become citizens, by an association of qualified persons, or by a domestic corporation. Locations can be made by minors who have reached the age of discretion, and without regard to the sex or residence of the locator. A locator may include as colocators other qualified persons who may or may not have seen the ground; moreover, a person may make valid locations as agent for other qualified parties.

No limit is placed by the Federal statutes on the number of locations that may be made by an individual or a company. Both lode and placer claims may be amended and the boundaries changed at any time, provided that such changes do not interfere with the rights of others.

A location notice must contain the names of the locator or locators, the date of location, and a description of the claim by reference to some natural object or permanent monument that will identify it. Lode claims must be marked distinctly on the ground so that their boundaries can be traced readily. State laws define how the location notice must be posted, what the size of the discovery cut or shaft shall be, and how the claim boundaries shall be marked. The location notice should be filed as required by the State laws.

Lode claims are limited to 1,500 feet in length and 300 feet on each side of the lode or vein at the surface. The maximum size of a claim is a parallelogram 600 by 1,500 feet (20.661 acres). The end lines of the claim must be parallel. A claim does not need to be of full size or to be rectangular. If a vein on which a claim is staked curves, the side lines of the claim may be broken to make the location fit the vein. A full-sized claim may be staked so as to embrace two or more noncontiguous fractions of open ground, and a discovery on one of such fractions is sufficient to validate the entire location.

Assessment Work

To hold the possessive title to a mining claim, not less than $100 worth of work must be done or an equivalent value of improvements made upon or for the benefit of each claim each year, regardless of its size. Where a number of contiguous claims are held in common, the aggregate expenditures for the group may be made on one claim, provided such expenditure tends to benefit or develop each claim of the group. Locations connecting only at the corners are held to be noncontiguous. The period within which the annual work must be done begins at noon of July 1 succeeding the date of location. Failure to do the annual assessment work will subject a claim to location by others unless work is resumed before such relocation. It has been held that a claim is not subject to relocation if work is being done on the ground at the end of the required period. In other words, if work is begun by noon of July 1, 1930, on a claim located in September 1928, and diligently carried on thereafter to completion, it is not subject to relocation. Additional work would be required for the period beginning July 1, 1930. Annual expenditure is not required after entry is made at the Land Office for patent.

Should the annual assessment work not be done on a claim for one or more years, the location will still be valid if work is resumed on the ground, provided there has been no relocation by another prior to such resumption of work. Most States have provided for filing proofs of labor for the annual assessment work.

Where one of several locators fails to contribute his share of the required expenditures made for the benefit of a claim, the co-owners, at the expiration of the period, may give notice personally, in writing, or by advertising in the newspaper published nearest the claim at least once a week for 90 days; if upon the expiration of 90 days after the personal notice or upon the expiration of 180 days after the first newspaper notice the delinquent co-owner shall have failed to contribute his proportion of such expenditures or improvements, his interest in the claim passes by law to his co-owners who have made the required expenditures.

Suspension of Assessment Work

Congress suspended the assessment work on all claims for the assessment year ended July 1, 1932. All claim owners who were exempt from the payment of Federal income tax were relieved from making the required annual expenditure for the years ended July 1, 1933, July 1, 1934, and July 1, 1935, subject to filing and recording of notice of intention to hold the claims. During 1934-35, the suspension applied to only 6 lode claims, or 120 acres of placer ground, held by an individual or 12 lode claims, or 240 acres of placer ground, held by a partnership, association, or corporation.

Indian Reservation

The Secretary of the Interior has been authorized by Congress (act of June 30, 1919, and amendment of March 3, 1921) to lease unallotted lands on Indian reservations for mining purposes in Arizona, California, Idaho, Montana, New Mexico, Nevada, Oregon, Washington, and Wyoming. After declaration by the Secretary that the lands are subject to lease, claims may be located as on the public domain; a duplicate of the location notice must be filed with the superintendent in charge of the reservation within 60 days. The locator has 1 year's preference right to apply for a lease through the reservation superintendent to the Secretary of the Interior. Leases are for 20 years, with provision for 10-year renewals.

In other States, unallotted lands not needed for allotment or agriculture may be leased for mining purposes for not more than 10 years.

Lands in the Papago Indian Reservation, in Arizona, are subject to location and patent under the provisions of the United States mining laws. Locators are required to pay a rental of not less than 5 cents per acre to the Papago Tribe, and, in event patent is desired, 1 dollar per acre; and the amount of any damages for loss of improvements made by the Indians must be paid, the damages to be determined by the Secretary of the Interior.[10]

National Forests in Middle Atlantic States

As stated before, mineral lands in national forests in the public land States may be entered as on the public domain. In the Middle Atlantic States (where the Federal mining laws do not apply) special regulations have been promulgated by the Department of Agriculture permitting prospecting, development, and utilization of the mineral resources on national forest lands.

Prospecting may be carried on without a permit, but no extensive excavations can be made or structures erected without a permit. For a fee of $5.00 exclusive prospecting permits, one to a person, will be issued to qualified persons to explore a specified area not to exceed 100 acres. Permits may be renewed.

Upon application, after the discovery of valuable mineral deposits, a mining permit will be granted for 5 to 20 years at a rental of not less than $1.00 per acre per year, and not less than $2.00 for any permit. In addition, a royalty of 2 to 8 percent of the value of the minerals mined will be charged. Rules are laid down by the Forest Service as to cutting timber and how the mining work shall be conducted.

---

10 See Circular 1347, General Land Office, February 27, 1935.

State Lands

When statehood was conferred upon the Western States, Congress granted to them sections 16 and 36 of each township for school, road-building, or other purposes. Sections 2 and 32 also were granted to Arizona, New Mexico, and Utah. Some special land grants also have been made to most of the Western States. By the terms of the original grant, the States were required to take lieu selections for lands already occupied at the time of the grant or known to be mineral at the time the land was surveyed. Congress, however, by the act of January 25, 1927, granted the States the mineral school sections subject to existing claims.

Most of the mining States provide for leasing minerals found on State land, but some provide for mining locations. After discovery, application for a prospecting or mining lease should be made to the authority having charge of State lands. Regulations on the granting of prospecting or mining leases vary in different States.

Mining Claims on Stock-Raising Homesteads

Patents to stock-raising or grazing homesteads reserve all minerals to the Government. Any qualified locator may go upon the lands entered or patented under the stock-raising homestead act to prospect for minerals, provided he does not damage the permanent improvements of the entryman; he also is liable for all damage he does to crops. (See General Land Office Circular No. 523.)

Anyone who has acquired the right from the United States to mine the minerals may reenter and occupy as much of the surface as is required for mining purposes (1) by obtaining a written consent or waiver from the homesteader; (2) by payment for crops or tangible improvements to the owner under agreement; (3) by posting a bond of at least $1,000 to cover any damages that might be awarded by a court of competent jurisdiction. The bond must be filed with and approved by the register of the Land Office. The Land Office will allow mineral applications on stock-raising homesteads, whether patented or held under entry, and patent will be issued in the regular manner except that it will contain the notation that the land is subject to occupancy and used in accordance with the act of December 29, 1916.

State Mining Laws

By the act of May 10, 1872, Congress authorized State and Territorial legislatures to pass laws regulating the location and holding of mining claims on the public domain. The States in which the mining laws apply have made regulations in addition to those passed by Congress regarding mining claims. By the act of May 17, 1884, the Federal mining laws were extended to Alaska.

Size of Claims

Except for South Dakota and North Dakota, the maximum size of lode claims prescribed by Congress (600 by 1,500 feet) is permitted by State laws. In South Dakota the claims are full width, except where a county at a general election may determine on a narrower width, but not less than 25 feet on each side of the center line. In North Dakota the standard lode claim is 150 feet on each side of the center line, except that by the same procedure as above the width may be increased to 300 feet on each side of the center line. In Colorado, claims were limited to 75 feet in Gilpin, Clear Creek, Boulder, and Summit Counties on each side of the center line, and to 150 feet on each side of the vein in the other counties of the State before 1921, at which time the law was amended to allow full-size claims.

Location Notices

All of the States that have enacted laws pertaining to the location of mining claims require location notices to be posted at the point of discovery, except in New Mexico, where the law provides that the notice shall be posted at a conspicuous place on the claim, and Oregon, where the law merely provides that the notice must be posted on the claim. The United States mining laws require that a location notice for a lode claim shall contain (1) the name of the claim, (2) the name of the locator or locators, (3) the date of location, and (4) such a description of the claim or claims located, by reference to some natural object or permanent monument, as will identify the claim. In addition, the laws of Arizona, California, Idaho, Nevada, Oregon, South Dakota, Utah, and Wyoming require that (1) the general course of the vein as nearly as can be determined, (2) the distance claimed on each side of the center line or discovery cut, i. e., the width of the claim, and (3) the distance claimed both ways along the lode from the point of discovery be shown either in the location notice posted on the claim or in the certificate of location filed for record. Also, the Idaho laws require that the location notice contain the name of the mining district, county, and State. In Alaska, in addition to the Federal requirements, a lode location notice must state the number of feet claimed along the vein each way from the point of discovery and the width on each side of the center line; in Colorado the notice must state the number of feet claimed on each side of the discovery shaft and the general course of the lode as near as can be determined; and in Montana the notice must give the approximate dimensions of the claim.

In most mining districts blank forms of location notices can be purchased from printing establishments or at stores handling stationery. These forms are a convenience in making locations and show the requirements to be followed in the particular State.

Filing Certificate of Location

All Western States require that a copy of the location notice or a certificate of location be filed for record. Thirty days from the date of making discovery or posting of location notice is allowed for filing lode claims in California and Utah; 60 days in Montana, North Dakota, Oregon, South Dakota, and Wyoming; 90 days in Alaska, Arizona, Colorado, Idaho, Nevada, New Mexico, and Washington. In Arizona, California, Colorado, Idaho, and North Dakota location notices or certificates and proofs of annual labor are filed with the county recorder; in Alaska, with the district mining recorder; in Montana, New Mexico, and Wyoming, with the county clerk; in South Dakota, with the recorder of deeds; and in Washington, with the county auditor. In Nevada and Utah notices are filed in duplicate with the district mining recorder if there is one; otherwise, with the county recorder. In Oregon the notices are filed with the recorder of conveyances, if there is one; otherwise, with the county recorder. Arkansas does not require the filing of notices for record, but provision is made for such filing with the county recorder if the mining claimant so desires. The cost of filing notices for record varies from \$1.00 to \$2.00. In Arizona and New Mexico the notice filed for record must be a copy of the location notice, and in Idaho and Utah the notice filed must be a "substantial" copy of the location notice. In Montana the notice filed must be sworn to, and in Oregon an affidavit must be attached stating that the discovery work has been performed. In Wyoming the notice filed for record must give the location of the claim by reference to section or quarter-section corners, if the claim is on surveyed land.

Marking Boundaries

In all Western States the boundaries of lode locations must be marked before the location notice is filed for record. In Montana and Oregon the boundaries must be marked within 30 days; in Idaho, 10 days; and in Nevada, 90 days after the location notice is posted.

In Utah there are no State regulations as to marking corners, except that they shall be marked distinctly. In Idaho, Montana, and Washington the corners or angles of the claims (four or more corners) shall be marked with substantial monuments. In Alaska, Arizona, California, Nevada, Oregon, and Wyoming each corner and center-end lines of lode claims shall be so marked. In North Dakota and South Dakota each of the four corners, the center-end line, and the center-side lines of lode claims shall be marked and the corners of placer claims by monuments. In Alaska and Washington the claim boundaries must be marked by cutting brush or blazing trees, if claims are covered by such growth. In Colorado the corners of the claim and the center-side lines must be marked.

In Alaska the claim corners of lode claims must be marked by posts at least 3 inches in diameter and 3 feet above ground or by mounds of earth or stone 3 feet high and 3 feet in diameter. The corners must be marked with the name of the claim, the number or position of the corner, and the direction of the boundary lines. In Arizona the monuments must consist of a 4-foot post or a mound of stone 3 feet high. In Colorado and Idaho when posts or trees are used they must be marked, and in Idaho if posts are used they must be at least 4 inches square. In Montana a corner can consist of an 8-inch tree blazed on four sides, a 4-inch square post 4-1/2 feet long surrounded by a mound 4 feet in diameter and 2 feet high, or a square stump with mound, a stone 6 by 18 inches two-thirds in the ground with a mound 4 by 2 feet nearby, or a boulder 3 feet above ground, and each corner is to be marked. Requirements for corners in Nevada are similar to those in Montana. In North Dakota and South Dakota corners are to consist of substantial posts hewed or glazed on the side or sides facing the claim and marked with the name of the claim and the corner. In Oregon the corners must consist of posts at least 4 inches square and 3 feet high or mounds at least 2 feet high; in Washington similar monuments are required, except that mounds must be 3 feet high. In Wyoming substantial corners of stone or posts sunk into the ground are required; they must be marked on the side or sides that face the claim.

Discovery Excavations

No discovery shafts or excavations are required on lode-mining claims in Utah. Alaska, California, Colorado, Idaho, Montana, Nevada, New Mexico, Oregon, South Dakota, Washington, and Wyoming require that a 10-foot shaft shall be excavated or an equivalent excavation made. In Alaska, the shaft or its equivalent shall be made within one year of the date of location; in California, Nevada, and New Mexico within 90 days; in Colorado, Idaho, Montana, and Oregon within 60 days; in South Dakota, Washington, and Wyoming before the location notice or certificate is filed for record. In North Dakota sufficient excavation must be made to show a well-defined vein or lode before filing the location notice. In Arizona and Nevada the shaft must be at least 4 by 6 feet in section; in Idaho it must be 16 square feet in cross-section or contain 160 cubic feet; in Montana it must contain 150 cubic feet, part of which can be elsewhere on the claim, but 75 cubic feet must be excavated at the point of discovery.

Proofs of Labor

In Alaska, Arizona, California, Idaho, Nevada, New Mexico, Utah, Washington, and Wyoming proof of labor, showing that the annual assessment work has been done on claims, is required to be filed. Such proofs should be sworn to and should be filed within 90 days after the first day of July in Alaska and Arizona; within 60 days after the first day of July in Idaho, New Mexico, and Wyoming; within 30 days after the first day of July in California and Washington; within 60 days after the work is done in Nevada and Wyoming; within 30 days after the work is done in Utah; and within 20 days after the work is done in Montana. In Arkansas, Florida, Louisiana, Nebraska, North Dakota, Oregon, and South Dakota it is not necessary to file proofs of labor.

Status of Unpatented Lode Claims

A valid lode claim held by right of location may be sold or leased like any other real estate. Ores may be mined and sold from claims held under location, as from patented claims. Patented and unpatented lode claims are taxable, as are buildings and their contents placed upon them.

Timber Rights

Timber and stone on national forests may be used free of charge by bonafide settlers, miners, residents, and prospectors for firewood, fencing, building, mining, prospecting, and domestic purposes under regulations set forth by the Forest Service. Timber on unpatented claims may be used for mining purposes but not sold.

Procedure to Obtain Patent to Lode Claims

Valid locations or groups of locations on which not less than $500 has been expended for the benefit of each claim may be patented. Proceedings for patent are instituted in the district Land Office. The claims or claim must be surveyed by a U.S. mineral surveyor; the application for a survey is made to the public survey office. Notice of the application is required to be posted on the land before the application is filed and published by the register of the Land Office after the application is filed. Information as to patent procedure can be obtained from the register of the local land office or from the General Land Office in Washington.

Adverse Claims

An adverse claim must be filed under oath with the registrar of the Land Office before the period of advertising expires. The adverse claim must set forth fully the nature and extent of the interference or conflict, whether the adverse party claims as a purchaser or as a locator. If the former, a duly certified copy of the original location notice, the original conveyance, or an abstract of title from the office of the proper recorder should be furnished; if verbal, the circumstances should be narrated. If as a locator, he must file a duly certified copy of the location notice from the office of the proper recorder.

The adverse claimant must also file a plat showing his entire claim and its relative position with regard to the one that he claims conflicts, unless the claims of both parties are located by legal subdivisions.

Upon filing the adverse claim, the register of the Land Office will give notice to the parties that adverse claim has been filed, informing them that the adverse claimant will be required, within 30 days from the date of such filing, to begin proceedings in a court of competent jurisdiction to determine the question of right of possession and to prosecute the action with reasonable diligence to final judgment, and that, should he fail to do so, his adverse claim will be considered waived and the application for patent will be allowed to proceed on its merits.

I. C. 6891 June 1936

Department of the Interior

United States Bureau of Mines
John W. Finch, Director

INFORMATION CIRCULAR

DESIGN AND OPERATION OF A FOUR-TON-PER-HOUR GOLD AND SILVER ORE-SAMPLING PLANT

BY

E. D. Gardner and W. A. Leddell

I.C. 6891,
June 1936

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

DESIGN AND OPERATION OF A FOUR-TON-PER-HOUR GOLD AND SILVER ORE-SAMPLING PLANT

By E. D. Gardner[2] and W. A. Leddell[3]

CONTENTS

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6891."

2 Supervising engineer, U.S. Bureau of Mines, Tucson, Ariz.

3 Mechanical and metallurgical engineer, El Paso, Tex., and one of the consulting engineers, U.S. Bureau of Mines.

INTRODUCTION

During the summer of 1935 a plan was proposed by which the Government should establish custom sampling plants for gold and silver ores in the mining districts of the West, where the construction of such plants would be justified as an unemployment relief measure. It was known that in many districts prospectors and small operators had small lots of smelting ore that could not be marketed for lack of sufficient capital to get out enough ore at a time to make carload shipments to distant smelters. Moreover, considerable ore of milling grade was available in many districts in which there were no custom milling facilities. It was felt that accurate sampling and the classification and consolidation of small lots of ore produced from prospects and small mines, with quick cash returns to the shippers on a nonprofit basis, would stimulate the working of small gold and silver properties. Idle mines would be put to work, and there was a possibility that some of the prospects could be developed into regularly producing mines.

Each sampling plant was to be centrally located so as to serve the greatest possible number of mines and prospects. Wherever possible, the plants were to be built on a railroad. As ore of smelting grade accumulated, it was to be shipped to a smelter in carload lots. Where a plant was not on a railroad, smelting ore was to be trucked to the railroad after being sampled.

Ore of milling grade was to be shipped by truck or rail to existing custom mills, if such there were within shipping distance. Field surveys showed, however, that in some districts not at present served by custom mills enough ore probably could be obtained to supply local plants once mining operations were properly organized. Mill ore could be purchased and stock-piled for subsequent treatment in those districts. It appeared probable that after a few thousand tons had accumulated and further shipments seemed assured, private interests would build a local custom mill to treat the ore.

The authors were assigned the task of formulating standard plans for the custom sampling plants of the project; their findings are contained in this paper. Although the project was not finally approved, it is believed that publication of the plan together with discussion of general sampling-plant practices will be of assistance to those who may be considering establishing private sampling works.

ACKNOWLEDGMENTS

The authors acknowledge the assistance of the various machinery houses in the West, who kindly provided data concerning their products and quoted prices on equipment. The Mine & Smelter Supply Co. and the Denver Equipment Co. of Denver and the Southwestern Engineering Co. of Los Angeles also submitted estimates on the cost of a complete plant. W. O. Vanderburg, mining engineer of the U.S. Bureau of Mines at Reno, Nev., drew the plans for and estimated the cost of the assay office.

GENERAL PRACTICE AT SAMPLING PLANTS

Although ore continues to be sampled by hand, the general practice is to do the work mechanically. Moreover, automatic mechanical sampling is more economical than hand sampling.

The point to which mechanical sampling is carried before hand working begins depends upon the mechanical perfection of the devices employed, principally as to cleanliness and loss of fines. In the final stages of mechanical sampling, cleanliness and the mixing of fines are very important.

The object of sampling a lot of ore is to determine its true value. There is no reason, however, for sampling to a degree of accuracy beyond that to be obtained in assaying the final sample.

A difference of 0.01 ounce between two assays by the same assayer of the same sample is considered a good check, yet 0.01 ounce of gold at $35 per ounce is worth $0.35, or 5 percent of the value of $7 ore. This percentage decreases, of course, as the value of the ore increases.

The samples obtained from lots of low-grade ore may vary by 5 percent of its true worth, but the discrepancy should be no greater than 1 percent on high-grade shipments.

Size of Products in Sampling

The fineness to which any lot of ore must be crushed before a predetermined proportion of its weight may be taken for a sample depends upon the size and grade of the lot and the distribution of the valuable minerals in it. Writers on the subject do not agree, and practices at sampling plants are far from uniform.

The problem has been attacked from both the mathematical and experimental sides. Taggart[4] summarizes the results of various investigators.

Table 1, by Woodridge,[5] based on extensive experiments and calculations by Brunton,[6] may be used as a guide for a medium-grade ($10-$20) gold ore.

For example, as shown by table 1, a 1-ton lot of medium-grade gold ore should be crushed to 1 inch before a sample cut is taken. Should the ore be "spotty" or high grade, finer crushing would be required before the initial cut is taken. On the other hand, a 1-ton lot of uniform, low-grade ore need not be crushed to this size.

Taggart[7] states that in sampling base ores the weights indicated in Woodridge's table probably may be multiplied safely by 10.

The size of crushing before each cut is taken at the contemplated plants discussed later in this paper is well within the limits set by Woodridge and other authorities. All lots would be crushed to 7/8 inch, and small lots of high-grade ore would be reduced to 1/4 inch before sampling is begun.

TABLE 1.- Smallest permissible weight of sample for varying sizes of crushing of medium-grade gold ore

Maximum size of products	Smallest permissible weight, pounds
2 inches	10,000
1 1/2 inches	5,000
1 inch	2,000
3/4 inch	1,000
3/8 inch	400
1/4 inch	300
3/16 inch	100
1/8 inch	75
6 mesh	50
10 mesh	25
18 mesh	10
30 mesh	4
50 mesh	1

4 Taggart, Arthur F., Handbook of Ore Dressing: John Wiley & Sons, Inc., New York, 1927, p. 1127.

5 Woodridge, T. R., Ore-Sampling Conditions in the West: Tech. Paper 86, Bureau of Mines, 1916, p. 57.

6 Brunton, I. W., Theory and Practice of Ore Sampling: Trans. Am. Inst. Min. Eng., vol. 25, 1897, p. 826.

7 Taggart, Arthur F., Work cited.

Size of Sample Cuts

Obviously, to obtain an accurate sample the size of the first cut must be governed by the size of the material to be sampled. For example, 1 percent of a very fine mill pulp may be sufficient to give accurate results for metallurgical control, while the same percentage of coarse ore would be not much more accurate than a grab sample.

By reducing the diameter of a cube of ore by one half, each fragment would be one eighth of the weight of the original cube. Theoretically, therefore, after each one-half reduction in size of the ore fragments, the minimum cut to be taken in sampling should be 12-1/2 percent. To allow for oversize getting through the crusher, probably 20 percent should be the minimum percentage considered for a cut after each one-half reduction. The percentage of the cut that may be taken safely for each intermediate size can be determined from Woodridge's table. The above was considered in the design of the proposed plants; the contemplated percentage for each cut would be ample.

The practice in regard to the size of cuts at different sampling plants varies greatly. The initial cut ranges from 5 to 25 percent, but is 20 percent at most plants. At the El Paso smelting plant of the American Smelting & Refining Co. five 20-percent cuts are taken, each after a reduction in size. The resultant product is mixed and quartered by hand for final grinding.

Moisture Samples

Accurate moisture samples are difficult to take, and generally are a bone of contention between the seller and the buyer of ores. The writers know of no entirely satisfactory method of taking moisture samples from run-of-mine ores. It would appear desirable that a sample for moisture determination should be taken at the time the ore is received and weighed, as some of the moisture in the ore will evaporate while the lot is being handled; on the other hand the ore may in some cases be rained upon between weighing and crushing. Obtaining an accurate sample at the time of weighing, however, depends upon the personal skill of the one taking the sample and may be no more than an estimate in many instances. Despite its limitations, this practice continues to be followed by many purchasers of custom ore.

At most plants where the moisture sample is taken on receipt of the ore, two or more holes are dug into the truck or car load after the top layer has been scraped off, and one or two 10-pound samples are obtained and placed in sealed containers until ready to be dried. Duplicate samples taken in this manner seldom check very closely. The principal chance for error lies in the fact that fines contain relatively more moisture than coarse material. Obviously, the personal equation cannot be eliminated; personal judgment must be depended upon to get a fair sample.

Occasionally grab samples for moisture are taken at the end of a conveyor belt from the primary crusher; if the ore has been thoroughly mixed before it reaches this point and a weightometer used on the conveyor, such a grab sample would reveal fairly accurately the moisture in the ore at the time the sample was taken. In most plants, however, little mixing of ore occurs in the initial crushing.

Fulton[8] considers grab sampling unfair and recommends that the regular plant sample prepared for final grinding be used in determining the moisture and that allowance be made for evaporation of moisture in the passage of the ore through the plant; this is considerable and must receive consideration.

---

8 Fulton, Charles H., The Buying and Selling of Ores and Metallurgical Products: Tech. Paper 83, Bureau of Mines, 1915, p. 11.

According to Brunton,[9] experiments have shown that the loss of moisture during machine sampling is about 10 percent in summer and 7 percent in winter in Colorado. Hence, an indicated, 5-percent moisture content of an ore sampled in the summer would contain actually 5.5 percent. He also states that 1 percent should be the maximum discount to be added to offset evaporation; if the sample contained 15 percent moisture, the deduction for moisture should be 16 and not 16.5 percent. Duplicate samples of relatively dry material should check within 10 percent.

Brunton's figures for loss of moisture in Colorado ores would be too low for damp ore in the Southwest, where summer temperatures are over 100°F. and relative humidity usually less than 20 percent during the day. Gold ores from near the surface in the Southwest usually contain about 1 percent water in dry weather. Some of the southwestern gold custom plants make a blanket deduction of 1 percent for moisture on all shipments in dry weather. Deductions for moisture in similar ores shipped to smelters usually range between 2 and 4 percent. Talcy ores from wet mines in the mountain regions may contain as much as 30 percent moisture.

Moisture samples are dried at a temperature of 212° to 250°F. until a constant weight is obtained.

PROPOSED METHOD OF SETTLEMENT FOR PURCHASED ORES AT CONTEMPLATED SAMPLING PLANT

Payment for purchased ores is based on the assays of the samples taken of the ores.

The final product from sampling a lot of ore generally is split into four 250-gram portions. One split is given to the shipper, one used for assay at the plant, one held for umpire, and the fourth for a recheck at the plant if needed.

Usually, if within certain limits, settlement is made on the average of the buyer's and the seller's assays; that is, the difference is split. If the discrepancy is beyond the limits set, a sample is submitted to an independent assayer for "umpire" determination. Should the result of the umpire fall between the disagreeing assays, this assay is taken as the basis of settlement. In this case the cost of the assay usually is shared between the buyer and the seller. Should the umpire assay be higher or lower, the one nearer the umpire assay is chosen for making the settlement; the loser pays for the assay. In some districts either the buyer or the seller may call for an umpire at any time irrespective of the difference between the two assays. Some purchasers of ore pay only on their own assay, except where an umpire is called for. In justification of this latter policy, it is pointed out that assay offices at most small mines generally are not properly equipped nor are competent assayers always employed, whereas up-to-date laboratories are provided by the purchasers and experienced assayers employed.

In assaying low-grade gold ores it was formerly considered that duplicate assays of the same sample by the same assayer should check within 0.02 ounce. With gold at $35 per ounce, however, this difference is $0.70 per ton, which appears to be too high. An attempt should be made to keep the discrepancy to 0.01 ounce per ton. Due to the inherent difficulties of assaying, however, results within 0.02 ounce can be considered a close check on the same sample of low-grade ore by two independent assayers. Obviously, therefore, differences within 0.02 ounce should be split. For high-grade ore a larger tolerance is allowable.

Control assays for low-grade silver ore should check within 0.2 ounce; for 50-ounce ore the discrepancy in the assays may be as much as 0.5 ounce.

According to Taggart,[10] the usual splitting limits for copper ore are 0.2 to 0.5 percent; lead ore, 0.5 to 0.6 percent; and zinc ore, 0.6 to 1 percent.

---

9 Brunton, I. W., Theory and Practice of Ore Sampling: Trans. Am. Inst. Min. Eng., vol. 25, 1897, p. 826.

10 Taggart, Arthur F., Handbook of Ore Dressing: John Wiley & Sons, Inc., New York, 1927, p. 1127.

In the plans for administering the plants considered in this paper, it was proposed to buy the ore outright and dispose of it later unless the shipper should prefer to do his own marketing, in which case it would be sampled and returned to him. The sampling, treatment, freight charges, and metallurgical losses would be calculated for each lot purchased and the proper deductions made at the time of settlement. If doubt should exist as to the recovery that might be made on any lot of mill ore received, or the cost of smelting a concentrate, a part payment would be made on receipt of a lot and final settlement after the final returns were in.

When ore was purchased outright, it was planned to pay the shipper on the basis of the dry weight for each lot as soon as the assays were out. Settlement was to be made immediately for small and intermittent lots on the assay at the custom mill when the shipper had no facilities for making the assay or did not want to go to the expense of having it made by a custom assayer. When the shipper had proper facilities for assaying and a competent assayer, or wished to have his assaying done by a competent custom assayer, settlement was to be deferred until the shipper's assay was received; if an umpire was required, settlement would be further delayed. In special cases, however, 90 percent of the settlement price would be paid when the buyer's assay was out.

Settlement for gold ores would be made as follows: If the difference between the shipper's and the buyer's assays was 0.02 ounce or less per ton, this difference would be split on all shipments that contained 1/2 ounce or less per ton. On ores containing between 1/2 and 1 ounce of gold, a difference of 0.03 ounce would be split; and containing over 1 ounce, 0.04 ounce. On small lots, however, by mutual agreement greater differences would be split. When the differences in assays were greater than the above, a sample of the disputed lot would be sent to a competent assayer agreed upon by both parties for umpire. Settlement would be made on the umpire assay when the results obtained by the umpire fell between those shown by the buyer's and the seller's assays; the cost of the umpire assay would be divided evenly between buyer and seller. When the umpire assay was higher than the seller's the seller's assay would be used as a basis of settlement, and when lower, the buyer's. The loser would stand the cost of the umpire.

The splitting limits suggested by Taggart would be followed for determining base metals in the ores. For silver ores that contained up to 20 ounces per ton, differences up to 0.5 ounce would be split. On high-grade ore the same procedure would be followed when the differences in assay did not exceed 1 percent of the value of the silver. Greater differences would be umpired in the same manner as for gold ores.

ESTIMATED CHARGES AGAINST PURCHASED ORES AT CONTEMPLATED SAMPLING PLANT

It was planned to run the plants at cost; neither a profit was to be made nor losses sustained on any of the lots of ore received. To come out even, should the ore be purchased outright, the following charges would have to be made against each lot: (1) Cost of sampling; (2) cost of treatment and transportation of the ore after sampling; (3) metallurgical losses incurred during treatment; and (4) cost of smelting concentrates when made.

Cost of Sampling

The cost of sampling at the small plants contemplated would be relatively high. The tons per man-shift handled would be low, and as most of the plants would be in isolated locations, the cost of power and supplies would be high. The cost of sampling, exclusive of assaying, at large private sampling works in the West is about $0.50 per ton; the cost at these small contemplated plants would be about three times that amount.

The proposed labor for running one of the contemplated plants on a one-shift basis with average wages would be:

Foreman - operator	$7.00
Assayer - clerk	6.00
Operator	4.50
Crusherman	4.50
Dumpman	4.00
	26.00

At an average of 4 tons per hour for 8 hours daily, the labor cost would be $0.81 per ton. When the plants run at part capacity or intermittently the labor cost would be higher. The plants discussed in this paper could handle considerably more than 4 tons of ore per hour if necessary, with a correspondingly lower cost per ton. Moreover the same crew could run a plant of several times that capacity.

Electric power would be used where available, but small gasoline or Diesel engines probably would be used at most of the plants contemplated. The sustained load would be about 26 kw. (35 hp.). At an estimated average of $0.02 per kw.-hr., the power cost would be $0.13 per ton. Supplies are estimated at $0.07 per ton of ore sampled.

The average cost of plants, as is shown later, would be about $20,000. Estimating the life of a plant to be 10 years and interest on the investment to be 4 percent, the yearly amortization would be $1,666; for 9,600 tons per year (32 tons daily for 300 days), the cost per ton would be $0.17. Cost of replacements would about equal cost of supplies, or about $0.07 per ton.

Although charges for administration were not contemplated in the original set-up, such costs would have to be included for private plants; they, together with general charges, are estimated at $0.10 per ton.

The estimated average total sampling cost per ton, then, would be:

Labor	$0.81
Power	.13
Supplies	.07
Amortization	.17
Replacements	.07
Administration and general	.10
	1.35

If enough ore were available to warrant it, lighting facilities could be provided and the mill run on a two- or even three-shift basis, which would reduce the administrative and amortization charges per ton.

As the work of handling the end products of the sampling and the cost of assaying would be the same for both small and large lots, a higher rate per ton would have to be charged for the small lots. Probably a flat charge of about $5 plus a fixed charge per ton would be made for each lot.

Cost of Treating Ores After Sampling

Smelter Rates

A part of the ore from the proposed sampling plants would be shipped to smelters.

Smelter rates in the West are competitive. Intermittent shipments of gold-silver ores usually are bought under open schedules. Commercial smelters, with both copper and lead furnaces, may have schedules for two or more classes of ore. In such cases, a shipper may sometimes choose the schedule that will bring him the highest return.

A base rate usually is quoted for each class of ore; the smelting charge increases with the value of the ore to a fixed maximum. Because of the advantages of receiving a steady supply of ore, smelters usually give lower rates to shippers who can contract a definite tonnage per month. When the same district is served by more than one smelter, concessions in rates can sometimes be obtained by bargaining. Special rates are offered by some copper smelters for siliceous gold ores, which are desired for their fluxing qualities or for converter linings. The minimum charge on this class of ore at one western copper smelter is $1.50 per ton. Special concessions also are granted sometimes for other special types of ores.

Smelter tariffs for siliceous gold-silver ores.- Only smelters with copper furnaces purchase ores under this schedule. The minimum base treatment rate for siliceous gold-silver ores in the open schedules ranges from $3.00 to $8.50 at most western smelters; the maximum ranges from $6.00 to $15.00. From 90 to 95 percent of the gold, depending upon the grade of the ore and the practice at the smelter, is paid for at the mint price. Usually 95 percent of the silver, if over 1 ounce per ton, is paid for at the mint price if the proper affidavits showing the ore to be of domestic origin and newly mined (1935) are supplied. About 90 percent of the copper is paid for at the market quotation, less 2.5 cents per pound, with a further deduction of 10 pounds per ton at most smelters. No payment is made for lead. The ore may contain a maximum of 5 to 10 percent zinc without penalty. Penalties of 30 to 50 cents per unit (20 pounds) are charged for zinc in excess of that amount. Arsenic, if over 2 percent, is penalized at $0.50 to $1.00 per unit. Penalties are also charged for antimony, bismuth, and other deleterious metals. A charge averaging about $1 per ton is made at most smelters for weighing, sampling, and other miscellaneous services. A special charge of $10 is usually made for assaying and sampling small lots.

Smelter tariffs for gold-lead ores.- The base treatment charge for ores containing lead is usually lower than for ores shipped under the siliceous gold-ore schedule. Gold and silver are paid for in a similar manner. Like penalties are charged for zinc and other deleterious metals and, in addition, a penalty of 5 to 12 cents for each unit of insoluble (silica plus other insoluble material) in excess over iron in the ore is charged. Lime in the ore is credited at 8 to 10 cents per unit at some smelters. Usually 90 percent of the lead is paid for at New York quotations minus 1.5 cents per pound. At most smelters lead is not paid for unless the ore contains over 3 percent. All copper is paid for less 10 to 20 pounds per ton, and a further deduction of about 6.5 cents per pound is made from the New York quotation on the day of sale.

Freight Charges

Commodity rates on carload lots are established usually by the railroads when regular shipments of ore are to be made. A base rate for low-grade ore is quoted; an increase is made for regular increments in the value of the ore. Freight schedules are usually competitive, at least to some extent; the same rate may be established from a mining district to two smelters at different distances. The base freight rate on ore from western mining districts to smelters may range from $0.25 to about $8.00 per ton on main-line roads and up to about $16.00 on branch lines. Usually it is cheaper to ship less-than-carload lots by truck from most western districts.

Milling Charges

Should mill ore be treated by custom mills in the district, the cost of trucking to the mill and the cost of milling must be charged against the ore sold.

Custom plants have been discussed in a recent paper by the senior author.[11] Custom charges, as shown in that paper, ranged from $1.75 to $9.60 per ton, depending upon the character and value of the ores treated, the milling method used, and to some extent upon what the traffic would bear.

Should a milling plant be built to handle the ore from the contemplated sampling works, the milling rates under average conditions are calculated to be about as follows:

Plant capacity	Rate per ton	
	Flotation	Cyanidation
50 tons per day..	$3.00	$4.00
100 tons per day	2.50	3.50

The above charges include amortization, overhead, and a milling profit of $0.75 per ton. A deduction of $0.25 per ton has been made on account of crushing the ore at the sampling plant. The direct costs included in the above figures are based on average costs in 1934 at a large number of gold mills of similar capacity in the West. With higher wage scales the labor costs would be increased.

If a satisfactory recovery could be made by amalgamation followed by table concentration, the milling rate would be less than for flotation.

Mills built to handle only the product from the sampling works would not need a crushing plant.

Metallurgical Losses in Mills

Obviously, only the metals saved in milling could be paid for. Based upon experience at small gold mills in the West, the recovery at cyanide plants would average about 92 percent, at flotation mills 87 percent, and at amalgamation plants 70 percent.

An estimate of the probable recovery would be necessary for each lot of mill ore received. In some cases a metallurgical test would be necessary.

The operators of most well-equipped custom mills are familiar with the different types of ores received and from past experience can anticipate closely the recovery that will be made on any lot. In such cases settlement is made on the assay value of the ore with standard deductions. Most small amalgamation and gravity-concentration custom plants turn over to the shipper the amalgam and concentrate from each lot after the milling charge is paid.

Cost of Smelting Concentrates

Where milling ores are purchased outright, allowance must be made for the cost of smelting flotation or table concentrates. The cost to be charged against each ton of a lot as received will depend upon the ratio of concentration, transportation, and smelter rate; it will average about $0.25 per ton of gold ore milled. Special schedules are issued by some smelters for treating concentrates; usually, however, the same tariffs apply as for crude ore.

11 Gardner, E. D., Gold and Silver Custom Plants: Inf. Circ. 6842, Bureau of Mines, May 1935, 4 pp.

Summary of Charges

As previously indicated, the charges against any lot of ore purchased by a sampling plant depend upon a variety of factors. The return for each lot must be figured separately.

As an example, the indicated direct charges and indirect deductions under average conditions at a sampling plant purchasing 32 tons per day of one type of $12 gold ore, which would be milled in a nearby 50-ton flotation or cyanide custom plant, would be:

	Milling method	
	Flotation	Cyanidation
Direct charges, per ton:		
Sampling charge	$1.35	$1.35
Trucking	.25	.25
Milling	3.00	4.00
Transportation and smelting concentrate	.25	....
Total	4.85	5.60
Indirect deductions, per ton:		
Milling loss	a/1.56	b/.96
Smelter deductions on gold in concentrate	c/.84	....
Total	2.40	.96
Grand total	7.25	6.56

a/ 13 percent of $12.
b/ 8 percent of $12.
c/ 8 percent of $12 minus 1.56.

The net return to the shipper of a flotation ore would be $4.75 ($12 minus 7.25) per ton and of a cyaniding ore $5.44 ($12 minus 6.56) per ton.

The average freight and smelting charges on $12 gold ore in western districts would be about $9 per ton, which would leave only $3 per ton for mining, trucking, and royalty charges. This grade of ore could be shipped profitably to a smelter only when concessions are given in the smelter tariffs, or when the freight rate is relatively low, or both.

The gold ore shipped to smelters under present conditions has an estimated average value of about $30 per ton. The smelting and freight charges on this class of ore from most western camps average about $11.50 per ton, including a 5-percent deduction in the price of the gold.

If purchased in small lots by the proposed sampling plants, the charges to the shipper would be:

Sampling	$1.35
Smelting and freight	11.50
	12.85

The net return to the shipper would be $17.15 per ton.

If the same ore is milled in a 50-ton plant, the charges to the shipper would be:

Charges	Milling method	
	Flotation	Cyanidation
Sampling	$1.35	$1.35
Trucking	.25	.25
Milling	3.00	4.00
Mill losses	3.90	2.40
Transportation and smelting concentrate	.40	0
Smelter deduction on gold (5 percent)	1.50	0
Total	10.40	8.00

The net return to the shipper of a flotation ore would be $19.60 and to the shipper of a cyanide ore $22.00 per ton. The shipper would get a higher return in both cases, under average conditions, if the ore were milled.

DESIGN OF PLANT

The authors were assigned the task of drawing standard plans for sampling plants that were to be built throughout the West under a wide range of natural conditions. The plans herein presented are for a plant on a level site, which is the most expensive from a construction viewpoint. The standard design used, however, could be adapted readily to slopes of various degrees.

The general layout of the sampling works is shown in figure 1.[12] The site shown is 330 by 330 feet and contains 2-1/2 acres. The arrangement of structures could be altered to fit plots of other shapes or sizes.

Provision is made in the standard plans for receiving the ore by trucks only. Any shipments that might be received by rail would have to be transferred to trucks for hauling to the receiving bins. As shown in figure 1, the purchased ore, after sampling, would be shipped by rail. Some of the contemplated plants, however, would not have rail connections; in such cases it was expected that smelting ore would be trucked to a railroad or a smelter. Mill ore would be trucked to local custom milling plants or stock-piled for future treatment.

It was expected that ore would be received in 1- to 20-ton lots. The ore would be weighed on a 9- by 22-foot, 15-ton platform scale and then dumped into the receiving bins.

The flow sheet of the sampling plant is shown in figure 2.

Approaches to Receiving Bins

Figure 3 is a detailed drawing of the approaches to the receiving bins. The ramps can be built in curved form if the site favors that construction. The radius of curvature of the part of the ramp carried on bents should be not less than 150 feet; the outside stringer on the outside of the curve should not have a span greater than 13 feet. The earthen part of the ramp, as shown in figure 3, may have a radius of curvature of 50 feet. If the works are built on a sloping site and the length of the incline is reduced by using a curved ramp, it is estimated that the economic point to which the earth fill may be carried is 8 feet.

12 The full-size tracings of the figures used in this paper are on file at the Southwest Experiment Station of the U.S. Bureau of Mines, where they are available for inspection.

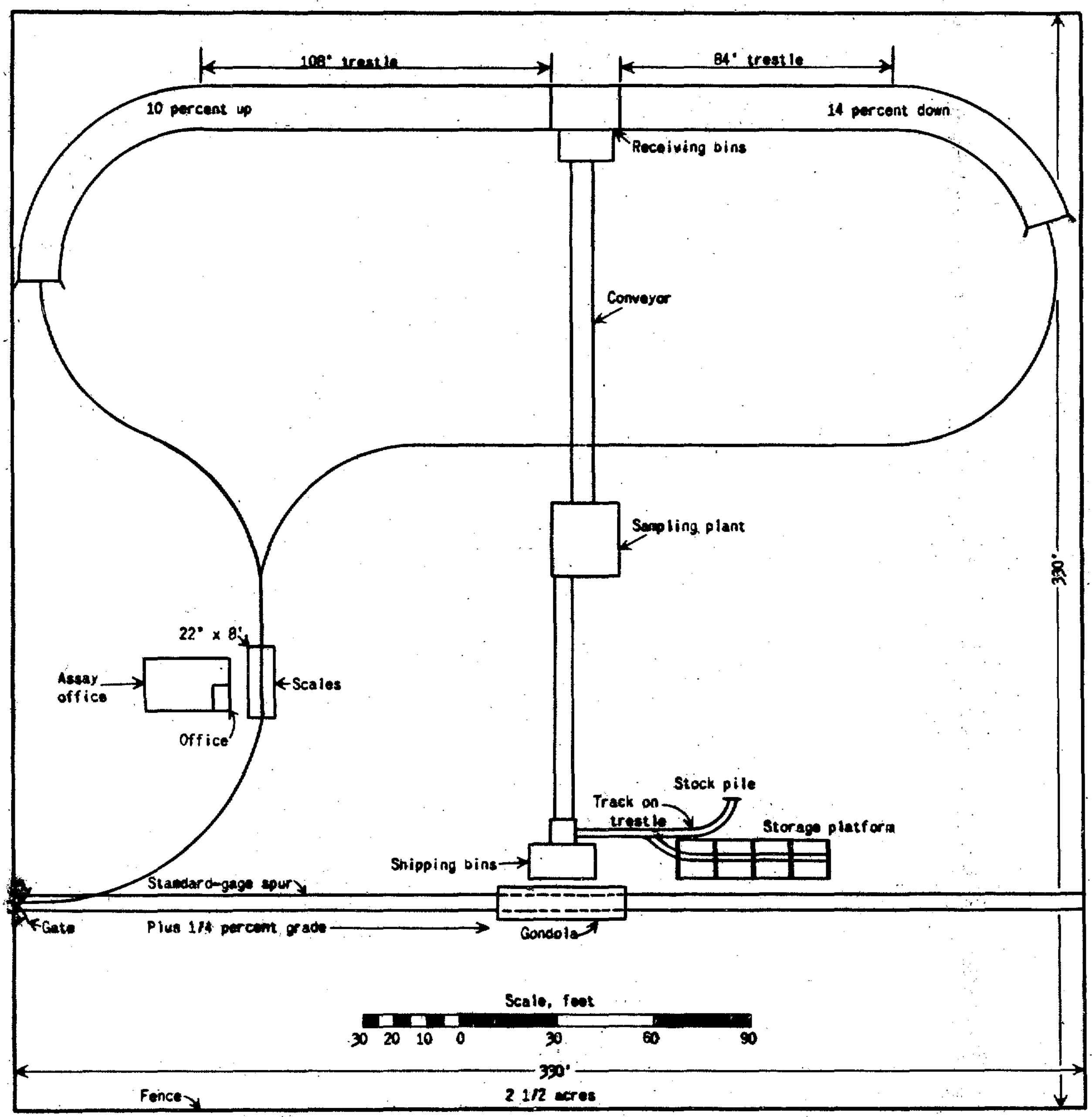


Figure 1.- General lay-out for sampling plant.

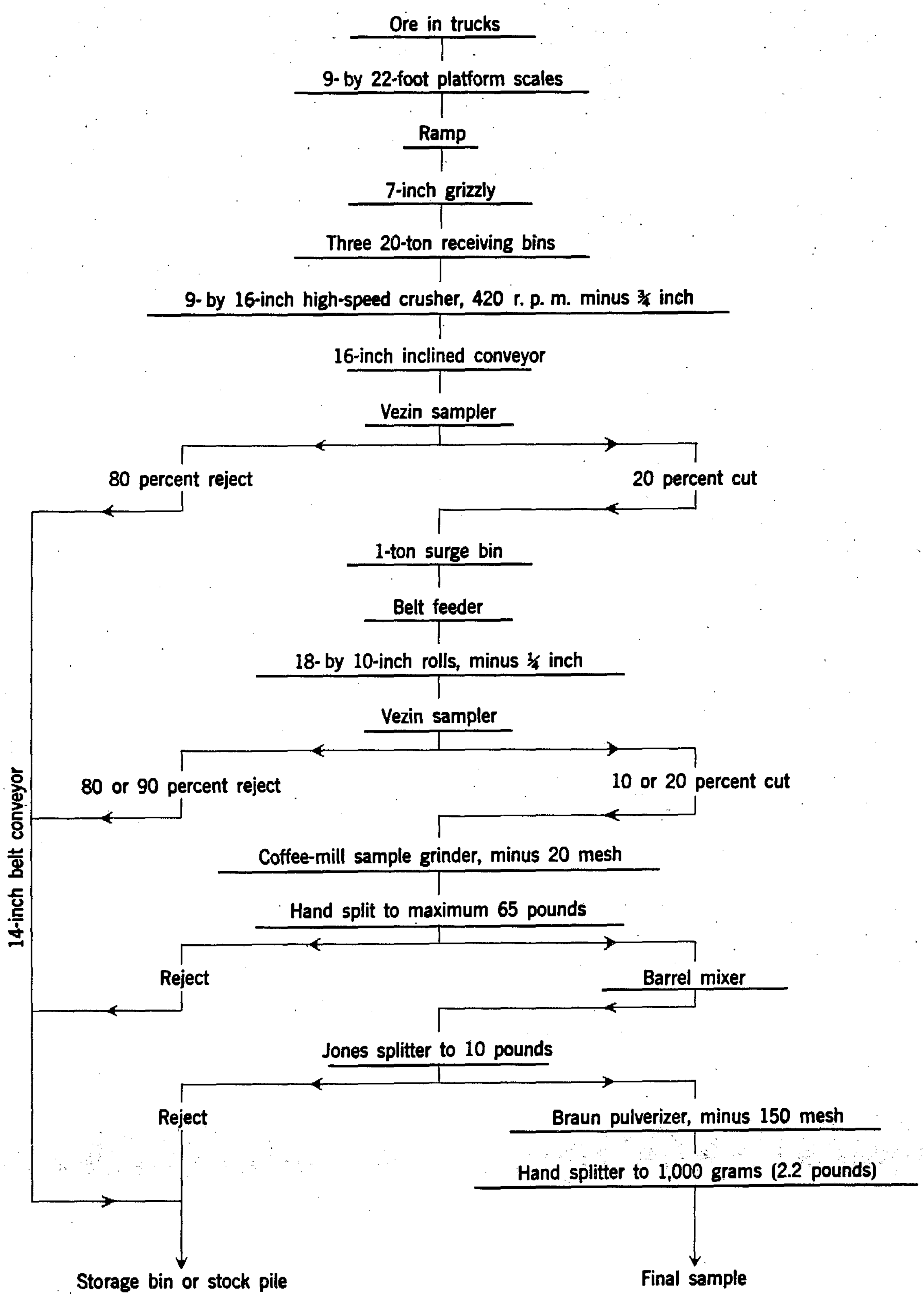


Figure 2.—Flow sheet of proposed Government sampling plant.

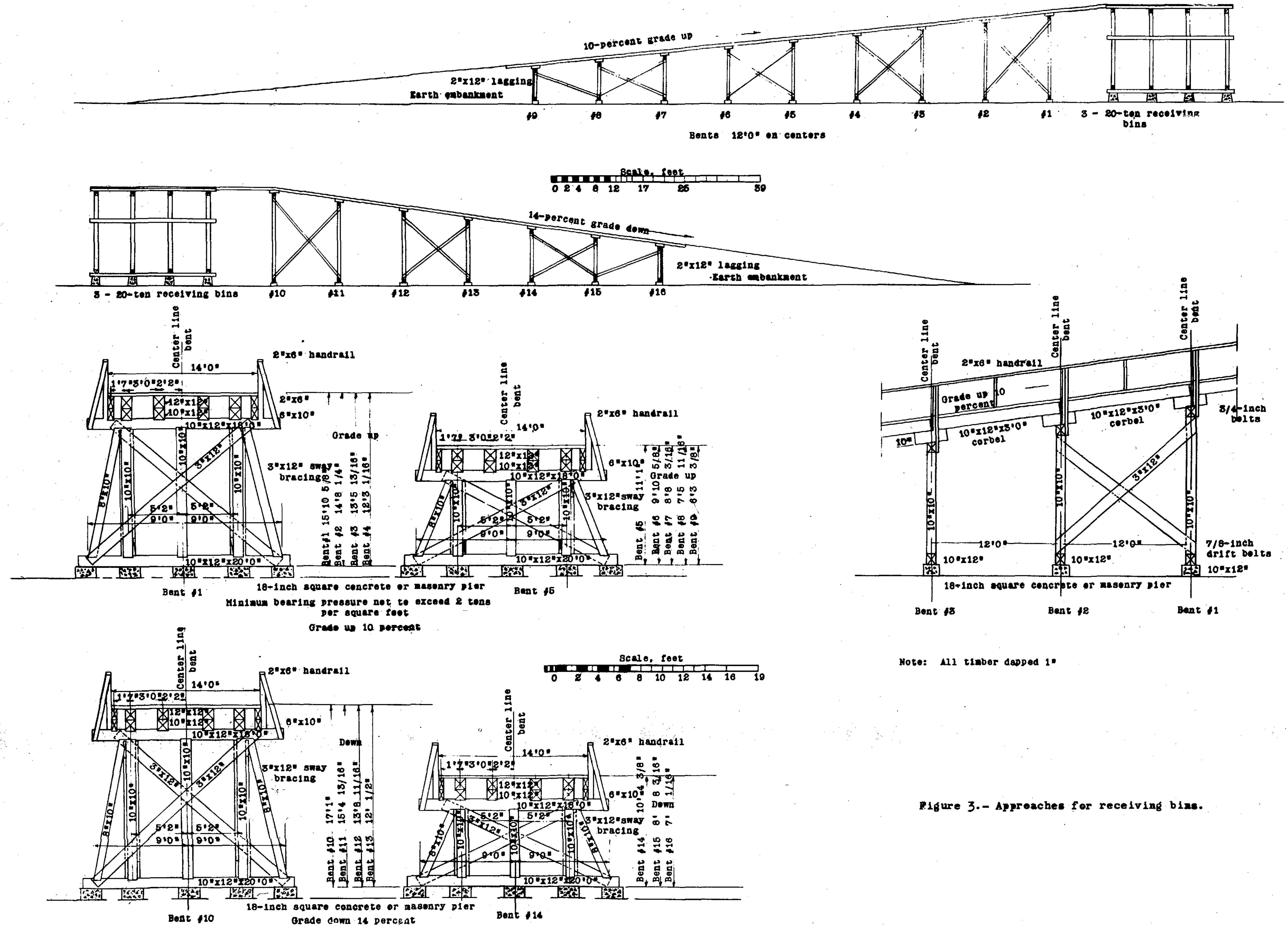


Figure 3.- Approaches for receiving bins.

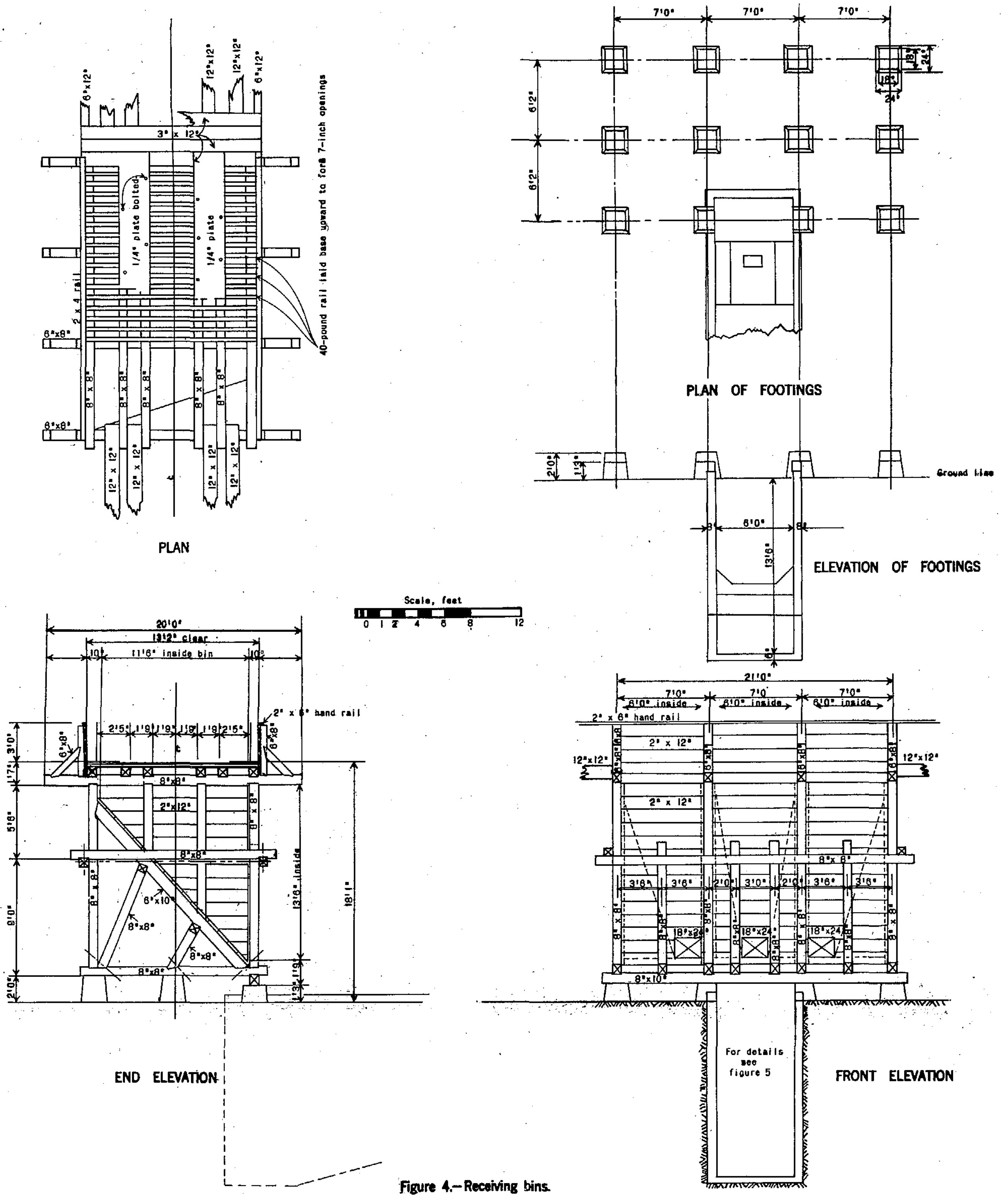


Figure 4.—Receiving bins.

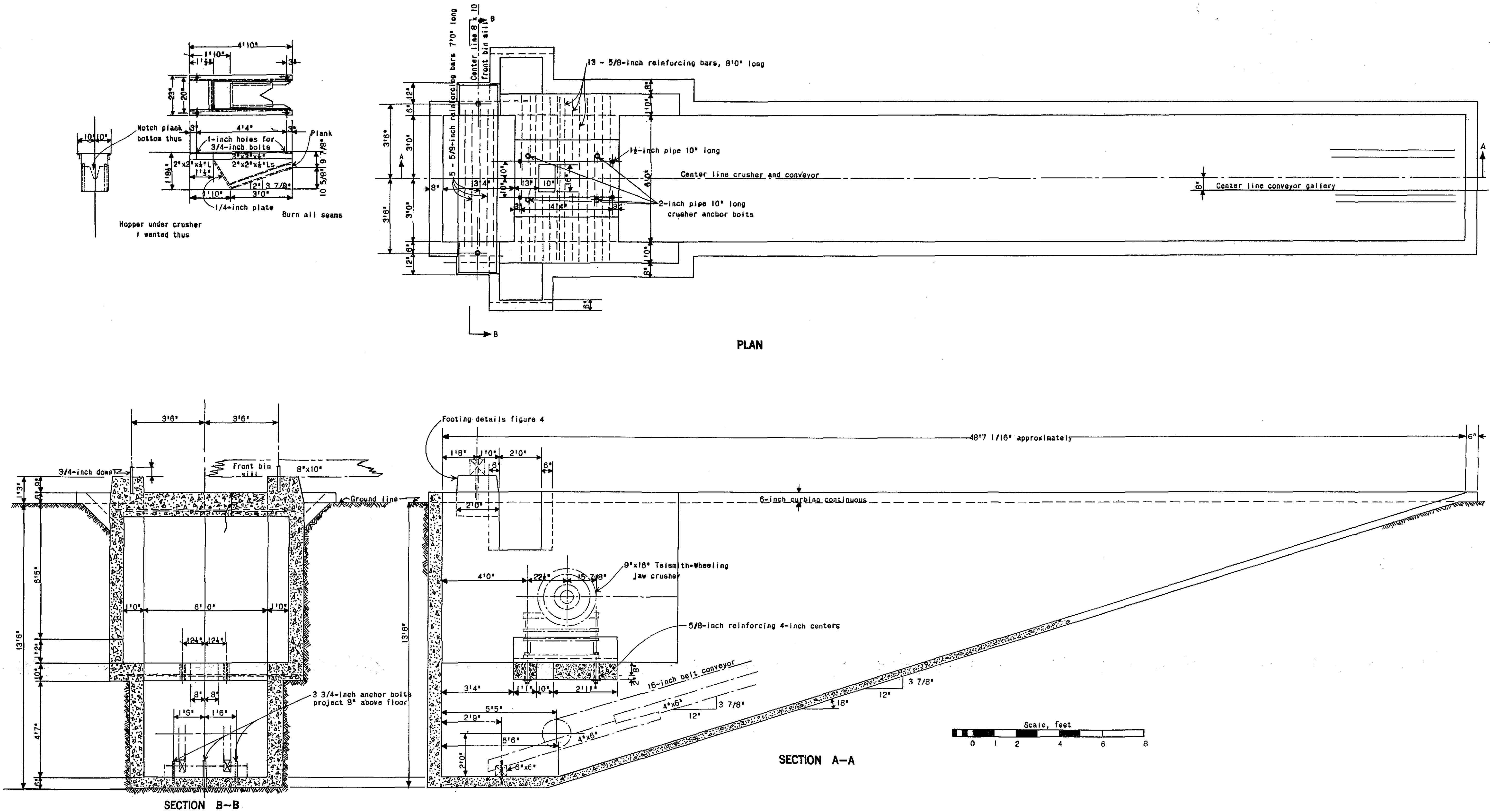


Figure 5.—Crusher and crusher pit.

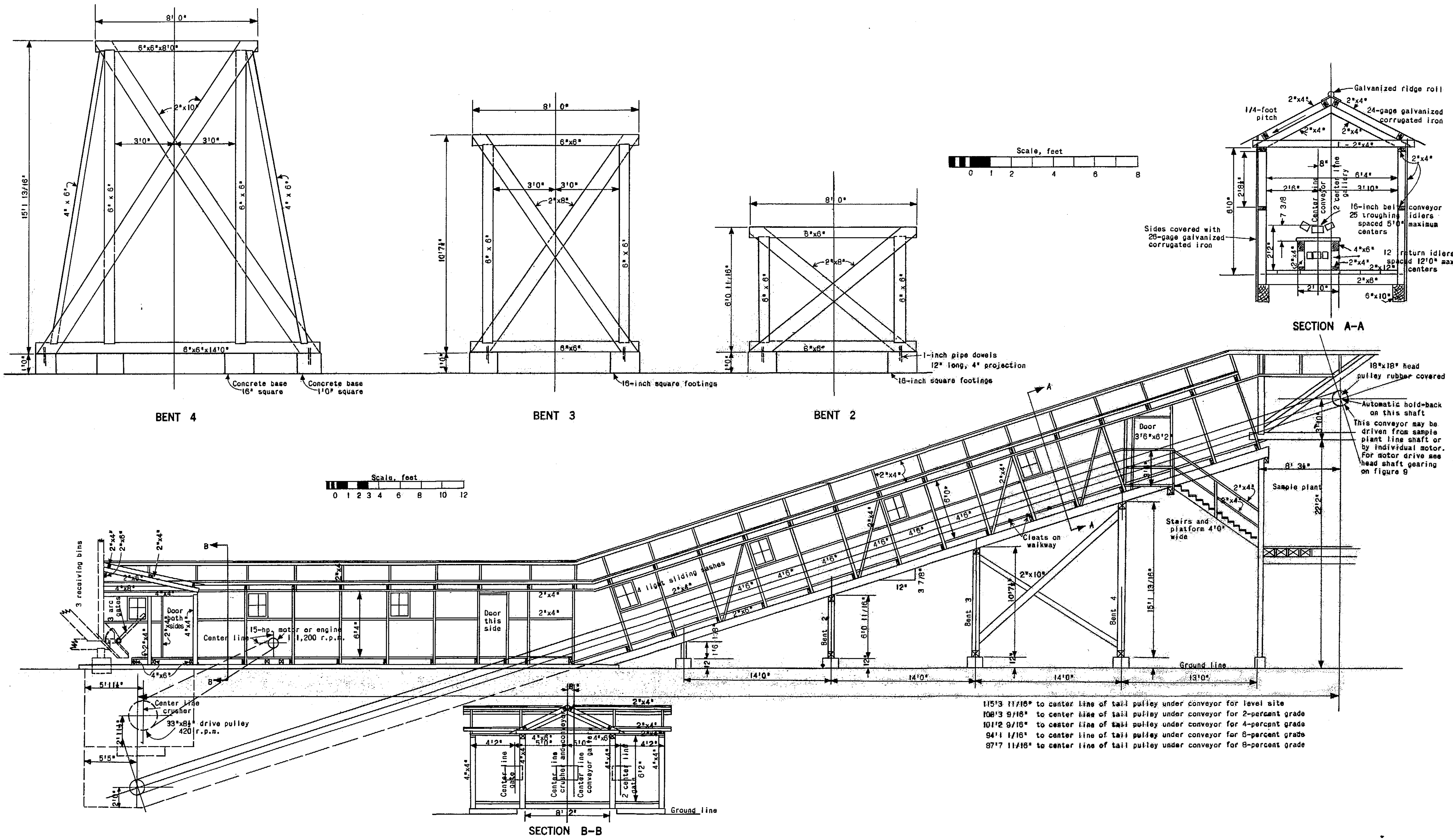


Figure 6.—Conveyor gallery from crusher to sampling mill.

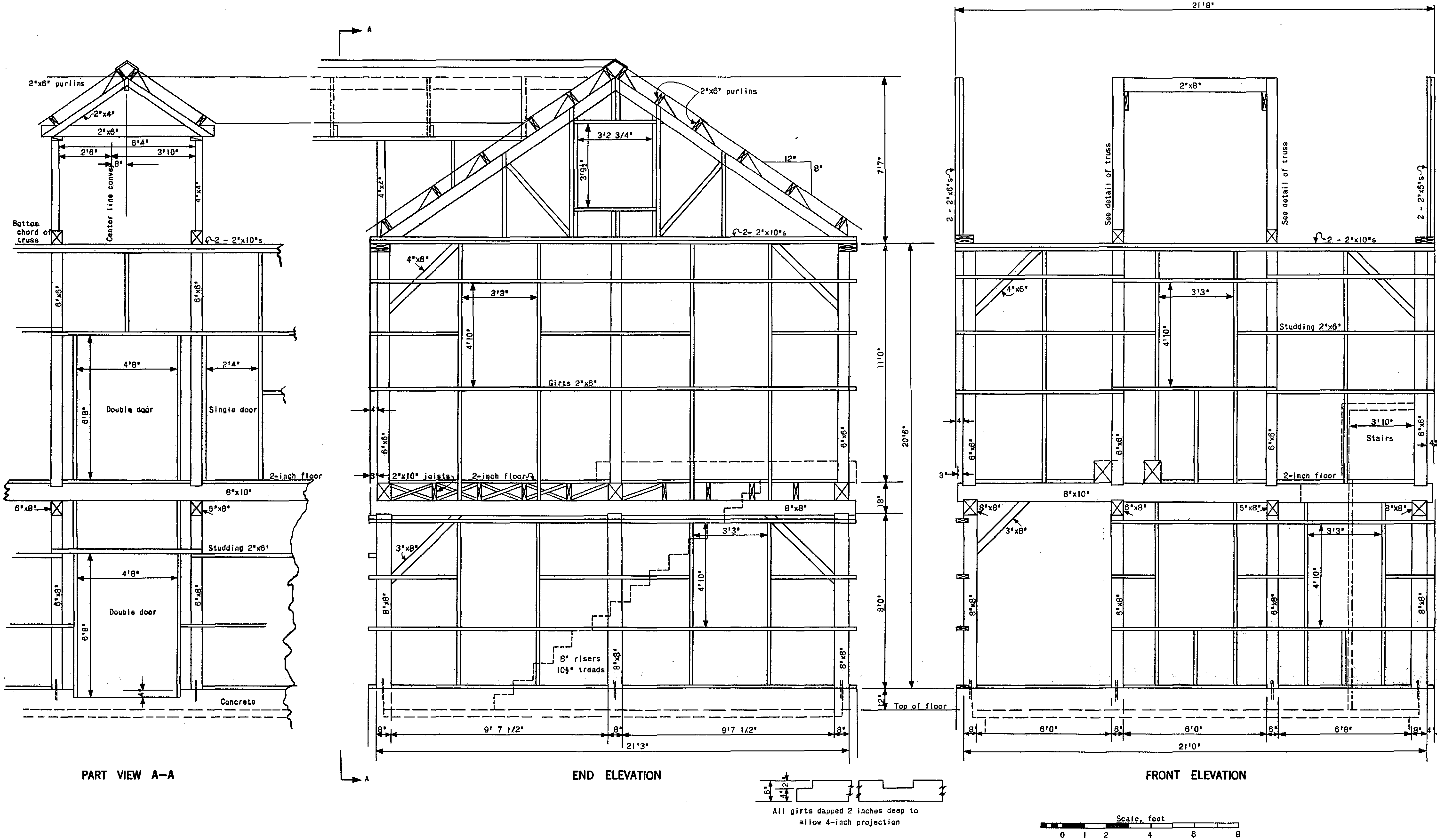


Figure 7.—Building details of sampling mill.

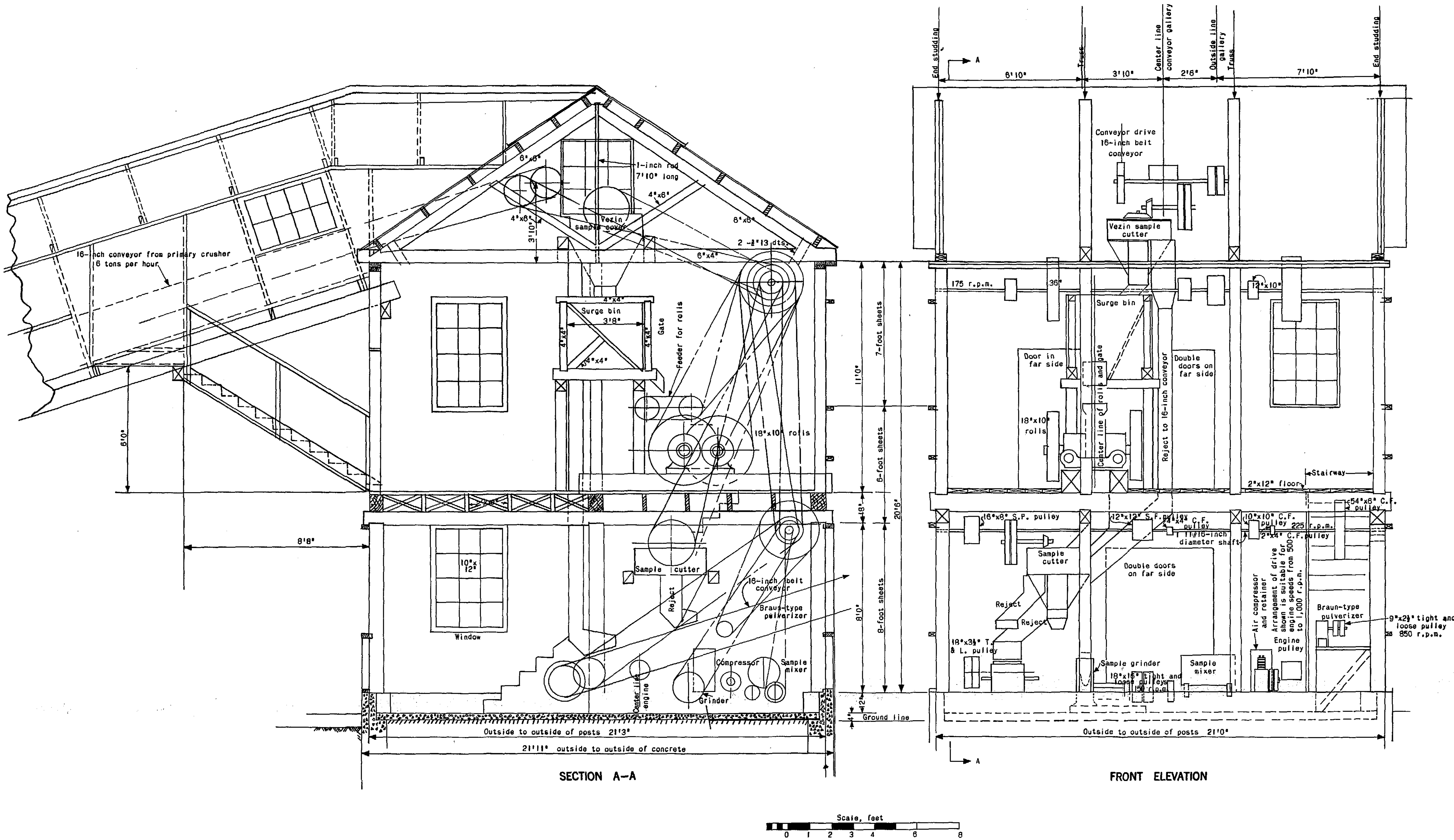


Figure 8.—General arrangement of sampling mill.

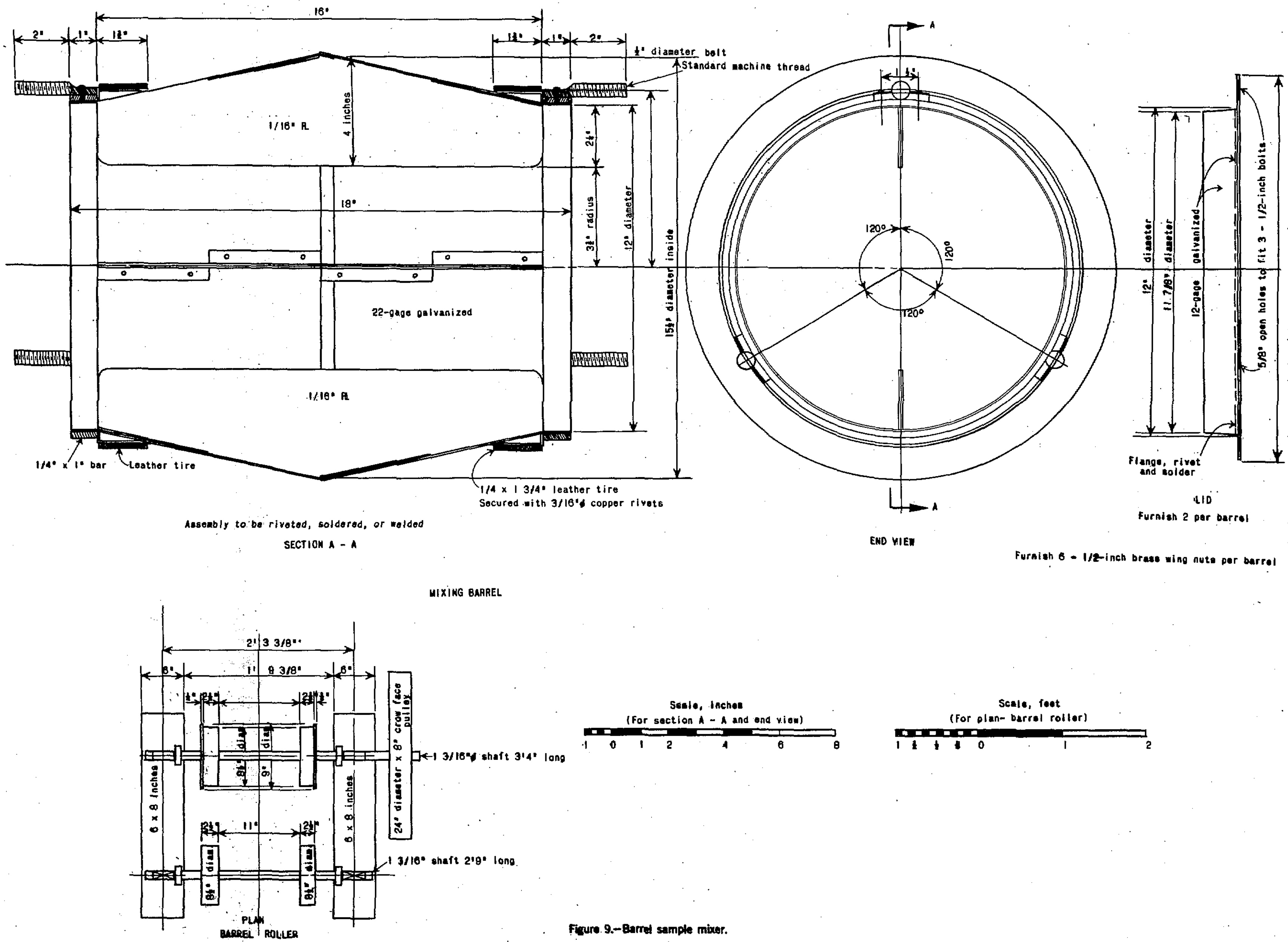


Figure 9.--Barrel sample mixer.

Receiving Bins

As it was expected that shipments would be received from a number of properties, provision was made for receiving ore from three shippers at a time. Three pockets with a capacity of 20 tons each are provided. All of the pockets could be used for the same lot, but few lots of over 20 tons were expected. The bins are self-cleaning. The details of the receiving bins are shown in figure 4.

A considerable saving in cost of construction could be made by eliminating the receiving bins and their approaches and substituting a small pocket or sloping platform at the coarse crusher. In this latter case a truck would be backed up a short earth ramp and its load dumped at the crusher. Ore from only one shipper could be received at one time.

Crusher

To save height in the receiving bins, the crusher was placed in a pit, as shown by figure 5. The gates of the receiving bins are so arranged that the ore can be fed directly into the crusher. Arc gates with 3-foot, 3-inch handles control the flow of ore. The operator at the crusher can stand on a platform at the crusher and manipulate the gates.

A high-speed, 9- by 16-inch crusher was chosen for the plant. It has a special toggle with two parts held together by rivets, which would shear if any tramp iron should get into the machine. This crusher can take any pieces of ore that pass through the 7-inch grizzly on top of the receiving bin and make a product that can be handled through a set of 18-inch rolls. The high ratio of reduction permits the omission of one crushing stage in the flow sheet of the plant.

According to the manufacturer's rating, the crusher has a capacity of 4 to 5 tons per hour, with a 5/8-inch discharge opening; with a 1/2-inch opening it is rated at 3 to 4 tons per hour, and at 5 to 6 tons per hour with a 3/4-inch discharge.

For the purpose of this paper it is assumed that the machine would have a capacity of 4 tons per hour with a setting that will pass particles with a maximum size of 3/4 inch, and will use a maximum of 15 hp.

The flow sheet of the mill was arranged for fairly high-grade ore. Should a larger hourly capacity be desired, the crusher and rolls could be opened to give a coarser product that would still be within the limits prescribed for accurate sampling of low- or medium-grade ores.

Conveyor to Sampling Mill

The conveyor and the conveyor gallery, with details of construction, are shown in figure 6. The housing on the crusher also is shown in that figure.

Sampling Mill

The details of the building to house the sampling mill are shown in figure 7. The general arrangement of the equipment in the mill, designed for a line-shaft drive, is shown in figure 8. This is discussed further under "power."

All equipment is standard and may be purchased from dealers of mining machinery, except the sample mixer, which would have to be made to order. The details of this mixer are given in figure 9; it is copied from one in use in the sampling plant of a southwestern custom smelter.

The ore to be sampled comes into the top of the plant by the conveyor and passes through a Vezin sampler. A 20-percent cut taken by the sampler drops into a 1-ton surge bin. The reject drops onto another belt conveyor, which takes it to a storage bin. From the bin the ore is fed by a belt feeder into 18- by 10-inch rolls, which are set 1/4 inch apart, and thence to another Vezin sampler. This sampler is so arranged that a 10- or 20-percent cut may be taken. As before, the reject goes to the conveyor. The cut from the second sampler goes to a sample grinder, which reduces it to minus 20 mesh.

The product from the sample grinder for each lot is accumulated and then cut down by hand on a large Jones splitter to a maximum of 65 pounds. This is mixed thoroughly in the barrel mixer and then cut down to 10 pounds on another Jones splitter. Should lots of 20 tons or over be run regularly, a mechanical sampler would be desirable after the grinder. The mixed sample is dried and then put through a laboratory pulverizer, which grinds it to 150 mesh. The ground pulp is then mixed by hand and split to 1,000 grams (2.2 pounds). The final product is used for assaying.

A moisture determination of the ore would be made at the time the sample is dried. This would be checked against the moisture determination made of the crude ore as received.

Conveyor to Storage Bins

The conveyor for taking away the rejects from the plant, and the conveyor gallery with construction details are shown in figure 10. The drawing shows the drive for the conveyor at the upper end, which is standard practice. Although not standard practice, the conveyor could be driven from the line shaft in the sampling plant; in this case the drive mechanism shown in figure 10 would be placed on the lower pulley of the conveyor in the sampling plant.

Should the works be built on sloping ground, there would be a saving in the cost of the plant, as the length of the two conveyors and the conveyor housings would be less. The reduction in length of the conveyors for slope of 2° to 8° is shown in figure 6.

Storage Bins

Provision is made in the design of the plant for handling several types of ores. Figure 11 is a detailed drawing of the proposed storage bins.

The ore could be deflected into any of the three bins by a gate arrangement at the end of the conveyor. The main bin (A) holds a carload (50 tons) and would be used for the principal type of shipping ore handled. Irregular lots of shipping ore would be dumped into bin C and thence trammed by hand to the platform shown in figure 1. The platform is large enough so that 50-ton lots of two different types of ore may be accumulated at the same time.

Mill ore that might be accumulated for future disposal would be trammed from bin C to a dump, as shown in figure 1. Should two classes of mill ore be received, a second stock pile could be used. As the tramming tracks are on a trestle, no shoveling would be required in stockpiling ore either on the platform or in the dumps. Ore from bin C could also be loaded into trucks.

Bin B would be needed only when the site had sufficient slope so that stock piles could be built up without shoveling. This bin could be omitted at most sites, with a saving ($55) in the cost of the structure.

POWER

The determination of the most satisfactory kind of power to be used in small milling plants usually is a difficult problem, and several factors must be balanced against each other.

Three alternate methods of driving the sampling plants were considered: (1) By purchased power, (2) by Diesel-electric power, and (3) by gasoline engines.

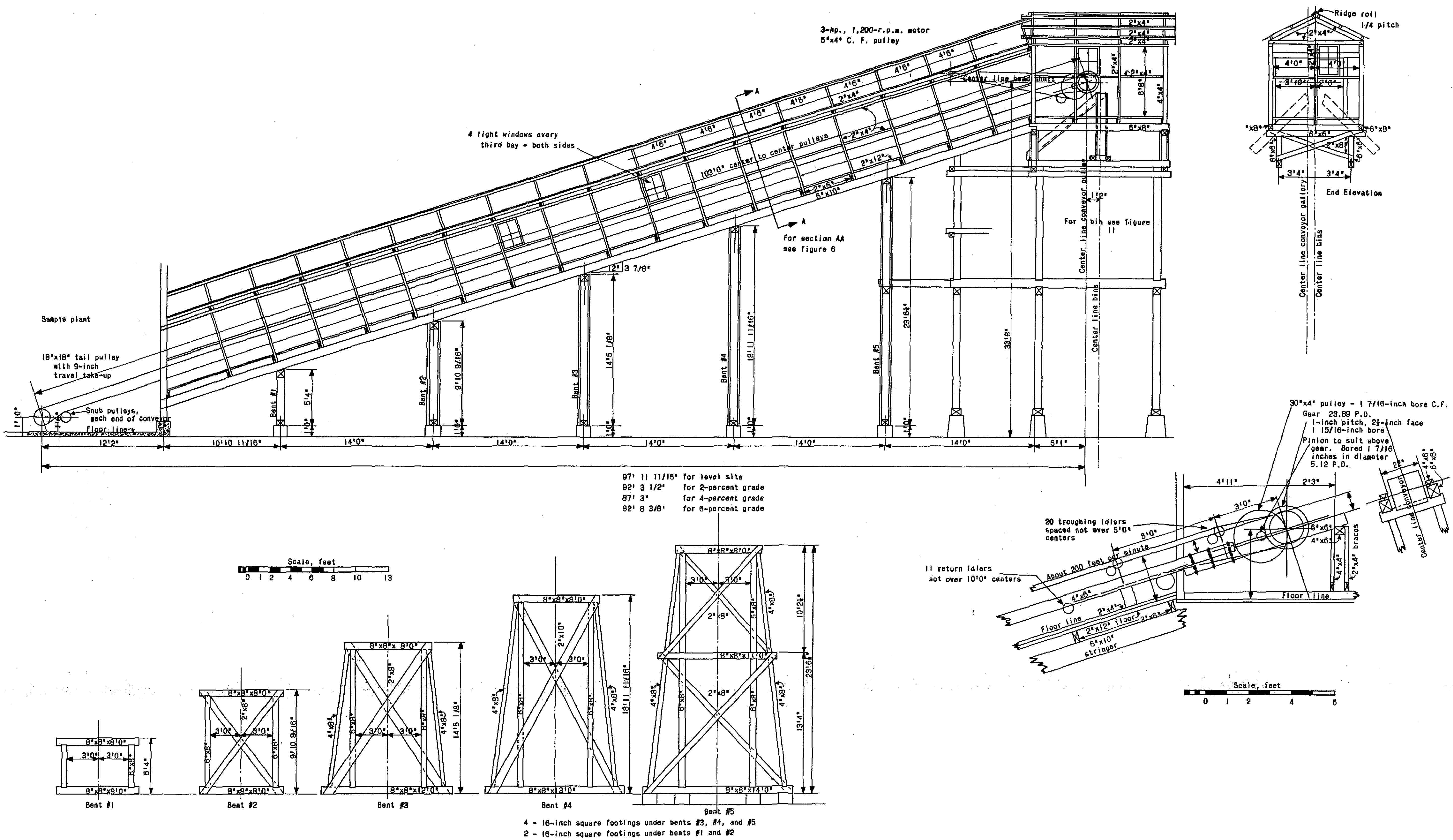


Figure 10.—Conveyor gallery from sampling plant to storage bins.

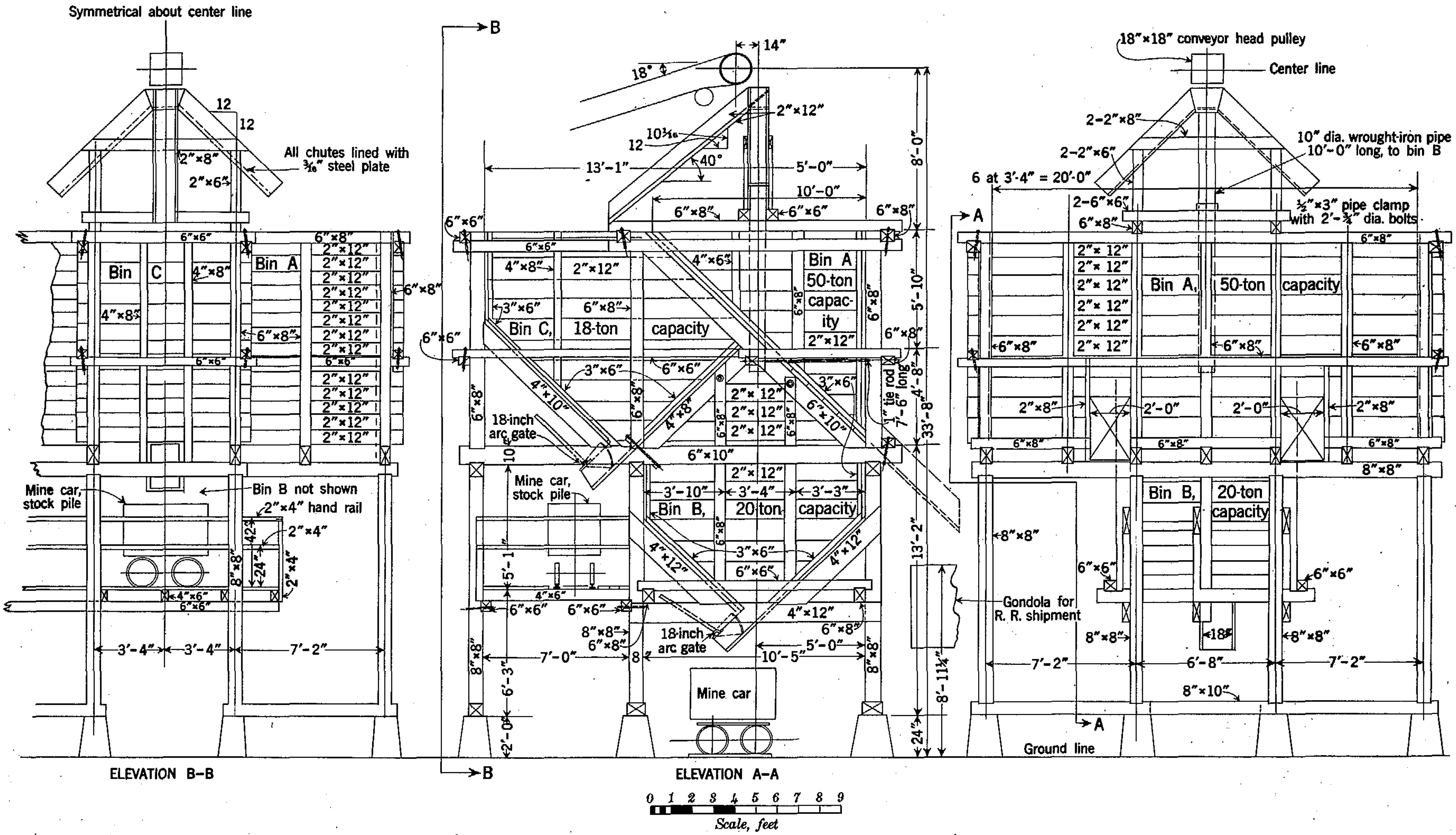


Figure 11.—Storage bins.

In most active mining districts electric power may be purchased from public utility companies, but usually this would not be the case in the districts in which the proposed sampling plants would be installed. The length of power lines required from the nearest source of electric power, together with the installation costs, usually make this form of energy uneconomical for users of relatively small quantities of energy.

Purchased electric power is usually more satisfactory than other forms of energy for small plants. Time lost on account of break-downs or failure of the power plants increases operating costs at most small milling plants. Although no trouble is experienced with Diesel or gasoline engines at some small plants, this is not usually the case.

In the contemplated plants, it was planned to use purchased electric power if available within a reasonable distance. As shown by figure 8, estimates were made for a line-shaft drive in the sampling-mill building.

The size and cost of the motors required would be:

15-hp. motor with accessories for driving crusher	$181.57
40-hp. motor with accessories for driving plant	440.86
5-hp. motor with accessories for driving discharge conveyor	256.74
Total	879.17

In addition to the motors, the following electrical equipment would be required:

3 conventional-type distributing transformers, 15-kv.-a., 1-phase, 60-cycle, 2,200-volt primary with two 5 percent tops on primary side to 220/110-volt secondary. The cost, complete with hanger irons and oil and freight allowed to designation for three, would be	$479.88
6 plug-type fuse cut-outs at $2.00 each	12.00
1 2-pole transformer station with cross arms and platform	45.00
Distribution poles in yard	75.00
Wiring to crusher and conveyor motor	40.00
	651.88

The line shaft and belting would cost about $445. The total for this method of drive would be:

Motors	$879.17
Transformers and poles	651.88
Line shafts and belting	445.00
Total	1,976.05

Individual motor drives for each piece of machinery in sampling plants are desirable whenever practicable, as line shafts with belt drives are a hazard to the workmen. Moreover, individual drives occasion less lost time and simplify the operation of the plant. The size and cost of motors for each machine with individual drives are shown in table 2.

Manufacturers of electric power equipment recommend squirrel-cage induction motors with constant speed (1,750 r.p.m.), 3 phase, 60 cycle, and 220 volt for all motors shown in table 2, except those for the conveyors. For the conveyors, gear motors equipped with general-purpose ball-bearing motors, with an output speed of 41 r.p.m., are recommended. The cost for this method would be:

Motors	$1,239.03
Transformers and poles	651.88
Extra wiring	100.00
Total	1,990.91

TABLE 2.- Horsepower, r.p.m., and costs of motors for individual drives

	Horsepower	Size of pulley, inches	Includes	Cost[1]
Crusher	15	8 by 6-3/4	Line starter, $28.56; safety switch, $11.05.	$181.57
Rolls	10	6 by 3-1/2	Line starter, $28.56; safety switch, $6.50.	150.98
Conveyors (2)	5		2 line starters at $28.56 each; 2 safety switches at $3.90 each.	513.48
Feeder	1	3 by 3	Line starter, $15.96; safety switch, $3.90.	59.34
Samplers (2)	1	3 by 3	2 line starters at $15.96 each; 2 safety switches at $3.90 each.	118.68
Grinder	3	4 by 3-1/2	Line starter, $15.96; safety switch, $3.90.	80.34
Pulverizer	2	4 by 3-1/2	Line starter, $15.96; safety switch, $3.90.	75.30
Air compressor.	1	3 by 3	Line starter, $15.96; safety switch, $3.90.	59.34
Total	44			1.239.03

[1]As of August 15, 1935, based on discounts applicable to the Federal Government.

The first cost of driving the plant by electric power with line shafts or by individual motors is about the same.

Although not standard practice, the conveyor to the storage bin could be operated from the lower end from the line shaft. Should this be done, a saving of $257 for the geared motor could be made with a line-shaft drive.

Should electric power not be available for purchase it could be generated on the ground, or the plant could be driven directly by Diesel or gasoline engines. The layout of the sampling works, as shown in figure 1, is such that one prime motor could not run the entire plant by belts without a long belt transmission line from the main building to the crusher. Apparently there would be a choice between one Diesel electric plant or two gasoline engines.

Diesel engines have the advantage of producing power at lower cost than gasoline engines, but the first cost is considerably greater per horsepower. For a plant with a long life ahead of it, a Diesel plant would probably be more economical than gasoline engines.

High-speed Diesel plants cost considerably less than low-speed machines but produce more energy per pound of fuel. Figures supplied by Fairbanks, Morse & Co. for a 60-hp. Diesel plant show that the fuel consumption of a low-speed plant would be 0.48 pound of fuel oil per brake horsepower-hour at full load as against 0.38 pound for a high-speed plant.

Fuel consumption of gasoline engines is 0.67 pound per brake horsepower-hour. Moreover, gasoline engines require 25 percent more lubricating oil than Diesels.

If a Diesel-electric drive were used, a 25-percent loss of power in the generator and motors should be considered. The total motor horsepower, as shown by table 2, is 44; the operating load would be about 35 horsepower. Table 3 contains quoted costs and other data for engines that could be used in the sampling plants.

If Diesel-electric power were decided upon, the best of the engines quoted would appear to be Model 36 with a generator, which costs $3,633 delivered at the nearest main-line railroad point. A similar assembly with a lower-speed engine, shown in the table, would cost $4,570.

TABLE 3.- Data on Diesel and gasoline engines suitable for sampling plants

Kind	Model	Horsepower		Number of cylinders	R.p.m.	Weight, pounds	Cost, f.o.b. factory	Freight at $4.50 a hundred pounds	Cost delivered
		Maximum	Sustained						
Diesel......	a/36	60	48	6	1,200	3,200	$2,340	$144	$2,484
Do.	a/, b/36	60	48	6	1,200	6,380	3,346	287	3,633
Do.	B-2	75	40	2	514	....	....	....	3,230
Do.	D7700	63	48	4	600	6,220	2,487	280	2,767
Do.	c/D7700	63	48	4	600	9,200	4,156	414	4,570
Gasoline..	d/Ford	....	....	4	625	....	....	....	205
Do.	d/Chrysler	....	....	....	....	1,240	562	56	618
Do.	1XA	e/36.5	25.5	4	1,800	....	302	....	....
Do.	f/FM3	22.3	17.8	....	1,800	585	370	26	396
Do.	f/FM4	28.3	22.8	....	1,800	825	430	37	467
Do.	f/FM5	37.8	33.6	....	1,800	1,000	475	45	520
Do.	f/G4A	g/29.2	24.8	....	1,000	2,265	h/825	102	927
Do.	f/, i/G4A	g/29.2	24.8	....	1,000	2,265	h/975	102	1,077

a/ Complete with radiator and starting battery.
b/ With 39.6-kw., 3-phase, 60-cycle, 240-volt alternator and excitor.
c/ With generator.
d/ Complete, but without mounting.
e/ At 2,600 r.p.m.
f/ Complete.
g/ At 1,200 r.p.m.
h/ Electric starting unit $122 extra.
i/ Converted to burn tops or Diesel oil.

Should two gasoline engines be used, FM5 at $520 for the main plant and FM3 at $396 for the crusher would appear to serve the purpose. The total cost would be $916.

Two 4-cylinder Ford engines apparently would supply sufficient power at lower first cost, but this equipment may not now be obtainable.

As shown by table 3, Model G4A gasoline engine can be converted to burn tops or fuel oil at an additional cost of $150, or $1,077 for the engine. Two of these engines would cost $2,154.

Assuming a sustained load of 35 horsepower, 25-percent loss in the generator and motors, and 15 horsepower-hours per gallon of Diesel oil (or 0.48 pound of fuel per hp. hr.), the consumption of fuel oil would be 3 gallons per hour for the Diesel-electric plant. Assuming 9 horsepower-hours per gallon of gasoline (or 0.67 pound of fuel per hp. hr.) for the two gasoline engines, the consumption of fuel would be 3.9 gallons per hour.

The fuel cost for the Diesel-electric plant would be three times $0.075 (average cost in western mining camps in 1935), or $0.23 per hour, or $1.80 per 8-hour shift. The gasoline would cost 3.9 times $0.13 (third-grade, exclusive of State tax), or $0.51 per hour, or $4.08 per 8-hour shift. A saving in fuel of $2.28 per day, or $0.07 per ton, would be indicated by using a Diesel plant. As the Diesel fuel could not be used in automobiles, the saving probably would be greater. The difference in fuel for 300 days' operation would be $684.00.

Assuming 11 horsepower-hours per gallon of fuel oil for the converted gasoline engines, the daily cost would be $1.91. Apparently two Model G4A's, or similar engines, would be the most economical to use, considering the first cost and power loss with the Diesel-electric set-up.

As running the plant only one shift per day was contemplated, lights would not be required in the plant. Should a night shift be worked, a 5-horsepower generator would have to be added to the set-up for gasoline-engine drive and consideration given to this additional power load.

A charge for labor must be made against gasoline or Diesel power; in this case, however, another operator would not be required. The chief operator could look after the power plant along with his other duties.

ASSAY OFFICE

As it would be expected that settlements for shipments would be made on the assays made at the sampling works, an adequate assay office would be required. Provision was made for both fire assaying and wet analytical work.

A plan of the proposed assay office is shown in figure 12. The building was to be of frame construction with a corrugated sheet-iron roof. The bill of material for the assay office is shown in table 4. The equipment and supplies for the assay office are shown in table 5. This is compiled mainly from a list suggested by the Denver Fire Clay Co. Prices are from the 1934 catalog, except balances, which are 1935 quotations. Table 6 is a summary of tables 4 and 5.

TABLE 4.- Bill of material for assay office

Quantity	Item	Unit price	Amount
	Construction materials		
84 linear feet	4- by 6-inch mud sill	$40 per M	$6.72
8 do	12- by 12-inch timber for balance support	40 do	3.84
304 do	2- by 6-inch floor joists	30 do	9.12
304 do	2- by 6-inch ceiling joists	30 do	9.12
488 do	2- by 6-inch rafters	30 do	14.64
32 do	4- by 4-inch corner posts	40 do	1.72
944 do	2- by 4-inch studs and girts	30 do	18.90
416 board feet	1- by 10-inch subfloor	27.50 do	11.44
616 do	1- by 10-inch roof sheathing	27.50 do	16.94
188 do	1- by 6- 8- 10-inch trim	40 do	7.52
676 do	1- by 8-inch shiplap	35 do	23.66
460 do	1- by 6-inch tongue-and-groove flooring	45 do	20.70
100 do	1-inch dressed lumber for shelves, etc.	45 do	4.50
260 do	1- by 4- 6-inch lumber for window and door frames.[1]	45 do	11.70
412 linear feet	1- by 4-inch scantling	35 do	4.83

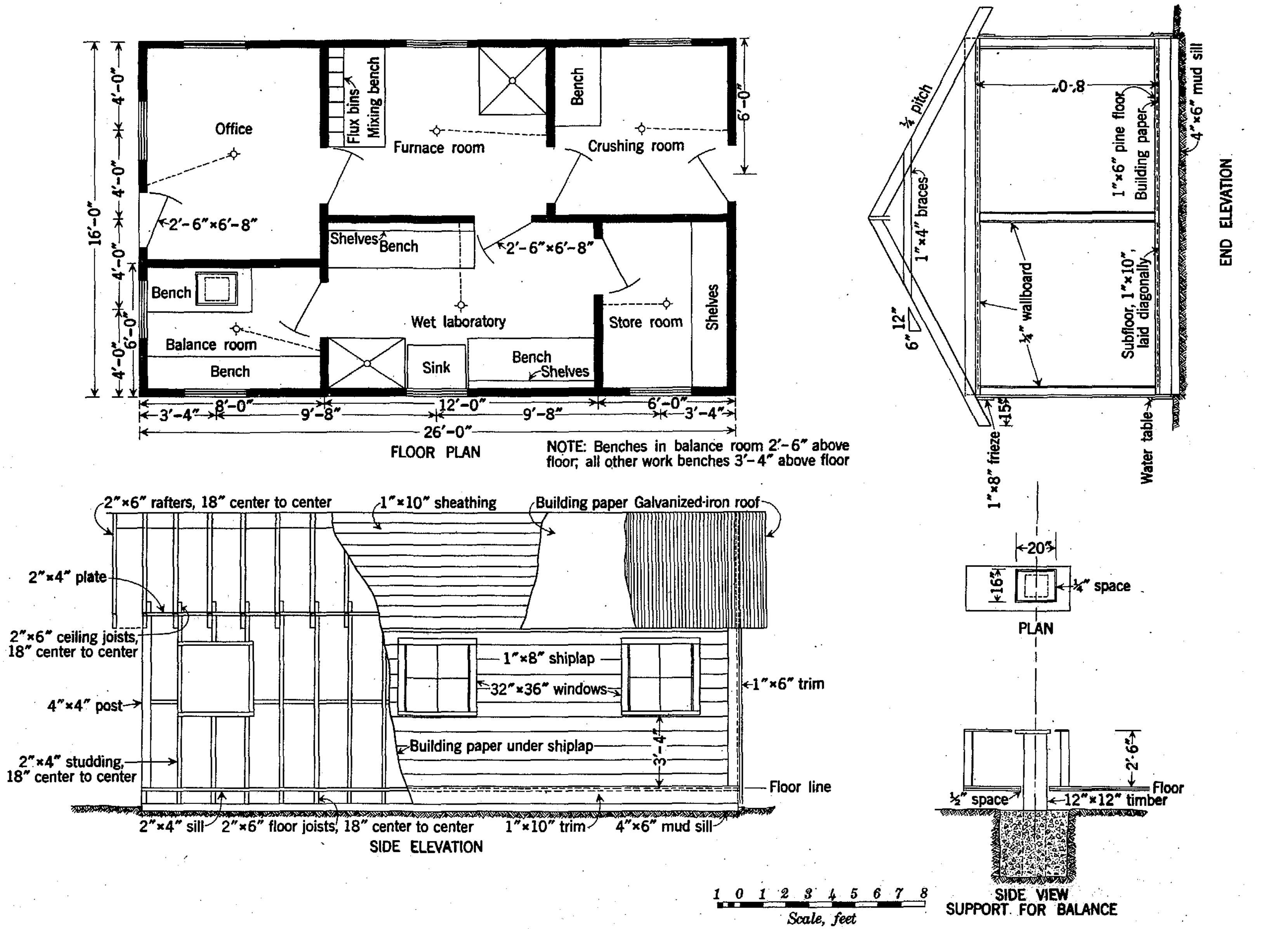


Figure 12.—Assay office.

TABLE 4.- Bill of material for assay office - Continued

Quantity	Item	Unit price	Amount
	Construction materials - Continued		
1,780 square feet	1/4-inch wall board	$50 per M	$89.00
48 sheets	G. I. roofing, 6-foot lengths	.72 per sheet	34.56
4 rolls	Building paper	1.35 per roll	5.40
8	Windows, 32- by 36-inch	4.00 each	32.00
7	Doors, 2-foot 6-inch by 6-foot 8-inch (5-panel)	4.14 do.	28.98
30 linear feet	Metal ridge roll	.10 per foot	3.00
200 pounds	Nails	.06 per pound	12.00
3 sacks	Cement	1.00 per sack	3.00
	Miscellaneous (molding, water table, etc.)		20.00
Total			393.29
	Hardware		
2 gallons	House paint	3.45 gallon	6.90
1	20- by 24-inch acid-resisting sink, flat rim	7.00 each	7.00
1	Acid-resisting trap to wall	9.00 do.	9.00
1	Faucet, 1/2-inch	1.15 do.	1.15
50 feet	1/2-inch water pipe[2]	.06 per foot	3.00
40 do.	2-inch soil pipe	.95 per 5 ft.	7.60
	Pipe fittings		4.50
2	Outside locks for doors	2.50 each	5.00
5	Inside locks for doors	1.25 do.	6.25
7 pairs	Door butts	.35 pair	2.45
20 do.	Steel shelf brackets	.30 do.	3.00
1	Heating stove (laundry)		14.00
1	Metal base for stove		1.00
12 lengths	Stovepipe, 5-inch diameter	.15 length	1.80
2	Elbows for stovepipe	.25 each	.50
1	Fume hood for laboratory		3.50
1	Hood for muffle furnace		3.50
2	Stovepipe roof vents	1.50 each	3.00
	Window catches, asbestos pipe covering, miscellaneous, etc.		15.00
Total			98.15
	Electrical supplies		
	Wire, switches, cleats, bulbs, etc.		20.00
	Furniture and fixtures		
1	Office desk (plain top)		25.00
2	Office chairs	3.00 each	6.00
1	Desk chair		5.00
4	Window shades	1.00 each	4.00

[1]Door and window frames to be made on the job.
[2]Laboratory sink to discharge into outside toilet. Toilet to be constructed of left-over material from assay office.

TABLE 4.- Bill of material for assay office - Continued

Quantity	Item	Unit price	Amount
	Furniture and fixtures - Continued		
1	Broom		$ 1.00
1	High stool for laboratory		1.50
1	Low stool for balance room		1.50
Total			44.00
Total materials			555.44
Labor for erection			300.00
Total cost of assay office			855.44

TABLE 5.- Equipment and supplies for assay office

FIRE ASSAYING

Balances and weights:	
DFC No. 384 Button, Heusser, sensitivity 1/500 mg	$250.00
Braun 12127 pulp, sensitivity 1/2 mg	50.00
DFC No. 462 scoop and platform, 25-pound, 240-pound platform	20.00
DFC No. 442 moisture balance	30.00
DFC No. 514 weights for Button balance	25.00
DFC No. 544 weights for assay-ton balance	6.50
1 rubber-sheeting scale cover	2.20
	383.70
Furnace:	
Braun No. 31050 furnace, type 40, gasoline	100.50
Miscellaneous equipment and supplies:	
Buckboard, 20 by 24 inches, and muller	18.00
Buckboard brush	2.25
Iron cupel tray	.90
1 dozen parting flasks	1.94
1 pair forceps	1.10
2 pairs forceps, slag, 6 inch	.60
1 pair asbestos mittens, 14 inch	2.50
1 slagging hammer, 15 ounce	2.00
1 slagging hammer, 9 ounce	1.60
1 box labels	.10
1 alcohol lamp, 4 ounce	.40
1 Dangler lamp	12.00
1 lead measure	.25
1 4-1/2 mallet	1.50
1 magnifier, 3 glass	1.75
1 magnet, 6 inch	.45
1 pouring mold	1.00
1 pair button pliers	1.25
1 blowpipe	.75
1/2 dozen bottles, 8 ounce	2.13
3 brushes	1.55

TABLE 5.- Equipment and supplies for assay office - Continued

FIRE ASSAYING - Continued

Miscellaneous equipment and supplies (Continued):	
1 button tray	$1.00
1 dozen charcoal sticks	.55
12 crucibles, 00 size	1.44
2 cartons, 96 each clay crucibles, 20 gram	10.37
12 roasting dishes, 5 inch	1.58
1 yard rubber sheeting	1.00
1 sampler, complete, 6 by 6 inch	3.50
12 sample pans, 6 inch	.76
150 scorifiers, 2-1/2 inch	4.14
1 set sieves, 8 inch, 20, 35, 48, 65, 100 mesh, with pan and cover	12.45
1 steel spatula, 6 inch	.50
1 pair tongs, 9 inch	.72
1 pair tongs, 30 inch	2.25
1 pair tongs, crucible	1.50
1 pair tongs, cupel	1.50
1 muffle scraper	.85
12 annealing cups	1.35
1 annealing tray	1.35
1 anvil, 6 by 6 inch	4.00
1 anvil, 2-1/2 by 2-1/2 inch	2.00
1/2 sheet, asbestos mill board, each 20 by 40 by 1/2 inch	1.75
2 extra muffles	4.30
125 mailing envelopes, 6 ounce	1.24
	114.12
Reagents, initial stock:	
5 pounds test lead, C.P.	.75
200 cupels, 1-1/4 inch	4.30
25 pounds litharge C.P.	3.75
10 pounds borax glass	1.80
1 pound lead foil C.P.	.40
1 ounce silver foil C.P.	1.50
1 bottle acid nitric, C.P., 7 pound	1.89
	14.39

WET ASSAYING

Analytical balance and weights:	
1 DFC analytical balance, 200 grams	70.00
1 set analytical weights, 50 grams down	15.00
1 scale cover	2.20
	87.20
Miscellaneous equipment and supplies:	
1 sheet 1/4-inch asbestos mill board, 15 1/2 pound	1.40
2 yards asbestos paper for hot plates, 1-1/2 pounds to yard	.45
12 pyrex beakers, lipped, 50 cm^3	2.04

TABLE 5.- Equipment and supplies for assay office - Continued

WET ASSAYING - Continued

Miscellaneous equipment and supplies (Continued):

13 pyrex beakers, lipped, 150 cm^3	$2.47
48 pyrex beakers, lipped, 250 cm^3	9.60
24 pyrex beakers, lipped, 400 cm^3	6.24
6 pyrex beakers, lipped, 1,000 cm^3	3.00
5 reagent bottles, 2 HNO_3, 1 HCl, 1 H_2SO_4, 1NH_3	3.29
5 caps for above	.98
12 reagent bottles, 8 ounce, for reagents that follow	6.00
12 reagent bottle caps for above	2.30
1 pyrex wash bottle, complete, 500 cm^3	.75
2 pyrex wash bottles, complete, 1,000 cm^3	2,00
6 CH pencil brushes, 1/2 inch	1.50
2 burettes, 50 cc, G.8	2.30
1 burette, dispensing, 500 cc	3.50
6 No. 3 casseroles	3.60
6 No. 3A casseroles	4.50
4 clamps, Mohrs, 2-3/4 inch	.60
1 12-hole color-test plate, 110 mm	.70
1 gross assorted corks	1.10
1 cork borer, 6 p.c.	.80
24 porcelain crucibles, no. 0	3.60
1 graduated cylinder, 100 cm^3	.40
2 graduated cylinders, 1,000 cm^3	2.40
500 filters, 11 cm	2.45
1 filter pump	2.25
1 first-aid outfit	15.00
12 pyrex Erlm flasks, 300 cm^3	2.28
1 filter flask, 500 cm^3	1.45
1 forceps	.60
1 funnel, 8-3/4 inch	1.00
100 white filters to fit 18 inch	1.55
2 pounds glass tubing, 7 mm	1.10
1 pound glass rod for stirrers	.60
1 asbestos gloves, no. 14	2.50
2 Vanning plaques	3.50
1 box labels no. 201	.10
12 funnels, 75 mm	3.67
250 sample envelopes, 4 ounce	2.10
6 pencils, blue, for glass	.90
3 pipettes, volumetric, 10	.75
2 rubber stirrers	.50
1 pound assorted rubber stoppers	.90
2 yards black sheeting for mixing	2.00
12 feet tubing, MW PG, 3/16 inch	1.08
12 feet tubing, MW PG, 1/4 inch	1.51
24 sample pans, 5-1/4 inch	1.30
18 sample pans, 10 inch	2.92

TABLE 5.- Equipment and supplies for assay office - Continued

WET ASSAYING - Continued

Miscellaneous equipment and supplies (Continued):	
1 spatula, no. 5	$.80
1 spatula, no. 3	.35
1 spatula	.50
1 support for 2 burettes	1.95
1 support, 4 ring	2.00
2 supports, 6 funnels	7.00
12 assorted test tubes, pyrex	1.10
2 tongs, beaker	.80
1 each, triangles, nos. 1, 2, and 3, lot	.85
6 dozen watch glasses to fit beakers, lot	2.50
1 H_2S generator	.55
1 shears, 5 inch	1.35
Gas hot plates	18.00
1 Bunsen burner	.40
Water still	16.00
Compressed-gas installation	60.00
	231.68
Chemicals, initial stock:	
2 5-pound bottles acetic acid, 99-1/2 percent	2.00
1 barrel 4 9-pound bottles CP sulphuric acid	6.00
1 barrel 4 6-pound bottles CP hydrochloric acid	4.50
1 barrel 4 4-1/2-pound bottles CP ammonia	4.50
10 pounds ammonium acetate CP	7.30
3 pounds ammonium chloride USP	1.29
1 pound potassium permanganate CP	.87
1 pound ammonium molybdate USP	1.75
1 pound sodium cyanide	.50
1 pound sodium hyposulphite CP	.53
1/2 pound acid oxalic CP	.72
1/4 pound acid tartaric	.43
1 gallon denatured alcohol and can	1.00
1 square foot aluminum, 1/16 inch	1.00
1/4 pound ammonium carbonate CP	.33
1 pound ammonium persulphate CP	.91
1 pound barium chloride CP	.65
1/4 pound copper foil CP 0.008	.35
1/4 pound copper sulphate CP, crystal	.34
1/4 pound ferrous ammonium sulphate CP	.33
1/4 pound ferrous chloride CP	.52
1/4 pound ferrous sulphate CP	.32
1/4 pound lead acetate CP	.33
1 ounce lead sulphate CP	.25
5,914 litmus paper, 13-books, 6 red, 7 blue	.65
1 box lubriscal for stopcocks	.40
1 ounce methyl orange	.54

TABLE 5.- Equipment and supplies for assay office - Continued

WET ASSAYING - Continued

Chemicals, initial stock (Continued):	
1 ounce phenolphthalein	$.21
1/4 pound potassium ferricyanide CP	.52
1/4 pound potassium iodide CP	1.00
1 barrel 5 7-pound bottles nitric acid CP	8.00
1 pound mercury	1.60
1 ounce silver nitrate CP	.55
1/4 pound sodium bicarbonate CP	.31
1 pound potassium chlorate CP	.61
1 pound sodium hydroxide CP, sticks	.71
1 ounce soluble starch	.25
1 pound zinc oxide CP, dry	.74
1 ounce uranium acetate	.62
1 ounce tannic acid CP, fluffy	.31
1 ounce iron wire CP, for standards	.35
5 pounds hydrogen peroxide USP	1.25
10 pounds silica powder	.60
10 pounds potassium carbonate powder for assaying calcines	1.70
	57.64

ORE TESTING

Laboratory flotation machine	150.00
Bottle rolls (estimate)	50.00
	200.00
Total cost of assay equipment and supplies	1,189.23

TABLE 6.- Summary of costs of building, equipment, and supplies for assay office

Building:		
Materials	$555.44	
Labor	300.00	$ 855.44
Fire assaying:		
Balances and weights	383.70	
Furnace, gasoline	100.50	
Miscellaneous supplies and equipment	114.12	
Initial stock reagents	14.39	612.71
Wet assaying:		
Analytical balance and weights	87.20	
Miscellaneous supplies and equipment	231.68	
Initial stock chemicals	57.64	376.52
Ore testing:		
Laboratory flotation machine	150.00	
Bottle rolls (estimate)	50.00	200.00
		$1,189.23
Grand total		$2,044.67

GENERAL SPECIFICATIONS

Timber

All timber used in the construction of bins and other structures for the sampling works shall be of No. 1 common timber, except where otherwise noted. The No. 1 common rough timber shall be free from loose and unsound knots and from large knots so located as to impair the strength of any member materially. Only the allowable commercial variation below the nominal size in cutting shall be allowed. The acceptance of such timber, as to the quality and size, will be subject to the approval of the engineer in charge.

The following allowable unit values for the timber have been assumed in the calculations for the design of the structures:

	Pounds per sq. in.
Extreme fiber stress in bending.............	800
Compression perpendicular to the grain	200

This low fiber stress has been used because local timber that might be used in the erection of plants in some portions of the United States is of very poor grade. Some of the plants, however, would be located in territory where douglas fir or longleaf pine would be available. In such cases the width of the stringers of the ramp and the tension members in the bins can be reduced in proportion to the increase in fiber stress allowable on the higher grade timber. For example, two 12- by 12-inch stringers are shown (fig. 3) under each wheel in the ramp. If shortleaf pine should be available, with a fiber stress of 1,100 pounds, the width of the above members could be reduced 37 1/2 percent; this would indicate a width under 8 inches. In other words, if shortleaf pine should be available, the stringers in the ramp could consist of two 8 by 12's placed on edge instead of two 12 by 12's under each wheel. Permission for the reduction in size of the timber should be obtained from the engineer in charge before such reductions are made.

Concrete

The materials for all concrete shall be first class in every respect and subject to the approval of the engineer in charge. The sand shall be clean and sharp, with no more than 1/2 percent earth or alluvial matter. The gravel or crushed rock used in the cement shall pass through a 2-inch opening. No large pieces of rock or "plums" will be allowed in the concrete construction. The cement used shall be of an approved standard brand in commercial use in the locality in which the plant is being built and shall, of course, be subject to the engineer's approval and test, if required.

All concrete used in the construction of these structures for floors, ordinary foundations, or footings shall be in the proportions of 1 part cement, 3 parts sand, and 5 parts gravel by volume. Concrete that will be subject to tension stresses, such as in the crusher reinforced foundation, is to be of a 1:2:4 mixture. All floors in the sampling plant are to be poured 6 inches thick of a 1:3:5 mixture; the top of this concrete is to be 1 inch under grade. A finish coat on the floor is to consist of 1 part cement and 3 parts of sand, and the surface is to be well-troweled with a steel trowel so as to be as smooth as possible. The materials used and the workmanship on this floor are to be subject to the acceptance of the engineer in charge.

Corrugated Roofing and Siding

Buildings shall be covered as shown on the drawings, with galvanized corrugated iron having standard 2-1/2-inch corrugations. Roofs shall be of no. 24 gage, and all siding shall be of not less than no. 26-gage iron. Both roofs and sides shall be laid with a minimum of 1-1/2 corrugations as side lap and a minimum of 6 inches on the end lap of sheets. The side lap shall be so placed that any leakage will be protected by two valleys or corrugations.

Conveyors

Conveyor materials, such as troughing and return idlers, gears, head and tail pulleys, take-ups, etc., may be furnished by any of the well-known manufacturers, subject to the approval of the engineer in charge. Conveyor idlers shall be of the 3-roller type and spaced not over 5 feet apart; return idlers may be placed as much as 10 feet apart. Side idlers must be placed as shown in the drawings. Each conveyor must be equipped with a hold-back or nonreturn stop, so that in case the conveyor drive fails, the conveyor belt will not run in the reverse direction. The take-ups on the tail pulleys shall be of standard design, with not less than 9 inches of travel. The head pulleys shall be as specified in the drawings and be rubber-covered. The conveyor belt shall be not less than 4-ply, 28-ounce duck, with 1/8-inch top, and 1/32-inch bottom rubber cover. All troughing and return idlers, head and tail pulleys, and countershafts shall have their bearings provided with suitable grease cups.

LISTS OF MATERIALS AND COSTS

The estimates given herein are for a plant to be driven by purchased electric power. Line shafts and belt drives would be used in the main sampling mill, which would be run by a 40-horsepower motor. A 15-horsepower motor would be used on the crusher and a 5-horsepower geared motor on the conveyor to the storage bins.

The following tabulations give the materials and equipment required and the estimated costs for each part of the sampling works. Quotations for equipment are of autumn of 1935, as applicable to Government purchases.

TABLE 7.- Timber list for approaches to receiving bins (fig. 3)

Number of pieces	Size, inches	Length, feet	Where used	Feet, board measure
16	10 by 12	20	Sills	3,200
16	10 by 12	18	Caps	2,880
16	10 by 12	12	Corbels	1,920
3	10 by 10	14	Posts	350
9	10 by 10	12	do.	900
6	10 by 10	20	do.	1,000
6	10 by 10	12	do.	600
4	8 by 10	14	Inclined posts	373
6	8 by 10	12	do.	480
2	8 by 10	16	do.	213
6	8 by 10	14	do.	560
2	3 by 12	22	Sway braces No. 10	132
14	3 by 12	20	Sway braces, Nos. 2, 3, 4, 11, 12, 13	840
16	3 by 12	18	Sway braces, Nos. 5, 6, 7, 8, 9, 14, 15, 16	864
8	3 by 12	20	Sway braces	480
8	3 by 12	18	do.	432
16	3 by 12	16	do.	768
64	12 by 12	12	Stringers	9,216
32	6 by 12	12	Stringers, outside	2,304
8	6 by 10	12	Corbels	480
32	2 by 6	12	Handrail	384
48	2 by 6	12	Handrail supports	576
184	3 by 12	14	Roadway deck	7,728
			Total feet, board measure	36,680

TABLE 8.- List of hardware for approaches to receiving bins (fig. 3)

200	Feet	7/8-inch round mild steel for drift bolts for bents.
140	Only	3/4-inch bolts, 26 inches long, for bolting stringers to cords, complete with square nuts and 4-inch thread each end.
150	do.	3/4-inch O. G. washers.

TABLE 9.- Cost of approaches to receiving bins (fig. 3)

Fill, 486 cubic yards at $1 per cubic yard	$ 486.00
Excavation	50.00
Concrete piers, 10 cubic yards at $12 per cubic yard	120.00
Timber, including construction, 37,000 board feet at $50 per thousand (table 7)	1,850.00
Hardware (table 8)	100.00
Total	2,606.00

TABLE 10.- Timber list for receiving bins (fig. 4)

Number of pieces	Size, inches	Length, feet	Where used	Feet, board measure
1	8 by 10	22	Front sill	147
8	8 by 8	14	Sills	597
8	8 by 8	14	Posts	597
4	8 by 8	10	do.	213
8	8 by 8	12	Sway braces	512
8	6 by 10	18	Bottom beams	720
2	8 by 8	22	Bottom supports	234
2	8 by 8	24	Front ties	256
2	8 by 8	16	End ties	171
4	8 by 8	20	Caps	427
6	8 by 8	22	Runway beams	704
4	6 by 8	14	Guards on top	224
3	2 by 6	16	Handrails	48
2	8 by 8	16	End verticals	171
40	2 by 12	22	Bin lining	1,760
48	2 by 12	12	do.	1,152
			Total feet, board measure	7,933

TABLE 11.- List of hardware for receiving bins (fig. 4)

2	Only	1/4-inch mild steel plates, 2 feet 6 inches wide by 22 feet long.
24	Pieces	40-pound T rail, 13 feet long.
24	Only	45° O. G. washers for 1-inch bolts.
100	Feet	1-inch round mild steel rods for tie rods and bolts.
12	Only	O. G. washers for 1-inch bolts.
24	do.	3/4- by 14-inch bolts.
24	do.	O. G. washers for 3/4-inch bolts.
		Necessary square nuts for above.
3	do.	24-inch wide Arc gates. Chutes to be of timber lined with 1/8-inch steel plate on bottom and sides.
1	do.	Hopper, complete
		Bin lining:
3	Pieces	3/16-inch mild steel plate, 6 feet wide by 17 feet long.
3	do.	3/16-inch mild steel plate, 6 feet wide by 13 feet long.
6	do.	3/16-inch mild steel plate, 6 feet wide by 15 feet long.

TABLE 12.- Cost of receiving bins (fig. 4)

Excavation, 2 cubic yards at $1 per cubic yard	$ 2.00
12 concrete piers, 5.35 cubic yards at $12 per cubic yard	64.00
Timber, including construction, 8,000 board feet at $50 per thousand (table 10)	400.00
Bolts, rods, and washers	25.00
2 only 1/4-inch M. S. plates, 2 feet 6 inches wide by 22 feet long, 1,200 pounds at $0.05	60.00
Laying plates	10.00
24 pieces 40-pound T rail, 13 feet long, 4,170 pounds at $0.04	167.00
Assembly	20.00
24-inch arc gates, 3 at $35 each	105.00
Erection	15.00
Front and bottom bin lining, 3/16-inch M. S. plate, 2,265 pounds at $0.05	113.00
Cutting and erection	50.00
Total	1,031.00

TABLE 13.- List of hardware for crusher pit (fig. 5)

5	Only	5/8-inch reinforcing bars, 7 feet long.
13	do.	5/8-inch reinforcing bars, 8 feet long.
4	Pieces	1-1/2-inch pipe, 10 inches long, for hopper under crusher.
1	Only	Hopper, complete, as shown in figure 5.

TABLE 14.- Equipment for crushing pit (fig. 6)

1	Only	9- by 16-inch Telsmith Wheeling jaw crusher. Drive pulley 33 by 8-1/2 inches.
4	do.	Anchor bolts, 1 by 18 inches.
4	do.	W. I. pipe sleeves, 1-1/2 by 10 inches, in foundation.
35	Feet	Transmission belt, 6 inches wide, 4-ply, of medium quality.

TABLE 15.- Cost of crusher pit (fig. 5) and equipment (fig. 6)

Excavation, 120 cubic yards at $1 per cubic yard	$ 120.00
Concrete, 40 cubic yards at $12 per cubic yard	480.00
Reinforcing bars, anchor bolts, etc. (table 13)	35.00
Steel hopper under crusher (table 13)	25.00
Telsmith Wheeling jaw crusher, 9 by 16 inches, weight 5,490 pounds	1,225.00
Freight	70.00
Erection	75.00
15-hp. motor, 1,750-r.p.m., complete with switch and starter	182.00
Anchor bolts, belt drive, and erection	40.00
Total	2,252.00

TABLE 16.- Timber list for conveyor gallery to sampling mill (fig. 6)

Number of pieces	Size, inches	Length, feet	Where used	Feet, board measure
2	6 by 10	10	Stringers	100
8	6 by 10	16	do.	640
12	2 by 6	14	Floor joists	168
23	2 by 4	14	Posts	215
4	2 by 4	16	Sway braces	43
18	2 by 4	14	Caps	168
30	2 by 4	18	Trusses	360
40	2 by 4	14	Purlins	373
20	2 by 4	14	Girts	186
3	4 by 6	20	Floor joists	120
63	2 by 12	14	Floor	1,764
1	4 by 8	10	Hoist support	27
3	4 by 4	16	Posts	64
2	4 by 6	10	Hoist support	40
6	2 by 4	12	Stair handrail	48
1	6 by 8	12	Stair support	48
1	6 by 6	12	Bent 2	36
1	6 by 6	16	do.	48
1	2 by 8	20	do.	27
14	4 by 6	20	Conveyor	560
14	2 by 4	20	Bent 3	187
1	6 by 6	16	do.	48
1	6 by 6	20	do.	60
2	2 by 8	14	do.	37
1	6 by 6	14	Bent 4	42
1	6 by 6	10	do.	30
2	6 by 6	16	do.	96
2	2 by 10	20	do.	67
2	4 by 6	16	do.	64
8	2 by 12	16	Stairs	256
			Total feet, board measure	5,922

TABLE 17.- List of hardware for conveyor gallery to sampling mill

16	Only	Sliding sash with 4 10- by 12-inch lights without frames.
1	do.	No. 2 H door.
2	do.	No. 7 H doors.
1	do.	3 foot 6 inch by 6 foot 2 inch door.
6	Pieces	1-inch pipe, 12 inches long, for foundation dowels.
126	do.	No. 24-gage galvanized corrugated iron, 8 feet long.
120	do.	No. 24-gage galvanized corrugated iron, 5 feet long.
8	do.	No. 24-gage galvanized corrugated iron, 10 feet long.
130	Feet	Galvanized-iron ridge roll.

TABLE 18.- Equipment for conveyor to sampling mill (fig. 6)

1	Only	Conveyor, 16 inches wide, 121 feet long, to consist of:
1	do.	18- by 18- by 1-15/16-inch bore, rubber-covered head pulley and shaft with automatic holdback.
1	do.	23.89 P. D. gear, 1-inch pitch, 2-1/2-inch face, 1-15/16-inch bore.
2	do.	1-15/16-inch common flat split boxes.
2	do.	1-15/16-inch split set collars.
1	do.	5.12 P. D. pinion, 2-3/4-inch face, 1-inch pitch, 1-7/16-inch bore.
1	do.	30- by 4- by 1-7/16-inch bore C. F. split steel drive pulley.
2	do.	1-7/16-inch common flat split boxes.
2	do.	1-7/16-inch split set collars.
1	do.	18- by 18- by 1-15/16-inch bore tail pulley complete with shaft.
2	do.	1-15/16-inch split set collars.
1	Pair	1-15/16-inch take ups, 9-inch travel.
25	Only	16-inch, 3-roll troughing idlers, complete with grease cups.
12	do.	16-inch return idlers, complete with grease cups.
247	Feet	16-inch wide, 4-ply, 28-ounce duck, 1/8-inch rubber top cover, 1/32-inch bottom cover conveyor belt.

TABLE 19.- Cost of conveyor and conveyor gallery to sampling mill (fig. 6)

Excavation	$ 5.00
Concrete, 1/2 cubic yard at $12 per cubic yard	6.00
Timber, including construction, 6,000 board feet at $50 per thousand (table 16)	300.00
Corrugated iron, 33 squares at $5.50 each	181.00
Laying, $1.75 per square	58.00
Sashes, 16 at $4 each, in place	64.00
Doors, 2, with hardware, in place	30.00
16-inch conveyor (table 18)	875.00
Erection	75.00
5-hp. motor, geared, complete with switch and starter	257.00
Erection and wiring	20.00
Total	1,871.00

TABLE 20.- Timber list for sampling mill (fig. 7)

Number of pieces	Size, inches	Length, feet	Where used	Feet, board measure
3	8 by 8	16	Posts	256
2	6 by 8	16	do.	128
2	8 by 8	22	Caps	235
2	8 by 10	22	do.	293
2	6 by 8	22	Beams	176
2	10 by 12	12	do.	240
12	2 by 10	16	Joists	320
8	2 by 10	22	Caps	293
80	2 by 6	12	Studs and girts	960
14	2 by 6	14	Cable ends	196
4	4 by 6	14	Trusses	112
8	2 by 10	20	do.	267
18	2 by 6	16	Purlins	288
40	2 by 12	12	Floor	960
1	3 by 8	20	Knees	40
1	4 by 6	20	do.	40
			Total feet, board measure	4,804

TABLE 21.- Sampling equipment for sampling mill (fig. 8)

2	Only	Vezin samplers, no. 1 size for 1-inch feed, complete with drive pulley and proper lag screws for mounting with 26- by 3-1/2-inch T. & L. drive pulleys.
1	do.	18-inch wide by 24-inch high ore-bin gate, rack and pinion type, complete with necessary lag screws.
1	do.	14-inch belt feeder for rolls, complete with gear reduction and drive pulley.
1	Set	18- by 10-inch "Economics" rolls.
1	Only	Braun pulverizer.
1	do.	Improved ore-sample grinder.
2	do.	Jones splitters.
1	do.	Sample mixer.

TABLE 22.- Hardware for sampling mill building

95	Pieces	No. 24 gage galvanized corrugated iron, 8 feet long.
42	do.	No. 24 gage galvanized corrugated iron, 7 feet long.
22	do.	No. 24 gage galvanized corrugated iron 10 feet long.
22	do.	No. 24 gage galvanized corrugated iron, 5 feet long.
16	Only	No. 4 sash with frames.
14	Pairs	Transom pivots.
4	Only	No. 2 H doors to form two double doors.
6	Pairs	T hinges, 8-inch size.

TABLE 23.- Transmission equipment (fig. 2), for sampling mill

		Lower shaft
1	Piece	1-11/16-inch cold-rolled steel shaft, 21 feet long.
4	Only	1-11/16-inch common flat split boxes.
1	do.	16- by 8-inch S. F. split steel pulley, 1-11/16-inch bore.
1	do.	12- by 12-inch S. F. split steel pulley, 1-11/16-inch bore.
1	do.	4- by 4-inch C. F. split steel or wood pulley, 1-11/16-inch bore.
1	do.	10- by 10-inch C. F. split steel pulley, 1-11/16-inch bore.
1	do.	34- by 6-inch S. F. split steel pulley, 1-11/16-inch bore.
2	do.	1-11/16-inch split safety set collars.
		Upper shaft
1	Piece	1-15/16-inch cold-rolled steel shaft, 21 feet long.
4	Only	1-15/16-inch common flat split boxes.
2	do.	12- by 8-inch S. F. split steel pulleys, 1-15/16-inch bore.
2	do.	36- by 6-inch C. F. split steel pulleys, 1-15/16-inch bore.
1	do.	6- by 5-inch C. F. split steel pulley, 1-15/16-inch bore.
1	do.	12- by 10-inch C. F. split steel pulley, 1-15/16-inch bore.
2	do.	1-15/16-inch split safety set collars.
1	do.	34- by 12-inch C. F. split steel pulley, 1-15/16-inch bore.
1	do.	16- by 8-inch S. F. split steel pulley, 1-15/16-inch bore.
		Transmission belting
		Conveyor..........No. 1..........30-foot, 4-inch, 4-ply.
		Sampler...........No. 1..........22-foot, 3-inch, 4-ply.
		Feeder..............................20-foot, 3-inch, 4-ply.
		Rolls..................................58-foot, 6-inch, 4-ply.
		Sampler.............................39-foot, 3-inch, 4-ply.
		Countershaft.... ....28-foot, 10-inch, 4-ply.
		Conveyor...........................29-foot, 4-inch, 4-ply.
		Grinder.............................23-foot, 6-inch, 4-ply.
		Mixer.................................19-foot, 4-inch, 4-ply.
		Braun.................................20-foot, 2-inch, 4-ply.
		Engine or motor...............50-foot, 12-inch, 5-ply.
		Compressor........................20-foot, 2-inch, 4-ply.
		Summary of belting
		40 feet 2-inch 4-ply.
		81 feet 3-inch 4-ply.
		78 feet 4-inch 4-ply.
		81 feet 6-inch 4-ply.
		29 feet 10-inch 4-ply.
		50 feet 12-inch 5-ply.

TABLE 24.- Cost of sampling mill (fig. 8)

Timber, including construction, 5,000 board feet at $50 per thousand (table 20)	$ 250.00
Hardware in place (table 22)	294.00
Primary No. 1 Vezin sampler, weight 600 pounds	300.00
Freight and hauling	25.00
Erection and supports	25.00
Drive	25.00
Surge bin, erected	50.00
12-inch rack and pinion gate, erected	25.00
14-inch belt feeder, erected	125.00
1 set 18- by 10-inch "Economic" rolls, weight 4,700 pounds	900.00
Freight	100.00
Erection	125.00
Drive	100.00
Discharge chutes	75.00
Secondary Vezin sampler	300.00
Freight and hauling	25.00
Erection and supports	20.00
Drive	25.00
Improved sample grinder	200.00
Freight	50.00
Foundation and erection	25.00
Drive	20.00
Sample mixer, weight 250 pounds	100.00
Freight	15.00
Drive	15.00
Braun pulverizer	135.00
Freight	10.00
Drive	10.00
Air compressor	78.00
Receiver and pipe	22.00
Installation	10.00
Chutes for all rejects, erected	150.00
Jones splitters, 2 at $26 each	52.00
40-hp. motor, 1,160-r.p.m., complete with pulley and slide rails, switch and starter	440.00
Lineshaft, pulleys, etc. (table 23)	250.00
Erection	125.00
Belting (table 23)	80.00
30 interior 50-watt lamps with sockets and switches Interior light wires to be installed in metal conduit 10 yards lights, wiring, and sockets to be weatherproof	200.00
Total	4,776.00

TABLE 25.- Timber list for conveyor gallery to storage bins (fig. 10)

Number of pieces	Size, inches	Length, feet	Where used	Feet, board measure
12	6 by 10	16	Inclined stringers	960
10	2 by 6	14	Joists	140
2	2 by 6	18	do.	36
2	2 by 6	14	Beams	28
2	6 by 6	20	do.	120
2	2 by 8	14	Sway braces	37
50	2 by 12	14	Floor	1,400
18	2 by 4	14	Posts	168
3	2 by 4	16	do.	32
2	4 by 4	16	do.	43
20	2 by 4	18	Trusses	240
8	2 by 4	12	do.	64
1	8 by 8	16	Bent 1	85
1	6 by 8	10	do.	40
1	8 by 8	16	Bent 2	85
1	6 by 8	14	do.	56
2	2 by 8	14	do.	37
1	8 by 8	20	Bent 3	107
2	6 by 8	14	do.	112
2	4 by 8	14	do.	75
2	2 by 8	18	do.	48
1	8 by 8	22	Bent 4	117
2	6 by 8	18	do.	144
2	4 by 8	18	do.	96
2	2 by 10	22	do.	73
1	8 by 8	20	Bent 5	107
1	8 by 8	14	do.	75
1	6 by 8	20	do.	80
2	6 by 8	12	do.	96
1	4 by 8	20	do.	53
2	4 by 8	14	do.	75
4	2 by 8	18	do.	96
12	4 by 6	18	Conveyor stringers	432
12	2 by 4	18	do.	144
3	2 by 6	18	Cross braces	54
			Total feet, board measure	5,555

TABLE 26.- Hardware for conveyor gallery to storage bins

12	Only	4-light sliding sash (10- by 12-inch lights), without frames.
133	Pieces	No. 24 gage galvanized corrugated iron, 8 feet long.
15	do.	No. 24 gage galvanized corrugated iron, 6 feet long.
86	do.	No. 24 gage galvanized corrugated iron, 5 feet long.
110	Feet	No. 24 gage galvanized ridge roll.
10	Pieces	1-inch W. I. pipe, 1 foot long, for dowels for trestle bent foundations.

TABLE 27.- Equipment for conveyor to storage bins (fig. 10)

1	Only	Conveyor 16-inches wide by 103-feet long, to consist of:
1	do.	18- by 18- by 1-15/16-inch bore rubber-covered headpulley with shaft and automatic holdback.
1	do.	23.89-inch P. D. gear, 1-inch pitch, 2-1/2-inch face, 1-15/16-inch bore.
2	do.	1-15/16-inch common flat split boxes.
2	do.	1-15/16-inch split set collars.
1	do.	5.12-inch P. D. pinion, 2-3/4-inch face, 1-inch pitch, 1-7/16-inch bore.
1	do.	30- by 4-inch C. F. split steel pulley, 1-7/16-inch bore.
2	do.	1-7/16-inch common flat split boxes.
2	do.	1-7/16-inch split set collars.
1	do.	18- by 18-inch tail pulley, 1-15/16-inch bore, complete with shaft.
2	do.	1-15/16-inch split set collars.
1	Pair	1-15/16-inch take-ups, 9-inch travel.
20	Only	16-inch 3-roll troughing idlers complete with grease cups.
11	do.	16-inch return idlers complete with grease cups.
212	Feet	16-inch wide, 4-ply, 28-ounce duck, 1/8-inch top rubber cover, 1/32-inch bottom rubber cover, medium grade.
12	Only	3/4-inch bolts, 16 inches long.
12	do.	3/4-inch O. G. washers.

TABLE 28.- Cost of conveyor and conveyor gallery to storage bins (fig. 10)

Excavation	$ 5.00
Concrete, 2-1/2 cubic yards at $12 per cubic yard	30.00
Timber, including construction, 5,600 board feet at $50 per thousand (table 25)	280.00
Corrugated iron, 27 squares at $5.50 each (table 26)	148.00
Laying at $1.75 per square	47.00
Sashes, 16 at $4 each, in place (table 26)	64.00
Door with hardware	15.00
16-inch conveyor (table 27)	800.00
Erection	70.00
5-hp. motor, geared, complete with switch and starter	257.00
Erection and wiring	20.00
Total	1,736.00

TABLE 29.- Timber list for storage bins (fig. 11)

Number of pieces	Size, inches	Length, feet	Where used	Feet, board measure
2	8 by 10	22	Sills	293
1	8 by 8	8	do.	43
10	8 by 8	12	Posts or columns	640
2	8 by 8	22	do.	235
1	8 by 8	8	do.	43
3	6 by 10	22	Beams	330
4	6 by 10	14	do.	280
7	6 by 8	12	Front verticals	336
10	6 by 8	12	Back verticals	480
4	6 by 8	16	Horizontals	256
3	6 by 8	20	do.	240
2	6 by 8	14	End horizontals	112
1	6 by 6	16	do.	48
1	6 by 8	16	do.	64
1	6 by 8	10	do.	40
1	6 by 8	16	End vertical	64
4	6 by 8	12	do.	192
1	4 by 8	16	do.	43
1	4 by 6	10	do.	20
7	6 by 10	16	Bin bottom beams	560
4	4 by 8	14	Diagonal braces	149
4	4 by 12	14	Bin bottom, Bin B	224
3	4 by 12	12	do.	144
4	4 by 10	18	Bin bottom, Bin C	240
2	6 by 6	16	Horizontal tie, Bin C	96
1	6 by 6	18	do.	54
2	6 by 6	16	Track runway	96
3	4 by 6	16	Track joists	96
8	3 by 6	18	Bottom cleats	216
2	6 by 8	14	Top ties	112
1	6 by 6	16	Top beams	48
36	2 by 12	16	Lining, Bin A	1,152
9	2 by 12	20	do.	360
5	2 by 12	20	Lining, Bin B	200
10	2 by 12	16	do.	320
21	2 by 12	18	Lining, Bin C	756
4	2 by 4	16	Handrail	43
7	2 by 12	16	do.	224
			Total feet, board measure	8,849

TABLE 30.- Hardware for storage bins (fig. 11)

1	Only	Pipe clamp, complete with bolts, as shown.
1	Piece	10-inch diameter, st. W. I. pipe, 10 feet long.
40	Linear feet	3/16-inch plate, 10 inches wide, punched 3/8-inch holes for spikes for chute lining.
2	Only	1-inch tie rods, 7 feet 6 inches long, 4-inch thread each end.
50	Feet	3/4-inch round mild steel for bolts.
42	Only	3/4-inch O. G. washers.
42	do.	3/4-inch nuts (square).
4	do.	1-inch O. G. washers.
4	do.	1-inch nuts (square).
2	do.	24-inch wide arc gates, complete with bearings.
2	do.	18-inch wide arc gates, complete with bearings. a/

a/ Deduct one 18-inch gate and 15 linear feet 3/16-inch plate, 10 inches wide, if bin B is not built.

TABLE 31.- Cost of storage bins (fig. 11)

Excavation	$ 5.00
Concrete, 3.55 cubic yards at $12 per cubic yard	42.00
Timber, including construction, 9,000 board feet at $50 per thousand (table 29)	450.00
24-inch arc gates, erected, 2 at $15	30.00
18-inch arc gates, erected, 2 at $12	24.00
Hinged chutes, erected, 2 at $20	40.00
Pipe chute to bin B, with clamps	20.00
Bolts, rods, and washers	25.00
Distributing chutes	25.00
Total	661.00

TABLE 32.- Timber list for ore storage platform and trestle (fig. 1)

Number of pieces	Size, inches	Length, feet	Where used	Feet, board measure
15	6 by 6	14	Parts for bents	630
16	6 by 8	12	Stringers for bents	768
5	6 by 8	12	Caps for bents	240
2	6 by 8	16	do.	128
4	4 by 6	14	Sills	112
2	4 by 6	16	do.	64
8	2 by 8	16	Braces	171
18	4 by 6	12	Ties	432
8	2 by 12	12	Walk	192
4	4 by 6	12	Sills for platform	96
36	2 by 12	12	Floor	864
16	2 by 12	12	Front and back walls of platform	384
			Total feet, board measure	4,081

TABLE 33.- Cost of ore-storage platform and trestle (fig. 1)

Timber, including construction, 4,000 board feet at $50 per thousand (table 32)	$ 200.00
Rails, 12-pound, 200 feet, 800 pounds at $0.04	32.00
Total	232.00

TABLE 34.- Cost of accessory equipment

Platform scale, steel, 22 by 9 feet, weight 5,400 pounds	$ 621.00
Freight	110.00
Platform, 600 feet, board measure, at $50 per thousand	30.00
Pit excavation, 28 cubic yards at $1 per cubic yard	28.00
Pit excavation, 8 cubic yards at $12 per cubic yard	96.00
Erection	75.00
Scale house	25.00
Shop and oil-storage house, 10- by 20- by 10-foot building, timber and corrugated iron construction	125.00
Sundry tools, saws, hammers, Stillson wrenches, etc.	100.00
Electric hot-plate dryer	40.00
Transformers and fused cutouts	482.00
Two-pole transformer station with cross arms and platform	45.00
Distribution poles in yard	75.00
Mine car, 20-cubic-foot, roller bearing	100.00
Total	1,952.00

TABLE 35.- Summary of timber lists a/

Structure	Table number	Figure number	Board feet
Approach to receiving bins	7	3	36,680
Receiving bins	10	4	7,933
Conveyor gallery to sampling mill	16	6	5,922
Sampling mill	20	7	4,804
Conveyor gallery to storage bins	25	10	5,555
Storage bins	29	11	b/8,849
Storage platforms	32	1	4,081
Total board feet			73,824

a/ Timber required for the sampling works (except assay office).

b/ If bin B is not constructed, deduct 1,184 board feet.

TABLE 36.- Summary of costs

	Table number	Cost
Assay office	6	$2,045.00
Approaches to receiving bins	9	2,606.00
Receiving bins	12	1,031.00
Crusher and pit	15	2,252.00
Conveyor to sampling mill and gallery	19	1,871.00
Sampling mill	24	4,776.00
Conveyor to storage bin and gallery	28	1,736.00
Storage bins	31	661.00
Ore storage platform and trestle	33	232.00
Accessory equipment	34	1,952.00
Total		19,162.00
Contractor's profit, 10 percent		1,916.00
Engineering and supervision, 5 percent		958.00
Grand total cost of complete sampling plant		22,036.00

The foregoing estimates were made for first-class construction and equipment throughout the plant on the most costly type of site for the construction of the plant; the design is for the most efficient operation of the works.

With Diesel-electric power, the cost would be increased by $2,981 ($3,633 minus $652), and with direct drives by two gasoline engines converted to use fuel oil, by $623, ($2,154 plus $445 minus $1,976). If two gasoline engines were used for power, the total cost would be decreased by $611 ($1,976 minus $920 minus $445). A further saving of $257 could be made by driving the conveyor to the storage bin from the line shaft.

A considerable saving in the cost of the ramps could be obtained with the same standards if the receiving bins were built on a hillside location. Moreover, if the trucks, after being unloaded, were required to back down, about 40 percent ($1,000) of the cost of the approaches to the receiving bins could be saved. Should the receiving bins be eliminated from the flow sheet, the costs of the ramp ($2,606) and of the receiving bins ($1,031) could be saved, with the additional expense of a short earth ramp and a platform, amounting to about $200 - a net saving of $3,437.

A further saving of about $2,850 could be effected by using bucket elevators instead of belt conveyors. A bucket elevator for the service required with the necessary housing, would cost $480 erected. Bucket elevators for this service are not recommended by the authors, however.

A plant driven by two gasoline engines, with no receiving bins, and two bucket elevators would cost:

Sampling mill	$4,776.00
Crusher and pit	2,252.00
Storage bins and platform	893.00
Two bucket elevators	960.00
Accessory equipment	1,952.00
Assay office	2,045.00
	12,878.00
Saving by using gasoline engines	611.00
	12,267.00
Contractor's profit, 10 percent	1,227.00
Engineering and supervision, 5 percent	613.00
Total	14,107.00

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

I. C. 6901

SEPTEMBER 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

GOLD MINING AND MILLING IN THE BLACK MOUNTAINS, WESTERN MOHAVE COUNTY, ARIZ.

BY

E. D. GARDNER

UNITED STATES

DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

WASHINGTON

900 F STREET NW.

The enclosed Information Circular entitled "Gold Mining and Milling in the Black Mountains, Western Mohave County, Ariz.," by E. D. Gardner is one of a new series being prepared by the United States Bureau of Mines dealing with mining operations in various districts of the Western United States. Among other circulars to be issued in the near future are "Gold Mining and Milling in the Black Canyon Area, Yavapai County, Ariz.," and "Reconnaissance of Mining Districts in Pershing County, Nev.". Information Circular 6876, "The Silver Belt and the Sunshine Mine of the Coeur d'Alene District, Shoshone County, Idaho," has already been published and is available upon request.

If you wish to receive the future circulars of this series as they are issued, please sign the coupon at the bottom of this page and return it to my office. The publications will be sent you free of charge.

In the interest of economy it is desired to distribute these circulars only among those to whom they are of real interest. Your co-operation in returning the slip promptly will assist in determining the size of editions.

John A. Davis,
Chief Engineer,
Information Division.

Information Division,
U. S. Bureau of Mines,
Washington, D. C.

Please include my name and address, as given below, in the mailing list to receive future Information Circulars dealing with mining operations in the various districts of the Western United States.

____

____

____

I. C. 6901
September 1936

INFORMATION CIRCULAR

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

GOLD MINING AND MILLING IN THE BLACK MOUNTAINS, WESTERN MOHAVE COUNTY, ARIZ.[1/]

By E. D. Gardner[2/]

CONTENTS

2/ Supervising engineer, U. S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

INTRODUCTION

This is one of a series of papers describing mining and related subjects affecting mining in western mining districts and mineralized areas. The parts of this paper pertaining to current production, mining and milling methods and practices, and general conditions affecting mining were collected principally during a field survey made in May 1935. Some original data obtained on previous visits to the area are also included. The history, geological background, and past production of the mines and district are largely abstracted from previous publications, for which due credit is given later in the text.

The principal districts in the Black Mountains are the Oatman or San Francisco and the Union Pass or Katherine. This range contains the principal producing mines of the State, in which gold is the only important metal. Although the total production of gold is relatively small as compared to other districts where the precious metals are obtained as a by-product in copper mining, the area is one of considerable economic importance to the State.

ACKNOWLEDGMENTS

The operators in the region freely supplied all requested information that was available.

Bemis Phelps, secretary of the Tom Reed Gold Mining Co. furnished considerable data concerning the mines in the Oatman district.

L. V. Root of Kingman accompanied the writer on his visit to the properties on the north end of the range and in addition gave considerable general information regarding the area.

The area was described by Schrader in 1909[3/] and again by Wilson in 1934.[4/] The Oatman and Katherine districts were described by Ransome[5/] in 1923, and by Lausen in 1931.[6/] Elsing and Heineman listed the production of the individual mines and the districts in 1936.[7/]

3/ Schrader, F. C., Mineral Deposits of the Cerbat Range, Black Mountains, and Grand Wash Cliffs, Mohave County, Ariz.: U. S. Geol. Survey Bull. 397, 1909, pp. 151-218.

4/ Wilson, Eldred D., Cunningham, J. B., and Butler, G. M., Arizona Lode Gold Mines and Gold Mining: Arizona Bureau of Mines Bull. 137, 1934, pp. 80-108.

5/ Ransome, F. L., Geology of Oatman Gold District, Arizona: U. S. Geol. Survey Bull. 743, 1923, 58 pp.

6/ Lausen, Carl, Geology of the Oatman and Katherine Districts, Arizona: Arizona Bureau of Mines Bull. 131, 1931.

7/ Elsing, Morris J., and Heineman, Robert E. S., Arizona Metal Production: Arizona Bureau of Mines Bull. 140, p. 96.

LOCATION

The Black Mountains extend north and south parallel to the Colorado River and from north of Boulder Dam to Toprock (see fig. 1). The range is 100 miles long and 20 miles wide, and its highest peaks attain an altitude of 5,000 feet. The altitude at the Colorado River at the foot of the range below Boulder Dam is between 600 and 700 feet. The eastern side of the mountains is drained through valleys lying at about 3,300 feet and running north and south into the river. Locally the northern section of the Black Mountains is called the River Range.

The town of Oatman, with a population of 647 (1930), is the principal settlement in the area. Another settlement, with a school, (1935) is situated at the Katherine mill. Camps are maintained at a number of other mines. The population of Oatman and the different camps varies with the fortunes of mining in the region. Until the price of gold increased in 1933, the whole range, with the exception of Oatman, was virtually deserted.

The Atchison, Topeka, and Santa Fe Railroad skirts the southern end of the range. Kingman, the county seat of Mohave County, is the main source of supplies. Highway 66, which is paved from Kingman to the coast, passes through Oatman (fig. 1). Timber and other supplies are trucked from Los Angeles to Oatman. Highway 69, from Kingman to Boulder, which serves most of the area was realined recently and graded. It is an excellent dirt road and probably will be oiled soon. A good dirt road extends to the Katherine from Kingman. Mines off the main highway can be reached by desert roads, usually in poor condition but passable for trucks and automobiles.

TOPOGRAPHY

The range is very rugged and deeply dissected by gulches and canyons. The rocks of the central mountains of the range are largely bare. The Oatman district is in a belt of rugged foothills, which flatten out toward the Colorado River to the west.

The Katherine mine is on fairly level ground. The principal producing mines in the region are in the foothills, as shown by the contours on figure 1.

CLIMATE

The region has an arid climate, high temperatures prevailing during the summer. The extreme range of temperature recorded at Fort Mohave[8] (altitude, 604 feet) is from 3° to 127°. The range at Kingman (altitude 3,326 feet) is 8° to 117°. The temperature at Oatman (altitude 2,700 feet) usually is between those at Kingman and Fort Mohave.

Rainfall, as temperature, depends to some degree upon altitude. The average annual rainfall at Fort Mohave is 5.21 inches and at Kingman 11.50 inches. As elsewhere in Arizona, the rainfall is seasonal; the heaviest rainfall, usually accompanied by electric storms, occurs in July and August. Another rainy season

8/ Smith, H. V., The Climate of Arizona: Univ. of Arizona Agricultural Exp. Sta. Bull. 130, 1930, p. 350.

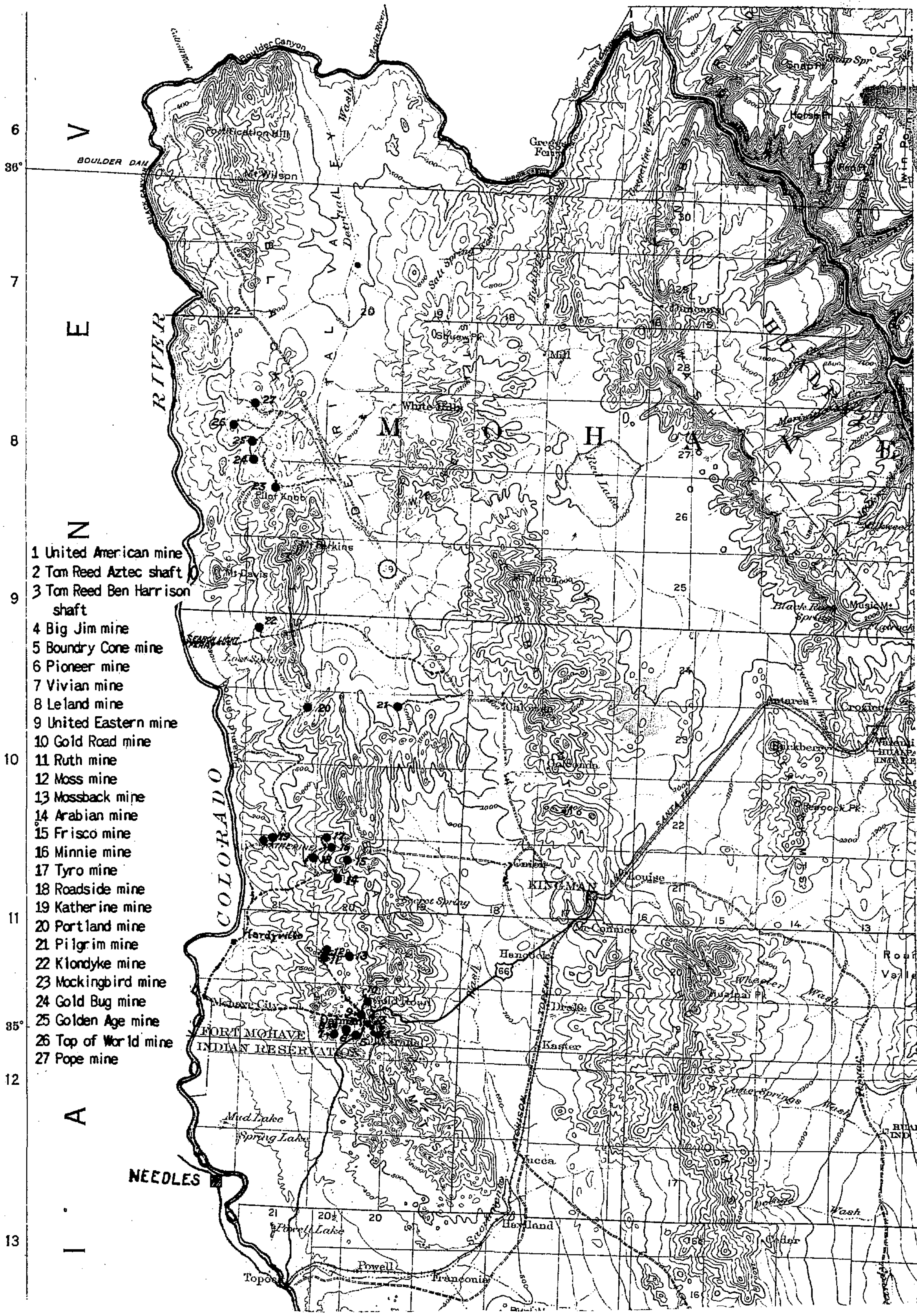


Figure 1.- Map from State contour, western Mohave County, Ariz.

occurs from December to March. Gulches are dry except after rainstorms. A few small perennial springs, the largest of which are in Silver Creek, occur throughout the southern half of the range.

HISTORY[9]

Oatman District

Soldiers stationed at Camp Mojave during the early sixties prospected part of the Oatman district. John Moss and party discovered the Moss mine about 1 mile north of Silver Creek in 1863 or 1864; it is reported that $240,000 worth of rich gold ore was taken from a pocket close to the surface during the next few years. Ore was milled first at the settlement on Silver Creek where water came to the surface and later at Hardyville on the Colorado River about 7 miles west of the mine.

The Hardy mine 2 miles west of Goldroad, the Leland on Leland Hill, and the Gold Dust 1 mile southwest of Oatman were discovered soon after. The Hardy mine was operated for a few years, the ore being treated in the Hardyville mill. After an uprising of the Hualpai Indians in 1866, the district was nearly abandoned for several years.

Activity revived in 1900, when rich ore was found in the Gold Road vein. Production has been intermittent since the mine was discovered. In 1901 the Gold Road Co. sank the Tom Reed and Ben Harrison shafts to 100 feet. The Leonora mill at Hardyville operated during part of 1901 and 1902 on ore from the Moss and Hardy veins. During 1903 and 1904 the Mohave Gold Mining Co. did considerable work on the Leland property. The Blue Ridge Gold Mines Co. produced ore from the Tom Reed vein during part of 1904-1905. In 1906 the Tom Reed Gold Mines Co. purchased the mine, developed high-grade ore, and in 1908 started production that has continued ever since, except from March 1932 to early 1934, when the property was shut down. It was operated by lessees from 1923 to 1927. The present cyanide mill was built in 1908. The town of Oatman was established about 1912.

During 1915 and 1916 an $11,000,000 ore body was developed in the United Eastern mine. The fact that this ore shoot did not outcrop encouraged scores of wildcat promotions in the district most of which, however, proved to be futile. In 1916 the Big Jim Mining Co. found an important ore body in the Tom Reed vein on the Big Jim claim immediately northeast of the Grey Eagle and Black Eagle claims of the Tom Reed Co. In 1917 the United Eastern Co. purchased the Big Jim ground; two years later the Tom Reed Co. brought suit to establish an apex claim to the Big Jim ore body. The courts decided in favor of the United Eastern. In 1924 the United Eastern ore body was exhausted, and the mine was closed. Considerable diamond-drill prospecting was done with unsatisfactory results.

9/ Mainly abstracted from Ransome, F. L., Geology of the Oatman Gold District, Ariz.: U. S. Geol. Survey Bull. 743, 1923, 58 pp. Schrader, F. C., Mineral Deposits of the Cerbat Range, Black Mountains, and Grand Wash Cliffs, Mohave County, Ariz.: U. S. Geol. Survey Bull. 397, 1909, 226 pp.
Wilson, Eldred D., Cunningham, J. B., and Butler, G. M., Arizona Lode Gold Mines and Gold Mining: Arizona Bureau of Mines Bull. 137, 1934, pp. 78-108.

During 1933 and early 1934 virtually the only work done in the district was carried on by Johnson and Witcher, lessees on the Big Jim; the ore was treated in the 45-ton-per-day Telluride cyanide mill.

With a higher price for gold in 1934 the Tom Reed started its mill and reopened the mine. The company announced a new low rate for custom ore and almost immediately the mill began to run at capacity. In the spring of 1935 it could not take all of the ore offered it. Capacity was increased early in 1936 by adding a new ball mill.

Katherine District

The Sheep Trails mine in the Katherine district about 7 miles east of the river was discovered in 1865 by Jack Mellen, captain of a steamboat running on the Colorado River. A 20-ton stamp mill for treating the ore was built at Pyramid.

The Katherine was discovered in 1900 and other mines in the district soon after. A 150-ton cyanide mill, later increased to 260 tons, was built in 1925 and operated on ores from the Katherine until 1929, when it was closed.

The mill was acquired by the Gold Standards Mines Corporation in 1933. It was running to capacity in the spring of 1935, treating both company and custom ore. The Tyro was the principal shipper of custom ore.

Other Districts

The Pilgrim (now Pioneer) was discovered in 1904, but production was insignificant until the mill was built in 1934.

The Gold Bug mine in the Gold Bug district was discovered in 1892. During 1893 and 1894, 50 tons of selected ore was shipped that netted $43,000, or about $860 per ton. Later a 24-ton amalgamation mill was built at the Colorado River. After a period of idleness the mine was being worked in 1907.

The Mocking Bird and Hall mines in the Mocking Bird district produced from 1900 to 1906 and the Burrows in the Eldorado Pass district from 1892 to 1906. Development work was being done at other properties in 1907. Intermittent small-scale work has been carried on in the district up to the present (1936).

PRODUCTION

Table 1[10] shows the production of gold and silver of the principal mines in the River Range up to the end of 1933.

10/ Elsing, Morris J., and Heineman, Robt. E. S., Arizona Metals Production: Arizona Bureau of Mines Bull. 140, 1936, p. 96.

TABLE 1. - Approximate production of gold and silver of the principal mines in the Black Mountains

	Gold value	Silver value	Total value
San Francisco district:			
United Eastern, 1917-1924	$ 13,665,000	$ 400,000	$14,065,000
Tom Reed, 1907-1933	13,100,000	100,000	13,200,000
Gold Road, 1903-1931	7,250,000	50,000	7,300,000
Moss, prior 1900	250,000	....	250,000
Telluride, 1922-1933	200,000	....	200,000
Pioneer or German-American, 1896-1906	40,000	....	40,000
Miller (Hardy), 1870-1906	100,000	....	100,000
Gold Ore	35,000	....	35,000
Homestake-Jack Pot	35,000	....	35,000
Total	$ 34,675,000	$ 550,000	$35,225,000
Katherine district:			
Katherine, 1900-1933	1,700,000	100,000	1,800,000
Frisco, 1893-1933	400,000	....	400,000
Sheep Trail-Boulevard, 1868-1906	50,000	....	50,000
Roadside, 1915-1933	20,000	....	20,000
Arabian, 1917-1933	20,000	....	20,000
Total	2,190,000	100,000	2,290,000
Mocking Bird district:			
Mocking Bird, 1900-1906	20,000	....	20,000
Gold Bug district:			
Gold Bug, 1892-1908	60,000	....	60,000
El Dorado Pass district:			
Burrows, 1892-1906	10,000	....	10,000
Grand total	36,955,000	650,000	37,605,000

GEOLOGICAL BACKGROUND

The general geology of the range has been described by Schrader.[11/] Ransome[12/] and Lausen[13/] have studied and mapped the geology of the Oatman and Katherine districts in detail. The northern deposits have received comparatively little geologic study.

The southern Black Mountains comprise an eastward dipping block of Tertiary volcanic rocks that rest upon a basement of pre-Cambrian gneiss and granite. The Tertiary formations are cut by numerous northwestward-striking faults. The rock

11/ Work cited (see footnote 3).
12/ Work cited (see footnote 5).
13/ Work cited (see footnote 6).

common to the western part of the Katherine district is a coarse-grained granite; the mountains to the east comprise the Oatman type of Tertiary volcanic rocks. Farther north, granite, schist, and gneiss predominate. The most important ore-bearing formation is the Oatman andesite termed by Schrader the "green chloritic andesite."

The veins of the Oatman district occur within fissures along which faulting has occurred. These veins are distributed widely, but the most productive are in the northeastern half of the district. The larger veins are essentially stringer lodes of complex structure. Many of the veins are lenticular. A strong vein may pinch down to nothing, or a stringer may thicken to a considerable width in a short distance. Some of the veins attain a maximum width of 90 feet; ore shoots up to 50 feet wide have been mined. Some of the outcrops, as that of the Gold Road vein, are prominent, but others, as the United Eastern, are inconspicuous. The vein system in the Oatman district is shown in figure 2 from Lausen.[14/]

The mineralization has not been shown to extend into the granite or gneiss in the Oatman district; most of the mines in the Katherine district are in granite and a few small mines have been worked in the granite and schist in the Mockingbird, Eldorado Pass, and Gold Bug districts to the north.

The gangue of the veins of the Oatman and Katherine districts consists mainly of quartz and calcite, either of which may predominate. According to Ransome,[15/] vein material that consists entirely of quartz and calcite generally is very low grade or barren. Microscopic adularia is a common constituent of the gold-bearing quartz. Fluorite occurs in some veins but is rare in large ore bodies.

The only valuable minerals found in the area are free gold and a minor amount of silver. The gold is characteristically fine-grained and can be seen only in the rich ore. Pyrite and chalcopyrite are found only rarely in the Oatman and Kingman districts. In the northern districts the gold may be associated with pyrite and galena. Lead vanadinite has been reported at one mine, the Pope. Copper minerals were associated with the gold at the Mockingbird mine.

The Pilgrim mine is in a spur of the range that heads toward Chloride. The gold occurs free, as elsewhere in the range; but the vein, which is in a highly fractured zone, differs from those found elsewhere in the range.

Taking the range as a whole, there is a remarkable similarity in the occurrence of the gold in the veins that are being worked. The ore bodies are usually steeply dipping; most of them are tabular in form and from 3 to 7 feet thick. Although most of the ore shoots are not extensive either on the strike or dip, a few, such as the Gold Road, have been worked for relatively long distances. One ore shoot in the Tom Reed has been worked to a depth of 1,400 feet. The United Eastern ore body, which occurred between the 300- and 850-foot levels, was unusual both in size and richness. Elsewhere in the range no production has come from below 700-feet from the surface.

14/ Work cited (see footnote 6).

15/ Work cited (see footnote 5).

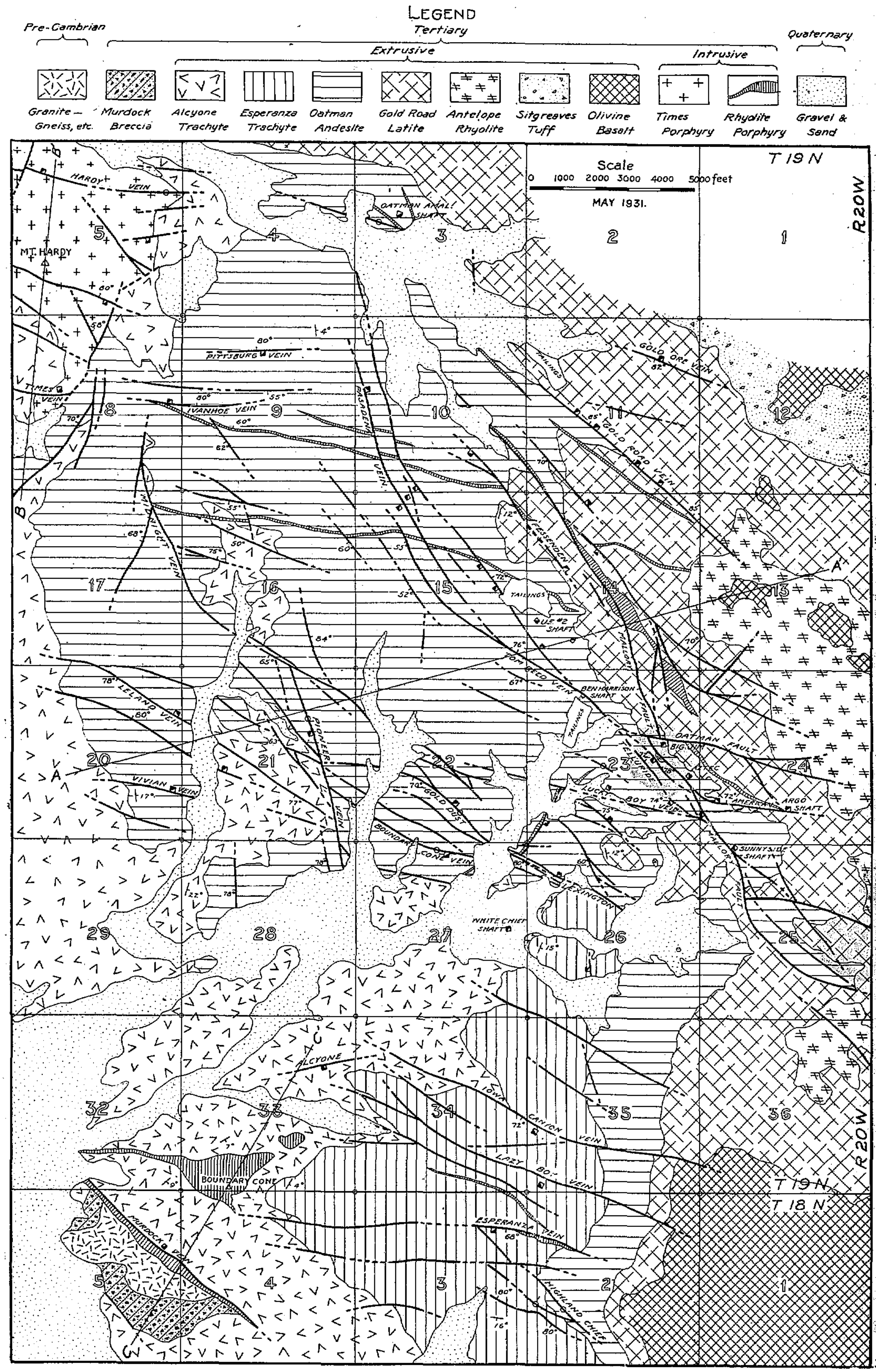


Figure 2.- Geologic map of the Oatman district, Mohave County, Ariz. (after Lausen).

On the whole, the ore in the area is not high-grade. Careful mining usually is necessary to maintain a milling grade. The average yearly grade of ore from the Oatman district from 1908 to 1928, inclusive, ranged from $3.71 per ton in 1908 to $16.98 in 1924; the average grade for the whole period was $12.37 per ton.

The wall rocks and vein material on the whole stand well, which permits relatively low mining costs. The only important exception is at the Pilgrim mine, where the ore occurs in a fracture zone and under gouge seams.

The mines (except the United Eastern) are ventilated naturally and where good air currents are maintained the workings are fairly comfortable. The temperature in dead ends below the 600-foot level of most of the mines is too high for efficient work. Mechanical ventilation was needed in all of the United Eastern workings to permit efficient operation. Rock temperature in 1920 on the 600-foot level of Aztec shaft of the Tom Reed was 86° F., and in a raise on the 300-foot level in the United Eastern it was 93° F.

The workings are dry on the whole. In 1920, 350 gallons of water per minute was being pumped from the Aztec shaft of the Tom Reed. Water was controlled in the United Eastern by pumping 32 gallons per minute from the tenth level for 16 hours daily.

TRANSPORTATION

None of the mines in the range have rail connections. All surface transportation of ore is by truck except that an aerial tramway is used from the Black Eagle shaft of the Tom Reed mine to the mill belonging to the same company.

Trucking

Trucking contractors at Kingman and Oatman have sufficient facilities to meet the needs of the range. Contract trucking prices for 1933 to 1935 are shown in table 2.

TABLE 2. - Trucking costs in Arizona

From -	To -	Distance, miles	Tons, daily	Year	Condition of road	Shovelling	Size loads, tons	Price per ton 1/	Cost per ton mile
Oatman district:									
Big Jim mine........	Telluride mill	1	45	1933	Good.....	None	7	$ 0.30	$ 0.30
Ruth-Rattan mine....	Tom Reed mill.	9	40	1935	Fair.....	do.	7	1.25	.14
Mossback mine.......	do......	9	.....	1934	do.....	do.	7	1.25	.14
German- American mine	do......	4-5	15	1935	do.....	do.	7	.50	.10
Do.................	do......	4-5	15	do.	do.....	In..	7	.75	.15
Midnight mine.......	do......	7	2	do.	Good.....	do.	7	1.25	.18
United Western mine.	do......	2	20	do.	Excellent	None	7	.35	.18
Katherine district:									
Tyro mine...........	Katherine mill	6	60	do.	Fair.....	do.	2/ 7	.50	.08
Roadside mine.......	do......	4	(3/)	do.	Good.....	do.	7	.30	.08
Arabian mine........	do......	8	(3/)	do.	do.....	do.	7	.40	.05
Portland mine.......	do......	15	(3/)	do.	Fair.....	do.	7	1.12	.07
Frisco mine.........	do......	10	(3/)	do.	Good.....	do.	7	.60	.06
River Range:									
Klondyke mine.......	Tom Reed mill.	82	15	do.	Fair.....	In..	7	5.00	.06
Eldorado mine.......	do......	75	(4/)	do.	Good.....	do.	7	6.00	.08
Pope mine...........	Kingman R.R. station...	53	(5/)	do.	do.....	do.	7	3.50	.07
Pilgrim mine........	Midvale, Utah.	515	(6/)	do.	Excellent	(7/)	......	8/ 15.00	.03

1/ All hauling done by contract.
2/ 3½ ton trucks loaded to 7 tons.
3/ Same contractor did all hauling to Katherine mill - total 250 tons daily.
4/ Odd lots.
5/ 50-ton lots.
6/ Odd lots, concentrate.
7/ Sacks.
8/ This includes $2 per ton highway tax in Utah. New rate, after new road built across dam which eliminates curves and hills, $11 exclusive road tax.

In May 1935 a single contractor hauled virtually all the ore (230 to 250 tons daily) to the Katherine mill with four 3-½ ton trucks loaded with 7 tons each and two 1½ ton trucks loaded with between 3 and 4 tons each. The trucks were worked one shift of from 10 to 14 hours daily; occasionally, however, two shifts were required for a part of the fleet. One driver and one mechanic were employed for each truck.

Freight Rates

Most of the smelting ore in the range is shipped to smelters in the Salt Lake Valley of Utah. The freight tariff on gold ore from Kingman to Midvale, Utah (March 1936), was as follows:

Value of ore	Rate per ton	Value of ore	Rate per ton
\$15 to \$20	\$ 5.25	\$70 to \$80	\$10.45
20 to 30	5.75	80 to 90	11.40
30 to 40	6.25	90 to 100	12.20
40 to 60	6.75	over 100	13.00

An added charge of 40 cents per ton, called an emergency charge, is assessed against all shipments.

Ore from the Pope mine (fig. 1) in the northern portion of the range was being shipped to the Midvale (Utah) smelter in May 1935. It was hauled 52 miles to Kingman by truck and then 855 miles by rail to Midvale. In January 1936, when the road over the Boulder Dam had been completed, it was trucked 34 miles to Boulder and thence by rail to Midvale a distance of 482 miles.

CUSTOM MILLS

The Oatman district is served by the Tom Reed and Telluride mills, both of which take custom ores. The Katherine mill in the Katherine district treats both company and custom ore. The Pioneer Gold Mining Co. has its own mill at the Pilgrim mine.

Custom rates

The custom rates at the Tom Reed mill in May 1935 were:

	Per ton
35-ton daily lots, or over (on at least 3 consecutive days, or 105 tons)	\$3.60
From 25- to 35-ton daily lots (on at least 3 consecutive days, or 75 tons)	3.80
25-ton lots or more as individual or irregular shipments	4.00
Less than 25-ton lots	4.50

The Tom Reed Co. paid for 95 percent of the gold in the ore received under the first three schedules. Ninety-two percent of the gold was paid for in ore shipments of less than 25 tons. No payments were made for silver as most of the ore in the district did not contain enough of this metal to justify the expense of assaying. In dry weather a flat deduction of 1 percent of the weight is made for moisture in ores from near the surface. Payment was made to shippers in May

1935, when the report was received from the Mint on the gold bar from any particular lot.

The Tom Reed also helps finance small operators who have ore in sight. A first lien is taken on the bullion. Repayment is made in installments and, however, is not all held out of the first shipments. The mill is equipped with a complete mechanical sampling plant.

The custom rate at the Telluride mill (May 1935) was $4 per ton; 95 percent of the gold was paid for. The milling rate at the Katherine in May 1935 for Tyro ore (60 tons daily) was $3.50 per ton. Ninety-two percent of the gold was paid for in shipments that assayed 0.15 ounces of gold to the ton; for each 0.01 ounce increment in the value of the ore the rate of payment was increased 0.1 percent, up to a maximum of 95 percent. For an ore containing 0.42 ounce per ton, 94.7 percent of the assay value would be paid. The shipper received payment for 30 percent of the silver in the ore.

SMELTER RATES

The following are two representative open-smelter schedules at a plant that receives ore from the northern districts in the area.

Lead schedule

Sampling is done free of charge by the smelting company or at the shipper's expense at a custom sampler. Payments are as follows:

Gold:

If from 0.02 ounce to 5 ounces per ton, at 92 percent Mint price.
If over 5 ounces and up to 10 ounces per ton, at 94 percent Mint price.
If over 10 ounces per ton, 96 percent Mint price.

Silver:

95 percent Mint price. No payment for less than 1 ounce.

Lead:

90 percent of dry assay (wet assay less 1-½ units) on Engineering and Mining Journal quotation for week preceding delivery less 1-½ cents per pound. No payments are made for less than 3 percent.

Copper:

90 percent of wet assay at Engineering and Mining Journal quotation week preceding delivery, less 5-½ cents per pound; minimum deduction 15 pounds.

Iron:

Paid for at 6 cents per unit (20 pounds).

Charges

Insoluble: Charged for at 10 cents per unit.
Zinc: 6 percent free; excess charged for at 30 cents per unit.
Sulphur: 2 percent free; excess charged for at 25 cents per unit; maximum charge $2.50.
Treatment; $2.50 per ton on the basis of 30 percent lead. Debit 10 cents for each unit under 30 percent and credit 10 cents for each unit above 30 percent.

Copper Schedule

Payments:

Gold and silver: - The same as for ore shipped under the lead schedule.
Copper. - 90 percent of wet asay at Engineering and Mining Journal quotation for week preceding delivery, less 3 cents per pound; minimum deduction 15 pounds.

Charges:

Treatment. - $3.70 per ton based on gross value of gold, silver, and copper of $20 per ton, with 10 percent of the excess gross value over $20 per ton added to the treatment charge, up to a maximum treatment charge of $7.70 per ton.

Zinc. - 6 percent free; excess charged for at 30 cents per unit.

On lots of less than 10 tons a sampling and assaying charge of $10 for the lot is made under both schedules. Concessions are made sometimes to a shipper who can supply a steady tonnage and for ores that have special fluxing qualities.

WATER SUPPLY

Water for mining and milling at Oatman is obtained mostly from mine shafts. Domestic water comes through a 6-inch pipeline from wells in Cottonwood Creek about 3 miles distant.

The water table in the Katherine shaft is 100 feet above the elevation of the Colorado River. In May 1935 water for milling was pumped from the river. Water is scarce in the northern part of the range; so far the only demonstrated supply is the Colorado River. Enough for camp use has been found in wells in a few localities. Two small mills were being operated with water pumped from mines in 1907.

POWER

The Oatman and Katherine districts are served with power by the Citizen's Utility Co. of Kingman, Ariz. A modern Diesel plant was erected in 1936 by the Pioneer Gold Mining Co. and a few smaller properties use semi-Diesel or gasoline engines. The following schedule was furnished by the utility company (March 1936):

Combined milling and mining power service

Applicable to mine power installations, 50 percent of the connected loads of which are used for milling purposes and which milling load exceeds 75 horsepower of demand (nameplate equipment rating) and where the customer owns and maintains the transformer substation.

Territory. - All territory served by the company in Mohave County.

Rate. - For consumption of stated quantities within a period of one calendar month or any part thereof;

From 1	to	20,000	kw.-hrs.	per	month	at	$0.0265	per	kw.-hr.		
20,001	to	100,000	"	"	"	"	.0225	"	"	"	
100,001	to	200,000	"	"	"	"	.02	"	"	"	
In excess	of	200,000	"	"	"	"	.0175	"	"	"	

Subject to a fuel oil differential, increased cost of $0.00025 per kw. hr. for each 5 cent variation above a base fuel oil cost of $1.50 per barrel, and a decreased cost of $0.00025 per kw.-hr. for each 5 cent variation below a base fuel oil cost of $1.25 per barrel, f.o.b. Kingman, Ariz.

The minimum monthly charge for service under this schedule shall be $1 per kv-a of transformer capacity for the first 100 kv-a, plus $0.50 per kv-a for each kv-a of transformer capacity in excess of 100 kv-a.

Less the applicable proportionate part of any decrease or plus the applicable proportionate part of any increase in taxes or governmental impositions which are assessed on the basis of gross revenue of the company and/or the price or revenue from the electric energy or service sold and/or the volume of energy generated or purchased for sale and/or sold hereunder as may be effective May 15, 1935.

Special conditions

(a) Service under this schedule shall be supplied at 44,000 volt, 3-phase, 60-cycle, alternating current.
(b) Metering shall be on the secondary side of the transformer.
(c) Energy under this schedule is supplied only on written agreement covering line construction, service term, minimum consumption, surety deposit, operating use of energy, etc.

Mine and industrial power service

Applicable to mine and industrial power where the customer owns and maintains the transformer substation.

Territory. - All territory served by the company in Mohave County.

Rate

Base rate - $0.03551425 kw.-hr.

Subject to a fuel oil differential of $0.000284 per kw.-hr. for each 5 cent variation in the cost of fuel oil above or below a base cost of $2.972 per barrel, the net being reduced to four decimal places.

Quantity discounts

Quantity discounts are allowed on bills paid on or before the tenth day of the month succeeding that in which service was rendered:

Kw.-hr. per month	Percent
From 1 to 20,000	net
20,001 to 100,000	7.5
100,001 to 150,000	10.0
150,001 to 200,000	12.5
200,001 to 250,000	15.0
250,001 to 300,000	17.5
300,001 to 350,000	20.0
350,001 to 400,000	22.5
400,001 and upwards	25.0

The minimum monthly charge for service under this schedule shall be $1 per kv-a of transformer capacity for the first 100 kv-a, plus $0.50 per kv-a for each kv-a of transformer capacity in excess of 100 kv-a

Rate calculation

Base rate	$0.03551425
Fuel oil differential (-30 x $.000284)	.008520
Difference	$ .026994425
Effective net rate	$.027

Less the applicable proportionate part of any decrease or plus the applicable proportionate part of any increase in taxes or governmental impositions which are assessed on the basis of gross revenue of the company and/or the price or revenue from the electric energy or service sold and/or the volume of energy generated or purchased for sale and/or sold hereunder as may be effective May 15, 1935.

Special conditions

(a) Energy under this schedule is supplied at 44,000-volt, 3-phase, 60-cycle, alternating current.
(b) Service will be metered on the secondary side of the transformers.
(c) Energy under the schedule is supplied on written agreement covering line construction, service term, minimum consumption, etc. Estimates and details on request.

The following notations regarding their application may be of interest:

1. On related mining operations under one ownership in the same district we combine meter readings from two or more banks of transformers or substations in computing the monthly charge.

2. There are no demand charges of any nature in either rate .

3. No load factor or power factor requirements are made.

4. There have been no taxes imposed since the effective date of these schedules affecting the rate by reason of the tax clause.

5. Service in December 1935 was rendered through approximately one hundred miles of transmission system and sixteen substations having a combined capacity of 4,155 kv-a. The average rate realized for that month was $.0192. The maximum in any instance was $.027.

6. Only one mine power customer in the group served in December 1935 was receiving service in November 1932, and the service taken at that time was not used in mining.

LABOR

In 1934 and 1935 skilled labor was plentiful in the Oatman and Katherine districts. All local mine workmen were employed, but many miners had drifted in from the copper camps in Arizona, which were then largely shut down. Eight-hour shifts were worked. The wage scale in the Oatman district was as follows:

	1933-1935		1933-1935
Miners............	4.50	Hoist engineers.....	5.00
Timbermen.........	4.50	Blacksmiths.........	5.00
Muckers...........	4.00	Cage tenders........	4.50
Carmen............	4.00		

LEASING

Considerable leasing is done in the Oatman district. The following tabulation shows the royalties charged by the Tom Reed Gold Mines Co. in March 1934.

Value of ore, per ton	Percentage royalty	Value of ore, per ton	Percentage royalty
Up to $10.........	5	From $30 to $35...	30
From $10 to $12...	6	From $35 to $40...	35
From $12 to $15...	9	From $40 to $45...	40
From $15 to $20...	15	From $45 to $50...	45
From $20 to $25...	20	From $50 and over.	50
From $25 to $30...	25		

The royalty charged by the Pioneer or German-American in May 1935 was as follows:

Value of ore per ton	Percentage royalty	Value of ore per ton	Percentage royalty
Up to $15.........	10	From $60 to $70....	35
From $15 to $25...	15	From $70 to $80....	40
From $25 to $40...	20	From $80 to $90....	45
From $40 to $50...	25	Over $90...........	50
From $50 to $60...	30		

MINING AND MILLING

Shrinkage, cut-and-fill, and open-stope methods are being used in the mines of the range. The gold in the ores is free, finely divided, and dissolves readily in cyanide solutions. The ore, except in rare cases, contains no cyanicides and on the whole is admirably adapted to treatment by the cyanide process.

Flotation is used at one mill (the Pilgrim), although the gold has the same characteristics as elsewhere in the range. The large amount of clay and gouge in the ore presents such a difficult settling problem that cyaniding has not proved economical.

Mining and milling practices, together with costs where available, are given in the descriptions of the individual mines.

The principal mines operating in the district in 1935 together with operating data are shown in table 3.

All of the ore produced in the Oatman district except that from the Mossback is treated in the Tom Reed and Telluride mills; that from the Katherine district goes to the Katherine mill. Odd lots also are received by these mills from elsewhere in the range. As shown by the table, about 600 tons of ore are being milled regularly in the range; occasionally additional shipments are sent to smelters outside the State.

TABLE 3. - Operating data, principal mines in Black Mountains, 1935.

Mine	Company		Agent in charge	Daily tonnage	Number of men employed: Surface	Underground	Total
Oatman district:							
Tom Reed 3/	Tom Reed Gold Mines Co.	Oatman	Jack Zwinge	1/225	45	56	2/100
Big Jim 3/	Big Jim Operating Co.	do.	Rae L. Johnston	45	6	13	2/ 19
Gold Road	United States Smelting & Refining Co.	do.	L. H. Duriez	4/ 33	......	....	40
Ruth-Rattan	Oatman Eastern Mines Co.	do.	James J. Moss	4/ 40	6	12	2/ 18
Mossback	Mollin Mining Co.	do.	Chas. A. Smith.	10	9	6	2/ 15
Pioneer or German-American	Amulet Mines Co.	do.	Geo. F. Moser	4/ 15	......	....	5/ 23
United Eastern and Telluride	Oatman Associates Mining Co.	do.	J. L. McIver	4/ 5	4	10	2/ 14
United Western 6/	Consolidated Gold Mines	do.	T. E. Wood	4/ 20	6	10	2/ 16
Lexington	do.	do.	do.	4/	......	....	5/ 4
Midnight	Hautier and Waters 5/	do.	L. V. Hautier	4/ 2	......	....	5/ 2
Katherine district:							
Roadside, Arabian, Frisco, Portland, Minnie	Gold Standards Mines Corp.	Kingman	Richard De Smet	1/250	28	25	2/ 53
Tyro	White Spar Mining Co.	do.	R. A. Elgin	7/ 60	4	20	2/ 24
Pilgrim district:							
Pioneer (Pilgrim).	Pioneer Gold Mining Co.	Chloride	E. F. Hastings.	75	......	....	73

1/ Mill.
2/ From 24th annual rept. of State mine inspector for year ended Nov. 30, 1935, last inspection.
3/ Closed January 1936.
4/ Included in Tom Reed tonnage.
5/ Lessees, May 1935.
6/ Closed May 1, 1935.
7/ Included in Katherine mill tonnage.

Oatman district

The location of the mines and geology of the Oatman district are shown in figure 2 from Lausen.[16]

Tom Reed

The Tom Reed has been the most consistent producer in the range and, next to the United Eastern, has had the largest production. The daily capacity has been from 20 tons in 1904 to 220 in 1931. With the fineness of grinding used in 1935, the mill had a capacity of 200 tons daily.

In May 1935, of the 200 tons being milled daily, 75 tons came from the Aztec and Black Eagle and 25 tons from the Ben Harrison. This ore was produced on company account. An additional 50 tons daily was being mined from Tom Reed ground by 20 sets of lessees (56 to 60 men) and about 50 tons of outside custom ore was being received.

In the year April 1, 1934, to March 31, 1935, 58,834 tons of ore was received and 58,791 tons milled. The value of the bullion shipped was $117,368. The mill heads contained an average of $11.41 in gold.

The company had a lease on the Argo, which adjoins the Tom Reed on the west, and was drifting on the Tom Reed vein in this ground.

Water for milling is obtained from the mine.

<u>Mining</u>. - The Tom Reed vein is continuous for about 1-½ miles on the strike (fig. 2). The three most important ore shoots are localized within 2,000 feet along the fracture. The fracture is barren for another 2,000 feet to the important Big Jim-Aztec ore shoot, which has a length of over 1,500 feet. The average width of all shoots is probably about 6 feet.

The property has a complete mine plant. The principal items of mine equipment are a 150 horsepower hoist, three compressors with a combined capacity of 4,000 cubic feet of air per minute, and an aerial tramway about 1 mile long.

The mine is developed by four main shafts - the Ben Harrison, 900 feet deep; Aztec, 700 feet deep; Grey Eagle, 425 feet deep; and Black Eagle, 1,100 feet deep. A winze from the 700-foot level of the Ben Harrison reaches the 1,400-foot level. The total underground workings amount to about 15 miles. Levels are 100 feet apart.

<u>Stoping</u>. - Shrinkage and horizontal cut-and-fill stoping have been the principal methods used in the Tom Reed mine. Chutes (raises in cut-and-fill stopes) are carried up 20 feet apart; manways are 125 feet apart.

Since reopening in 1934, much of the mining has been done in narrow parts of the vein between worked-out shoots and around old stopes; more timbering than formerly has been necessary. Open stoping with stulls is used in the narrow parts of the vein together with a little shrinkage. The open stopes and emptied shrinkage stopes are filled when development rock or sorted waste is available.

16/ Work cited (see footnote 6).

Mining cost. - The mining cost for the 1934 fiscal year was $1.73 per ton. The cost of transportation on the tramway was $0.20 per ton.

Milling. - Standard countercurrent cyanidation is used in the Tom Reed mill. Flow sheets similar to those employed in the United Eastern plant are used in the Katherine and Telluride mills.

The flow sheet of the mill (1936) is shown in figures 3, 4, and 5. These figures were supplied by P. U. Brough, mill superintendent. In May 1936 a new 40-by 12-foot thickener was being installed at a higher elevation. This will be the future tailing thickener. The present tailing thickener will be converted into an agitator as longer contact is needed at the present tonnage rate.

The run-of-mine ore was crushed in the Traylor crusher to minus 2 inches. Grinding was done in solution in a (primary) 6- by 5-foot ball mill and two (secondary) 5- by 6-foot ball mills. The first mill was in a closed circuit with a Model C Duplex Dorr classifier 4 feet 6 inches by 14 feet 6 inches that made 26 strokes per minute. The overflow, which was minus -10 mesh, went to two other classifiers. The drag product went to the two secondary ball mills which, in turn, were in closed circuit with two classifiers. The overflow from these last classifiers was 85 percent through 200 mesh.

The overflow from the two intermediate classifiers and the last two went to a 40 by 12 primary thickener, the rake of which revolved one-sixth revolution per minute. The overflow went to a gold-storage tank. The pulp went through three 30- by 12-foot agitators, the arms of which revolved at 3 r.p.m. Air under a pressure of 20 pounds per square inch was used for agitating. Cyanide was added to the No. 1 agitator to maintain a strength of 2.3 pounds of cyanide per ton of solution. The pulp in the agitator contained 40 percent solids. The solution in the mill circuit contained 1-½ pounds cyanide and 1 pound lime to the ton of solution.

In 1934, when 122 tons were treated daily, the ore was agitated for 5 days. From the third agitator the pulp was removed by a diaphram pump to the suction of a Wilfley pump which, in turn, delivered it to the head of a series of five thickeners that comprised the countercurrent system of the mill. Wash water, which was also the make-up water of the mill, was added to the no. 5 thickener, and barren solution was added to no. 3. The overflow from no. 1 went to the mill-solution tank. The discharge from no. 5 tank, which was the mill tails, contained 60 percent solids.

The solution in the discharge contained 0.4 pound cyanide, 0.5 pound lime, and 20 cents gold per ton of solution. The gold lost in solution equaled 12 cents per ton of tailings. The total gold in the tailings amounted to 43 cents per ton (March 1934). The over-all mill saving was 95.5 percent. Seventy percent of the gold was dissolved from the ore during grinding.

The solution from the gold-storage tank was divided - part was pumped to the mill-solution tank and part went through a clarifier, thence to two batteries of Merrill filter presses for the removal of gold. Each press contained twenty-six 36-inch triangular leaves. With the exception that the Tom Reed mill has no Crowe vacuum, the gold-precipitation plant is the same as at the United Eastern, described by Bagley on page 436 of volume 119, Engineering and Mining Journal, March 14, 1935.

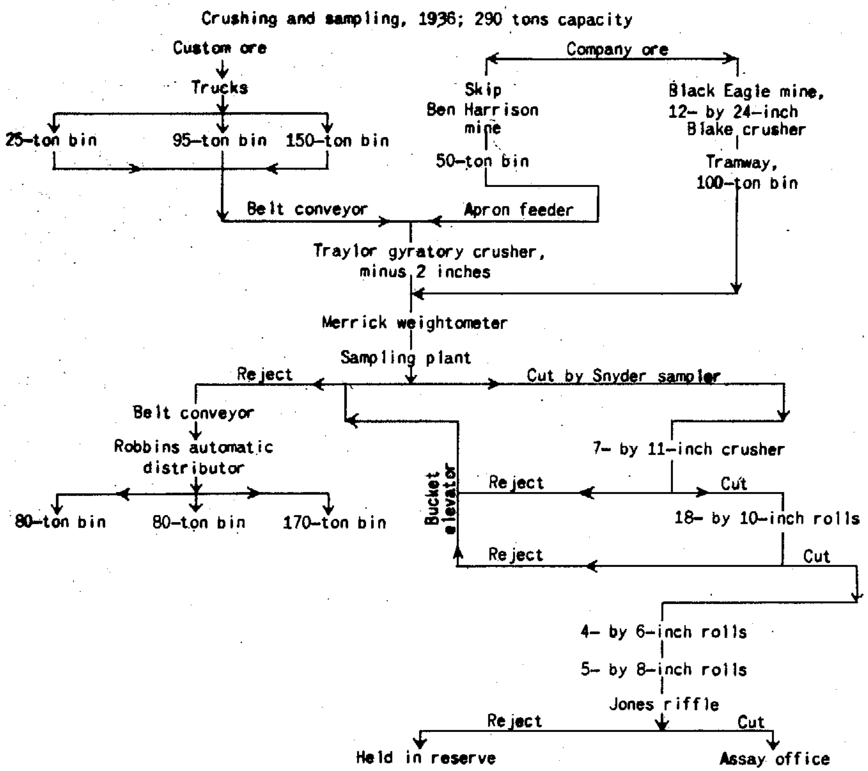


Figure 3.- Tom Reed flow sheet

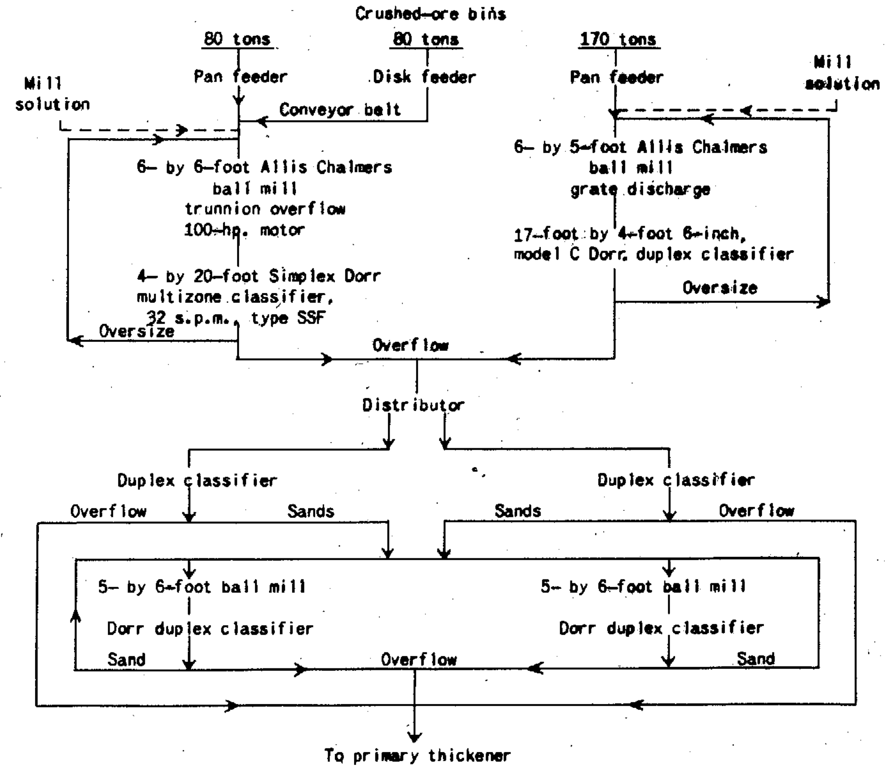


Figure 4.- Flow sheet, Tom Reed grinding circuit.

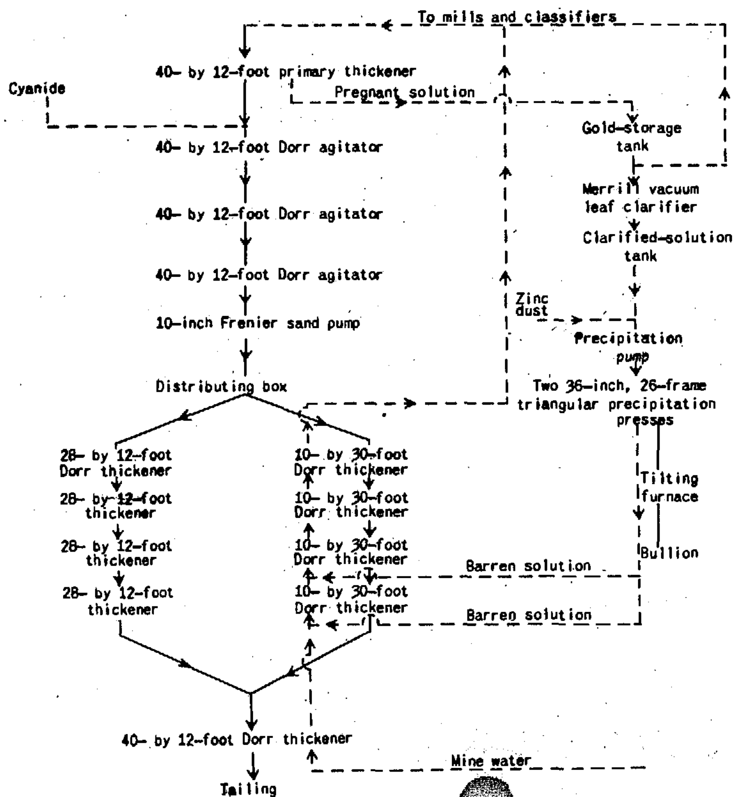


Figure 5.- Flow sheet of Tom Reed treatment plant, 1936.

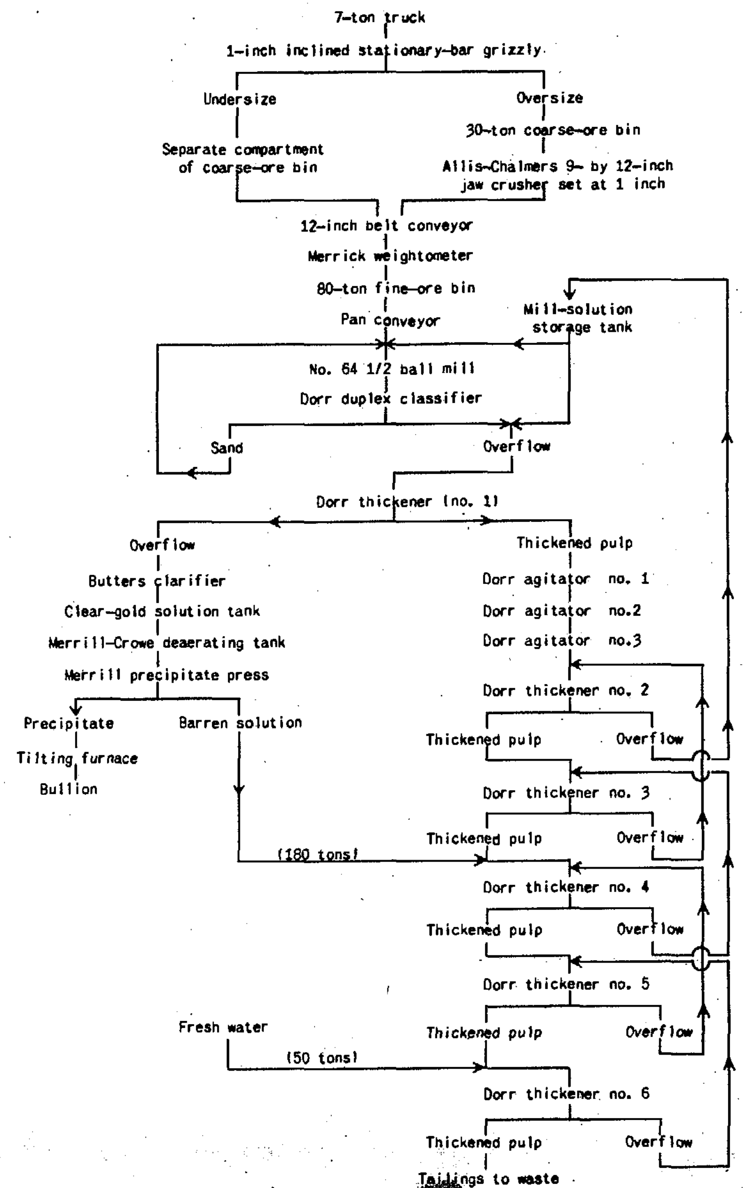


Figure 6.- Flow sheet of Big Jim (telluride) mill.

The filters were cleaned up twice a month. The method of cleaning followed at the United Eastern was used. The bullion is 880 fine with one melt.

Metallurgy - The screen size and value of the tailings in April 1935 are shown in the following tabulation. The tailing discharge was 53 percent solids.

	Percent of total	Value		
		Ton of sample	Ton of ore	Percent of total
+ 150-mesh sand..........	4.6	$ 1.75	$0.0805	15.8
+ 200-mesh sand..........	12.2	1.40	.1708	33.5
- 200-mesh sand..........	22.0	.70	.1540	30.3
- 200-mesh slime.........	61.2	.17	.1040	20.4
	100.0		$0.5093	100.0

In the fiscal year April 1, 1934, to March 31, 1935, recovery was as follows, figured against bullion recovery and composite tail sample.

Average daily tonnage.........	$161.00
Average mill heads...........	11.408
Average mill tails...........	.549
Average recovery gold........	95.19 percent
Average silver recovery......	60.00

The company does not regularly assay for silver, as it occurs in too small amounts to justify the expense.

In March 1935 the tailing loss was as follows:

	Gold	Silver
Washed tailings..................	$0.49	$0.129
Solution loss....................	.052	.005
Total loss.......................	$0.542	$0.134

The indicated recovery was 95.69 percent.

Supplies - The ball and reagent consumption (April 1935) was as follows:

Ball consumption, pounds per ton:

5-inch balls..................	1.93	
2-½-inch balls................	2.63..........	4.56
Cyanide (Aerobrand), pounds per ton ore.........		1.41
Lime, pounds per ton ore........................		4.44
Zinc dust, pound per ton ore....................		.47
Zinc dust, pound per ton solvent precipitated...		.145
Zinc dust, pound per ton, fine ounce bullion....		.71

Power. - Power is purchased from the local utility company at the rates previously shown. The consumption of power in May 1935 was 32.8 kwh. per ton ore. The motor set-up in the mill follows.

Equipment	Horsepower of motor
Traylor gyratory	40
Pan feeder	3
Belt elevator	10
Sampler	25
Belt conveyor	5
Crusher	40
Ball mill (primary)	100
Ball mills (secondary, two 75)	150
Ball-mill feeder and classifier	5
Secondary classifiers	5
Agitators (three 9)	27
Thickeners (six 5)	30
Wilfley pump to thickeners	5
Solution pump (four 3)	12
Mill circulating solution pump	15
Precipitating pump	10
Clarifying pump	10
Air compressor (some air to mine)	60
	552

Labor. - The number of men employed in the mill in March 1934 (122 tons per day) was as follows:

Aerial tramway	3
Crusher and sampler	1
Ball-mill operator (2 shifts)	2
Solution men (3 shifts)	3
Refinery man	1
Tailing man	1
Superintendent	1
Assayer	1
Total	13

One-half the wages of the two upper terminal men and one-half of the salary of the assayer were charged to mining.

Milling costs. - The milling cost in 1934, when 122 tons was treated daily, was $2.60 per ton. Amortization and depreciation for taxation purposes was figured at $0.70 per ton. For the year April 1, 1934, to March 31, 1935, the direct milling cost was $1.73 per ton (161 tons daily). The grinding cost was $0.60 per ton.

Total costs. - The costs per ton in the fiscal year 1934 were:

Mining	$4.34
Aerial tram	.20
Milling	1.73
Other costs	.79
Total	7.06

These do not include depreciation or depletion.

United Eastern

Production in the United Eastern by the United Eastern Mining Co. was begun in January 1917, and the known ore bodies (United Eastern and Big Jim) were exhausted in May 1925[17] and the mines closed.

The Big Jim part of the property was leased to Johnston and Witcher in 1932; this operation is described in a subsequent section of this paper.

In May 1935 Oatman Associates Mining Co. had a lease on the United Eastern; some selected old tailing and clean-up material from the site of the United Eastern mill had been shipped to the Tom Reed mill. The main shaft of the United Eastern had been retimbered to the 300-foot level and a drift begun around the old filled stopes with the hope that with the higher price of gold it would pay to work the margins.

The mining methods and records of the United Eastern Mining Co. have been recorded by Moore.[18] The mine surface plant, mill construction, and plant operations have been described by Wartenweiler.[19] Mill operations have been described by North[20] and Bagley.[21]

Surface plant. - The equipment at the main hoisting shaft consisted of a double-drum electric hoist direct-connected to a 150 horsepower motor. Ore was hoisted in skips working in balance, each skip being suspended below a cage used to handle men, supplies, and waste. The skip loads averaged 2.2 tons; the maximum hoisting speed was 800 feet per minute. Air was supplied by two belt-driven 19- by 12- by 16-inch compressors; each furnished 888 cubic feet of air at 100 pounds per square inch pressure. Shops and the main surface plant were at the hoisting shaft.

Mining. - The No. 1 shaft was sunk to the sixth level for prospecting and development. Later No. 2, a main working shaft with eight levels, was sunk to a depth of 1,000 feet.

The top of the United Eastern (called Tom Reed Extension by Moore) ore body was on the 300-foot level and the bottom was on the 850-foot level. The maximum dimensions were: 750 feet high, 950 feet long, and 48 feet thick.

The horizontal cut-and-fill method of stoping was used almost exclusively in the United Eastern ore body. In preparing for stoping, the vein was silled out on the drift level and timbered with square-sets. Cribbed manways and chutes were run on 22.5-foot centers where the vein was not over 18 feet wide. Where the vein was wider double rows of manway chutes were constructed with the same longitudinal spacing. Waste for filling was obtained from development and waste raises run into the walls between chutes. Ten-inch grizzlies were used at the shaft pockets.

17/ Moore, Roy W., Mining Methods and Records at the United Eastern Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 76, 1928, p. 71.
18/ Work cited (see footnote 17).
19/ Wartenweiler, Otto, The United Eastern Mining and Milling Plant: Trans. Am. Inst. Min. and Met. Eng., vol. 59, 1918, p. 274.
20/ North, W. O., Mill Operations at the United Eastern During 1917 and 1918: Trans. Am. Inst. Min. and Met. Eng., vol. 63, 1920, p. 548.
21/ Bagley, E. M., Operations at United Eastern Mill: Eng. and Min. Jour., vol. 119, 1925, p. 436.

Tramway. - Ore from the Big Jim shaft was transported to the United Eastern mill by a Riblet aerial tramway with 35 buckets of 10-cubic-foot capacity; the loads averaged 0.52 ton each. The terminal ore bins were 5,080 feet apart; the buckets discharged at an elevation 8 feet higher than the loading plant. The tramway easily handled 300 tons per 8 hours with two men.

Milling. - The mill capacity was 260 tons daily in 1918 and was gradually increased to 325 tons per day in 1922.

The flow sheet was similar to that used at the Tom Reed. The average recovery was 96.59 percent.

A typical screen analysis as given by Bagley,22/ is shown in table 4.

TABLE 4. - Typical screen analysis

Coarse grinding

Mesh	Marcy discharge	Classifier "drag over"	Classifier overflow
Plus 4...........	1.0	2.5	. . .
Plus 6...........	2.5	9.5	. . .
Plus 8...........	7.5	19.0	. . .
Plus 10..........	16.0	35.0	. . .
Plus 14..........	28.0	54.0	. . .
Plus 20..........	37.0	67.0	4.0
Plus 30..........	54.0	81.0	19.0
Plus 48..........	61.0	85.0	28.0
Plus 65..........	66.0	88.0	37.0
Plus 100.........	70.0	89.5	44.0
Plus 150.........	73.0	91.0	48.0
Plus 200.........	77.0	93.0	66.0
Minus 200........	23.0	7.0	34.0

Fine grinding

Mesh	Ball-mill discharge	Classifier "drag over"	Classifier overflow	Cone overflow
Plus 48........	4.07	5.43	. . .	. . .
Plus 65........	11.12	13.71	. . .	. . .
Plus 100.......	24.18	30.00	. . .	. . .
Plus 150.......	43.75	57.30	5.01	0.12
Plus 200.......	63.75	81.40	23.55	7.08
Minus 200......	36.25	18.60	76.45	92.92

22/ Work cited (see footnote 21).

Consumption of supplies was as follows:

	Pounds per ton
Cyanide............	0.713
Zinc...............	1.675
Zinc dust..........	.368
Lime...............	4.849
5-inch balls........	1.049
2-inch balls........	2.013

Power consumption was 23.021 kw.-hr. per ton.

Costs and production data. - Production and cost data at the United Eastern mine are shown in tables 5, 6, 7, 8, 9, and 10, inclusive. The tables include both the United Eastern and Big Jim ore bodies.

TABLE 5. - Production at United Eastern mine, 1917-24

Ore milled, United Eastern, tons	511,976
Value..	$10,770,606
Ore milled, Big Jim, tons........................	220,552
Value..	$ 3,804,128
Total, ore milled, tons..........................	732,528
Gross value ore milled............................	$14,558,210
Value of ore, per ton.............................	$ 19.874
Loss in tailing...................................	$ 495,065
Loss per ton......................................	$ 0.676
Values recovered..................................	$14,063,145
Total costs amount................................	$ 5,920,211
Total costs per ton...............................	$ 8.082
Income from direct operations.....................	$ 8,142,934
Miscellaneous income..............................	$ 17,941
Administration expense............................	$ 143,705
Net income from operation.........................	$ 8,017,169

TABLE 6. - Mine operating costs at United Eastern mine, 1917-24

Labor.....................	$ 2.581
Timber....................	.453
Explosives................	.502
Other supplies............	.298
Power.....................	.284
Miscellaneous.............	.214
Total.................	4.332

TABLE 7. - Milling costs in labor and supplies, United Eastern mill, 1917-24

Labor.....................	$0.507
Supplies..................	.831
Power.....................	.593
Miscellaneous.............	.063
Total.................	1.994

TABLE 8. - Average milling costs, United Eastern mill, 1917-24

General	$ 0.161
Lighting expense	.005
Water supply	.108
Coarse crushing	.055
Coarse grinding	.356
Fine grinding	.469
Cyaniding	.523
Tailing disposal	.025
Clarification	.033
Precipitation	.095
Refining	.074
Sampling	.003
Assaying	.031
Retreating skimmings	.014
Mill heating	.031
Solution recovery system	.010
Experimental work	.001
Total	1.994

TABLE 9. - Indirect cost at United Eastern, 1917-24

Superintendence	$ 0.121
Office expense	.133
Legal expense	.119
Taxes	.727
Liability expense	.080
Accident expense	.079
Fire insurance	.037
Miscellaneous	.119
Total	1.415

TABLE 10. - Summary of costs at United Eastern, 1917-24

Mining	$ 4.332
Milling	1/ 2.011
Shaft sinking	.140
Indirect	1.415
Tramway and mill addition	.062
Marketing	.122
Total	8.082

These costs are exclusive of depletion, depreciation of plant, prepaid development, federal income taxes, and litigation expenses.

1/ For a slightly longer period than covered in table.

In May 1935 an overhand open-stope method was being used to mine an ore shoot 3 to 4 feet wide in a parallel fracture in the hanging wall and 12 feet from the main vein.

Mining costs in April 1933 are shown in table 11.

TABLE 11. - Mining costs per ton, April 1933, 1,457 tons

Breaking ore	$ 1.522
Tramming	.196
Hoisting	.275
Timbering	.079
General underground	.237
Steel sharpening	.146
Compressed air	.138
Drill steel hose, etc	.047
Total	2.640

The above costs do not include track, drill repairs, tools, miscellaneous supplies, or water. Development is included in the above costs and amounted to $0.91 per ton of ore mined. A total of 85 feet of raising was done at a cost of $15.86 per foot. Total direct costs were as follows:

Mining	$ 1.73
Development	.91
Trucking, contract.	.30
Milling	2.44
	5.38

This does not include office or general expense or miscellaneous supplies on hand when operations began.

Milling. - The ore was treated in the Telluride mill 1 mile distant, the rental cost of which was $0.612 per ton. The mill was in good condition and required but little overhauling. The ore contained no sulphides and consisted of about equal amounts of calcite and quartz; it was resistant to grinding. The gold was in the quartz and unusually fine grinding was required to liberate it enough for cyanidation. The flow sheet of the mill is shown in figure 6.

Grinding was done in cyanide solution, which was drawn from the main storage tank at the rate of 330 tons per 24 hours. About 50 percent of the value recovered from the ore was dissolved in the ball mill. The classifier overflow contained 15 percent solids; it was 80 to 82 percent minus 200-mesh.

No. 1 thickener was 24 feet in diameter and 8 feet in depth; nos. 2 and 3 agitators were 13 feet in diameter and 14 feet in depth. The thickened discharge contained 40 percent solids. No. 1 agitator was 19 feet in diameter and 14 feet in depth. Air for agitation was used under 30 pounds per square inch pressure. About 95 percent of the gold recovered was taken in solution by the time the pulp left no. 3 agitator. Nos. 2, 3, 4, 5, and 6 thickeners were 15 feet in diameter and 10 feet in depth. The thickened pulp discharge contained 50 percent solids.

Big Jim

The Big Jim was acquired by the United Eastern in 1915, and an aerial tramway was erected in 1922 to transport the ore to the United Eastern mill about 5,000 feet distant.[23]

From 1921 to 1924, when the United Eastern ceased operation, 220,000 tons of ore with an average value of about $17.25 per ton was produced from the Big Jim.

The Big Jim was leased in 1932 by Rae L. Johnston and Roy S. Witcher. This was the only operation in the district at the time and until the Tom Reed started up in 1934.

The ore was treated in the Telluride mill, which had an average capacity of about 45 tons daily; the range was from 35 tons in winter to 60 in summer. In April 1933, 50 tons per day was being mined, with 10 men underground and 3 on the surface. In May 1935 the mine had difficulty in keeping the mill operating full time. In early 1936 operations in the Big Jim had become unprofitable and ore was being mined elsewhere for milling.

The Big Jim was described by Moore[24] as a fissure vein in andesite. The ore body was 450 feet high and 350 feet long and had a maximum width of 35 feet. The vein dipped 70°. The Johnston and Witcher operations were confined mainly to stoping at the ends of the old workings and mining parallel fractures.

A three-compartment shaft was sunk on the hanging-wall side of the vein to a depth of 730 feet. The ore body, which did not outcrop, was developed by five levels; the first level was above the ore and the lowest below the bottom of the shoot.

Surface plant. - Two Imperial-type compressors constituted the surface equipment one 19- by 12- by 16-inch and the other 17- by 10- by 14-inch. The capacities were 1,100 and 600 cubic feet of free air per minute, respectively. The smaller compressor, with a 75-horsepower motor, was used by Johnston and Witcher. In place of the large double-drum hoist used in the former operations the lessees installed a single-drum hoist run with a 60-horsepower, distillate-burning engine capable of raising a 1-ton skip on the cage at 250 feet per minute.

Stoping. - The original ore body was mined by the United Eastern by a horizontal cut-and-fill and shrinkage method. The ore mined by Johnston and Witcher was 3-½ to 6 feet wide and between strong walls. A block between the 300- and 600-foot levels was mined by the open-stope method with stulls to hold working platforms. Later an extension of the ore body on the 600-foot level was developed and mined by underhand open stoping. A raise was first put up to the level above, and, as stoping progressed, a second and then a third raise was started at intervals of 50 feet.

A series of benches was carried downward by drilling down-holes with jackhammers; the rill was maintained steep enough that the ore would run by gravity into the chutes. One miner would drill 15 to 20 holes and break 20 to 25 tons of ore per shift.

23/ Johnson, C. H., Mining and Milling Methods at the Big Jim Mine, Oatman, Ariz.: Inf. Circ. 6824, Bureau of Mines, 1935, p. 1.
24/ Moore, Roy W., Mining Methods and Records at the United Eastern Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 76, 1928, p. 56.

Fifty tons of fresh water were added to No. 6 thickener daily; the barren solution from the precipitation section, 180 tons daily, was added to the feed launder of No. 4 thickener. This solution contained 2.2 pounds of sodium cyanide and 2 pounds of lime per ton. Part of the cyanide solution as drawn from the mill storage tank was added to the feed launder at the head of the ball mill and the rest at the lower end of the classifier. The overflow from No. 1 thickener went to Butters clarifier filter in a 11- by 5-$\frac{1}{2}$-foot-deep tank and thence to the clear-gold solution tank 11 by 6 feet in depth.

Metallurgical results. - The recovery was 97 to 98 percent. The bullion was 640 parts gold, 300 parts silver, and 60 parts base metal and impurities. The consumption of reagents was as follows:

Sodium cyanide....................	0.75 to 1 pound per ton of ore
Lime..............................	3.75 pounds per ton of ore
Zinc dust.........................	8 pounds per day, or 0.7 ounce per ton of ore
Lead acetate......................	1 ounce per ton of ore

Ball consumption was 2.5 to 4.5 pounds chrome steel to a ton of ore.

Labor. - The mill employed the services of 1 superintendent and assayer, 1 refiner and assayer assistant, 1 crusherman (day shift only), and 6 shiftmen (two on a shift). Experienced men were paid $4.50 and ordinary labor $4 per shift.

Power. - Power cost $0.0267 per kw.-hr. (May 1935). The total installed electric power in the mill was 200 horsepower, divided as follows:

Motor	Horsepower
Crusher......................	50
Ball mill....................	75
Main conveyor................	10
Apron feeder.................	5
Compressor...................	10
Cyanide plant................	25
Pumps........................	25
Total....................	200

The crusher operated only 5 hours per day and the ball mill at about 70 percent capacity. The average power load was 90 horsepower and power consumption was 30 kw.-hr. per ton of ore milled.

Water. - Water was purchased from a nearby shaft. The cost was $100 per month; for 50 tons of water daily the unit cost would be about 30 cents per 1,000 gallons.

Milling costs. - The milling costs for July 1933 are shown in table 12.

TABLE 12. - Milling cost, Johnston and Witcher lease on Big Jim mine, July 1933 (1,635 tons milled)

	Labor		Electric power		Water		Lime		Miscellaneous chemicals		Cyanide		Zinc	
	Total	Per ton	Total	Per ton	Total	Per ton	Total	Per ton	Total	Per ton	Total	Per ton	Total	Per ton
Assaying 1/..	$283.00	$0.173	$10.15	$0.006	.....	......	....	......	$20.11	$0.012	......	......	.....	......
Crushing....	140.13	.086	30.43	.018	.....	......	....	......	......	......	......	......	.....	......
Grinding....	393.88	.241	507.26	.310	.....	......	....	......	29.10	.018	......	......	.....	......
Cyaniding...	425.19	.260	456.54	.279	$78.61	$0.048	$116.51	$0.071	4.91	.003	$172.62	$0.106	$32.00	$0.020
Repair and replacement	......	......	......	......	.....	......	....	......	......	......	......	......	.....	......
Refining and marketing..	31.50	.019	10.14	.006	.....	......	....	......	......	......	......	......	.....	......
Rental of mill	......	......	......	......	.....	......	....	......	......	......	......	......	.....	......
Total......	1,273.70	.779	1,014.52	.620	78.61	.048	116.51	.071	54.12	.033	172.62	.106	32.00	.020

	Replacement and repair parts		Balls		Fuel oil and lubricants		Insurance and shipping		Mill rental and miscellaneous		Total cost of milling	
	Total	Per ton	Total	Per ton	Total	Per ton	Total	Per ton	Total	Per ton	Total	Per ton
Assaying 1/...	......	......	....	......	$9.99	$0.006	.....	......	....	......	$323.25	$0.197
Crushing.....	......	......	....	......	.....	......	.....	......	....	......	170.56	.104
Grinding.....	$58.25	$0.036	$130.77	$0.080	2.02	.001	.....	......	....	......	1,121.28	.686
Cyaniding....	19.92	.012	....	......	4.50	.003	.....	......	....	......	1,310.80	.802
Repair and replacement.	......	......	....	......	.....	......	.....	......	....	......	....	......
Refining and marketing...	......	......	....	......	10.05	.006	$8.64	$0.005	....	......	60.33	.036
Rental of mill	......	......	....	......	.....	......	.....	......	$1,000.00	$0.612	1,000.00	.612
Total.......	78.17	.048	130.77	.080	26.56	.016	8.64	.005	1,000.00	.612	3,986.22	2.437

1/ Cost of assaying includes cost of superintendence.

Gold Road

This mine was discovered in 1902 and soon after the Gold Road Mining & Exploration Co. was incorporated to work it. According to Schrader,[25] 180 men were employed on the property in 1908. The mine was sold to the Needles Smelting and Refining Co., a subsidiary of the United States Smelting Co., in 1911. The known ore bodies were exhausted in 1916, and the mine was closed. It was again worked on company account in 1922 and 1923. Some prospecting was done and some ore produced intermittently by lessees in 1926 and 1928 and again in 1929 above the water level, which is at 300 feet.

In May 1935 the mine was being reconditioned by the company and about 1,000 tons monthly was being produced from rehabilitation work, sorting over of old dumps and leasing operations, and trucked to the Tom Reed mill. The mill at the Gold Road had been junked. The mine was being pumped out for the first time since 1917.

As shown in table 1, $7,250,000 had been produced from the mine up to the end of 1931.

Geology. - The Gold Road vein dips about 85° and occupies a fault in latite. It is a stringer lode with a prominent outcrop nearly 100 feet wide in places. The vein filling that occurs in two zones separated by barren latite consists mostly of chalcedonic quartz. Most of the ore mined has come from three shoots on the north strand of the vein. The largest shoot was 900 feet long by a maximum width of 22 feet and extended from the surface to the 700-foot level. The Sharp ore body, which was 600 feet to the southeast, was mined from the 300- to 500-foot levels and on the 800-foot level. The Rice ore body, about 200 feet farther southeast, was 400 feet long and occurred between the 300- and 500-foot levels. The Lime Road shoot was higher up the mountain and several hundred feet above the Rice ore body. Most of the ore was from 2 to 10 feet wide and occurred between good walls. According to Schrader, the ore mined in 1908 ran $10 to the ton.

Mining. - The mine has been developed to a depth of 800 feet by two shafts, an adit, and levels at the 300-, 500-, 600-, and 700-foot levels.

In May 1935 a plan for one year's development was in force. Two drifts on the vein in virgin territory were being run and the old workings were cleaned out as the water was lowered.

The main oreshoots have been worked by the shrinkage method of mining. Most of the new production in May 1935 came from leasing operations in open cuts on the outcrop of the Lime Road shoot and from development work. The Lime Road ore would have to be trucked in any event either to a custom mill or a new mill should development show enough ore to justify building one on the property. (Recent reports from the Kingman district state it is known from authentic sources that the results of development have justified the erection of a 300-ton mill.) An old shrinkage stope left in 1906 was also being pulled. At the old price of gold it did not pay to hoist this.

25/ Work cited (see footnote 3).

The cut-off point of the ore in the early workings was 0.3 ounce of gold. With gold at $35 per ounce, material of this grade would be worth $10.50 per ton and could be worked at a profit. It was expected that a part of the vein between the old stopes can now be worked.

Plans for further stoping call for open stopes, shrinkage, and cut-and-fill methods; the method to be used at any one place will depend upon ground conditions and the need for selective mining.

Milling. - The old mill originally had a capacity of 200 tons daily, which was increased to 400 tons toward the end of its life. The ore was cyanided; grinding was done at first with Huntington mills and later with ball mills. Water for milling was pumped from the mine and from a well at Little Meadows on the east slope of the range.

Ruth-Rattan

In May 1935 the Oatman Eastern Mining Co. was mining 1,200 tons per month (30 days) from this mine and shipping it to the Tom Reed mill. Fourteen men were employed.

Prior to 1907 several hundred tons of ore had been taken from the mine and were milled at Hardyville. The mine was worked during the Oatman boom; a shaft was sunk, levels were run, and a stamp mill with electro-amalgamation was built. According to Ross Barkley, mine superintendent, about 25,000 tons were mined on the 100-foot level and milled. The ore body was cut off by a fault. The tailing from the old mill contains $1.70 per ton (gold at $20 per ounce).

In 1933 Barkley and two partners obtained a bond and lease on the property, found the ore on the other side of the fault, and during 1933 and 1934 shipped $25,000 worth of $14.70 ore to the Tom Reed mill. A shipment of 100 tons was made soon after the property was taken over from above the 100-foot level near the shaft. The option was turned over to the present company in 1935.

Geology. - The country rock is porphyry. The vein filling is breccia, calcite and quartz; the dip is 60°. The vein is cut by a fault that crosses the shaft at a sharp angle. The part of the ore shoot mined at the time of the Oatman boom was 3 to 5 feet wide and 86 feet long.

The ore shoot opened up by the present operators averages 12 feet in width on the 200-foot level; the range is 8 to 15 feet. In May 1935 it had been drifted on for 150 feet; a 6-inch streak of ore in the face assayed $50 per ton in gold.

Shipments from development and stope preparation during the first 3 months were as follows: February, 500 tons at $9.45; March, 900 tons at $13, and April, 1,200 tons at $14.

The ore shoot directly above on the 100-foot level is 80 feet long; it was stoped to a width of 3 feet. On the 500-foot level the vein pinches to 2-½ feet.

Mining. - The mine is developed by a 60° incline shaft and drifts on the 100-, 200-, and 300-foot levels. In May 1935 over 600 feet of drifting had been done.

The mine had but one entrance. Ventilation was supplied through 10-inch galvanized-iron tubing with a branch line into the new stopes on the 200-foot level.

In May 1935 the equipment consisted of a 40 horsepower gasoline hoist, a 309-cubic foot compressor run by a 50 horsepower gasoline engine, and a no. 5 fan run by a gasoline engine. Mining done during the Oatman boom and by Barkley was by open-stope methods.

In May 1935 preparations were being made to mine the ore on the 200-foot level by a shrinkage stope, but the chutes had not been built.

Water was bailed from the shaft just below the 200-foot level with a 1,000-pound bucket. The water in the shaft was lowered 20 feet in 6 hours of bailing and came back in 15 to 18 hours.

Costs. - Gasoline cost 10 cents a gallon delivered (without State tax). The gasoline bill for April 1935 was $350. The wage paid in May 1935 was $5 for all workmen except the hoist engineer, who received $6 per day. Mine costs in April 1935 were $3.50 per ton.

Mossback

The Mossback mine is about 7 miles northeast of Oatman in rolling country at an elevation of 2,400 feet. The Mossback claim was located in 1863. Prior to 1907 the mine was developed by a 330-foot shaft, in which the water stood at the 170-foot level. The shaft was sunk to the 800-foot level in 1918 and 1919. M. B. Lauzon worked the mine from 1927 to 1935 under a lease and bond from the Empire Consolidated Gold Mining Co. The Mollin Mining Co. became interested in the mine in May 1935 and began development work which continued until the end of the year. On his last inspection in 1935 the State mine inspector reported 14 men employed.

Geology. - The Mossback vein is in andesite and up to 90 feet wide; it dips at 80°. The vein filling consists of brecciated andesite and calcite with a subordinate amount of quartz. A 65-foot crosscut on the 300-foot level and a 164-foot drift on the 400-foot level disclosed calcite throughout. The gold occurs in fractured areas in the veins; the ore is bunchy, and values are spotty. An irregular ore shoot with an average cross section of 14 by 20 feet corkscrews upward on the 400-foot level. Eighty-six tons from this shoot milled $12.88 ($35 gold) per ton in gold. A total of 435 tons of ore taken from the 65-foot crosscut averaged $10.85 in gold per ton.

Production. - The mine is credited with no production up to 1933, during which year Lauzon mined and milled 200 tons. A 15-ton mill was run intermittently up to the spring of 1935. Lauzon reports a gross production of $15,000 in gold prior to May 1935. From May to December 1935, 1,309 tons of ore containing an average $7.14 in gold per ton was produced from development work.

Development. - The mine is developed by an 80° incline shaft and 2,000 feet of drifts and crosscuts. Work from 1932 to 1935 was done on the 400-foot level and the levels above. During 1935, 325 feet of development work was done on the 500-foot level in addition to work above.

Equipment. - The mine equipment in May 1935 consisted of a compressor run by a 75- horsepower gasoline engine and a 40- horse power gasoline hoist. An auxiliary 7 1/2- by 6-inch compressor also was run by the hoist engine. The hoisting equipment had a capacity of 25 tons per 8 hours.

Stoping. - No systematic stoping had been done up to the end of 1935. The small ore shoots that had been found had been gouged out as open stopes. In March 1933 a stope 8 feet wide and 15 feet high was being worked near the shaft on the 400-foot level. The ore was broken down in benches or by rounds drilled from the top of the pile of broken ore. Drilling was done with either a stoper or mounted drifter. Eighty-eight tons of ore were stoped by one man in 15 shifts. Sixty-four holes that averaged 4-1/2 feet deep were drilled in eight shifts; seven shifts were required to do the shoveling and tramming. No deadwork or sorting was done. One hundred thirty-four buckets of 1,300 pounds each were hoisted. The direct cost was as follows:

	Amount	Per ton
Labor, mine, 15 shifts at $4.50		
Hoist engineer, 5 shifts at 4.50		
Total labor, 20 shifts..............	$90.00	$ 1.02
73-1/2 pounds explosive at 15 cents...	11.02	
Fuse, 330 feet at 1 cent............	3.30	
Caps, 65 at 3 cents.................	1.98	
	16.30	.19
Fuel, 110 gallons at 6 cents........	6.60	.08
Other supplies......................	8.80	.10
Total..........................	$ 121.70	$1.39

Milling. - A mill equipped with a jaw crusher, Hardinge ball mill, and amalgamation plates was used in 1933. In the latter part of the year a mill of 15 tons daily capacity was rebuilt. The mill was started when enough ore for a run had accumulated in the bins. The flow sheet was rearranged in 1934 as shown in figure 7. The mill as remodeled by the Mollin Mining Co. is shown in figure 8.

The ore was stored at the collar of the shaft in a 60-ton bin divided into two compartments of 40 and 20 tons capacity, respectively.

A 1-inch grizzly, 30 inches wide and 12 feet long, was set over the larger compartment; the undersize fell into a mine car. About one-eighth of the ore passed through the grizzly. It was stated that the fines generally were much lower grade than the coarse; they were either milled or dumped on the waste pile, according to the need for ore and their probable value.

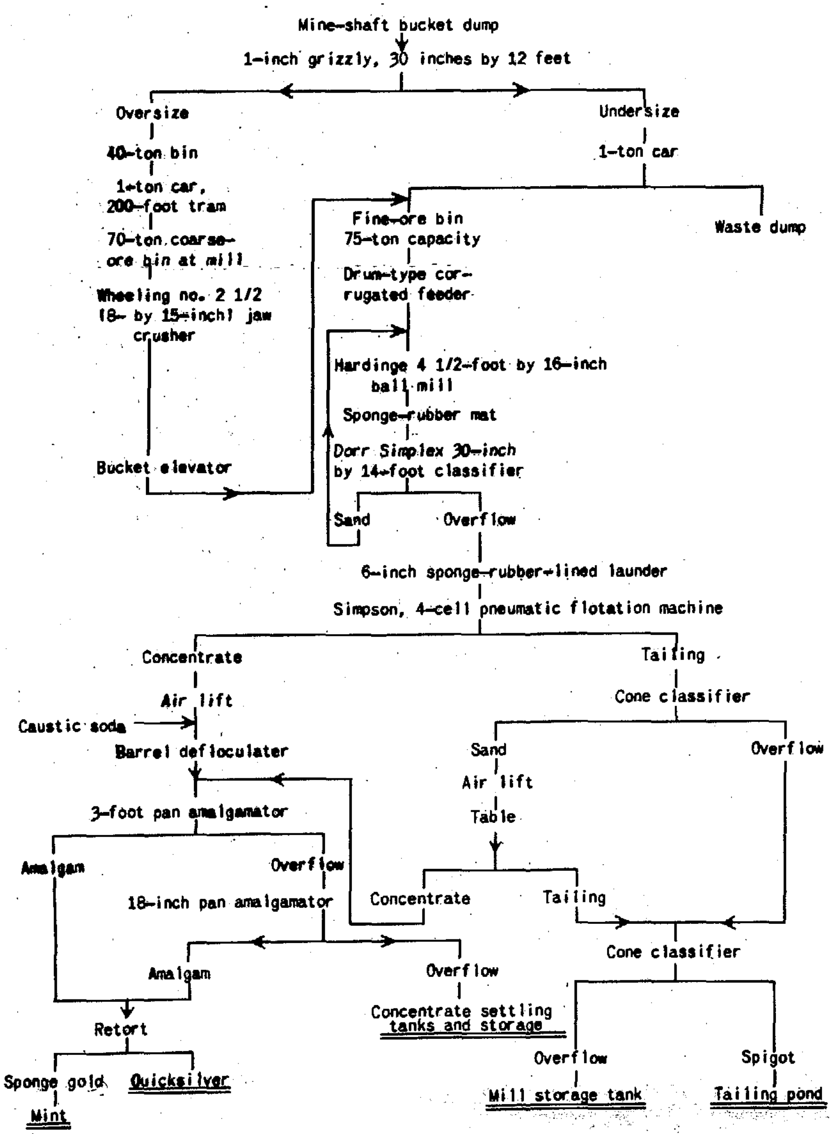


Figure 7.- Mill flow sheet, Mossback mill, 1934.

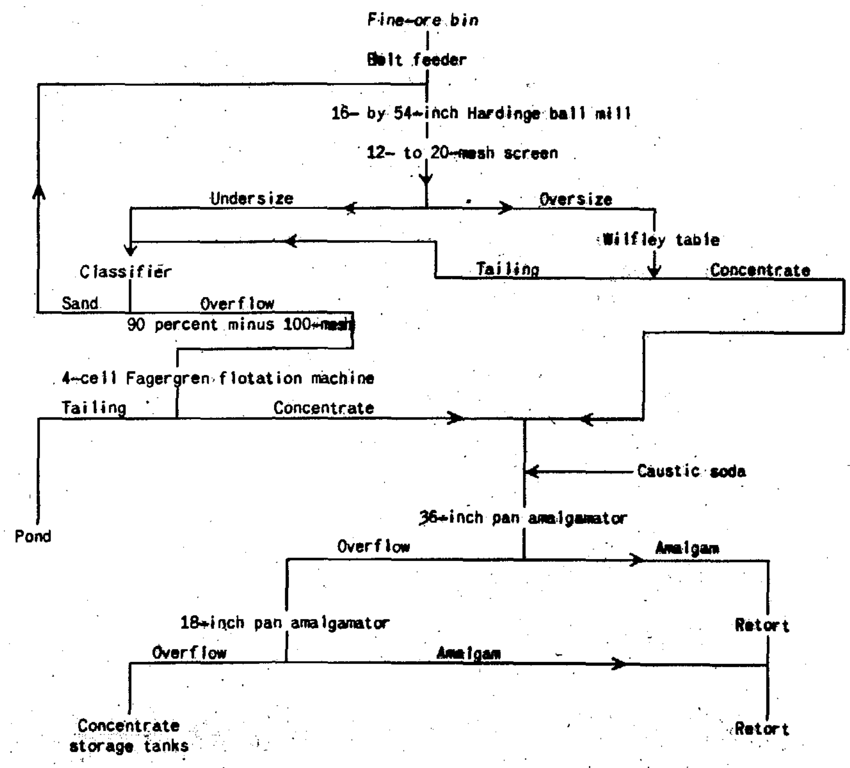


Figure 8.- Mollin Mining Co. flow sheet, Mossback mill, 24 tons per day.

From the fine-ore bin a belt feeder fed the ore to a 4½ foot by 16-inch Hardinge conical ball mill turning at 33 to 34 r.p.m. The ball mill was run by a 40-horsepower motor; the actual load was about 25-horsepower. The ball mill was said to have a capacity of 1 ton per hour on ore from near the surface but only about 0.8 ton per hour capacity on ore from the 400-foot level, which contained a higher proportion of quartz to calcite. The classifier was set at a slope of 2-3/8 inches in 12 and was run at 26 strokes per minute. The classifier overflow was about 95 percent minus 65 mesh.

The sponge-rubber mat under the ball-mill discharge was washed off in a tub of water every 2 hours, and the rubber lining of the flotation-feed launder once each day; these concentrates were treated in a large pan amalgamator. About 50 percent of the gold content of the ore was caught on the sponge-rubber and the rubber mats.

Several combinations of flotation reagents had been tried. From 0.01 to 0.02 pounds of American Cyanimid reagent No. 301 per ton of ore was added in the classifier. Copper sulphate was added at the rate of a teaspoonful every 15 minutes. Ammonium phosphate added at about the same rate was believed to improve silver recovery appreciably. P. E. collector (Southwestern Engineering Co.) was being added in the ball mill and pine oil in the flotation-feed launder at the time of the visit, but this combination was said to be less effective than a previous use of Aerofloat no. 15 (0.005 lb. per ton of ore) and American Cyanimid B 22 frother (an alcohol). The latter gave a brittle froth and, being nonoily, did not interfere with amalgamation. Air at 3 pounds per square inch pressure for the flotation cells was supplied by a small blower. A finished concentrate was taken from the first three cells, the froth from the fourth cell being returned to the head of the machine.

The flotation concentrate was raised by an air lift to a tank made of a steel barrel, where a very small stream of saturated solution of caustic soda was added to break down the froth. The overflow from the barrel passed by gravity to the larger of two pan amalgamators, thence to the smaller, and finally to a settling tank. The concentrate was stored for future treatment or shipment to a smelter.

The amalgamators were said to be removing 60 percent of the gold from a 5-ounce feed, leaving a shipping product assaying about 2 ounces of gold per ton.

The scavenger table made a small quantity of concentrate containing 1.5 ounces of gold per ton; this was fed periodically by hand to the amalgamators.

Mill water supply came partly from pumping from the mine and partly by gravity through a pipe line from a spring in the foothills about 2 miles away. The mill water was stored in a 24,000- and a 11,000-gallon tank. About 8,000 gallons of fresh water was needed daily when the mill was operating two shifts. From 20 to 25 percent of the water in the tailings was returned from the dewatering cone to storage by a centrifugal pump driven by an electric motor that was turned on and off automatically by a float-operated switch. Enough water is available to treat 15 tons daily in summer and 25 tons in winter in the mill with the new flow sheet (fig. 8).

Amalgam was retorted over a wood fire and the sponge shipped to the mint without melting or refining. Although the ore contained about equal weights of gold and silver, the mint return showed only 1 ounce of silver to 3 of gold in the bullion.

Power for operating the mill was purchased from the public utility company. The mill required an operator on each shift. The toplander at the mine tended to the crushing on day shift. No operating data are available for the mill after the flowsheet was changed, as shown by figure 8, except that two-thirds of the recovered gold was in the amalgam.

Pioneer or German-American

The Pioneer, formerly called the German-American, mine is 1-½ miles southwest of Oatman in the foothills of the main range. The Gold Roads Co. did some work on the property in 1902. During 1903 to 1906 the German-American Mining Co. produced 2,700 tons of ore that averaged \$10 per ton in gold. In 1925, 312.4 tons of ore with a value of \$6,051 was produced by lessees from the property.[26]

In May 1935 the property was held by the Amulet Mines, Inc., and was worked by 12 sets of lessees (23 men). Between 400 and 500 tons was mined monthly. The ore was treated at the Tom Reed mill, the value ranging from \$16 to \$35 per ton; the average was a little over \$16.

Geology. - The Pioneer vein, which also goes through the Treadwell and 35th parallel patented claims, dips 80° to the east. The hanging wall is Oatman andesite, and the footwall for 2,000 feet is trachyte. The vein intersects the Gold Dust-Boundary Cone fault at an angle of 40° but neither vein is offset by the other. The maximum width of the vein is 18 feet, at a point near the Pioneer shaft. The vein material here is coarsely grained gray calcite and quartz.

Several small ore shoots were mined near the southern end of the Pioneer vein. One of them, near the Treadwell shaft, was 400 feet long and 3-½ feet wide. The ore consisted of quartz and calcite. At the Pioneer shaft near the northern end of the property narrow parts of the vein assayed more than \$10 (at \$20 gold) and yielded some rich ore near the surface.

The vein is strong to the 400-foot level, where developed, but no ore has been mined below the 200-foot level. Most of the ore produced is along a slip in the vein. A number of small ore shoots up to about 40 feet in length occur along the Pioneer vein at the surface.

Development.[27] - The underground development is as follows:

Pioneer No. 1 shaft: 420 feet deep with 400-foot drift on 100-foot level, 1,150 feet of drifts and crosscuts on 200-foot level, and 1,230 feet of drifts and a 70-foot winze on 400-foot level.

26/ From report on mine by Geo. F. Moser, Oatman, Ariz.
27/ Moser, Geo. F., work cited (see footnote 26).

Pioneer no. 2 shaft: 50 feet deep with short drifts.

Treadwell shaft: 340 feet deep with 405 feet of drifts on 100-foot level, 150 feet of drifts on 200-foot level, and 25 feet of drifts on 300-foot level.

35th Parallel shaft: 220 feet deep, with connection with adit on 150-foot level and two raises to surface.

Snowball shaft: 90 feet deep.

In addition numerous open cuts have been dug on the property.

Equipment consists of a 22-horsepower gasoline hoist and 300-cubic foot compressor run by an 80-horsepower gasoline engine at no. 1 shaft.

Mining. - The ore produced in 1903 and 1906 was mined underground in open stopes. The work being done by lessees in 1935 was in open cuts or from shallow shafts that had been sunk by the lessees. The leases were at the surface for blocks of ground beginning at the surface along the Pioneer vein and on sections of other veins on the property. The leases being worked in May 1935 were verbal only. The royalty charged is shown in a previous section under the heading "Royalties." The methods used are shown in the following representative leases.

Ferra lease on Treadwell claim. - E. J. Ferra and three partners started work in November 1934 and began shipping in January 1935; up to May 1935, 270 tons running about $14 per ton had been shipped to the Tom Reed mill.

The vein at this point is about 4 feet wide. The vein filling is calcite with some black hematite. The ore occurs in streaks 6 to 12 inches wide.

A stope was started on the surface and up to May 1935 had been mined underhand over a length of 40 feet and to a depth of 30 feet. A part of the stope is under an old drift run in the vein and partly cutside in front of the portal of the drift. About half of the material broken was sorted out and discarded. Control of the grade was obtained by a large amount of panning, all work being done by hand; hoisting was done by means of a hand windlass.

Bridges lease on Pioneer: - The lessee found a high-grade shoot of ore in a strand of the main vein at the surface to the south of the no. 1 Pioneer shaft. The first ore was mined in an open-cut and from a short adit. In May 1935 he was mining underhand below the floor of the adit at a depth of 20 feet. Drilling was done with a jackhammer. Air was supplied by means of a small portable gasoline compressor. Hoisting was done in a 300-pound bucket. The hoist consisted of an old automobile with one rear wheel replaced with a small drum. Lowering was done in gear; breaking was done by the compression in the engine. The car was set facing the working.

United Western

The United Western is on the Oatman-Kingman highway and adjoins the United Eastern on the east. According to Frank Waring of Oatman, the mine was operated in 1927 and 1928 by the Consolidated Gold Mining Co. About 12,000 tons was produced from the 700-foot level. A little ore also was produced at this time on the 850-foot level through the United Eastern shaft.

From March 1 to May 1, 1935, about 2,000 tons was produced from above the 500-foot level by the same company. The water stands at 560 feet in the western shaft, which was not pumped out. A connection has been made with the United Eastern workings, which was under water during the last operation. At the time of the author's visit only a watchman was on the property.

The ore was treated in the Tom Reed mill 2 miles distant. The trucking charge was $0.35 per ton. The ore was loaded from chutes.

The Consolidated Gold Mining Co. also had the Lexington and Del Rey mines. Lessees were working the Lexington.

The vein in the United Western is reported to be a spur of the Tom Reed-United Eastern vein. It is 3 to 12 feet wide, averaging 6 feet. The vein material is massive calcite with a little quartz; the values occur mostly with quartz. Development work consists of a 735-foot shaft with levels at 300, 500, and 700 feet. The 850-foot level, which connected with the United Eastern workings, was reached by a winze from the 700-foot level.

Lexington. - The Lexington mine is 2 miles west of Oatman at Tex's Camp. A little ore was shipped in to the Tom Reed mill during the summer of 1934. In May 1935 four lessees were working in the mine but had not begun to ship. The mine is developed by a 375- and a 200-foot shaft connected on the 250-foot level of the deeper shaft. The vein is reported large, the ore hard, and the walls soft. The ore occurs in small bunches in the vein; that shipped in 1934 averaged about $9 per ton.

Telluride

The Telluride mine adjoins the Tom Reed on the south. It was active from 1922 to 1925 and was operated in a small way from 1930 to 1934. The estimated production to the end of 1933 was about $200,000. In May 1935 140 tons per month were being mined by the Oatman Associates Mining Co. and trucked to the Tom Reed mill for treatment. The mill, which was built to treat the Telluride ore, was under lease to Johnston and Witcher. The Oatman Associates Mining Co. started development work in January 1934. Shipments, starting with 35 tons per month, were begun in January 1935.

The Telluride vein, which joins the Tom Reed vein south of the Ben Harrison shaft, has an inconspicuous outcrop and is about 3,000 feet long. The vein is about 3 feet wide. An ore shoot 200 feet long on the 300-foot level was being worked in 1935. The ore consisted of 4 to 10 inches of calcite (no quartz) frozen on the hanging wall. Production has been between the 300- and 500-foot levels. Some development work was being done on the 700-foot level, but the vein was too low grade to mill.

The mine is developed by a 700-foot shaft and four levels. It is connected with the Tom Reed underground.

In May 1935 a cut-and-fill method of stoping was being used. The stoping width was 3 feet - the distance between walls.

Midnight

The Midnight mine is 2-1/7 miles northwest of Oatman. A moderate production was made between 1900 and 1907. The property was then idle until 1935, when Hautier and Waters began leasing operations. Up to May 165 tons, which averaged $17 per ton, was shipped to the Tom Reed mill. It is reported that some ore from this mine was milled in the Leland mill after that mine closed down in 1904.

The Midnight vein, which is the footwall member of a wide lode, dips 45° to the west. It occurs in the Oatman andesite and probably also in trachyte. The vein filling consists of quartz, calcite, and fluorite. Development work consists of a 300-foot 45° shaft with three levels.

The hoist was run by an old automobile engine, and the ore was hoisted in a 600-pound bucket. Three and one-third gallons of gasoline was used per 8-hour shift. Compressed air was supplied by a 120-cubic foot portable compressor. A rental of $75 monthly was paid for the compressor; these payments, however, could be applied on the purchase price.

The old mining was done underhand from the 50-foot level. The present work is underhand from the 100-foot level. A crosscut was being run in May 1935 to get the junction of two leads where an ore shoot was expected.

Principal Nonproducing Mines, May 1935[28]

Moss. - The Moss mine is about 7 miles northwest of Oatman and 2 miles north of Silver Creek. It was one of the first mines to be worked in the Black Mountains. About $240,000 in gold was produced in the early days. Since that time considerable intermittent development work has been done but little ore has been mined. The vein, which can be traced for over a mile on the surface, dips 80° to the south, is 20 to 100 feet wide, and occurs in quartz-monzonite-porphyry. The vein filling consists of fine-grained white quartz and calcite with stringers of fluorite. The principal ore shoot extended to a depth of 65 feet.

Development workings include a 230-foot shaft with about 750 feet of drifts and crosscuts, about a 900-foot adit, and irregular surface openings. Ransome states that the vein on the 220-foot level appears to be 90 feet wide and carries from 0.15 to 0.2 ounce of gold to the ton.

Gold Dust. - The Gold Dust, formerly known as the Victor-Virgin and the Orion, is about 1 mile southeast of the Ben Harrison shaft. The property, which was located in the early days, produced a small amount of shipping ore prior to 1907. Some ore was produced in 1923, 1926, and 1932.

The vein consists of solid quartz and calcite with a width up to 7 feet. It splits to the northwest into a series of small stringers. Two ore shoots were mined, the largest having a length of 200 feet and extending from the surface to a depth of 160 feet. The mine has been developed to a depth of 500 feet.

Leland. - The Leland mine is about 2 miles west of Oatman and was one of the early mines worked in the district. Some high-grade ore was produced in 1902. In 1903 a 17-mile railroad was built and a 40-stamp mill erected at Milltown about one-half mile from the Colorado River. The venture proved unsuccessful, and the mine was closed in 1904. It is reported that $40,000 was produced from 4,500 tons of ore.

28/ Abstracted from Wilson, work cited (see footnote 4).

Two veins, the Leland and the Mitchel, occur on the property. The vein filling consists of quartz and calcite. It is reported that a large tonnage of $3 to $4 ore (at $20 gold) has been indicated.

A shaft over 700 feet deep has been sunk on the Mitchel vein.

Sunnyside. - The Sunnyside mine is about 1-½ miles southwest of Oatman at the edge of the district. The mine is on a fault. A small ore shoot was mined in 1928 from the 500-foot level. The ore consisted of quartz with a very little calcite. Some production was made by lessees in 1929 and 1930.

Iowa mine. - The Iowa mine is in the southern part of the district about 3 miles south of Oatman. Prior to 1907 it was developed by a 200-foot shaft and 100 feet of drifts. A little further development was done in 1916. The vein occurs in trachyte. Schrader states that gold values occur in a 3-foot width adjacent to the hanging wall.

Hardy. - The Hardy mine is about 5 miles northwest of Oatman. It is reported to have yielded rich ore near the surface during the early days. Prior to 1907 it was explored to a depth of 300 feet by several shafts, over 1,200 feet of drifts, and several hundred feet of adits. The vein, which occurs in granite porphyry, ranges in width from 2 to 30 feet. The vein filling is quartz, calcite, and fluorite.

Gold Ore. - The Gold Ore property, about three-fourths mile northeast of the Gold Road mine, has had some small production at various times from 1918 to 1926. The country rock is latite. The vein occurs in a fault fissure; the dip is 82°. The vein material is mostly quartz. The mine is developed by an 800-foot shaft with levels at 100-foot intervals. The ore mined has come mostly from between the 300- and 600-foot levels.

Katherine or Union Pass District

The location of the principal mines in the Katherine district is shown in figure 1. The geology is shown in figure 9 (from Lausen).

Gold Standard Mines Corporation

The Katherine mine and mill are 1-½ miles east of the Colorado River and 33 miles from Kingman. The collar of the shaft is 990 feet above sea level and 450 feet above the river. The water table in the shaft is at the 350-foot level.

As stated in the chapter on "History," the Katherine mill and water rights were purchased by the Gold Standard Mines Corporation in 1933. The company also acquired the Roadside and Arabian Mines at that time and later the Minnie and Frisco.

The Minnie-Sheeptrail-Boulevard group was reported under lease in May 1935. The Portland, up the river, about 12 miles as the crow flies was acquired late in 1935. Ore from the Roadside, Arabian, Minnie, and Frisco, together with custom ore, was being treated in the Katherine mill in 1935. In January 1936 the development ore shoot at the Roadside had been exhausted, and the mine was closed.

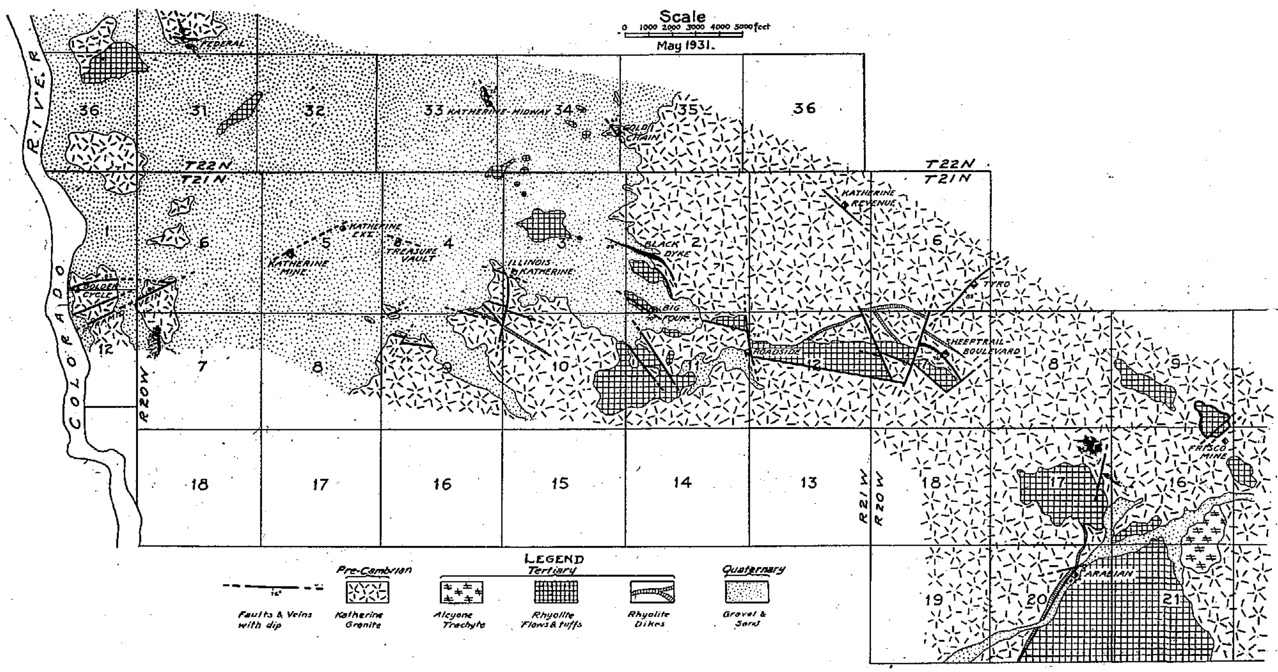


Figure 9.- Geologic map of the Katherine district.

At that time about 100 tons daily was being shipped from the Portland, 60 tons of custom ore was being received from the Tyro, and the remaining tonnage required to bring the mill up to capacity came from the Arabian (or adjoining Philadelphia), Minnie, and Sheeptrail mines.

Katherine mine. - The Katherine mine and mill were being operated by the Katherine Gold Mine Co. in 1927; it has been described by Dimmick and Ireland.[29]

The vein at the Katherine occurs in pre-Cambrian granite, which is largely buried under surface wash. It is nearly vertical and consists of a stringer lode that is about 60 feet wide at the surface but narrows underground. The vein filling consists of closely spaced stringers of quartz and calcite in the granite. At some places the vein is solid quartz and calcite up to 10 feet thick; the quartz is more abundant than calcite. The gold is finely divided in the quartz. The lode has been opened to a depth of 900 feet and for 1,700 feet along the strike. The ore shoots were found between the 400 level and the surface.

The mine is developed by a 900-foot shaft with levels at 100, 200, 300, and 400 feet.

Shrinkage was the principal method of stoping followed; when the walls were weak the cut-and-fill method was used. Shrinkage stopes 700 feet long and 200 feet high were worked successfully. The mining cost was $1.50 per ton with shrinkage stopes and $2.25 per ton with cut-and-fill stopes. The total cost of operation was:

General expense.........	$0.59
Milling.................	1.11
Mining..................	1.63
Mine Development........	.62
Total..............	3.95

Katherine mill.

History. - Construction of the Katherine mill was begun January 1, 1925, and it was put in operation on June 29 of that year. The cost of the mill as it stood in 1927 was $95,000, or $365 per ton of daily capacity (260 tons). The metallurgical data and costs in 1927 reported by Dimmick[30] follow.

Consumption of power balls and reagents per ton of ore was:

Power, kw.-hr.	16.0
5-inch balls, pounds....	2.0
2-½ inch balls, pounds..	2.2
Lime....................	3.0
Water, ton..............	.62
Zinc dust per ton solution, pounds......	.07

29/ Dimmick, R. L., and Ireland, Eugene, Mining and Milling at the Katherine Gold Mine: Eng. and Min. Jour., vol. 123, April 1927, pp. 716-720.
30/ Work cited.

The ore was ground to 57 percent minus 200-mesh. The tailing loss averaged \$0.15 per ton of ore of undissolved gold and \$0.045 for dissolved gold. Milling costs were:

Labor	\$0.359
Supplies	.462
Power	.291
Total	1.112

The mill was shut down in 1930; it was reopened for a short time in 1932 and 1933 but soon closed again for lack of ore. E. F. Nieman and associates purchased the Katherine mill from the Federal receiver soon after it closed, formed the Standard Gold Mines Corporation, and began to ship ore from their Roadside mine on November 13, 1933.31/ One year later a fire wiped out the crushing plant, power house, ore bin, head frame, and timber of the Katherine shaft down to the water level. The fire started in the Diesel plant the day before it was planned to cut in electric power from the Utility Co. The plant was again put in operation on November 26, 1934.

Ore treatment. - Standard countercurrent cyanide practice is followed, except that, due to clarifying trouble, the solution from No. 2 thickener rather than No. 1 thickener goes to the precipitation plant. In May 1935 from 230 to 250 tons per day were being treated. The flow sheet of the mill as it was at that time is shown in figure 10.

The decantation system consists of four Dorr and one Hardinge superthickeners at the end of the line. Two 4-inch Dorrco pumps are used on each tank. An auxiliary gasoline engine is installed to supply power to the agitators should the electric power go off. Water is added to the tails for sluicing them to a tailing pond.

The ore was in contact with cyanide solution 48 hours. The mill was cleaned up weekly.

Water. - Water is pumped through a 5-inch pipe from a well sunk at the edge of the Colorado River 2 miles distant. The rise in elevation is 445 feet. An average of 100,000 gallons was pumped daily in May 1935 on a 10-hour shift. The supply for the mill and camp is stored in three 18,000- and two 32,000-gallon tanks.

Power. - Power is purchased from the public utility company at Kingman by way of the Moss mine. It is conducted at 44,000 volts and stepped down to 440 volts at the mill. Three 150 kv-a transformers are at loaded capacity. Over 500 horsepower is used in the mill. The average rate is 2 cents per kw.-hr. The motor set-up in the crushing and grinding plant is as follows:

31/ Banks, Leon M., Gold Mines at Katherine: Explosives Eng., May 1935, p. 147.

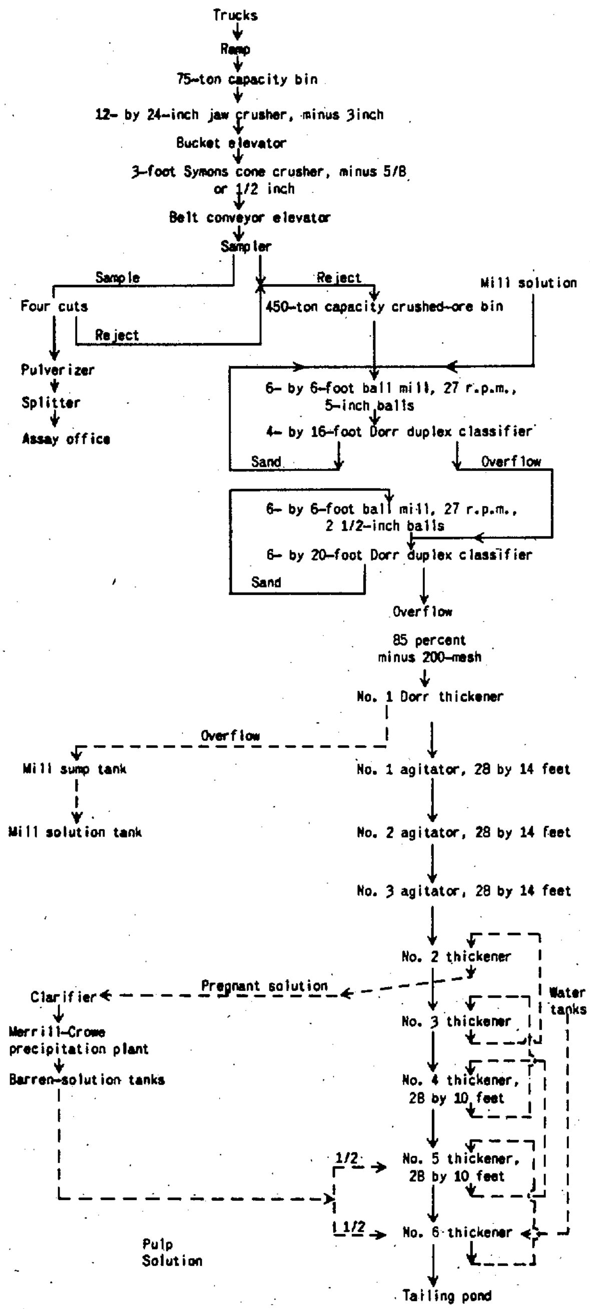


Figure 10.- Flow sheet, Katherine mill.

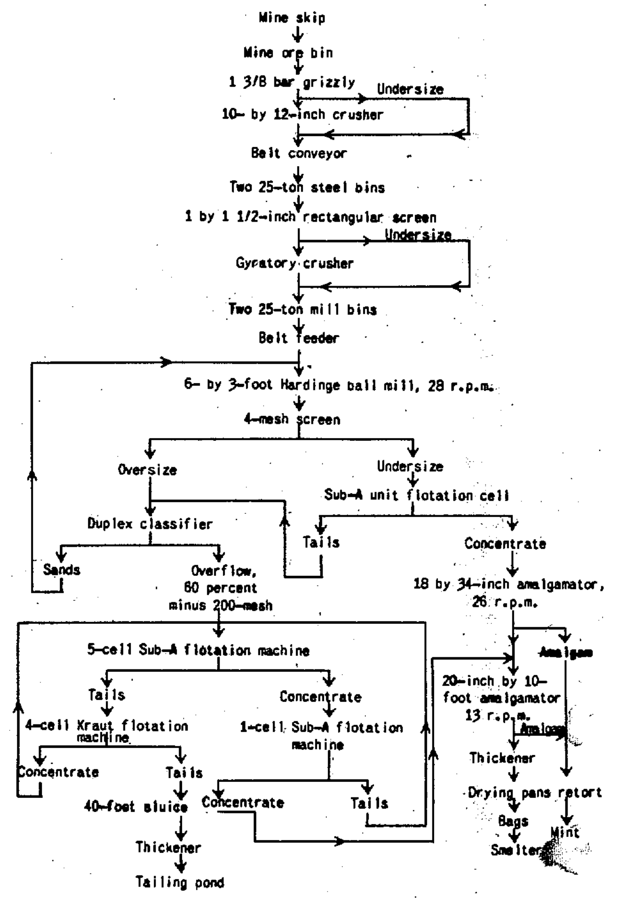


Figure 11.- Flow sheet, Pilgrim mill, capacity 75 tons per day.

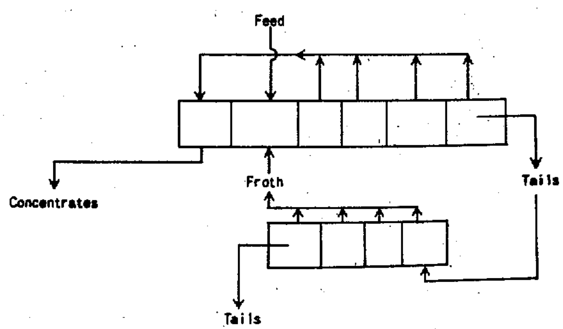


Figure 12.- Flotation circuit, Pilgrim mill.

	horsepower	
Crusher	40	
Symons	60	
Primary ball mill	125	
Secondary ball mill	125	
Belt conveyor	10	
First Dorr classifier	2	
Second Dorr classifier	5	
Bucket elevator	5	
Sampler:		
Rolls	20	
Other sampling equipment	3	23
Total		395

The motor set-up in the cyanide plant is as follows:

Agitators and two 5-horsepower pumps	10	
Thickener and first agitator	5	
Second thickener	5	20
Air compressor, 12- by 14-inch		50
Water-supply pump		50
Total		120

The motors on the solution pump in the gold room are as follows:

Centrifugal circulation pump, 3.7 1/8 horsepower	22½
Triplexsolution pump	10
Vacuum pump	3
Vacuum pump	2
Total	37½
Grand total	552½

Metallurgy. - In March 1934 95 percent of Arabian ore and 92 percent of Tyro ore were recovered. Tyro ore is harder and is not ground as finely as the other. Silver recovery was 50 percent.

The reagent consumption was as follows:

Cyanide consumption, Aerobrand	2 pounds per ton
Lime consumption, cold weather	5 pounds per ton
Lime consumption, hot weather	3 pounds per ton
Zinc	25 pounds daily

The Tyro ore requires more lime than the Arabian. The ball consumption was 3½ pounds per ton of ore.

Mill labor, May 1935. -

Mill and crusher men	10
Foreman	1
Assayer	1
Assayer's helper and sampler	1
Scale and warehouse	1
Blacksmith	a/ 1
Master mechanic	a/ 1
Watchman	1
Clerk	1/2

a/ Also does work for mine.

The scale for mill men in May 1935 was $4.50 per 8-hour shift.

Milling cost. - Chemicals cost 30 cents per ton. The milling cost in 1935 was as follows: Direct, $1.76; indirect, 64 cents; total, $2.40. Power cost was $0.49, chemicals, $0.30, and labor $0.22 per ton.

Roadside mine. - The Roadside mine is on the Kingman-Katherine road about 4 miles east of the Katherine mill.

The present shaft was sunk to the 100-foot level in 1915 and 1916 and later extended to the 300-foot level. It has two compartments and is at an incline of 70°. Up to January 1934 the Gold Standards had done about 1,000 feet of development work on the 100-foot level and produced 890 ounces of gold and 1,734 ounces of silver. In May 1935 the last of the ore from a shrinkage stope was being drawn; in January 1936 the mine was closed.

The country rock is granitic gneiss. The vein dips 33° to 38° and is in a fault zone in a block of rhyolite against a fault. The vein material is irregular stringers and bunches of quartz and calcite in shattered silicified rhyolite. A chimney-shaped ore shoot with a flat rake to the north extended from the surface to the 100-foot level. It was 20 to 35 feet wide and 75 feet long on the strike. According to Banks,[32] the ore averaged 0.28 ounce of gold to the ton.

Despite the flat dip of the vein, the ore was mined by a shrinkage method. In March 1934 about 10,000 tons had been broken, but only the swell had been pulled. Plans called for using a scraper for emptying the stope.

Chutes were 25 feet apart with pillars left between them. Although only a few pillars were left in the stope, no caving of the back occurred.

In the spring of 1935 the ore was being mined by one machine miner, two men with hammers in the stope, two trammers on the 50-foot level, one bucket loader, and one topman. Hoisting was done in a 1,000-pound bucket and tramming in 16-cubic-foot cars running on 18-gage track. The water was kept down in the shaft by pumping two shifts each day with a boiler-feed pump.

The mine equipment consisted of a 20-horsepower gasoline hoist and a 10- by 12-inch compressor driven by a 40-horsepower gasoline engine. An old automobile engine was belted to the compressor engine for starting purposes.

32/ Work cited (see footnote 31).

The costs for the 4-month period, November 1933 to February 1934, are shown in table 13. The costs shown in the table are apparent rather than actual. A reserve of broken ore was built up in the stope, to be drawn later and on which the breaking cost had been charged out.

According to Banks,[33] the cost for the 6 months preceding the fire of September 1934 was $2.15 a ton delivered in the surface bin.

TABLE 13. - Cost of mining 3,278.15 wet tons (3,186.14 dry tons), Roadside mine, November 1933 to February 1934

Item	Material	Labor	Total	Per dry ton
Supervision	....	$1,200.00	$1,200.00	$ 0.377
Mining	$1,119.88	2,785.34	3,905.22	1.226
Mucking	6.11	1,062.23	1,068.34	.335
Compressed air and hoisting	711.76	900.10	1,611.86	.506
Equipment and repairs	126.31	....	126.31	.040
Carbide	24.00	....	24.00	.008
Blacksmithing	65.51	210.05	275.56	.086
Pipe fittings	13.29	....	13.29	.004
Sampling and assaying	113.51	477.07	590.58	.185
Trucking	213.77	171.36	385.13	.121
Miscellaneous	511.24	....	511.24	.160
Insurance	....	278.28	278.28	.087
Total	2,905.38	7,084.43	9,989.81	3.135
Cost per ton of ore	0.912	2.223	3.135	....

Arabian mine. - The Arabian mine is on the Kingman-Katherine road about 8 miles from the Katherine mill. Intermittent work has been carried on at the mine since before 1917. It was taken over by the Gold Standards late in 1933. The 1933 production amounted to 593 ounces of gold and 1,156 ounces of silver.

The country rock is granite in which a rhyolite-porphyry dike has been intruded. Rhyolite tuffs have been faulted against the hanging-wall side of the dike. The Arabian vein occurs in the dike close to the fault; the dip is 82°. A portion of the dike south of the underground workings occurs as a bold outcrop. Between 60 and 70 feet of the outcrop next to the hanging wall is reported to run 0.10 to 0.11 ounce of gold to the ton.

A mineralized zone 30 feet wide, consisting of a number of quartz stringers, occurs in the dike and to some extent in the granite footwall. A shaft on the north end of the property has exposed a stringer vein 3 to 8 feet wide that contains 0.25 to 0.40 ounce of gold to the ton on the 80-foot level.

Development work consists of a 280-foot shaft with levels at 80 and 180 feet. The shaft and ore shoot dip at 53°. Most of the ore is above the 100-foot level and comes to the surface in contact with the gravels in the wash as the hanging

33/ Work cited (see footnote 31).

wall. In March 1934 a shrinkage stope 125 feet long was being worked from above the 80-foot level. Chutes were 25 feet apart. Triangular pillars 15 feet long and 10 feet high were left between the chutes. Three manways built of stulls with outside lagging were maintained in the stope. Except for the manways and chutes, no timber was used.

In May 1935 drifting and crosscutting were being done to get under the shoot on the 180-foot level, and the upper portion of the ore shoot in contact with the gravels was being mined. About 50 tons per day, including development rock, was being milled from the underground workings.

The ore was raised in a 1-ton skip by a hoist run by a 25-horsepower gas engine. Compressed air was furnished by a 8- by 10-inch compressor. The ore was dumped into an 18-ton bin at the shaft and then trammed to a 25-ton storage bin.

The underground costs for 4 months, November 1933 to February 1934, are shown in table 14.

TABLE 14. - Cost of mining 4,208.80 wet tons (4,122.69 dry tons), Arabian mine, November 1933 to February 1934

Item	Material	Labor	Total	Per dry ton
Supervision	....	$1,087.50	$ 1,087.50	$ 0.264
Mining	$1,519.80	1,598.77	3,118.57	.756
Mucking	5.10	1,270.71	1,275.81	.309
Compressed air and hoisting	918.13	1,496.90	2,415.03	.586
Equipment and repairs	449.10	....	449.10	.109
Carbide	18.00	....	18.00	.004
Blacksmithing	65.51	228.05	293.56	.071
Pipe fittings	8.00	....	8.00	.002
Sampling and assaying	113.51	477.07	590.58	.143
Trucking	213.77	171.36	385.13	.094
Miscellaneous	465.92	....	465.92	.113
Insurance	....	252.14	252.14	.061
Total	3,776.84	6,582.50	10,359.34	2.512
Cost per ton of ore	0.916	1.596	2.512	....

Arabian open-cut. - The Arabian open-cut was about one-quarter mile from the Arabian shaft mine. The dike is 150 to 200 feet wide and outcrops for a distance of 700 feet. The hanging-wall side of the vein has been eroded away by a large wash; the dip was 58°. The face originally stood up as a cliff 150 feet high.

Mining was done by breaking 10-foot benches, starting at the top, over a width of 50 to 60 feet. Drilling was performed by two men with jackhammers on one shift per day at a contract price of 20 cents per ton. Air was furnished by a portable compressor. The broken ore fell to the base of the cliff and large pieces were broken up by hand, which cost about 30 cents per ton of ore mined.

The ore was loaded into trucks on contract at 20 cents per ton by a 1/4-cubic yard, full-revolving power shovel mounted on a tractor. The boom was 14 feet long and dipper stick 12 feet long. The contractor who owned the shovel came out from Kingman each day that ore from the open-cut was wanted at the mill. According to Banks, in December 1934, 100 tons was loaded on alternate days in 4 hours. The trucking cost was 40 cents per ton, which made a total direct cost of $1.10 per ton before milling.

In March 1934 a reserve of several thousand tons was broken ahead of the shovel; in May 1935 the open-cut work had been discontinued.

Minnie-Sheeptrail-Boulevard mine. - The Sheeptrail and Boulevard were the earliest recorded locations in the Katherine district; they are credited with approximately 15,000 tons of ore that was hauled to a mill on the Colorado River prior to about 1906. The Gold Standards Mines Corporation acquired a lease on these two properties and a central one known as the Minnie in 1935. In May 1935 the Minnie was being prepared for production.

A crosscut was run to intersect several quartz-calcite veins that outcrop on the hillside near a granite-rhyolite contact.

In January 1936 lessees had begun shipping from the Sheeptrail to the Katherine mill.

Frisco. - The Frisco mine is about 10 miles by road from the Katherine mill. A small mill was formerly at the mine and operated intermittently from 1894 to 1914; about 14,000 tons of $14 ore was treated. This mine was being operated by the Gold Standards Mine Corporation in the Spring of 1935. About 75 tons of 0.14 to 0.30 ounce gold was being produced daily with a crew of six men and was trucked to the Katherine mill.

The rock formation consists of a small granite hill capped by a rhyolite flow and rhyolite tuffs. Stringers of banded vuggy quartz occur at the contact to form a vein in previously shattered rhyolite; in some places the mineralization occurs in the underlying granite.

An adit enters the ore on the west side of the hill but passes 35 feet under the ore on the east side due to a fault. Mining formerly was done by a room-and-pillar method with a maximum room height of 18 feet. In the spring of 1935 the stope height had been increased to a maximum of 40 feet in some places by taking out roof and floor pillars. The old pillars were being removed by a retreating system. The ore above the upper adit was transferred through a chute raise to a lower level 120 feet below, along which it is trammed to a truck bin at the surface. The tramway was done on a bonus system.

The faulted part of the vein above the upper level was tapped by a series of short raises from laterals on that level. Ore on the east was mostly drawn off through this chute while that on the west was shoveled. Compressed air was supplied by a small portable one-machine compressor.

The direct mining cost[34] was about 60 cents. The minimum grade of ore mined was $3 per ton.

34/ Banks, Leon M., Gold Mines at Katherine: Explosives Eng., May 1935, p. 147.

Portland. - The Portland mine is about 15 miles by road north of the Katherine mill and 6 miles from the Colorado River at an elevation of about 2,200 feet.

J. E. Potter obtained an option on the property in 1934 and later turned it over to the Portland Mines, Inc., who in turn transferred it to the Gold Standard Mines Corporation late in 1935.

In May 1935 the Pioneer Mines, Inc., which had just finished sampling the mines, was building a road to the Katherine mill and had contracted to deliver 50 tons daily to the mill. The milling charge was to be $3.50 per ton, and 93 percent of the gold was to be paid for.

The country rock is granite and andesite. Two ledges occur on the property. The larger dips 53° and is 160 feet across and 210 feet long. It was cut at a depth of 65 feet by a crosscut. It was reported that the ore ran $7.60 per ton. The values in a crosscut 166 feet below the top of the outcrop were reported to contain $4 per ton.

The smaller ledge is a vein that has the same dip, is 6 feet wide, and lies exposed on a side hill of the same dip. It was opened up for 300 feet on the strike and to a depth of 86 feet. The ore was reported to run $15 per ton. The bottom of a 70-foot shaft was in ore.

The large ore body was sampled by drilling and blasting 6-foot holes on the surface and by crosscuts. The cost of the sampling job, including a compressor and the underground workings, was $20,000.

The equipment in May 1935 consisted of a 3/8-cubic yard gasoline shovel and a 500-cubic foot compressor run by a 100-horsepower semi-Diesel engine. The power cost was reported to be 2 cents per horsepower-hour. There is spring flowing about 4 gallons per minute near the camp.

In January 1936 the outcrop of the small ore body was being mined by a gasoline shovel. Drilling was done by jackhammers. The ore was broken to the footwall; the hanging wall was exposed. Later in the season work was transferred to the large ore body. In April a pit 75 feet long and with a face 20 feet high had been excavated. Down holes were being drilled with jackhammers; the ore was being loaded with the yard shovel into trucks for transportation to the Katherine mill.

Road building. - Eleven miles of road from the Katherine mill to the mine was built under contract for $2,500. The contractor used a No. 80 caterpillar tractor and a 12-foot bulldozer with a crew of eight men. The workmen were paid a bonus when over half a mile of road was built in a day. The road crossed a series of deep arroyos and was in generally rolling country. No rock work was necessary. The right of way was covered with desert growth. The contractor surveyed the road by eye as he built it.

Tyro

The Tyro mine is 6 miles by road from the Katherine mill. The drop in altitude from mine to mill is 2,000 feet. During 1915 and 1916 a 500-foot shaft was sunk and some drifting done on the 200-foot level. A small tonnage of ore was taken out at this time from pockets at the surface, but the operators missed an ore body that outcropped just below where the shaft was started. In 1933, C. F. Weeks and W. E. Whalley obtained a lease and option on the property and formed the Whitespar Mines Co.; work at the mine began in September 1933.

The country rock is a gneissic granite cut by numerous narrow dikes of rhyolite porphyry. The Tyro vein consists of a silicified fracture zone up to 60 feet in width in the granite. The gold occurs in quartz bands in the vein. Calcite also occurs in the vein but in smaller quantities than the quartz. In places quartz-calcite streaks reach a width of several feet. Where work began the oreshoot was 25 feet wide, but it narrowed down to 10 feet 20 feet below the surface. The minimum width of the shoot where worked is 6 feet. In March 1935 the shoot had been opened up 120 feet on the dip and 240 feet on the strike. The ore shipped in 1935 averaged 0.42 ounce per ton in gold. Surface sampling along the outcrop for about 900 feet indicated a value of $5 to $6 per ton.

The development work consists of an old 500-foot shaft with a drift at the 200-foot level and a new adit 175 feet long. The adit is a crosscut for 75 feet and a drift for 100 feet. Raises have been put up from the drift for ore chutes to an underhand stope above. The drift will be extended and other raises run as needed.

In May 1935 electric power had been brought to the mine. The equipment consisted of a 500-cubic foot electrically-driven compressor, two 100-cubic foot portable gasoline compressors (standby), mine blacksmith shop, and a 100-ton ore bin.

Stoping. - In March 1934, 40 tons of ore was being mined daily from an open-cut on the vein. A part of the dump from the old shaft had to be moved to get at the ore. The mill took the ore in lots of 80 tons.

The old dump and some overburden were removed from the upper part of the vein by a power shovel on contract for a lump sum of $600. Between 2,000 and 3,000 tons of overburden was removed, including 500 tons of solid rock. The contract price did not include breaking the rock. In March 1934 an underhand stope from the surface 10 feet wide by 35 feet long had been extended to a depth of 20 feet.

The ore was drilled with a jackhammer and hoisted in a 10-cubic foot bucket by a small gasoline hoist. The bucket was dumped into a car, which was pushed by hand to a 60-ton bin. The mining was done on contract at $2.50 per ton. The crew consisted of 10 men - 2 miners, 4 muckers, 3 hoistmen, and the contractor. Wages were $4.50 for miners and $4.25 for muckers. The output was 6.6 tons per man-shift for the men in the stope and 4 tons per man-shift for the entire force.

Forty tons of ore was broken with twelve 5½ foot holes; 1-1/3 pounds of explosive was used per ton. Bits were sharpened at the Katherine mill at 10 cents each. Detachable bits were tried but did not prove satisfactory in the hard drilling ground.

The compressor was rented at $2.50 per day and the hoist at $20 per month. The compressor used 15 gallons and the hoist 5 gallons of gasoline per day.

When mining was done as an open-cut the contract price was $1 per ton. The output was 8½ tons per man-shift.

In May 1935, 100 tons per day were being taken out of the mine through the adit. The ore was mined in 6-foot benches. The blasted ore was drawn through three chutes, without shoveling, into cars that were pulled by a mule to the ore bin in trains of four cars each.

The stope crew consisted of one jackhammer man and one helper on each of two shifts. Two small shrinkage stopes had been run at the ends of the open stope where the ore would not run without shoveling. Shrinkage stoping will be used below the adit level.

No timber had been used for supporting the walls in the open stope up to January 1936. All loose rock had been cleared away at the surface and a wire guard placed around the excavation. The walls were watched closely, and loose blocks were barred down as found.

In March 1934 the crew consisted of: 2 miners, 2 miners' helpers (for stoping), 1 drifting miner, 2 raising miners (for development), 2 trammers, 1 blacksmith, 1 superintendent, and 1 mule.

The mining cost, at 100 tons per day in 1935, was $1.20 per ton. On a 50-ton basis the cost was $1.75.

Panning. - Close control of stoping operations in the open-cut and early operations in the underhand stope was obtained by panning. A good understanding of the ore habits and of the margins of the ore shoot was obtained by this procedure. A close approximation of the value of each part of the vein was also obtained before the material was broken, without waiting for assays; moreover, an appreciable saving was made over the cost of assaying. In the wide portion of the ore-shoot ribs of low-grade material were discarded. Samples across sections of the vein were panned and a composite of the samples was assayed each day. Nearly the full time of one man (an official of the company) was used in taking the samples and doing the panning.

Samples weighing about 6 pounds each were obtained by moiling. About twenty were taken daily; one moil was used for each sample. They were crushed to about minus 3/16-inch in a No. 2 Wheeling crusher run by a 5-horsepower gasoline engine; a sample was crushed and run through a second time in about one minute. After crushing, the sample was screened through 40-mesh and a measure of the fines taken in a cap box. The sample was settled in the box and struck off at the top; it was then panned.

The panning was continually checked against assays. Rejects from assayed samples were used for the control. It had been found that 60 to 70 percent of the gold was in the fines of the crushed sample. The estimate made by panning was usually within 10 percent of the value shown by assays.

Road building. - One-half mile of road up a sandy gulch was built for $150. A power shovel was used for grading. Little blasting was required. A single-track road in disintegrated granite in which no blasting was necessary was built by hand. One man at $4 averaged 16 feet of road construction per day - making the cost per linear foot $0.25.

Nonproducing mines (May 1935)

Pyramid. - The Pyramid mine probably is the oldest location in the district. It is situated near the Colorado River in some very low hills of granite. The vein consists of stringers in the granite. Some rich ore is reported to have been stoped from a 70-foot shaft.

Golden Cycle. - The Golden Cycle is northwest of the Pyramid mine and has a similar vein. Lausen[35] reports assays of $1 to $3 at the surface and up to $14 underground. A 115-foot shaft with lateral workings has been sunk.

Other properties. - The San Diego, O K Expansion, Union Pass, Monarch, New Chance, and Sunlight properties have been described by Schrader.[36] More or less prospecting has been done on the Black Dyke, Gold Chain, Burke, Mandalay, Bonanza, Banner, Tin Cup, and Quail properties.

Low-grade deposits. - The gold-bearing rhyolite dikes in the Katherine district offer an interesting possibility if the mining and milling costs could be lowered sufficiently. It is estimated that 500,000 tons of $3.50 ore is exposed in the Arabian dike. The "Black dike" about 3 miles to the east of the Katherine mine is exposed for about one-half mile and has a maximum width of 150 feet. It is a calcite vein cut by stringers of quartz in shattered rhyolite. Lausen[37] reports that it runs $2.40 to the ton. The result of sampling done in 1935, however, gave a reported value of over $0.95 per ton.

The Gold Chain about 3 miles from Katherine is similar to the Arabian and is reported to be over 100 feet wide and to run $3 per ton. In addition to the gold-bearing dikes, low-grade deposits have been indicated in several other localities in the district.

Pilgrim district

Pilgrim mine

The Pilgrim mine, the only mine in the Pilgrim district, is 9 miles west of Chloride on the eastern slope of a spur of the main range at an altitude of about 3,600 feet. The deposit was discovered in 1903; a 360-foot inclined shaft was sunk, and a few tons of rich ore were shipped prior to 1907. Further development work was done intermittently until 1934, when the property was acquired by the Pioneer Gold Mining Co. Previous to 1933 about $3,000 had been produced from the mine.

Construction of a mill was begun in July 1934, and mining and milling were started April 1, 1935. In January 1936, 75 tons was being treated daily, and 73 men were employed on the property.

35/ Lausen, Carl, Geology and Ore Deposits of the Oatman and Katherine Districts, Arizona: Arizona Bureau of Mines Geol. Ser. 6, Bull. 131, 1931, p. 119.

36/ Schrader, F. C., Mineral Deposits of the Cerbat Range, Black Mountains, and Grand Wash Cliffs, Mohave County, Arizona: U. S. Geol. Survey Bull. 397, 1909, pp. 207-214.

37/ Lausen, Carl, Geology and Ore Deposits of the Oatman and Katherine Districts, Arizona: Arizona Bureau of Mines Geol. Ser. 6, Bull. 131, 1931, p. 119.

The rocks in the vicinity consist of andesites and rhyolites intruded by dikes of rhyolite porphyry. The mineral deposit occurs within a fault zone that dips 30° and is 50 feet wide. A vein occurs on the footwall and another on the hanging wall of the zone.

A streak of red gouge up to 3 feet in thickness occurs over the ore of the hanging-wall vein. Moreover, both veins are fractured and contain considerable gouge. The veins are made up of irregular stringers and masses of fine-grained quartz with some calcite in silicified country rock. The gold occurs mainly with a green-tinted quartz.

In May 1935 three ore shoots had been opened up in the mine, 125, 70, and 30 feet long, respectively. Two occurred in the footwall vein and one at the hanging wall. The ore averages 3½ feet thick.

All gold is in the free state and in very fine particles. No sulphides occur in the ore but a small amount of pyrolusite is locally associated with the calcite.

<u>Equipment</u>. - The surface mine equipment consists of a 400-horsepower Fairbanks-Morse Diesel plant (cost $55,000) installed late in 1935, a compressor with 100-horsepower motor, an electric hoist, and a 5-horsepower fan. Two 7½-horsepower pumps are used for drainage.

Previous to installing the new power plant, power was supplied by an old 280-horsepower Diesel; the actual output was about 150 horsepower. Air was compressed in a 10- by 12-inch compressor run by a 40-horsepower gasoline engine and hoisting done with a 15-horsepower gasoline hoist. The small compressor did not supply enough air to permit stoping and development work to be carried on at the same time. The gasoline bill for the hoist and compressor was $600 per month.

The mine is developed by a 360-foot, 35° incline working shaft, an air shaft, and four levels. In January 1936 five development headings were being run.

<u>Stoping</u>. - An open-stope method is used. In the hanging-wall stopes the ore is hard, and the walls are soft and heavy. One to two feet of red gouge falls down on the ore if extreme care is not taken to hold it up. In these stopes, stulls with headboards are placed close together. The back is stronger in the footwall stopes and less support is needed.

The dip of the vein is too flat to permit the broken ore to run by gravity. To obviate pulling the ore down chute raises for over 100 feet, crosscuts were run back under the vein and then vertical raises put up to the ore, or sublevels were used.

Stope sections in May 1935 were 40 feet wide. Combined manway and chute raises were put up on the vein on 40-foot centers and timbered with stulls. Each raise was used for two sections. The face of a stope was advanced as an inverted V, the apex midway between two raises. A 5-foot cut was taken and as room was made, stulls with headboards were placed on 5-foot centers in the hanging-wall stopes and, at less frequent intervals in the foot-wall stopes. The ore was shoveled into the ore passes and then pulled down with shovels to chutes, whence it was loaded into cars. The stopes were run about 6 feet high. Extra rock was broken at the bottom of the stope to make working room. This waste and the red clay that fell down on the ore were backfilled, but were not enough to fill a stope completely. Plans were made to salvage the stulls as the face advanced and let the back come in but at the time of visit had not been put into effect.

The ore is treated in a flotation mill at the collar of the shaft. Although the gold is readily soluble in a cyanide solution, the clay in the ore is so difficult to settle that the cyanide process did not appear practicable.

The flow sheet as of November 1935 is shown in figure 11. The flotation circuit is shown in figure 12.

The flotation reagents were Aerofloat 15, Dupont B-23, and Amyl and Ethyl xanthate.

A Gibson amalgamator was used in the original flow sheet of the mill ahead of flotation; it recovered 7 percent of the gold. The small amalgamator was first put in to treat the concentrate from the unit cell. This worked so well that the second and larger one was installed for treating all of the concentrate. In May 1935, before the amalgamators were installed, the ratio of concentration was 57 to 1 and the recovery 92 percent. The heads ran 0.476 ounce gold and the tails 0.04 ounce. The flotation concentrate contained 20 to 25 ounces gold and 10 ounces silver. In January 1936 the ratio of concentration was 700 to 1. The concentrate for a month's run consisted of 3-½ tons of 30-ounce material. The overall recovery was 91 percent, of which 90 percent was obtained by amalgamation. Ninety-five percent of the gold recovered in this way was obtained from the first amalgamation.

On March 29, 1936, amalgamation was being done in cyanide solution, according to a letter from C. F. Hastings. It was then run through zinc boxes. All of the gold was obtained as bullion.

In May 1935 the net received by the company for the gold in the concentrate, after deducting trucking and smelter charges, was $29.25 per ounce. The net received for the gold sent to the Mint, after deducting melting, retorting, express, and Mint charges, was $33.75 per ounce.

The motor set-up in the mill in May 1935 was as follows:

	Horsepower
Jaw crusher	10
Gyratory crusher	20
Conveyor	5
Belt feeder	5
Ball mill	100
Unit cell	5
Classifier	5
Sub-A flotation machine, 6 - 1½	9
Kraut flotation machine, 4 - 5	20
Pump	10
Concentrate, thickener	3
Tailing, thickener	10
Total	202

New water for milling and other purposes was obtained from Willow Springs 7 miles from the mine. It was pumped through 2 miles of 2-inch pipe over a divide with a 400-foot vertical rise and then run by gravity to the mill through 5 miles of 5- and 3-inch pipe. Thirty-five gallons per minute was pumped for 22 hours daily. The pumpmen lived at the spring. Thirty-five percent of the milling water was reclaimed for reuse.

Labor and costs. - The labor required in May 1935 for mining 60 tons daily was 22 men, including 1 blacksmith, 1 hoist engineer, and 1 mine superintendent. The surface labor consisted of 3 Diesel operators at \$5, 1 master mechanic at \$200 per month, 1 carpenter at \$5, 1 pumpman at \$125 per month, 1 pumpman at \$100 per month, 1 general utility man at \$5, 1 manager, and 1 clerk. The total mine and mill labor was 38 men.

The wage scale for the mine was as follows: Miners, \$4.50, muckers, \$4.00, and hoistmen, \$5.00. The mine scale of wages has been raised since May 1935.

The mill required 1 foreman, at \$5.50, 3 operators, at \$5.00, 1 crusherman, at \$4.00, 1 swamper, at \$4.00, and 1 assayer, at \$5.00, a total of 7 men, at \$33.50.

The direct cost during the first half of April 1935 was \$3.47 for mining and milling. Due to breakdowns of the power plant and changes in the mill the total cost was considerably higher.

Virginia district

The Virginia district as designated by Schrader[38] lies 25 miles northwest of Chloride near the middle of the west slope of the River Range and nearly opposite the Searchlight district in Nevada; the elevation is about 1,500 feet. The country rock consists mostly of rhyolite and green chloritic andesite. The veins dip southwestward and usually have a calcite gangue; they grade into the country rock locally.

Klondyke

The Klondyke mine was first operated in about 1900. According to Wilson,[39] about 4,500 tons of ore from the property was treated in an amalgamation plant on the Colorado River, and later the tailings were cyanided. It is reported that the ore ran \$13 per ton.

In the summer of 1935 a lease was taken on the property by Peter Vukoye and associates and shipments were made regularly to the Tom Reed mill; 1,500 tons had been shipped up to January 1936, according to Peter Vukoye.

The ore was mined from four ore shoots in a vein that was 60 feet wide in places. The average grade of ore shipped was \$26 per ton. The ore also contained 3 ounces of silver to the ton, but no payments were made for this metal.

38/ Schrader, F. C., Mineral Deposits of the Cerbat Range, Black Mountains, and Grand Wash Cliffs, Mohave County, Ariz.: U. S. Geol. Survey Bull. 397, 1909 p. 214.

39/ Wilson, Eldred D., Cunningham, J. B., and Butler, G. M., Arizona Lode Gold Mines and Gold Mining: Arizona Bur. of Mines Min. Tech. Ser. 37, Bull. 137, 1934, p. 80.

The direct mining cost was $1.85 per ton; trucking cost $5 (82 miles) and milling $4 per ton. The first 10 miles of road was poor.

Golden Door

The Golden Door or Red Gap is about 1 mile north of the Klondyke. Lessees made a few small shipments in 1933 and 1934. The property was idle in May 1935.

The vein is 2 to 4 feet wide; the vein filling is quartz with some calcite. One oreshoot about 50 feet long was the last worked. The mill heads from this shoot ran about $10 per ton.

The ore was treated in a cheaply constructed mill near the Colorado River about 8 miles by road from the mine and about 1 mile from the Searchlight Ferry.

Mill. - The Golden Door mill, of about 25-ton-per-day capacity, was idle in May 1935. It consisted of a 7- by 12-inch crusher, a 4-½ by 4-foot ball mill, a classifier, a flotation cell, and a sluice box. The machinery was run by a 47-½-horsepower gasoline engine. The flow sheet also contained amalgamation plates, but during the last run the plates were cut out and a sluice box was used instead. The mill was in poor condition, and apparently a poor recovery was made. About 250 tons of tailings, including some from an old stamp mill, was piled below the mill.

Dixie Queen

The Dixie Queen is about 2 miles south of the Klondyke. According to Wilson, ore was produced in 1927-28, and some shipping and milling ores were produced several years before. During 1933 and 1934 lessees treated old tailings. In May 1935 the property was idle.

Mocking Bird district

The Mocking Bird district lies about 25 miles northwest of Chloride in a re-entrant parallel side valley in the last foothills of the range at an elevation of between 3,000 and 4,000 feet. Except for some desultory prospecting, no work was being done in the district in May 1935. Three mines, two with mills, were being operated, and several other properties were being developed in 1907 when Schrader visited the district. Relatively little work has been done in the area for the past 20 years.

Mocking Bird

The Mocking Bird mine is in the northern part of the district; the surface is gently sloping. The vein lies nearly flat in a local sheet of flat-lying dike of minette. The vein is about 6 feet thick and consists of red and green quartz and breccia. The gold is free and usually associated with hematite. In March 1935 pillars left at the surface showed considerable copper stain. The ore averaged about $10 per ton.[40/]

40/ Schrader, F. C., Mineral Deposits of the Cerbat Range, Black Mountains, and Grand Wash Cliffs, Mohave County, Ariz.: U. S. Geol. Survey Bull. 397, 1909, p. 214.

The principal development in 1907 consisted of 12 or 15 shafts ranging from 25 to 60 feet deep and about 500 feet of drifts.

The mine has long been idle. It was worked by a room-and-pillar method; the cover over the flat vein was relatively shallow. Prospecting was done through shallow shafts. The ore was pulled up a slight incline out of the mine then up to the mill bins.

The remnants of a cyanide plant are still on the ground.

Hall

The Hall mine is situated in the southern end of the district in granite. The vein is steeply dipping, varies in thickness up to 2 feet and is associated with diabase dikes. The vein filling is mostly quartz, some being of the honeycomb variety. Some of the ore was very rich.41/ In 1907 a 24-ton mill was being operated at the mine.

The mine was developed in 1907 by a 210-foot shaft and two levels with 200 feet of drifts.

Great West

The Great West mine is near the Hall. The vein is about 3 feet wide and consists of iron-stained quartz. The ore in 1907 was reported to run from $10 to $80 per ton. Fifty tons were shipped to the Kingman sampler in 1926.

Pocahontas

The Pocahontas mine is near the Hall. It is developed to a depth of 200 feet by a shaft and drifts. A cyanide plant had just been built in 1907 to replace an amalgamation mill.

Gold Bug district

The Gold Bug district is near the summit of the range 3 miles north of the Mocking Bird district, 3 miles south of Eldorado Pass, and 30 miles northwest of Chloride. The veins dip steeply and occur in a volcanic rock. The district was busy in 1907, when Schrader visited the area. It was long idle, but in 1935 some interest was again being shown in the area. In addition to the following mentioned mines, work has been done on a number of properties in the district.

Gold Bug

The Gold Bug has been the principal producer in the district. Early operations ceased in 1908. Some development work was done in 1931 and a small tonnage of ore was mined in 1932. In May 1935 Joseph Gardner and partner, lessees, were shipping to the Tom Reed mill. In a year's time 290 tons had been shipped; a 5-ton lot of selected ore ran $72 per ton and 285 tons ran from $14 to $26. The ore came from a shoot 60 feet long in a vein 2 feet wide on the 90-foot level.

41/ Schrader, F. C., work cited (see footnote 40).

The mine is developed to a depth of 512 feet by a shaft and 5 levels. Open-stope method of mining was used; some waste was sorted out and left in the stopes.

Top-of-the-World

The Top-of-the-World is 29 miles from Chloride and 2-½ miles from the old Kingman-Boulder road. The mine had been idle for about 20 years. The dump shows that considerable underground work had been done; no production data are available.

In May 1935 Arthur Black and three associates had a bond and lease on the property. Three men were taking out some high-grade ore about one-half mile from the main workings.

The country rock is gneiss and granite. A ridge of rhyolite and another one of andesite are near the new workings. The vein consists of a number of fractures cutting the formation. The ore consists of a 6-inch streak of quartz; it contains about 2 ounces of gold to the ton. The gold is free and visible to the naked eye in places. The ore shoot is small. No values are found beyond the 6-inch streak. An open-cut was being made on the ore at the side of an old 30-foot shaft. The cut was down 15 feet at the time of visit. A small shipment had been milled with Gold Bug ore at the Tom Reed.

Mojave Gold

The property of the Mojave Gold Mines Co. is about 1 mile west of the Gold Bug. A camp was built in 1934 and a 112-foot shaft sunk and 500 feet of lateral work done on the 40-foot level. In May 1935 only a watchman was on the property.

Golden Age

The Golden Age is about 1 mile north of the Gold Bug. J. H. Omie, owner, with one man was getting out a shipment from the vein at the surface in May 1935.

The country rocks are schist, granite, and porphyry. The vein dips 25° and is 1 to 6 feet wide, averaging 3 feet. The ore occurs in bunches. A drift has been run 125 feet on the vein.

Eldorado Pass district

The Eldorado Pass district is in the northern part of the range at the Eldorado Pass at an elevation of 2,500 to 3,000 feet. The topography is one of gentle relief. The country rock is granite intruded and locally overlain with volcanic rocks. Schrader briefly mentioned the district but did not describe the individual mines. There has been some minor activity in the area during the past few years.

Pope (Expansion)

The Pope mine is at the side of the old Kingman-Boulder road 32 miles from Chloride on the east slope of the range.

In May 1935 Ben Fortner and Peter Eilson were working the property under a lease and option. In the 4 months previous to May 1935 10 cars of ore had been shipped to a Utah smelter. Water for mining purposes was hauled 10 miles from the Colorado River.

The mine is developed by a 100-foot 1-$\frac{1}{2}$-compartment shaft with drifts on the 50- and 100-foot levels. The shaft was sunk in 1927 and ore was shipped from a stope on the 50-foot level and from another on the 100-foot level. One 50-ton car of ore gave a return of $1,100.

The vein occurs in the granite, dips 70°, and on the 50-foot level is 1 to 3 feet wide. The high-grade ore occurs in lenses in the vein. Low-grade ore adjoins that which can be shipped. A shoot 50 feet long and with a rake of 40° had recently been mined from above the 50-foot level. One carload ran $100 per ton and nine others about $50 per ton; the total net from the smelter was $11,000. The cut-off for shipping ore was $40 per ton.

The ore, besides gold, contains some lead carbonate and a reported trace of vanadium. No galena had been found up to May 1935. The ore is different in character from any other found in the range. The ore was being mined by an open-stope method in 1935. Two chutes were used in each stope. The ore was hoisted in an 800-pound bucket.

The equipment consisted of an 8-horsepower gasoline engine, a one-machine-capacity portable compressor, and a 20-ton bin.

The ore was trucked 53 miles to Kingman for $3.50 per ton. It was loaded from the bin and shoveled into cars.

Hoover

The Hoover mine is near the Pope. It is developed by a 100-foot shaft. A lessee shipped 5 tons of $60 ore to the Tom Reed in the spring of 1935. The shaft had been sunk within the last 2 years.

Other mines

Schrader mentions the Burrows, Bogg, Young, and Pauley as the principal mines in the district in 1907. He states that the Burrows had a reported production of $10,000.

SUMMARY

Gold deposits occur throughout the Black Mountains of western Mohave County. Gold is the only valuable metal (except a minor amount of associated silver) found in the range; there is a remarkable similarity in the occurrence of gold in the veins.

Several periods of activity have occurred in the range with relatively quiet periods between. The first mining was in the early sixties, when some rich surface deposits were found. At the beginning of the century work was being done throughout the range at a large number of deposits. The greatest activity in the Oatman district was between 1917 and 1924 during the life of the United Eastern, from which $14,000,000 in gold and silver was produced. The so-called "Oatman boom" occurred at this time, and considerable unproductive work was done on wildcat promotions. After the boom, production fell off gradually, and at the beginning of 1933 the area outside of Oatman, where one small mine was operating, was virtually deserted except for desultory work by a few lessees.

Interest was revived in the range when the higher price of gold was established. The Tom Reed and Katherine mills were again put in commission and began taking custom ore; before long more custom ore was being offered than could be accepted. This condition persisted up to the time of writing (spring, 1936). The Tom Reed, Gold Roads, and other old mines were reopened. Important new ore bodies were discovered in the Tyro, Ruth-Rattan, Portland, Minnie, and other mines. One new mill, the Pilgrim, was built in 1934. This was at an old mine with a negligible previous production. The total production to the end of 1933 was $37,000,000 in gold and over $600,000 in silver.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.