I. C. 6787 OCTOBER 1934

Matthew Shumaker

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

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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

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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.

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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-

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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:

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

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

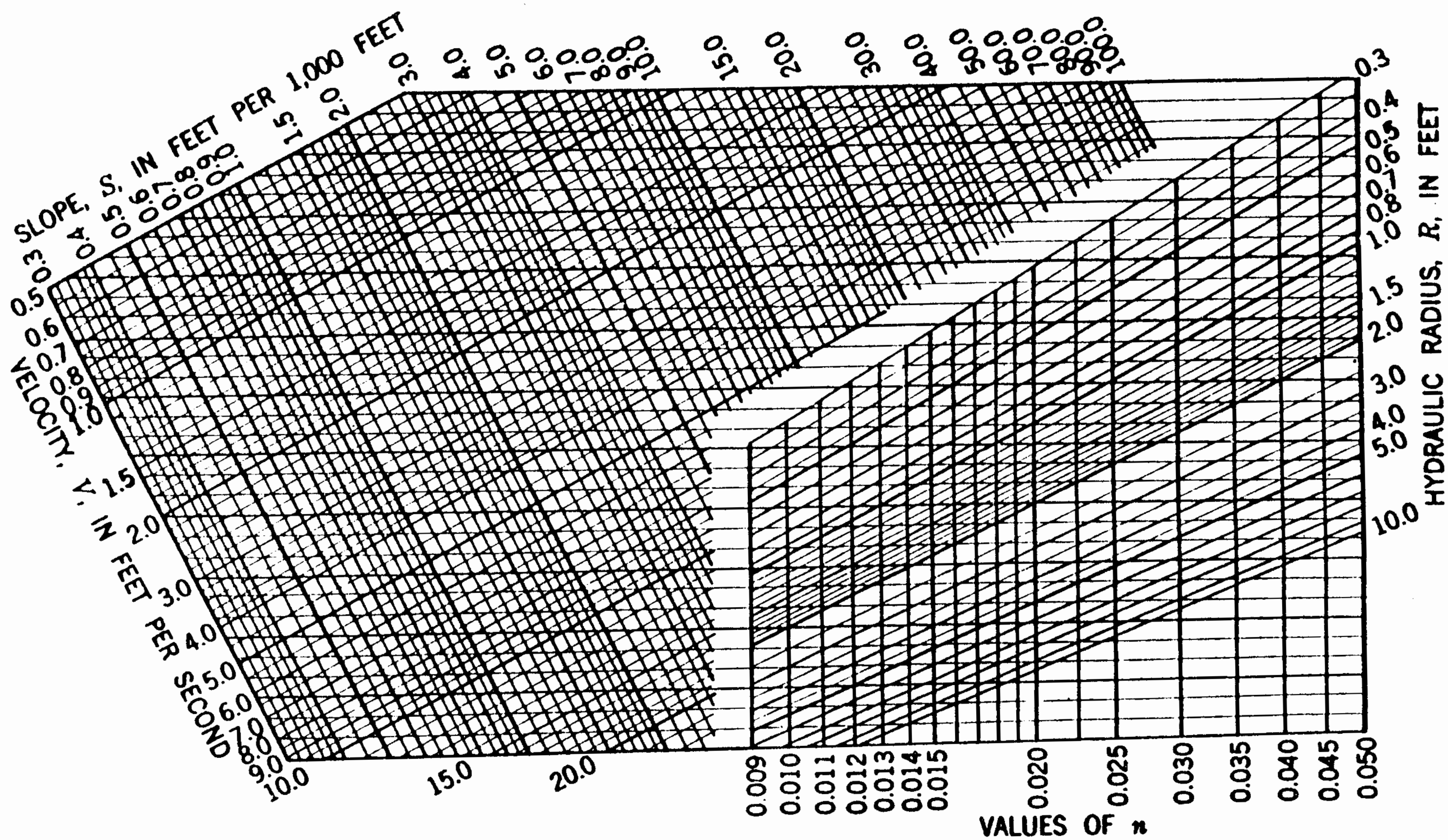
Surface	Best	Good	Fair	Bad
Coated cast-iron pipe	0.011	[1]0.012	[1]0.013	....
Commercial wrought-iron pipe:				
Black	.012	.013	.014	0.015
Galvanized	.013	.014	.015	.017
Smooth brass and glass pipe	.009	.010	.011	.013
Smooth lock-bar and welded "OD" pipe	.010	[1] .011	[1] .013	....
Riveted and spiral 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	1.166	.876	1.370	1.756	.941	1.484	1.909	2.263	1.569	2.019	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.

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

where

Q = quantity of water in cubic feet per second,
w = width of notch in weir,
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.

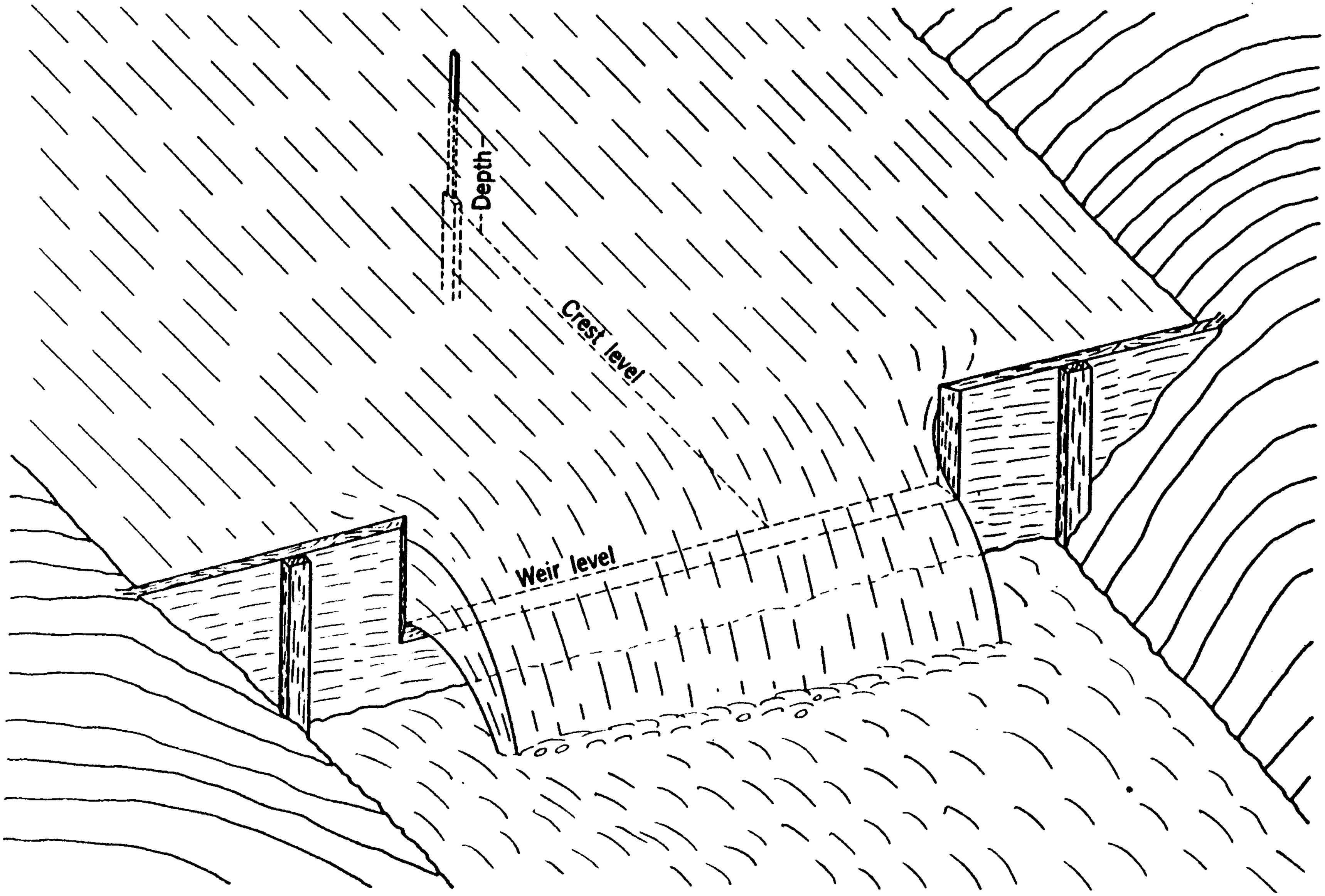


Figure 2.—Weir for small stream.

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

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

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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.

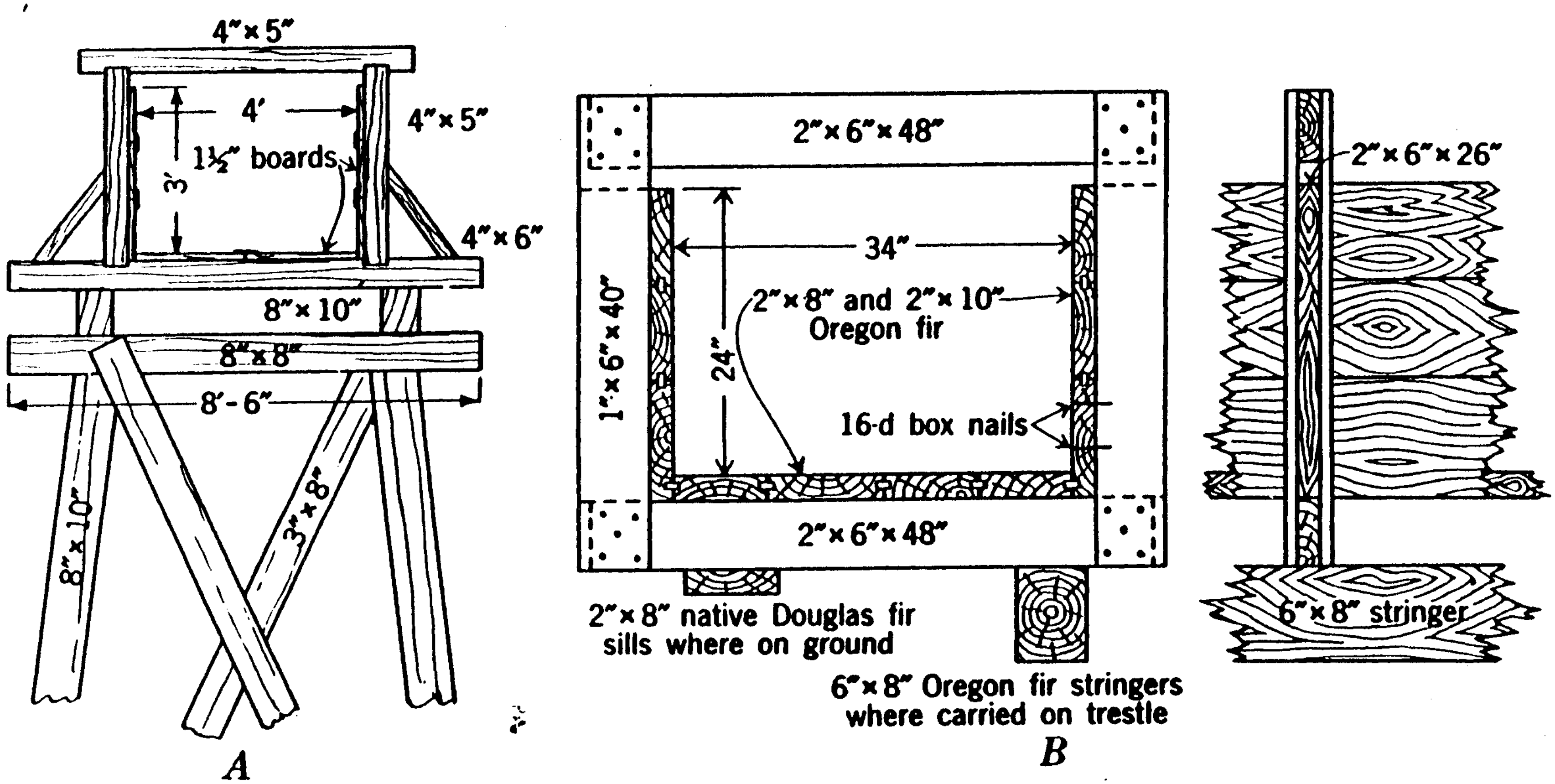

Figure 3.—Flume for hydraulic mines: *A*, Flume and trestle used in California; *B*, flume at Questa, N.Mex.

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	1.15	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.

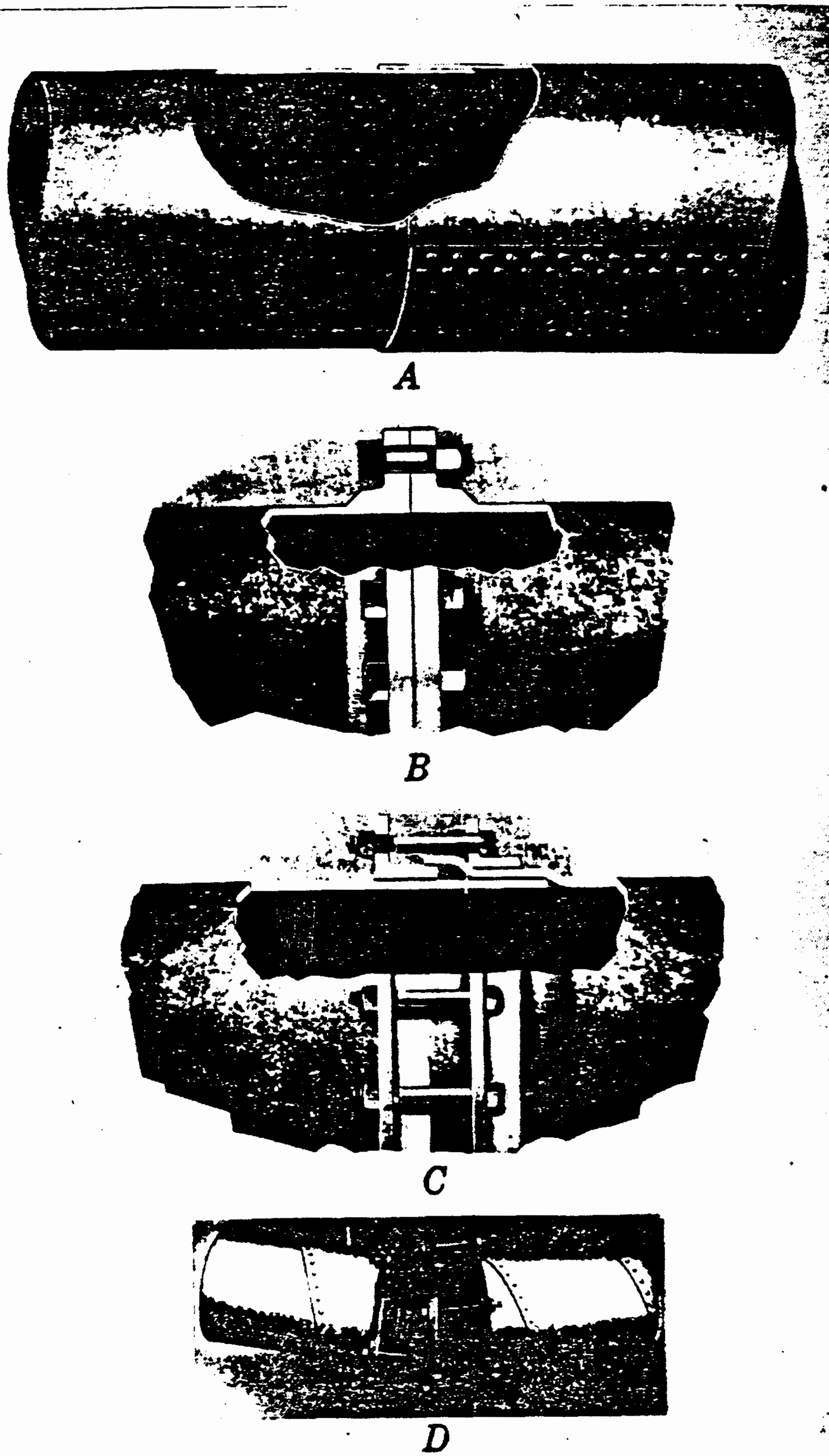

Figure 4.—Joints for placer pipe: *A*, Slip joint; *B*, forged-steel slip joint, on flanges welded to pipe; *C*, bolted-socket joint without retaining shoulder; *D*, bolted joint elbow.

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	750	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.

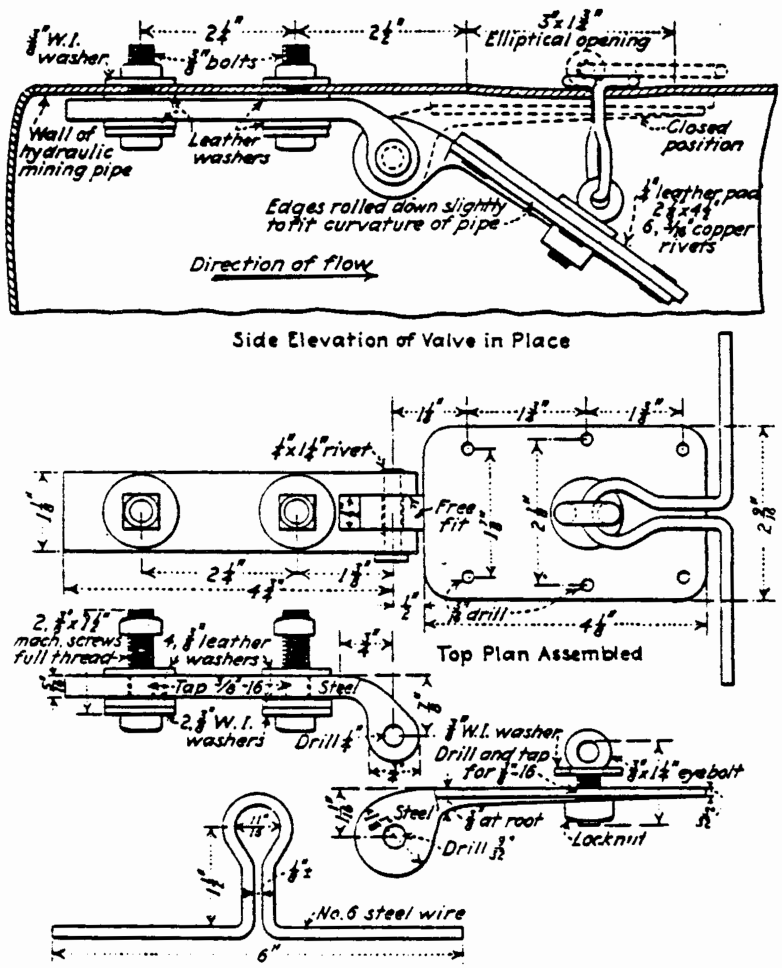

Figure 5.—Air-vent valve for pipe lines, Salyer, Calif.

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:

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

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

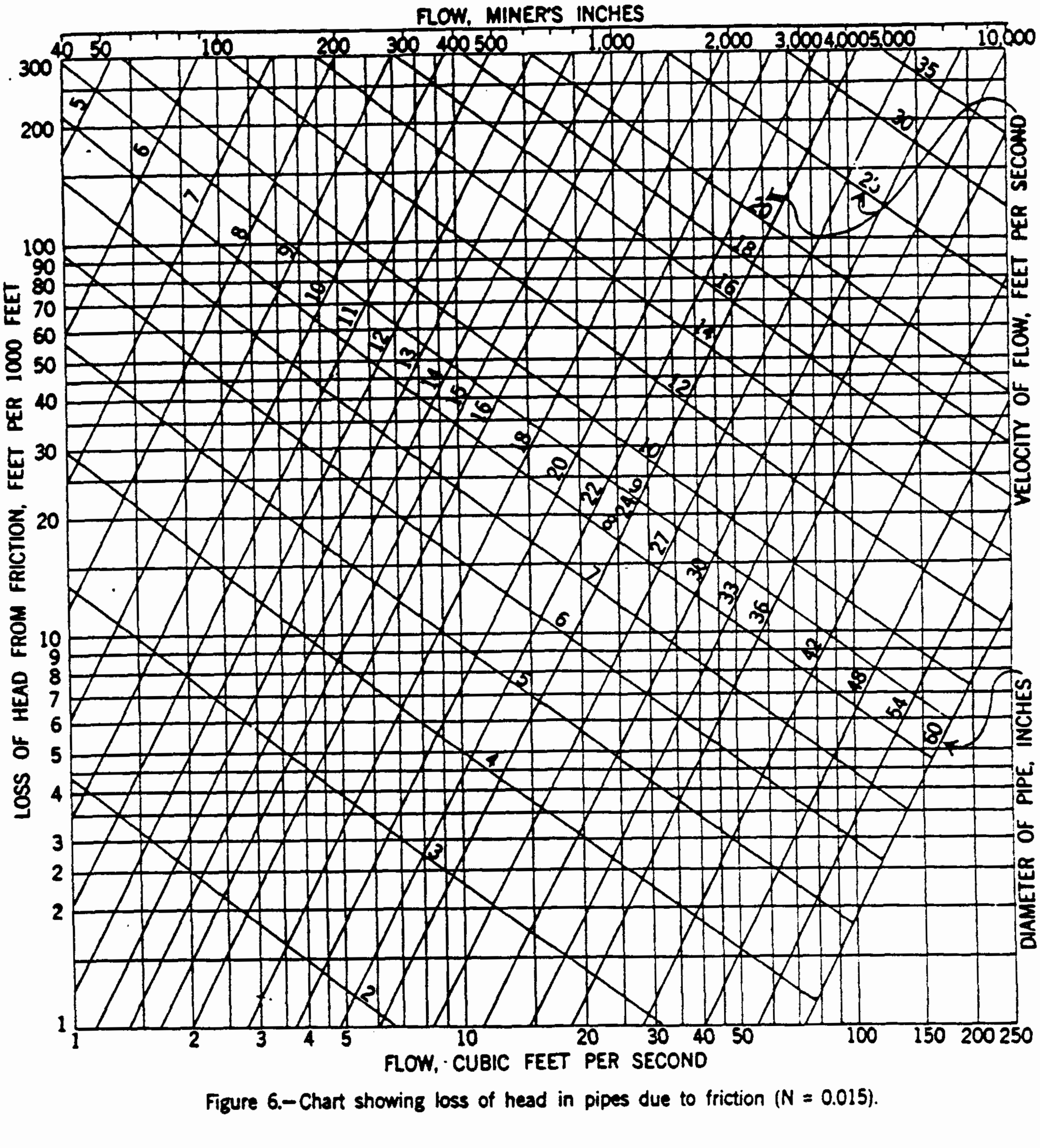

Figure 6.—Chart showing loss of head in pipes due to friction (N = 0.015).

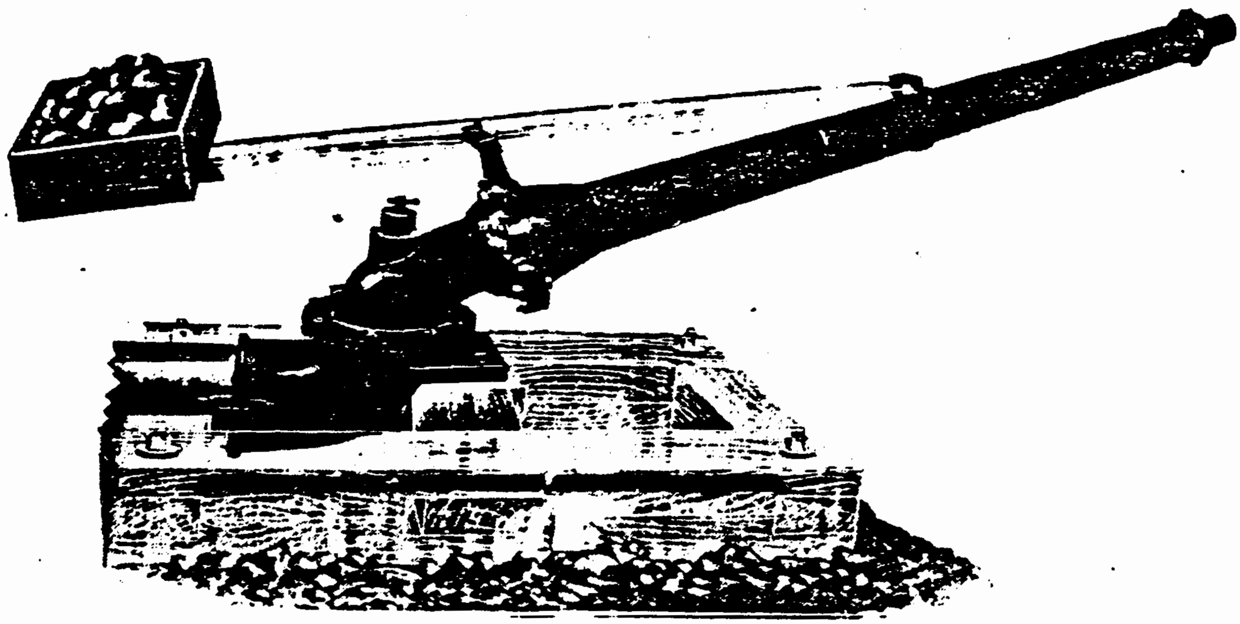

Figure 7.— Hydraulic giant, no. 1, 2, and 3.

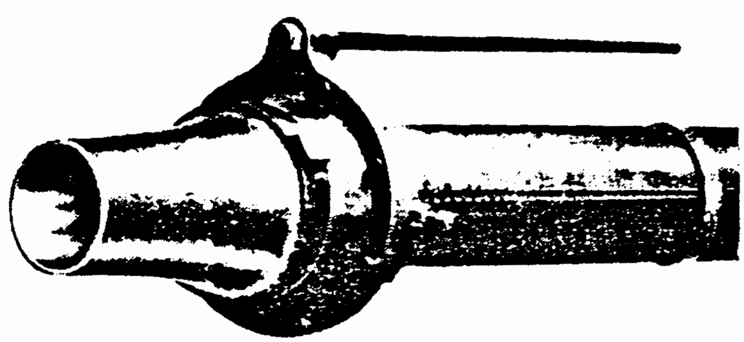

Figure 8.— Deflector for hydraulic giant.

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	690	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.

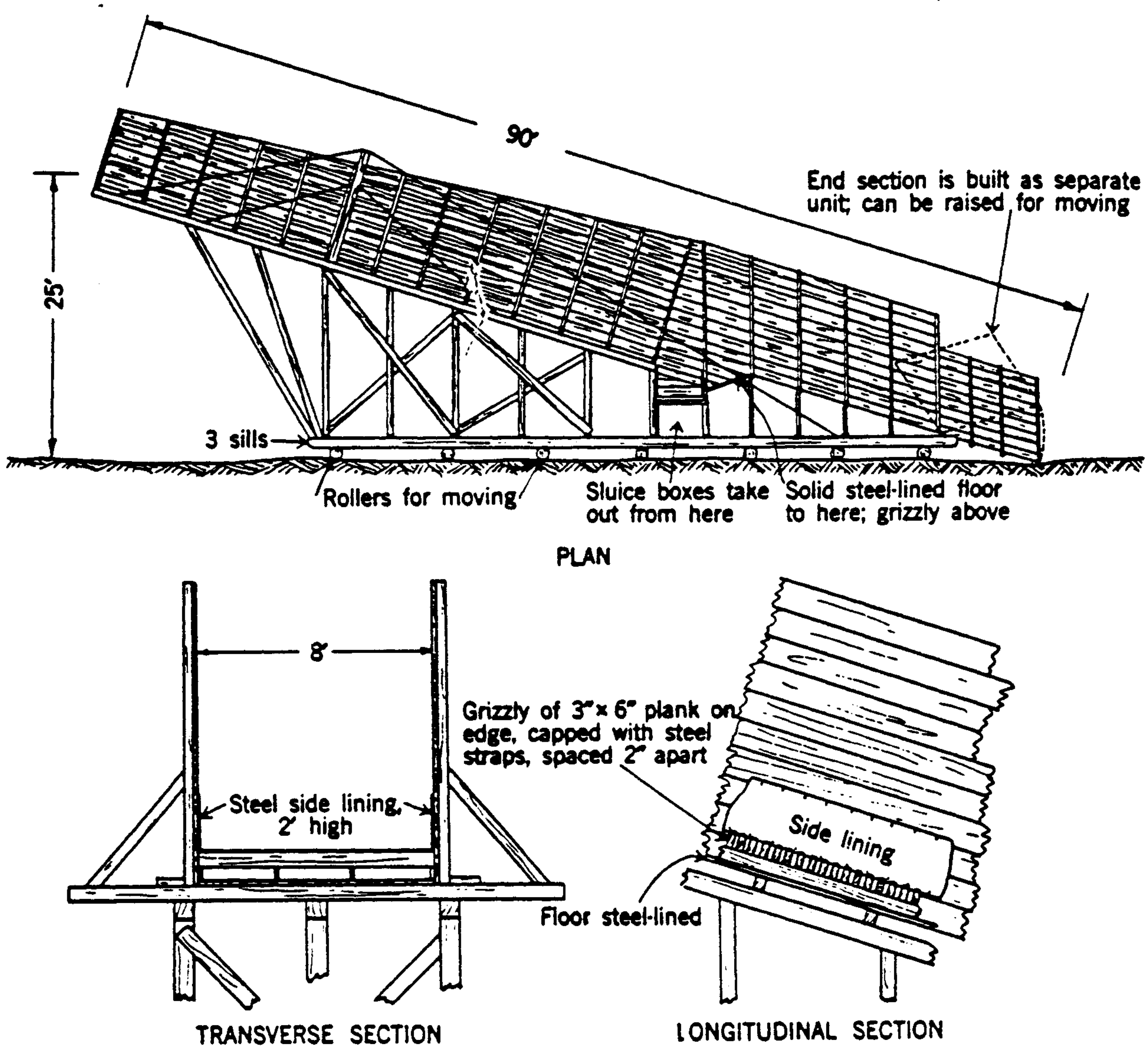

Figure 9.—Ruble elevator used at Redding Creek mine, Douglas City, Calif.

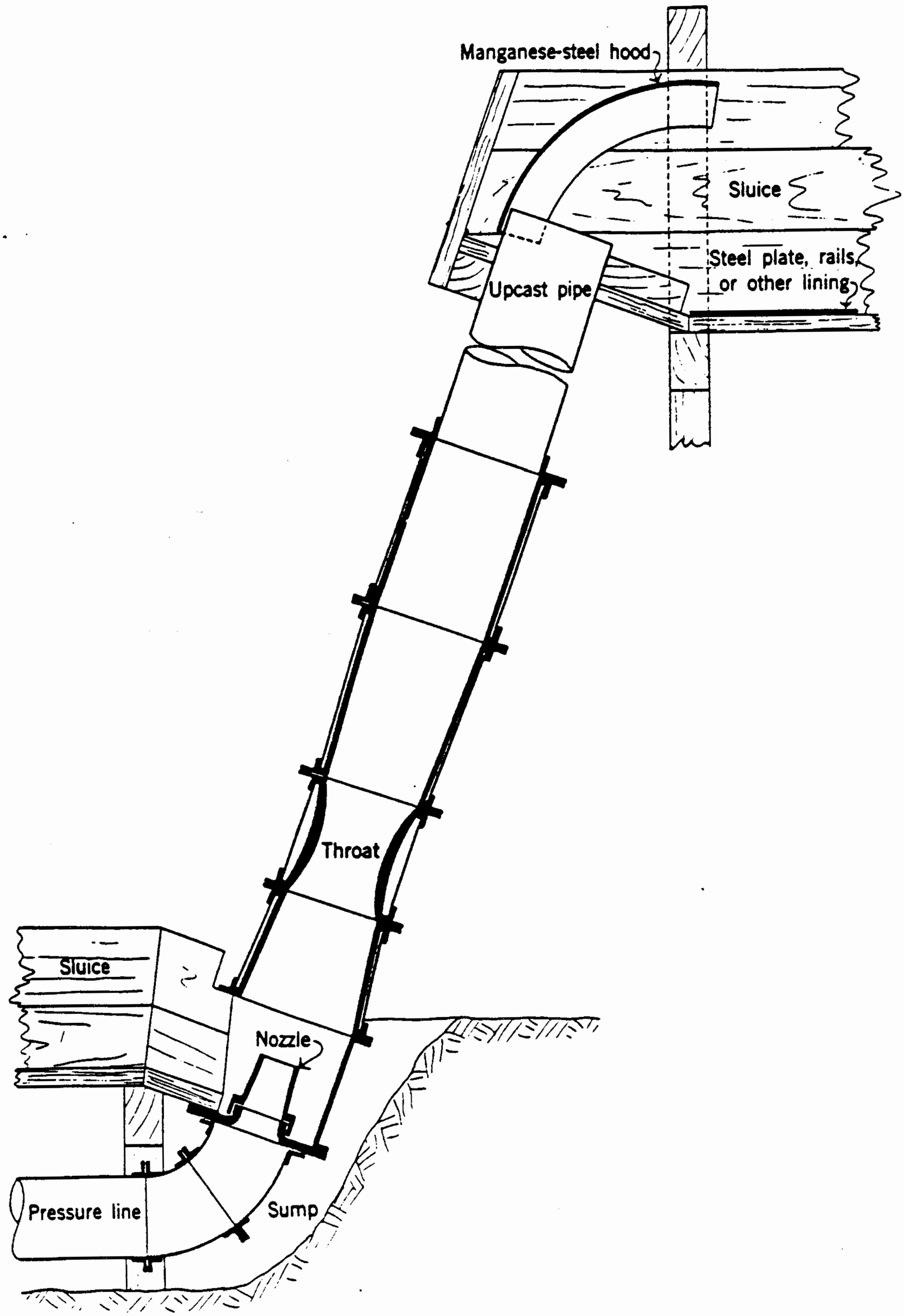

Figure 10.—Hydraulic elevator.

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.

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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.

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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: Name	Mine: Location	Operator	Gravel: Thickness, feet: Maximum	Gravel: Thickness, feet: Minimum	Gravel: Thickness, feet: Average	Gravel: Physical condition	Boulders: Over 1 foot in diameter, percent in gravel	Boulders: Maximum diameter through sluices, inches	Clay, percent in gravel	Bedrock: Kind	Bedrock: Physical condition	Gold: Character	Gold: Value, per ounce (at $20.67 per fine ounce)
Songer	Weaverville, Calif.	M. A. Songer			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	$18.58
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	Camptonville, Calif	B. T. Dyer			35	do.	10	14		Slate	Rough		
Depot Hill	do.	Fred Jourbert			60	do.	5	4		do.	Uneven	Fine	18.80
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	18.80
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.	W. 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 20 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		Clay, percent in gravel	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		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	6	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
Steamwinder	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.	6		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			
		Quantity, miner's inches[1]											
	Hours used	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
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			Reservoirs, capacity, acre-feet
	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	
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)														Height gravel elevated, feet		Hours cutting per day	Hours driving per day	Hours piping tailing per day	Method of handling boulders
	Cutting			Driving			Tailings disposal			Rubble elevator			Hydraulic elevator, diameter nozzle, inches	Total number of nozzles used at one time						
	Number used	Size no.	Diameter of nozzle, inches	Number used	Size no.	Diameter of nozzle, inches	Number used	Size no.	Diameter of nozzle, inches	Number used	Size no.	Diameter of nozzle, inches			Ruble	Hydraulic				
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	18	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)												Hydraulic elevator, diameter nozzle, inches	Total number of nozzles used at one time	Height gravel elevated, feet		Hours cutting per day	Hours driving per day	Hours piping tailing per day	Method of handling boulders
	Cutting			Driving			Tailings disposal			Ruble elevator					Ruble	Hydraulic				
	Number used	Size no.	Diameter of nozzle, inches	Number used	Size no.	Diameter of nozzle, inches	Number used	Size no	Diameter of nozzle, inches	Number used	Size no.	Diameter of nozzle, inches								
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 derrick.
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	8	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	Riffles: 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	1,200	1.0	48	40	168	1	8.3	Rails		3 1/2	12	0	4 1/4	do.	1
Salyer	2,600	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	160	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	600	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	160	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	160	1.3	24	18	240	7/12	4.8	Pole	4	4	5	6		None	0

TABLE 11 - Sluice boxes and riffles at eastern 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					Riffles							
			Width, inches	Depth, inches	Total length, feet	Grade		Type	Width, inches	Height, inches	Length		Center to center, inches	Point at which quicksilver used	Under-currents
						Inches per foot	Per-cent				Ft.	In.			
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	500	.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
Callia	[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	Super-vision	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	159,000	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]263	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 cleanin. 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: Cubic yards washed: Per day	Cubic yards washed: Per season	Cubic yards washed: Per man-shift	Washing, days, season of 1932	Total man-shifts	Men employed: Per shift	Men employed: Total	Daily wage scale	Length of shifts, hours	Total costs per cubic yard: Labor	Super-vision	Supplies	Total operating
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: Width, inches	Undercurrent tables: Length, inches	Riffles: Type	Riffles: Size, inches	Riffles: Distance, center to center, inches	Gold recovered on undercurrents, percent of total
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.

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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

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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.

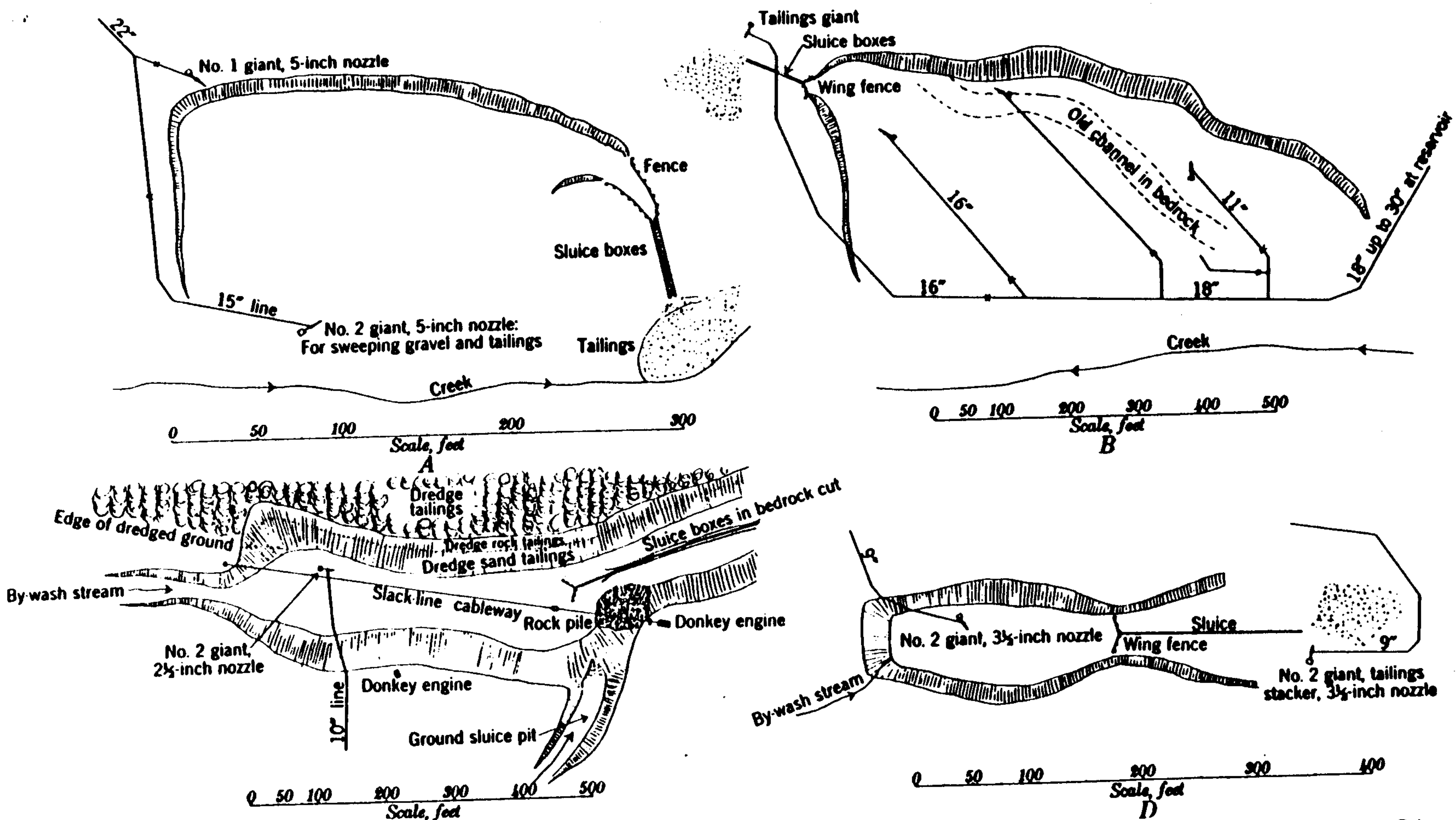

Figure 11.—Lay-outs of hydraulic mines: *A*, Indian Creek mine, Douglas City, Calif.; *B*, Blue Channel mine, Wolf Creek, Oreg.; *C*, Superior mine, Superior, Mont.; *D*, Golden Rule mine, Warren, Idaho.

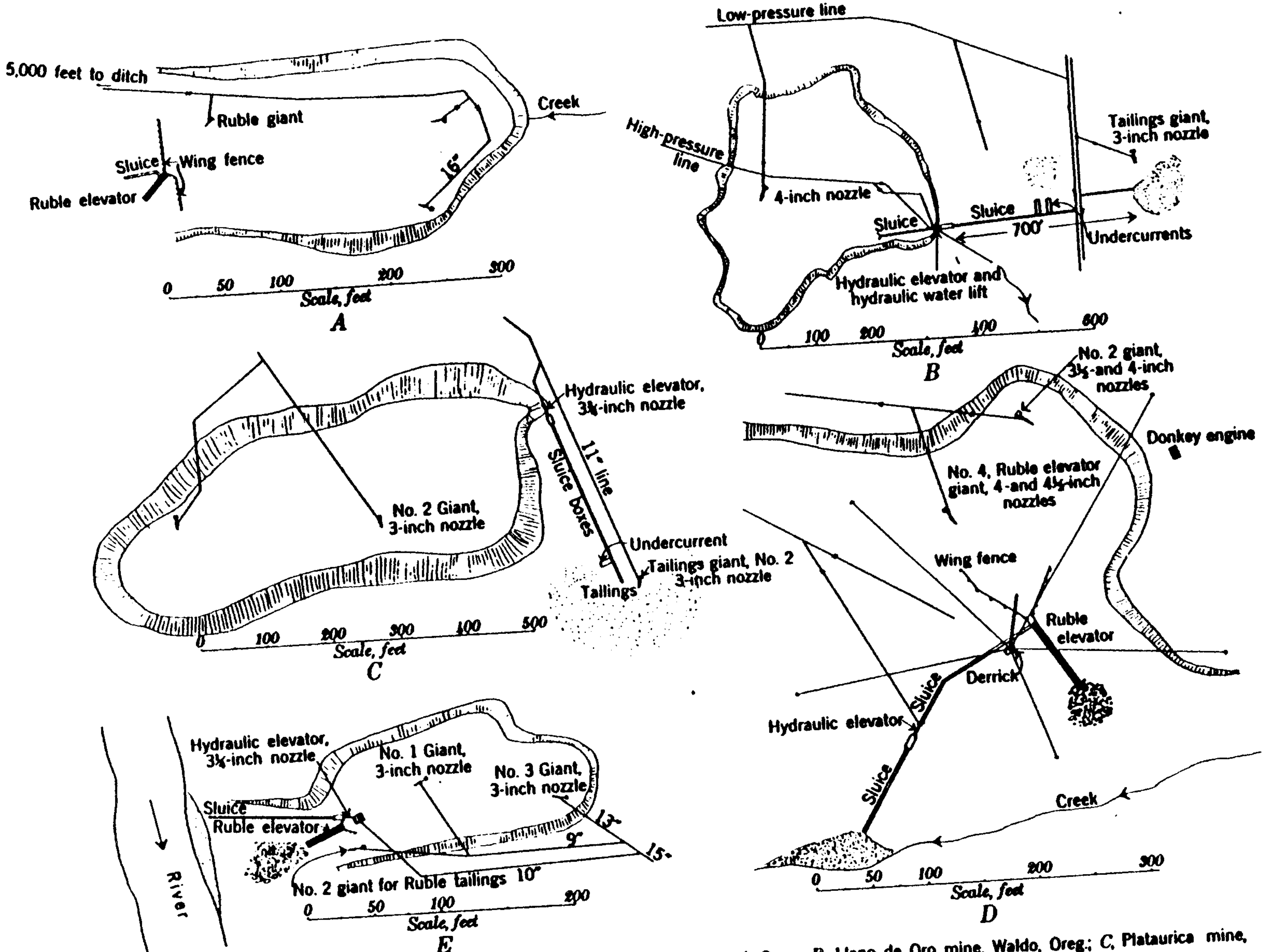

Figure 12.—Lay-outs of hydraulic mines using elevators: *A*, Browning mine, Leland, Oreg.; *B*, Llano de Oro mine, Waldo, Oreg.; *C*, Plataurica mine, O'Brien, Oreg.; *D*, Gallia mine, Sawyers Bar, Calif.; *E*, Lewis mine, Calice, Oreg.

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]

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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 Gold.- The Yellowstone Gold 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 12,000 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 hydraulicking is pumped and the gravel elevated from the pit by means of a gravel pump.[28] A 10-inch Byron Jackson centrifugal pump operated by a steam engine furnishes water for a monitor with a 3-inch nozzle. Steam is produced in two locomotive-type, 60-hp. boilers. Wood is used for fuel. The pump delivers 1,800 gallons of water per minute (4 second-feet) under a pressure of 45 pounds per square inch.

An 8-inch Byron Jackson gravel pump run by an automobile engine is set up over a sump in the pit and elevates the piped gravels to a sluice box. This pump requires 25 hp. to operate it; it seldom gives trouble by choking.

A grizzly with 4- by 5-inch openings is placed over the sump. The water from the sluice is settled and pumped back. Only seepage water is available; the water supply permits the plant to be operated 10 hours out of each 24.

The gravel on bedrock is 15 to 20 feet deep, contains much glacial clay, and is very tight. It is blasted before it is piped. No mention is made of boulders; apparently relatively few occur.

The capacity of the plant is stated to be about 300 cubic yards per 10-hour shift. The crew consists of 1 engineer at $5 per day, 1 monitor man at $5, 2 pitmen at $3, and 1 gravel-pump attendant at $4. The total daily labor cost is $20. Supplies used daily are 3 cords of wood at $5, 40 gallons (Imperial) of gasoline at $0.37, 2 kegs of black blasting powder at $3. Other supplies and expenditures, including amortization, amount to $27.90, making a total of $63.70 for supplies and a grand total of $83.70 daily expense. This is equivalent to $0.28 per cubic yard of gravel handled.

Summary of Yardages Mined and Operating Costs at Hydraulic Mines in 1932

Yardages

The quantity of gravel handled in 1932 at the mines listed in this paper ranged from 2,000 to 718,900 cubic yards. (See table 12.) Disregarding four mines where 100,000 or more cubic yards was washed, the average yardage was about 25,000. The total washed at all the mines listed was about 2,000,000 cubic yards. The total hydraulicked in the Western States during 1932 was probably about 2,500,000 cubic yards. This was less than the monthly capacity of the dredges in operation.

The daily capacities of the mines during the season ranged from 18 to 7,400 cubic yards. The cubic yards washed per man-shift ranged from 6 to 500. (See table 12.) The cubic yards washed per season, as shown in the table, usually were estimated on the ground by the authors.

Enough gold was recovered at only a few of the mines worked in 1932 to pay operating expenses if the prevailing wage in the district was paid to all labor. As with other forms of placer mining other than dredging, most of the hydraulic mines were worked by individual owners or lessees. At some mines lessees made wages at properties where company operation had failed, partly because of the elimination of overhead and partly because of the extra effort they put forth when working for themselves.

Operating costs

Conditions are directly comparable at very few mines, and average costs cannot be indicated correctly. The operating costs at 28 mines where no elevators or pumps were used ranged from 2.63 to 37.5 cents. (See table 12.) Labor was the major expense at all of these mines. At two mines - Indian Hill and Depot Hill in California - 2 cents per cubic yard

28 Ann. Rept., Minister of Mines of British Columbia, 1932, p. A112.

washed was spent for storing tailings back of a power dam. The charge was reduced from 3 to 2 cents in 1932. Other mines also had tailing-disposal charges in connection with mining operations. The cost of supplies and incidentals usually ranged from 1 to 2 cents. At one mine supplies cost 5 cents per cubic yard, but a power shovel was used in connection with hydraulicking. Timber for boxes is the main item of operating supplies at most hydraulic mines. Explosives at one mine - Golden Rule - cost 2 cents and at another 1 cent per cubic yard washed. At most mines where explosives were used, however, the cost of explosives was well under 1 cent per cubic yard. The cost for supervision ranged from 0.3 to 10 cents at mines where supervising officials were employed. Unless the gravel is unusually rich, the salary of a nonworking supervisor at most small hydraulic mines raises the operating cost to a prohibitive figure.

Operating costs at mines where elevators were used are as follows:

Two mines with Ruble elevators, 19 and 6 cents per cubic yard.

Three mines with hydraulic elevators, 8, 8, and 27 cents per cubic yard.

Two mines with both Ruble and hydraulic elevators, 17 and 10 cents per cubic yard.

At four mines where water was pumped for hydraulicking the operating costs were 93, 29, 38, and 28 cents per cubic yard. At two of these where electric power was used, the power cost 15 cents with a 200-foot lift and 21 cents with a 100-foot lift; in the second mine half the water was used in a hydraulic elevator. In both mines a 40-pound pressure was maintained on the nozzles. At the third mine, where gasoline engines were used, the fuel cost for a 225-foot lift was 22 cents per cubic yard. A 65-pound working pressure was maintained at the nozzle. At the fourth mine water for hydraulicking was pumped by steam at a cost of 5 cents per cubic yard. Fuel for a gravel pump cost 4.9 cents per cubic yard.

Sufficient data are not available to calculate total costs at any of the mines. In all but a few mines old ditch and pipe lines were used, which were charged off years ago. Depreciation or amortization charges could not be figured, as the quantity of gravel to be washed was known at only a few mines.

SLUICE BOXES AND RIFFLES (GENERAL)

The sluice box serves a double purpose in placer mining; it collects the gold or other heavy minerals sought within the riffles of the sluice and conveys the washed material to a dumping ground. It is an efficient gold saver and is universally used in hydraulicking and ground-sluicing. The principle of the riffled sluice is used for recovering most of the gold on dredges and in other forms of placer mining where the gravels are excavated mechanically.

Other types of gold savers have not proved generally successful in placer mining, although as an auxiliary method and under special conditions some of these gold-saving devices have been found useful. This subject is further discussed in a subsequent paper under the heading of Excavation of Placer Gravels by Power Equipment.[29]

Sluices are built in accordance with the service to be demanded of them. Riffles are of varied forms and are made of different materials. Although the form of riffle is chosen largely to fit particular conditions custom in various districts and materials at hand have a bearing upon the practices followed.

The following discussion has a general application and is not confined to any region or method of mining.

29 Gardner, E. D., and Johnson, C. H., Placer Mining in 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.

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:

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30 Bowie, A. J., Hydraulic Mining in California: Van Nostrand Co., New York, 3d ed., 1889, p. 220.

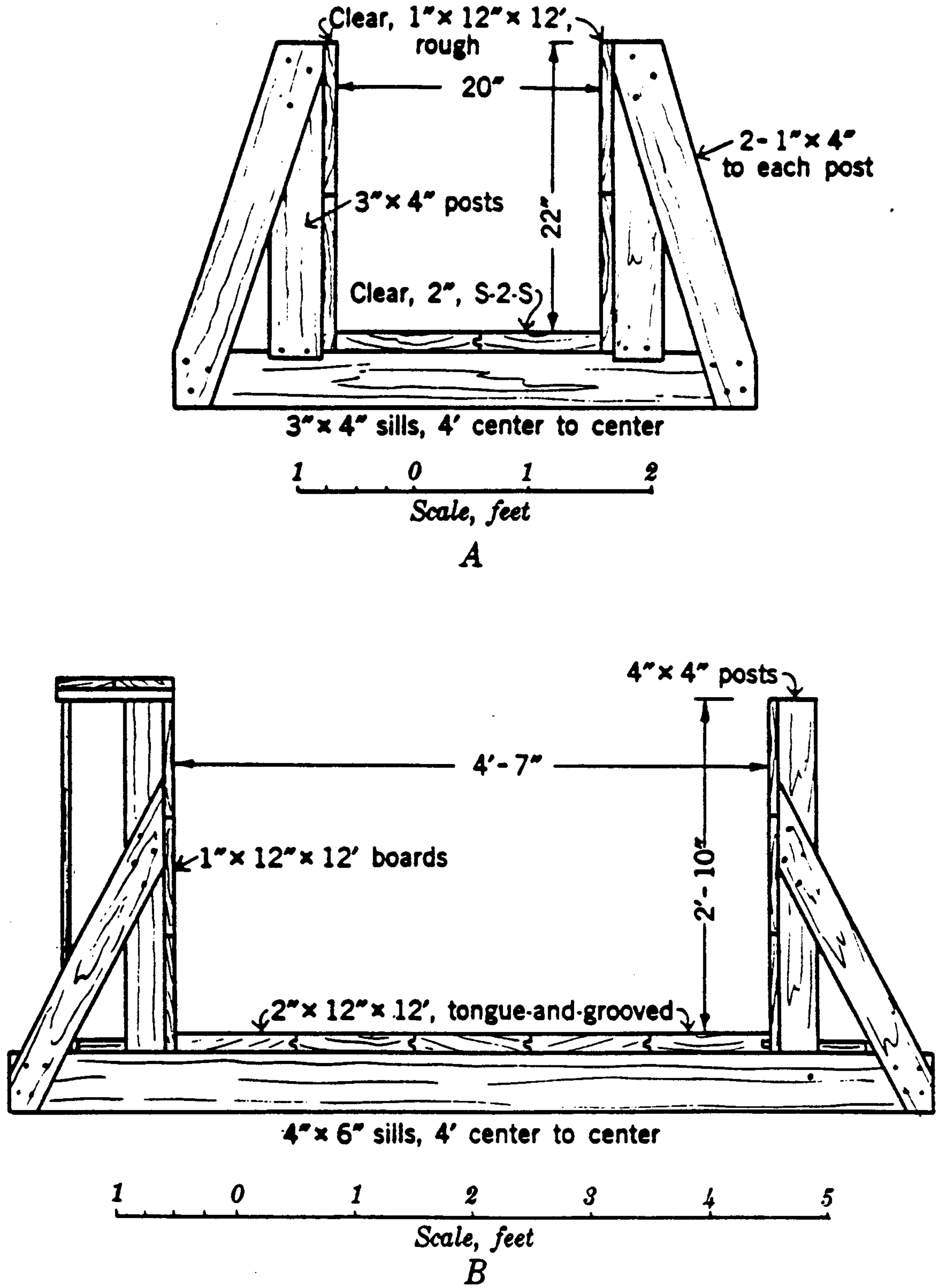

Figure 13.—Sluice-box construction: *A*, Twenty-inch box at Henderson mine, Gold Creek, Mont.; *B*, five-foot sluice box.

	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 conteur 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

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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. 523-525.

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

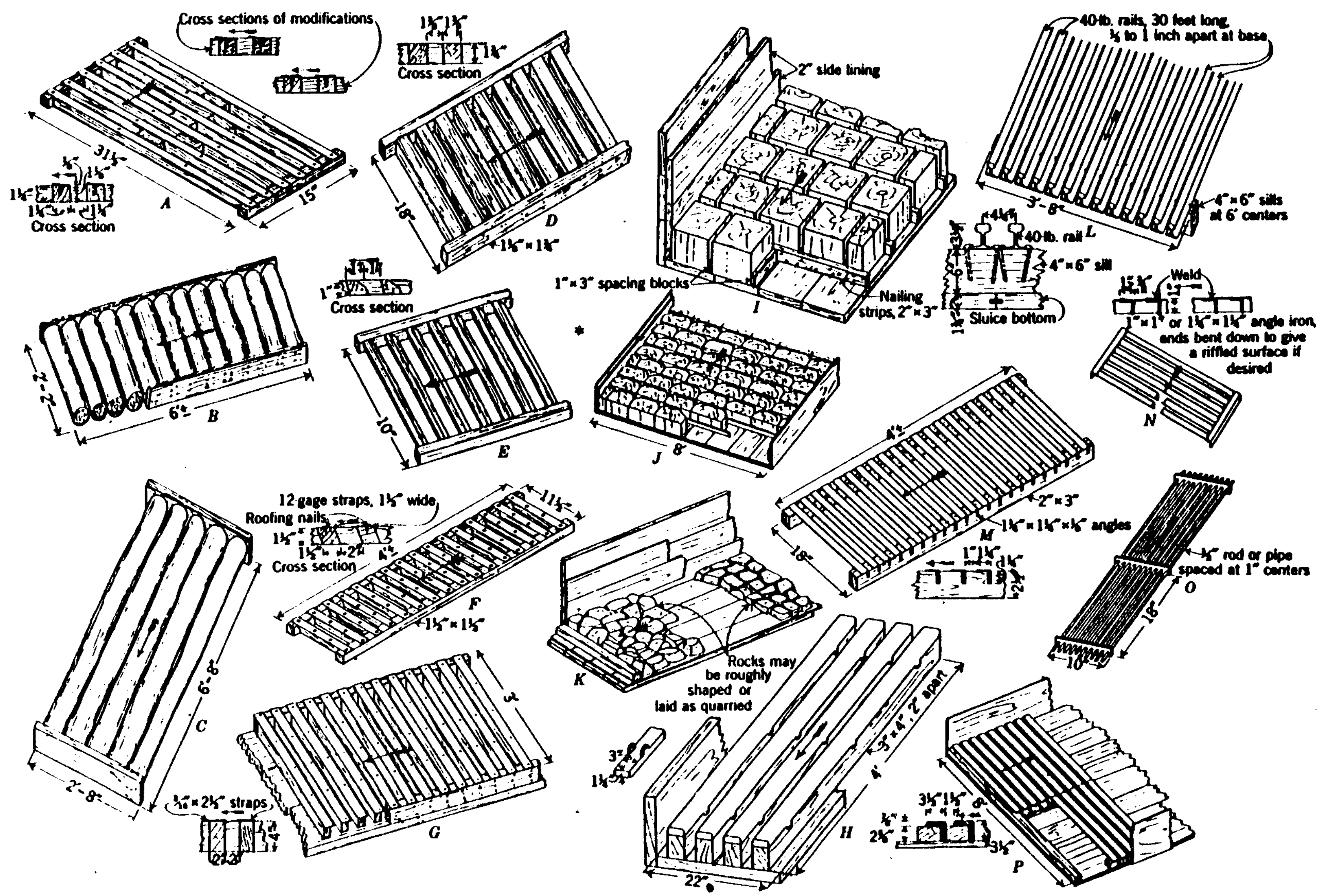

Figure 14.- Types of riffles. A, transverse wooden, steel-capped riffles used on dredges; B, transverse pole riffles; C, longitudinal pole riffles; D, transverse wooden riffles, square section; E, transverse wooden riffles, beveled section; F, transverse wooden riffle, steel-capped, inclined section; 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 toe 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,E, 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.)

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37 MacDonald, D. 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

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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.

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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, nontechnical 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 ove 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.

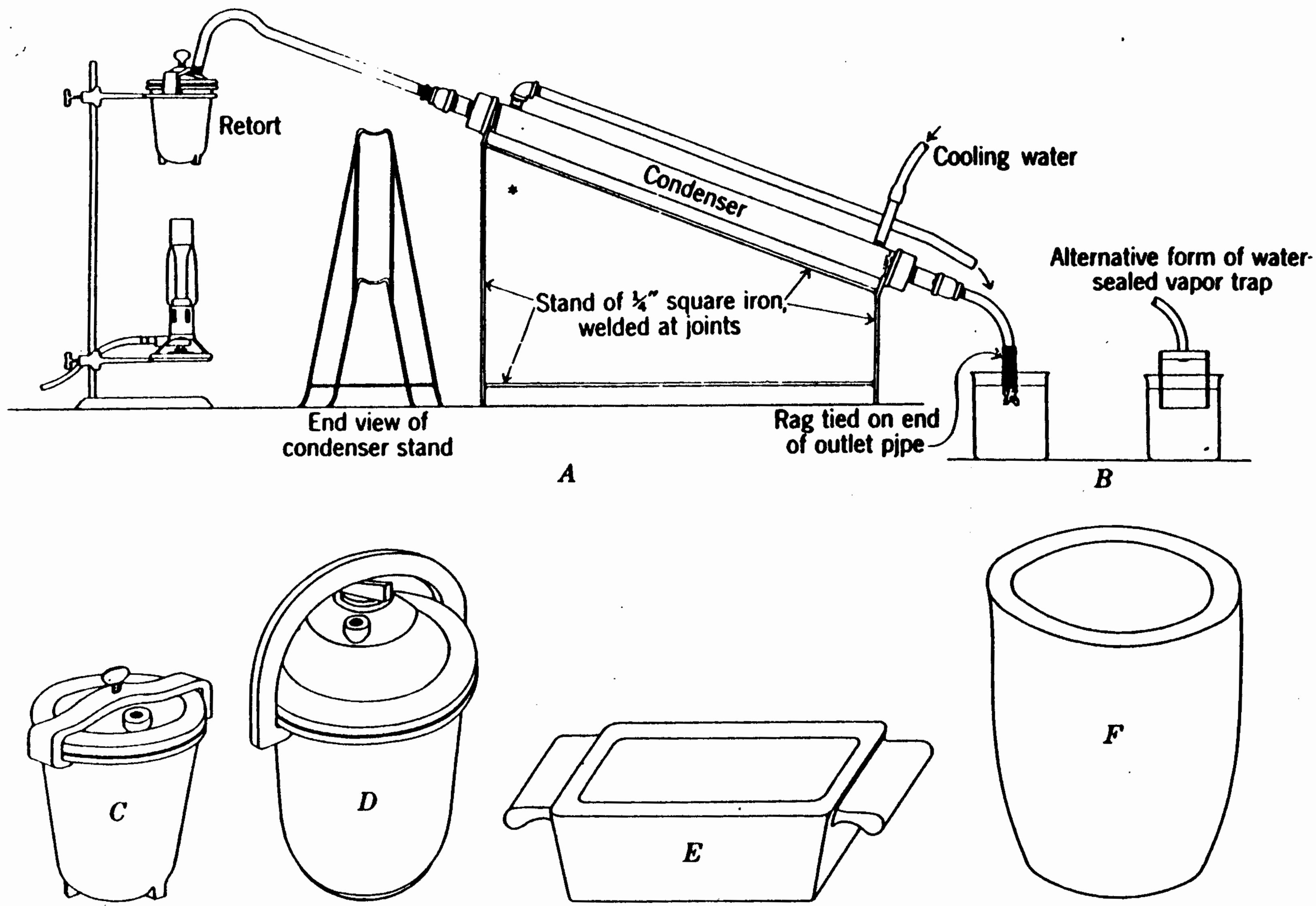

Figure 15.—Apparatus for retorting amalgam and quicksilver: *A*, Amalgam retort; *B*, Nevada-type retort; *C*, set-up of small retort; *D*, water-sealed vapor trap; *E*, graphite crucible; *F*, bullion mould.

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							
					1,000 fine		900 fine		800 fine		700 fine	
	Grams	Grains	Penny-weights	Troy ounces	At $20.67	At $35.00	At $20.67	At $35.00	At $20.67	At $35.00	At $20.67	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.- <u>Conversion table of metric and troy weights and equivalent values of gold</u>

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 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 f 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.

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52 Ricketts, A. H., American Mining Law: Bull. 98, Calif. Bur. of Mines, 1931, pp. 640-646.

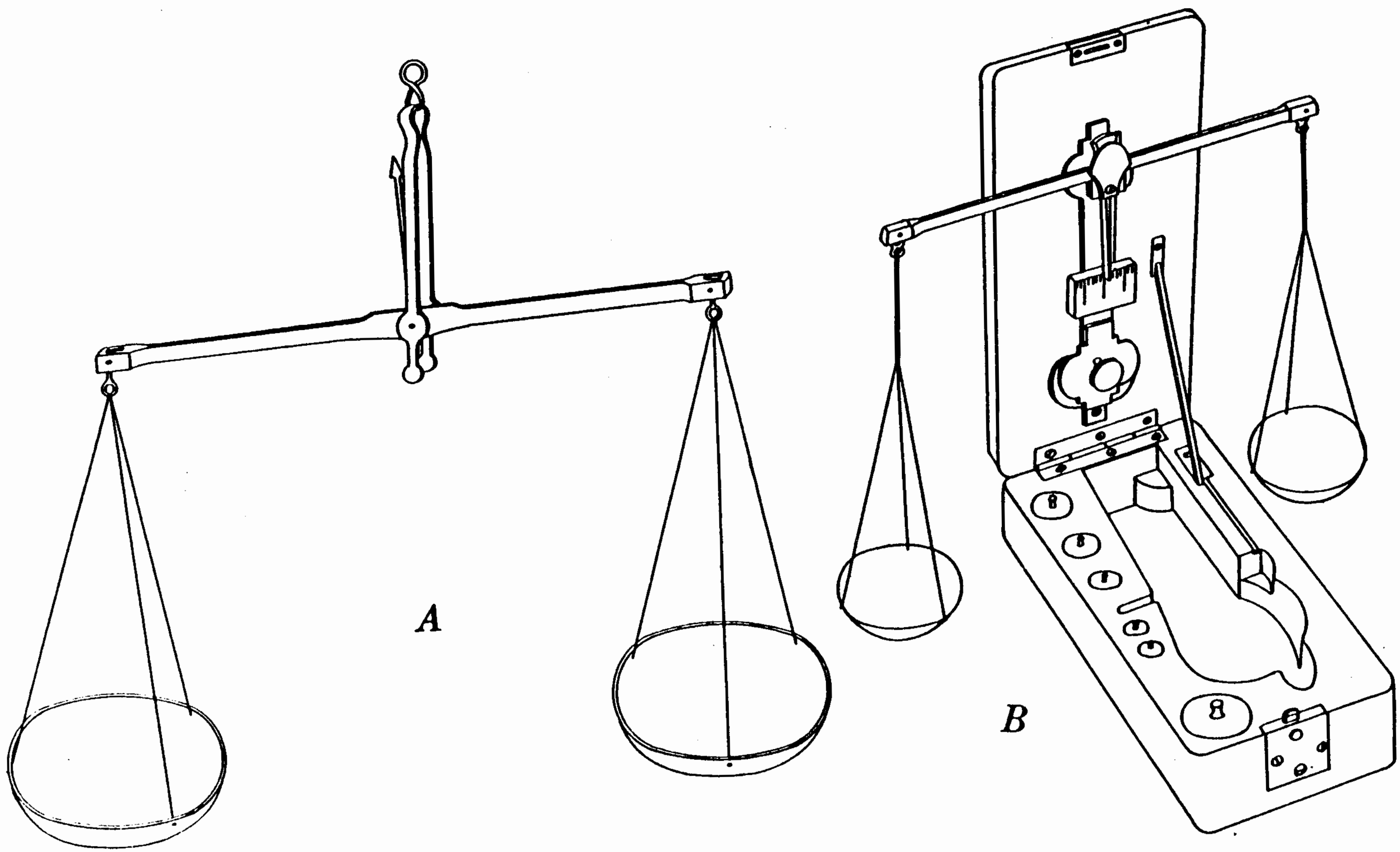

Figure 16.—Prospector's gold scales: *A*, Hand scales; *B*, folding pocket scales.