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Hydraulic Mining Methods

Compiled by

STUART S. HOLLAND

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INTRODUCTION

When placer gold was first discovered in British Columbia much of the gravel was mined by methods other than hydraulicking. Subsequently, however, with the working out of rich shallow gravel, extensive yardages of lower grade gravel were left which under favourable conditions were mined by hydraulicking. This type of mining produces the largest proportion of placer gold at present.

Placer gold was discovered in the Province at an early date so that in the well known districts most of the creeks have been prospected and worked wherever possible, leaving the more undesirable and lower grade ground for subsequent miners. All the rich ground that is known has been, or is being, worked. In the past, failure to sample and properly estimate the available yardage of placer deposits has resulted in a tremendous waste of money and effort. A large proportion of placer operations have failed because the gold in the gravel was insufficient to repay the cost even of the most efficient mining. In consequence, preliminary testing and sampling is more important now than ever before and it is particularly necessary where the investment of a large amount of money is involved.

The expense of preliminary sampling and test work should be regarded as a type of insurance from which it is possible to get valuable information necessary in making definite plans for operation and by which it is possible to save the loss of considerably more money if certain unfavourable facts and conditions can be known beforehand.

Prior to mining a placer deposit by hydraulicking, it is important to obtain information regarding the following points.

(1) The area and depth of the deposit, its average value per cubic yard, the distribution and the nature of the valuable mineral content. The minimum economic value per cubic yard depends entirely on local conditions.

(2) The supply of water available and the head or pressure obtainable, as this forms a basis for estimating the daily yardage that can be mined.

(3) The total length of ditches, flumes and pipe-lines required to bring the water to the giants at the working faces.

(4) The nature and grade of bedrock as well as its depth from the surface.

(5) The height available for the dumping or disposal of tailings; also the available area so that the tailings will not interfere with adjoining operations.

A placer deposit may be sampled by any one or a combination of methods; by panning gravel from natural exposures; by drifting, by test-pitting, by shaft-sinking, or by Keystone-drilling. In every instance in order to get reliable results the work should be done carefully and systematically so that the information may be compiled to give as complete a picture of the deposit as it is possible or economical to obtain.

A placer deposit of any considerable size can seldom be adequately sampled from its natural exposures. Nevertheless there probably are some exposures along creek banks or gulches where some panning may be done to advantage. The results from panning depend for their value largely on the man who takes the gravel for the sample. Consequently, they can be very misleading unless the sample is properly taken and the results interpreted in the light of experience. In the first place it is necessary to assume the volume of a pan full of gravel, this is generally taken as 150 to 170 to the cubic yard. Secondly it is necessary to estimate or assume the total yardage of material the pan represents; it cannot be taken for granted that a single pan will represent the average of any large volume of gravel. Nevertheless panning is valuable in determining from scattered exposures the range of values to be expected, as well as some indication as to the distribution of values in various sections of the deposit.

Drifting on bedrock, where so frequently the greatest proportion of values is concentrated, is the best method for getting average values in one part of a deposit. If the placer deposit is a buried channel, then it may be possible to obtain information as to bedrock values and consequently average values across the full width of the channel. This is especially important in hydraulic mining where it may be necessary to wash the entire channel filling and not mine the deposit selectively. If the drift be run from side to side of a channel, it is best to sluice the entire spoil from the workings, making it possible by measuring the yardage excavated to calculate the average value per yard of gravel mined. By taking pan samples in the drift it is possible to locate any pay-streak present, and by sluicing separately the material from each timbered set any variation in values will be found. A foot to 18 inches of bedrock should be taken up with the gravel to be certain that all gold is being recovered.

An alternative method of calculating the values rather than the common one based on a cubic yard of gravel is to transpose the values in terms of square feet of bedrock uncovered or cleaned, or alternatively in terms of per lineal foot of channel. In this way, particularly when most of the values are on bedrock and the overburden is barren and of variable thickness, it may be found that there is a uniformity of values which would not be apparent when the calculation is expressed in terms of cubic yards of gravel.

Shallow untimbered test pits may be dug in instances where the placer deposit is shallow and no trouble is experienced from ground water. If it is possible all the material excavated from the pit should be sluiced and the gold from it recovered and weighed. By this means a more representative sample is obtained than by attempting to take a sample of exact volume from the side or bottom of the pit. If the volume of the pit is measured as well as the area of the bottom, the value per cubic yard, or alternatively the value per square foot on bedrock may be readily calculated.

If possible, numerous test pits should be dug and should be laid out in lines according to a definite plan established in large part by the type of placer occurrence but of course modified and extended as more information is obtained.

In many deposits it is not always possible to drift on bedrock; it then becomes necessary in order to ascertain the depth to bedrock, and the values there, either to sink a shaft or put down a Keystone drill-hole. A shaft is preferable to a single drill-hole because the greater area of bedrock cleaned gives a much better sample. However, the comparative benefits should be balanced between a shaft and the number of drill-holes that might be put down for the same cost. Shafts should be sunk where possible to check drill-holes.

In loose material, a shaft must be timbered but may be sunk either by driving lagging or by cribbing, whichever is preferable or more expedient. The vertical distribution of values is found by sluicing the spoil as the sinking advances and again if possible all the material should be washed as a better average sample is thereby obtained.

Shafts almost invariably encounter water to a greater or less extent and a pump should be available to keep it under control. Where an excessive flow is encountered or can be predicted with certainty in advance then the only alternative is to do the testing by Keystone drill.

In deep placer ground where bedrock is not accessible by a drift, the longer time necessary and greater expense involved in shaft-sinking make it advisable to drill the ground. Drilling gives accurate information as to the depth of bedrock, as well as the values in the material drilled. However for dependable value-results much depends on the skill, experience and reliability of the driller. Close supervision is most important for the reason that drilling results must be adjusted and interpreted in the light of the physical conditions which are encountered; for example hard packed gravel and loose caving sand or fine gravel may give values that are either too low or too high unless the physical conditions are taken into account when the final calculation is made.

In order to determine the vertical distribution of values, the cuttings from the drill-hole are collected and panned or put through a rocker at regular intervals of depth, or it may be found convenient to have the bailer discharge directly into a small sluice-box, which is cleaned up after each pumping and which will give the values for each drive of the casing and its respective bailing.

The value of preliminary sampling and testing cannot be over-emphasized but inasmuch as no two placer deposits are identical, conditions may vary widely and distribution of values be so different, no specific method of testing can be outlined. However, the method should be adapted to each particular deposit, the programme being laid out systematically and the work directed by a capable and experienced man.

The following material in quotation marks is reprinted from Information Circular No. 6787 (now out of print) of the United States Bureau of Mines, "Placer Mining in the Western United States, Part II," by E. D. Gardner and C. H. Johnson, to which due acknowledgment is made herewith.

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

HYDRAULICKING

History[1]

Hydraulic mining was developed in California as early as 1852 when Edward E. Mattison, in order to reduce labour costs, used a rawhide hose to which a wooden nozzle was attached to direct a stream of water against the gravel bank. This step was followed by the use of a canvas hose bound with wire and rope and the use of a metal rather than a wooden nozzle. The canvas hose was replaced by 100 feet of stove-pipe by R. R. Craig at American Hill, Nevada County, California. The first metal pipe made specifically for hydraulicking was made of wrought iron in 1856 by a company in San Francisco. Difficulty was encountered with the first pipe because of the rapidity with which it rusted.

As hydraulicking became more and more used, high water pressures were wanted and the strength of the pipe led to its being made of sheet-steel in short lengths that could be packed by mules or burros.

The first nozzles were attached to the discharge end of the pipe by a short length of canvas hose, but later as higher water pressures were used a flexible iron joint was devised using two elbows working over each other and held together by a king-bolt. This arrangement was improved by using a radius plate. Various types of machine were devised to which the name "giant" was applied, culminating finally in the modern double ball-bearing giant equipped with a deflector at the end of the discharge barrel.

Application[2]

"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

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1 See Bowie, A. J. Hydraulic Mining, Van Nostrand Co. New York, 1898, p. 42.

2 Reprinted from United States Bureau of Mines Information Circular 6787.

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."

As a consequence, therefore, a proposed hydraulic operation must fulfill three essential physical conditions: (1) An adequate water supply must be available for the scale of operation intended; (2) There must be sufficient dump space to dispose of the tailings from the sluice, or lacking that, sufficient additional water and space to enable the tailings to be stacked; (3) The bedrock grade should be such that the sluices can be laid with the proper grade to carry the water and gravel efficiently, or, height must be available so that a bedrock cut may be made in which to place the sluice-boxes on their proper grade. All these, in addition to the prime requisite of sufficient average recoverable values per yard to warrant the operation.

"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." As an illustration of the value of obtaining the most water possible it is claimed that a 5 per cent increase in the amount of water used increases the yardage sluiced by approximately 10 per cent. "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-sluicing, 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

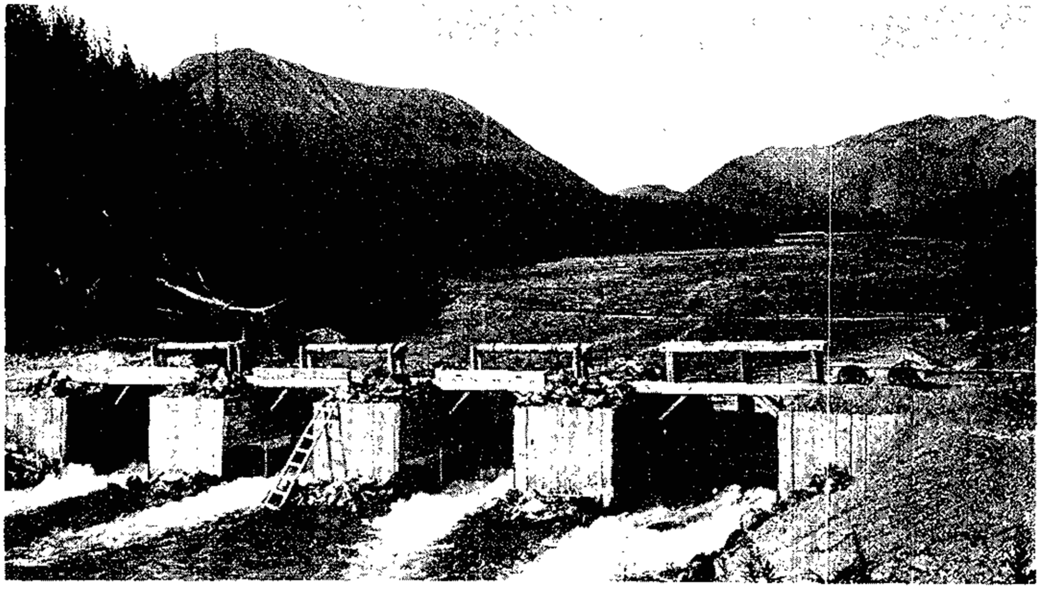


Plate I A. A storage dam at the outlet of Germansen Lake. The three spillways at the left are controlled by timbers held in vertical guides, the main one at the right by two gates operated by hand wheels. This dam is 7 feet high and impounds about 20,000 acre feet, sufficient water for a full season's operation.

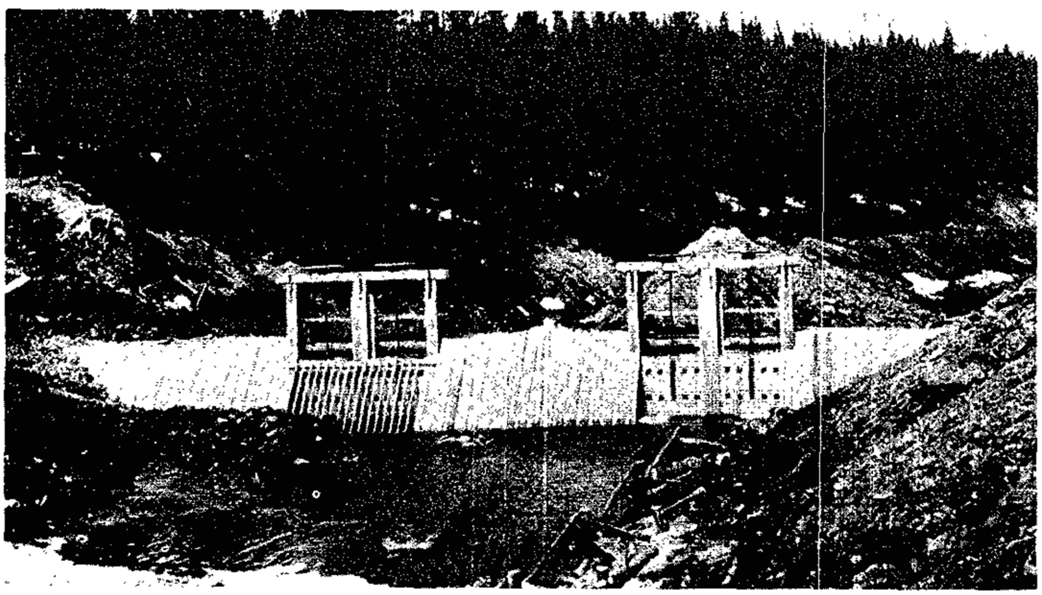


Plate I B. A diversion dam on the ditch-line of Germansen Ventures Ltd. The trash rack is in front of the gates leading into the ditch; the waste gates at the right control flow in a natural water course.

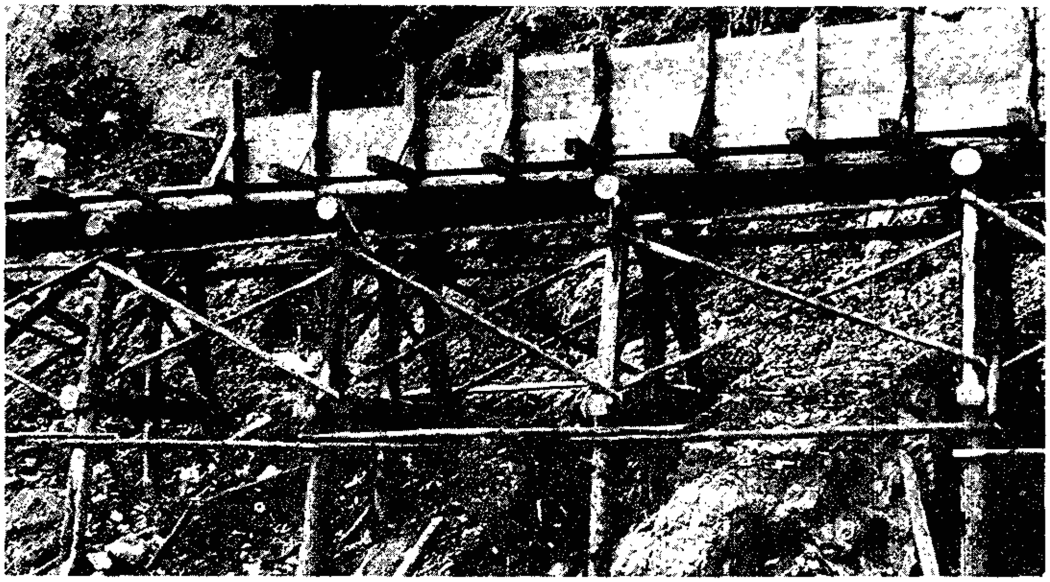


Plate II A. A flume built on a trestle across a steep slide. The flume is 6 feet wide and 5 feet deep; note the well-supported bents of heavy timbers with centre posts.

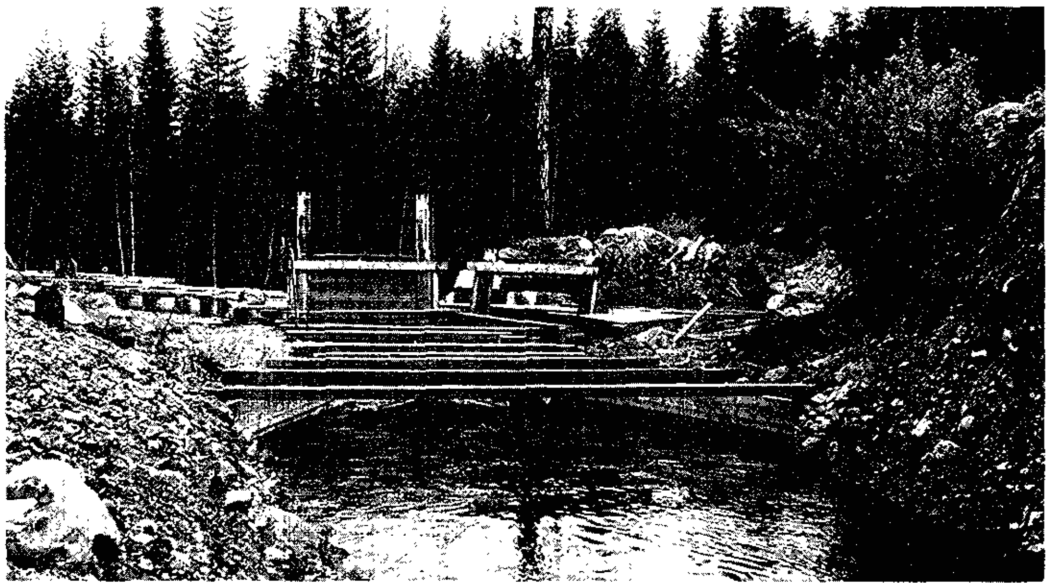


Plate II B. Control gates at the end of the ditch at the intake to the penstock at Bullion Mine.

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 tail races.

"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 promotors 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 may be 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 the quantity and head of the hydraulic water affect the costs, but no general grouping is possible in accordance with any of these heads.

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."

In British Columbia the Bullion Mine has about 23 miles of ditch-line, the Lowhee 25 miles and Germansen Ventures Ltd., 10 miles of ditch and flume.

"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 if such can be obtained. 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." Weeds will hardly grow in water flowing 2 to 2 1/2 feet per second.

"The following are recommended as maximum mean velocities for ditches in various materials:

Material	Mean velocity
	Feet per second
Silt	0.5
Loose sand	1
Sandy soil	2
Firm soil, firm sandy loam	3
Stiff clay, gravel	4
Coarse gravel, cobbles	5
Cemented gravel, soft rock	6
Hard rock	10

"The figures on the preceding page represent mean velocities, the corresponding bottom velocities being 20 to 30 per cent lower, and the corresponding surface velocities, as measured by floating objects, possibly being 25 to 35 per cent 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 Kutter formula:

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

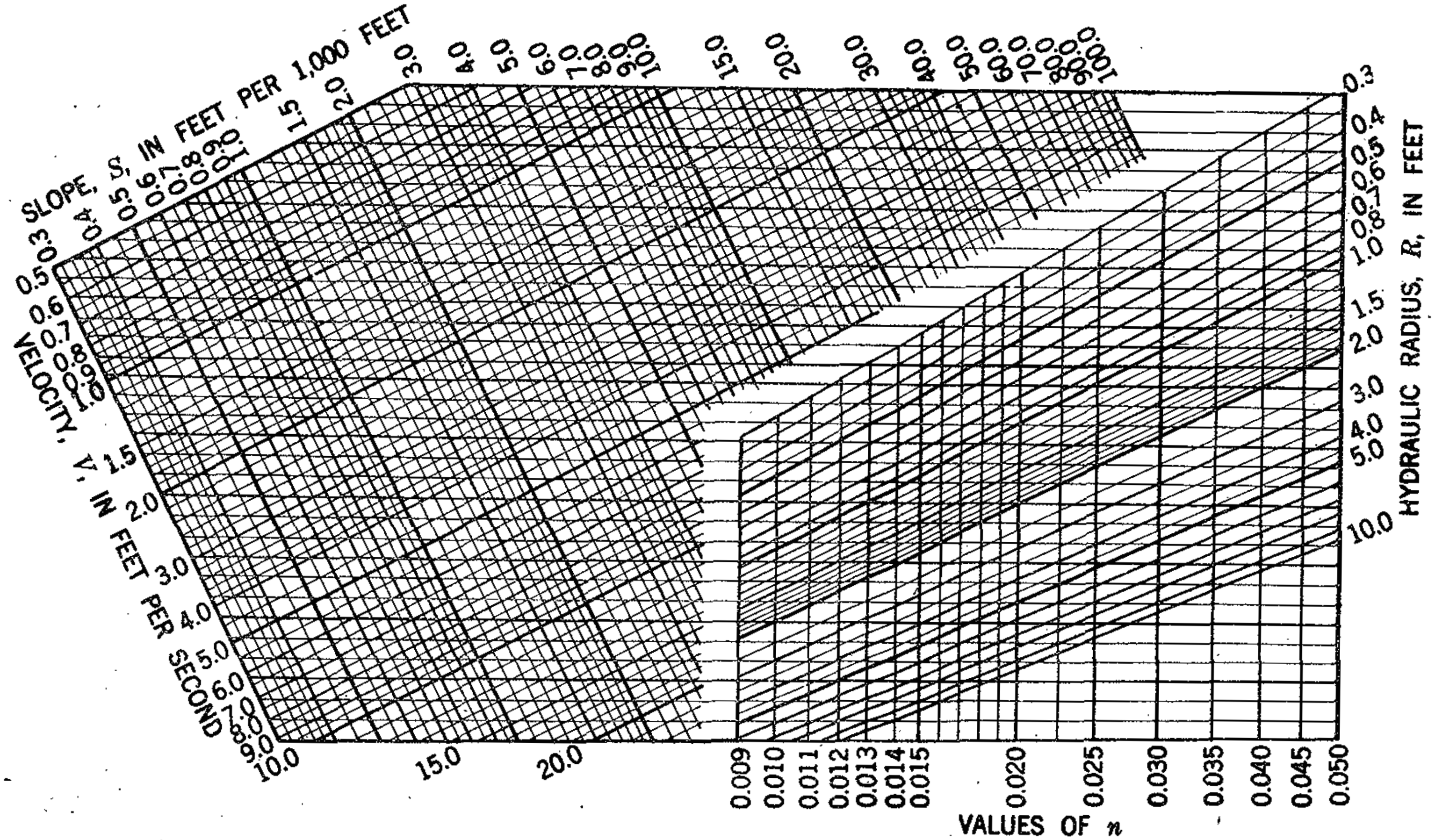
In which

V = mean velocity (in feet per second)
"n" = coefficient of roughness
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 F. C. Scobey of the United States Department of Agriculture. The proper value to use for the coefficient 'n' is a matter of judgment. The following values of 'n' are recommended by modern designers.

Values of Roughness Coefficient "n"

Surface	Best	Good	Fair	Bad
Coated cast-iron pipe	.011	0.012x	.013x	
Commercial wrought-iron pipe				
Black	.012	.013	.014	.015
Galvanized	.013	.014	.015	.017
Riveted and spiral steel pipe	.013	.015x	.017x	
Wood-stave pipe	.010	.011	.012	.013
Plank flumes				
Planed	.010	.012x	.013	.014
Unplaned	.011	.013x	.014	.015
With battens	.012	.015x	.016	



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.

Reprinted from United States Bureau of Mines Information Circular 6787.

Semi-circular metal flumes				
Smooth	.011	.012	.013	.015
Corrugated	.0225	.025	.0275	.030
Canals and ditches				
Earth, straight and uniform	.017	.020	.0225[x]	.025
Rock cuts, smooth and uniform	.025	.030	.033[x]	.035
Rock cuts, jagged and irregular	.035	.040	.045	
Winding sluggish canals	.0225	.025[x]	.0275	.030
Dredged earth channels	.025	.0275[x]	.030	.033
Canals with rough stony beds; weeds on earth banks	.025	.030	.035[x]	.040

[x] 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. Thus the velocities and capacities shown in Table 1 might be increased 15 or 20 per cent 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 degrees, 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[1] who tabulates data on 35 Alaskan ditches, states that side slopes of 45 to 65 degrees 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[2] 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 per cent of the intake water by seepage, even in good soil, and in porous soil, as much as 20 per cent." As an illustration, at Germansen Ventures Limited, Germansen Creek, B. C., starting with an initial flow at the intake of 135 cubic feet per second there was a loss of about 25 cubic feet per second through seepage in 3 miles of flume and 6 miles of ditch largely in fairly impervious boulder-clay.

"Remedies where the loss is serious are to decrease the size of ditch and increase the velocity; to reduce the velocity to a

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

2 Peele, Robert, Mining Engineers' Handbook: John Wiley and Sons, 3rd. ed., 1941, p. 38-26.

point at which 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. Puddling when sufficient clay is available will tend to seal much of the seepage.

"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 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 the 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 water flowing over the crest. The bottom of the notch should be level and the sides vertical. The weir notch is bevelled 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

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

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

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

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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.

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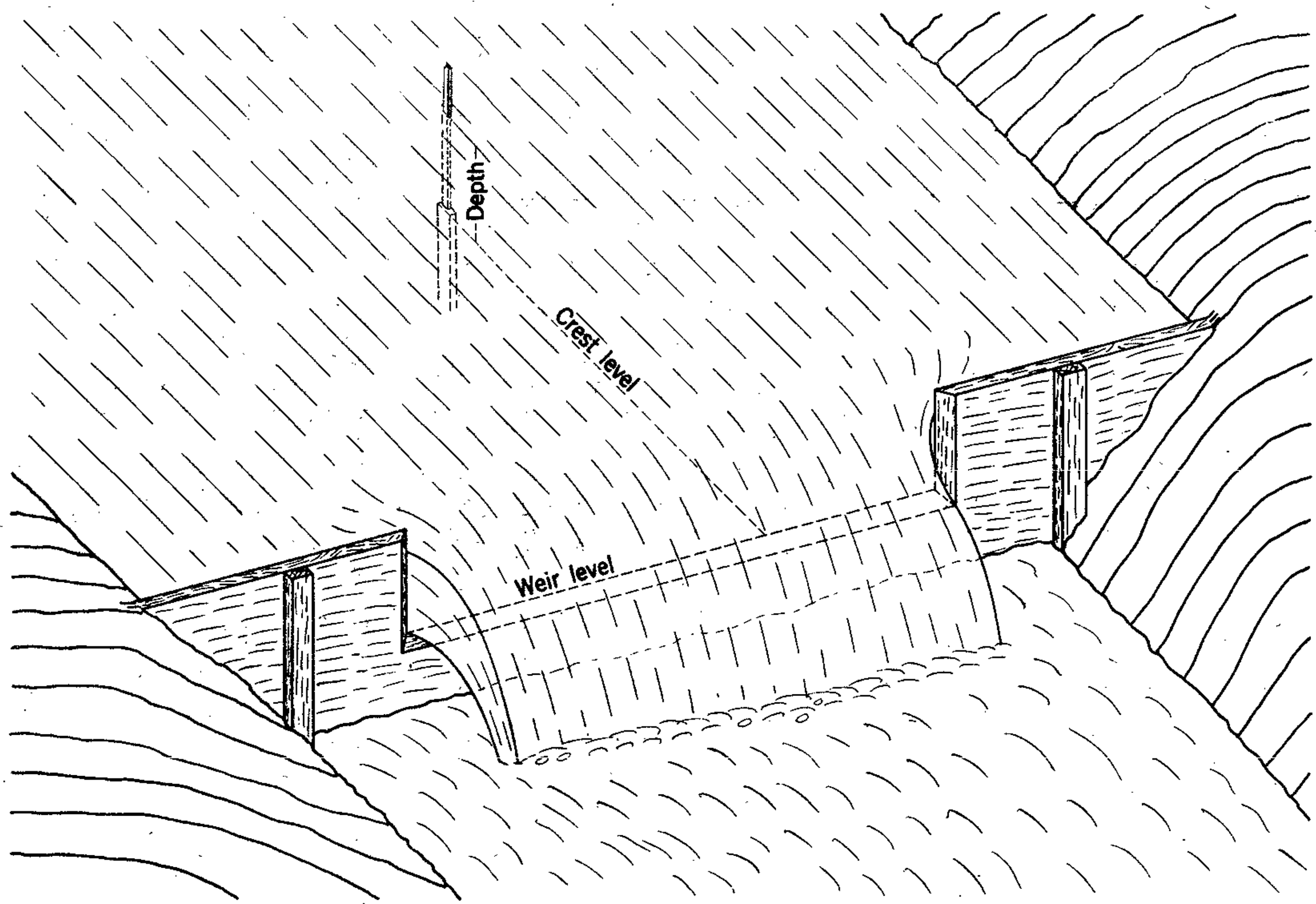

Figure 2.—Weir for small stream.

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there is free access of air to the under side 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 in cubic feet per second and miner's inches[1] 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

"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 (see Plate IIA). 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 page 9) 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 to less than 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. For lowest friction losses the width of the flume should be made twice the water depth and a freeboard of 1 to 2 feet allowed. According to Egleston[2] the usual water velocity

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1 In this bulletin a flow of 1 cubic foot per second is equivalent to 40 miner's inches.

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

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

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

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is 4 to 9 feet per second (3 to 6 miles per hour). The same author gives the range in grade in 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[1] 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 to give the same head.

"The construction of wooden flumes has changed little since the early days of placer mining in California. Figure 3 A 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 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."

An adequate number of spillways should be provided to divert water in case of damage or break in the flume-line.

Diversion Dams and Reservoirs

"Diversion dams for hydraulic ditch-lines usually are earth-filled timber cribs or rock-filled cribs faced with boards (see Plate I). Small streams often are dammed by throwing logs across

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

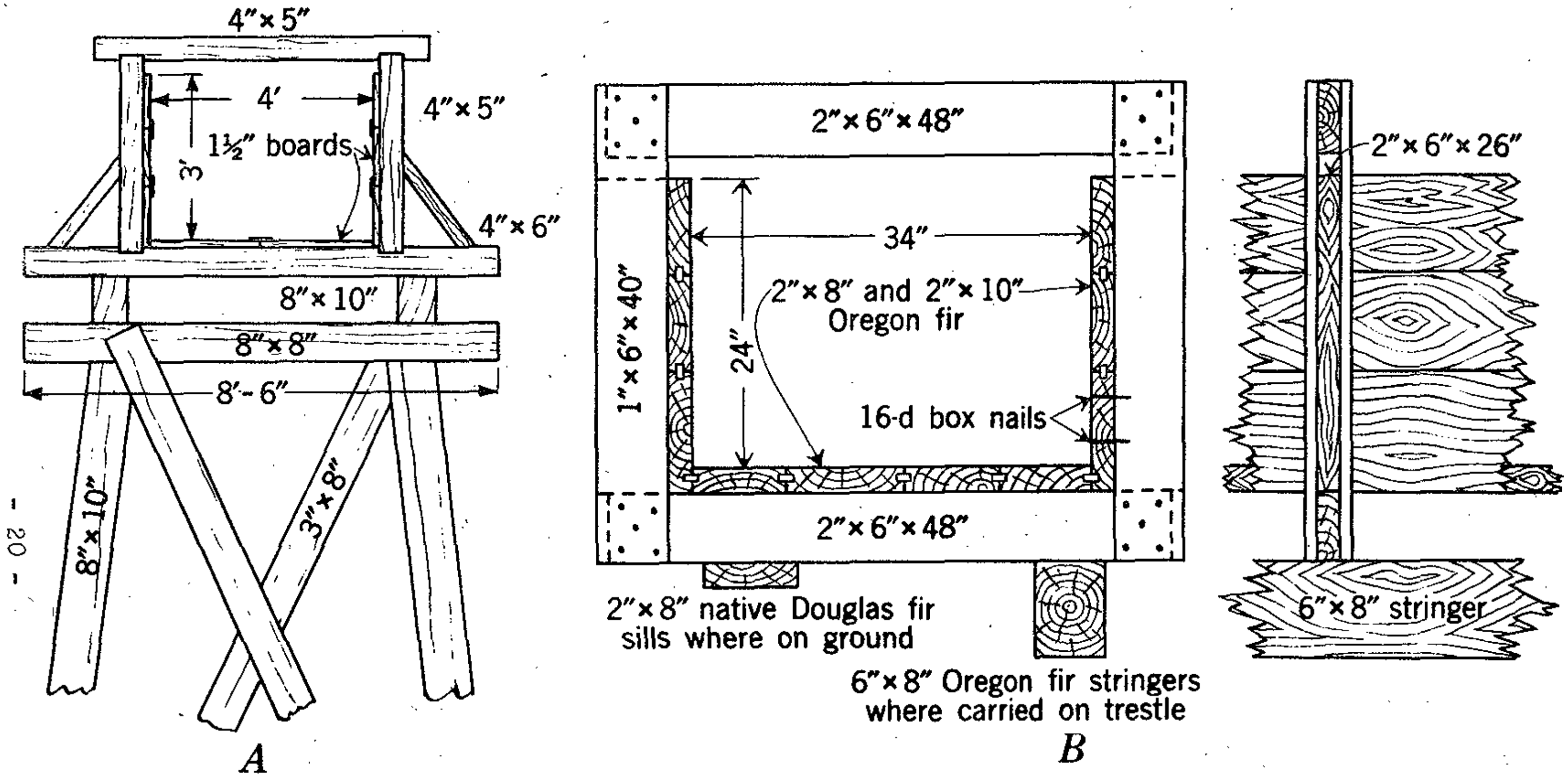

Figure 3.—Flume for hydraulic mines: *A*, Flume and trestle used in California; *B*, flume at Questa, N.Mex.

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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 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 may be used if the gravel contains an excessive quantity of large boulders (see Plate XIV A). 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, or may be used for drilling tough boulder clay preparatory to bank blasting should that be found expedient.

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 (see Plate III B). Occasionally, where the grade of a creek is steep, the water will be diverted from the stream directly into a pipe-line. Although wood stave pipe is used at a few properties, steel pipe is preferred at nearly all hydraulic mines."

The installation of the pipe-line is a very important part of the hydraulic plant. Sharp bends and angles should be avoided as well as rapid reduction in size of the pipes. Above all, the line should be laid so that no point rises above the hydraulic gradient.

The first consideration in design is to have a pipe of sufficient size to carry an ample margin of water over and above immediate requirements.

"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 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; 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 and strength of riveted pipe with slip joints. The weights are for pipe double dipped with asphaltum coating.

Table 3.

Pipe diameter inches	Gage No.	Wt. per foot, pounds	Safe head, feet	Pipe diameter inches	Gage No.	Wt. per foot, pounds	Safe head, feet
6	16	5.3	340	16	16	13.4	158
6	14	6.4	490	16	14	16.3	198
7	16	6.2	325	16	12	22.3	277
7	14	7.4	450	16	10	28.2	356
8	16	7.0	315	18	16	15.1	140
8	14	8.4	394	18	14	18.3	175
8	12	11.6	553	18	12	24.9	246
9	16	7.8	280	18	10	31.6	316
9	14	9.4	350	20	16	16.7	126
9	12	12.9	490	20	14	20.3	158
10	16	8.6	252	20	12	27.5	221
10	14	10.4	316	20	10	35.0	284
10	12	14.3	443	22	16	18.3	115
10	10	18.1	568	22	14	22.2	143
11	16	9.5	230	22	12	30.1	201
11	14	11.4	287	22	10	38.5	258
11	12	15.6	402	24	16	19.8	105
11	10	19.7	517	24	14	24.2	131
12	16	10.3	210	24	12	32.6	184
12	14	12.4	263	24	10	41.9	237
12	12	16.9	368	26	14	26.2	121
12	10	21.4	473	26	12	35.1	170
13	16	11.1	194	26	10	45.3	219
13	14	13.4	243	28	14	28.2	113
13	12	18.3	340	28	12	37.6	158
13	10	23.1	437	28	10	48.7	203
14	16	11.8	180	30	14	30.1	106
14	14	14.4	226	30	12	40.1	147
14	12	19.6	317	30	10	52.0	189
14	10	24.8	406	32	12	42.5	137
15	16	12.6	168	32	10	55.4	177
15	14	15.4	211	34	12	45.0	129
15	12	20.9	297	34	10	58.7	166
15	10	26.5	379	36	12	47.5	122
				36	10	62.0	156

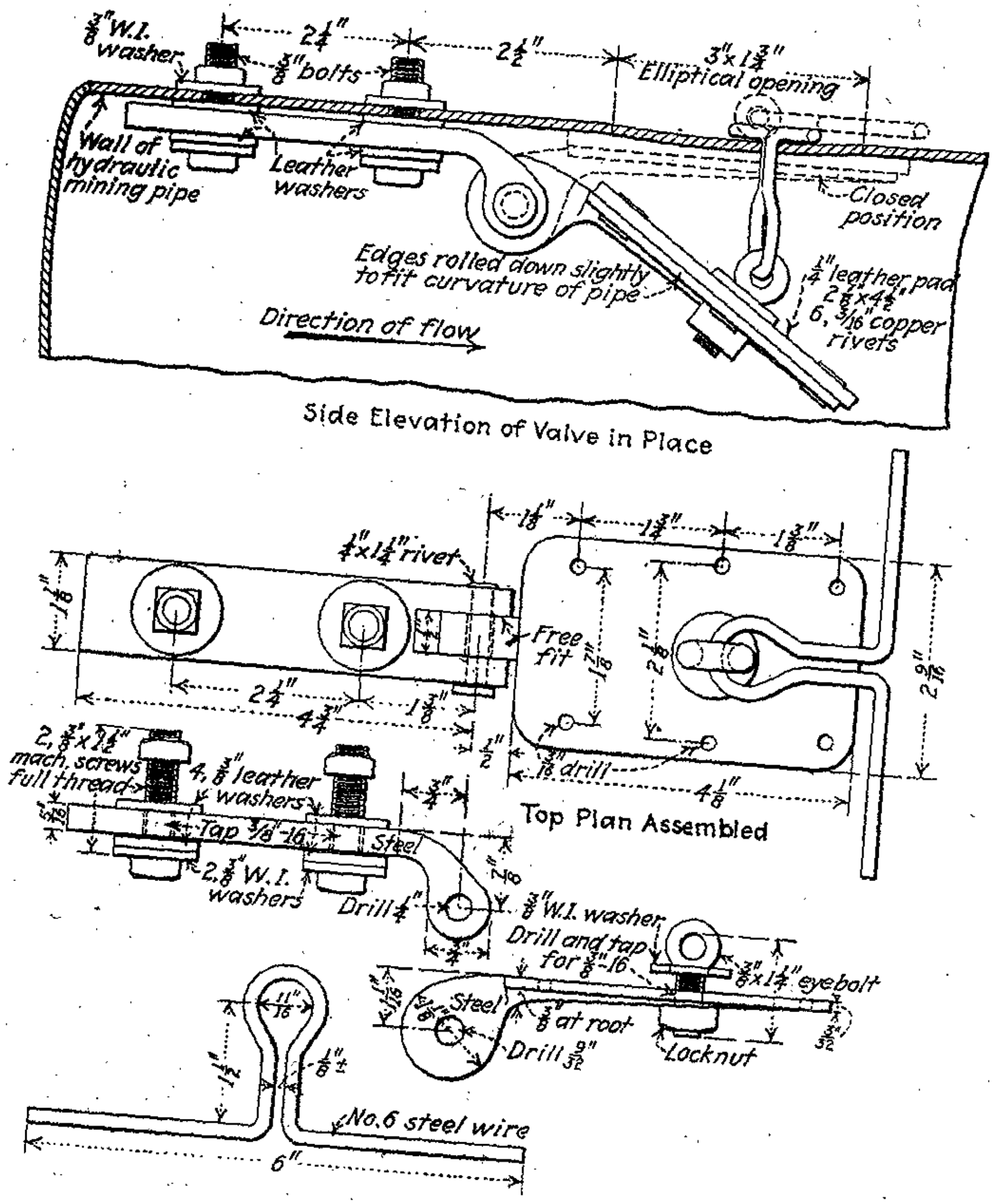

Figure 4.—Air-vent valve for pipe lines, Salyer, Calif.

Reprinted from United States Bureau of Mines Information Circular 6787.

Joints and Valves. "In making pipe with slip joints the diameter of one end is slightly contracted. This joint in straight pipe-lines will withstand most pressures encountered at placer mines. Slip joints however become battered from frequent laying but may be hammered back to shape.

"Riveted elbows furnished by the pipe manufacturers generally are used for making turns in pipe-lines. It is good practice to make bends so that the radius of the bend is equal to 5 times the diameter of the pipe. Taper joints are used where reductions are made in lines. Sudden reductions in size should 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. They also allow air to escape when the line is being filled. Venting also is necessary to prevent air pockets in the line, and it is hardly possible to have too many air valves. Fig. 4 shows an air vent used at the Salyer mine in Trinity County, Calif.[1] 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 (see Plate IV A). 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 or grizzly should be placed at the head of the line to keep out trash, also a settling box should be provided where sand and gravel may settle before the water goes into the pipe, as such material may cause rapid wear of the nozzles of the giants. A spillway is necessary in case too much water is turned into the pipe-line.

Laying pipe-lines. "Pipe-lines are laid by beginning at the bottom and working upwards. 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 should be

[1] Eng. and Min. Journ., Vol. 131, 1931, p. 161.

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, using chisel-edged pipe tools. The upper pipe is driven home by hammering on a block of wood placed at the upper end. Where the pipe has been battered from previous handling, wetted burlap or sacking may be wrapped around the joint before driving. If leaks develop they may be stopped by shovelling sawdust into the pressure box, or by driving in thin wooden wedges; sometimes an outside band is required."

Timbers should be placed at fairly close intervals beneath the pipe-line to keep it off the ground and to give it a firm foundation (see Plate IV B). All bends in the pipe-line either lateral or vertical as well as all gate valves should be firmly anchored to withstand the thrust of the water by being loaded with rock or firmly braced against stumps.

"In placing pipes with flanged joints they are laid end to end and the bolts put through and tightened. 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."

In filling a pipe-line for the first time care should be taken not to admit the water too fast otherwise shock from included air or contraction may cause severe damage. New pipe-lines, when first put in use, often show numerous small leaks; these can often be stopped by feeding in small quantities of sawdust at the intake.

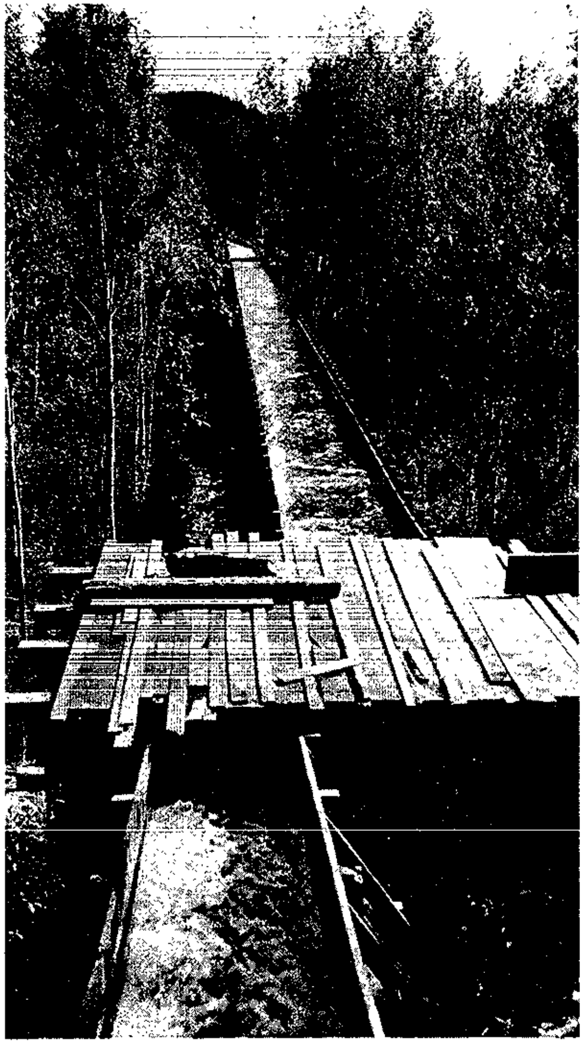

Plate III A. A baffle box to check the water velocity at the bottom of section of flume having a steep grade.

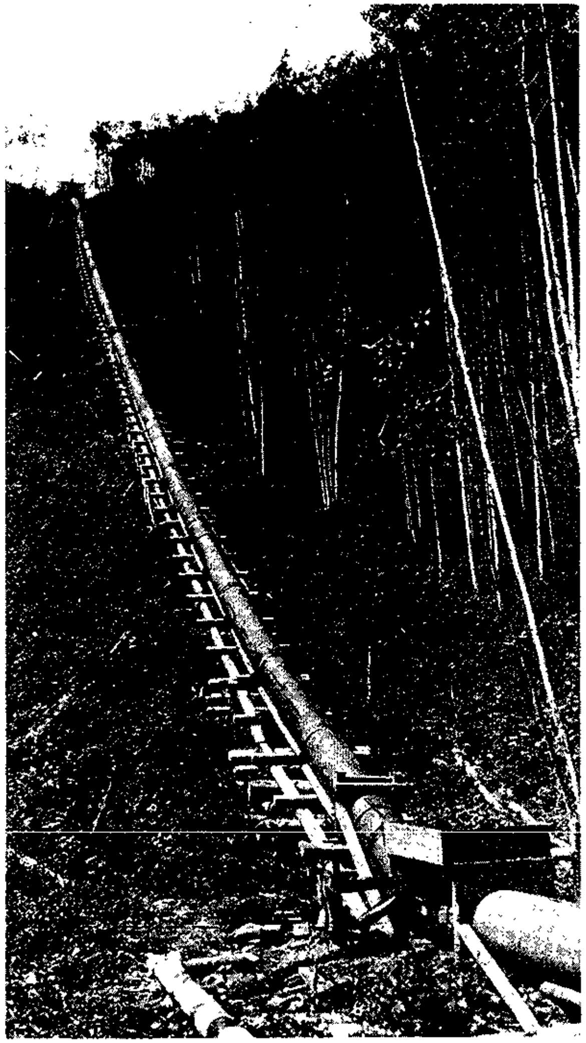

Plate III B. A section of 24-inch pipe-line leading from the penstock at Cariboo Cottonwood Placers Ltd. This is a well-laid pipe-line; notice the supporting cribbing for the pipe and the bracing at the change in grade of the pipe.

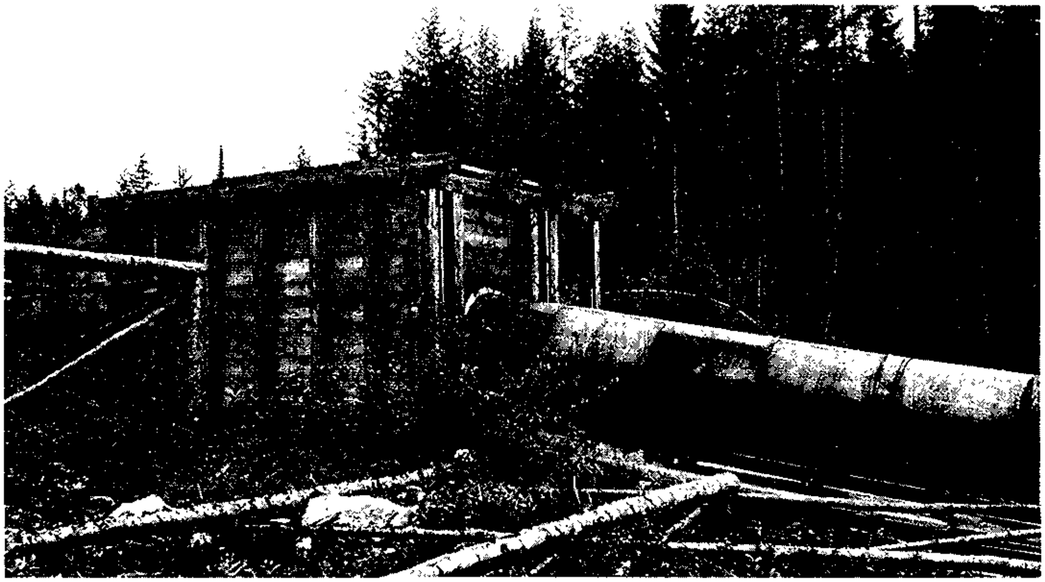

Plate IV A. The pressure box at the Bullion Mine. The outlet pipe is 54 inches in diameter.

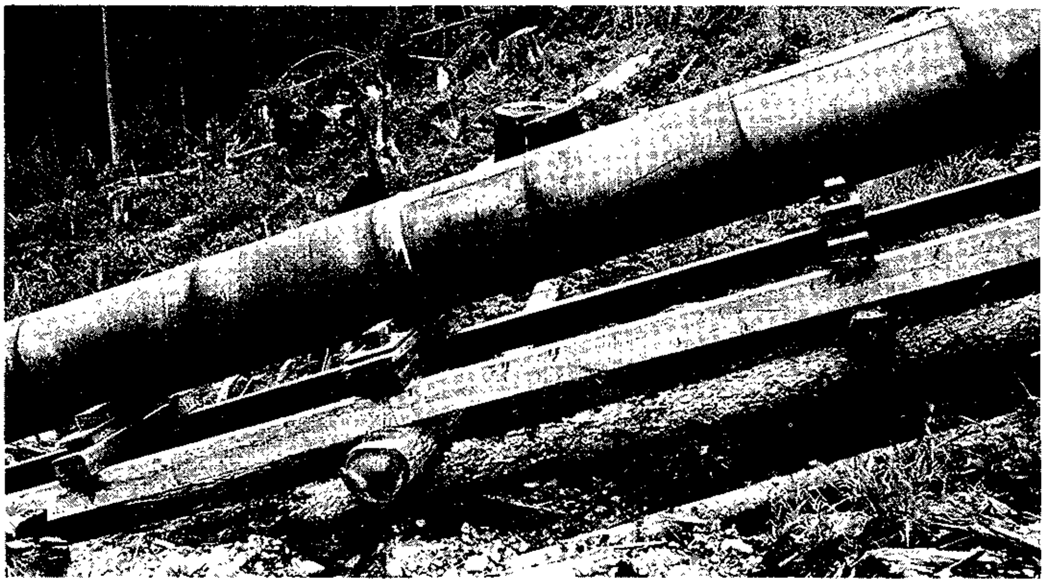

Plate IV B. Part of the main 30-inch section of the Bullion pipe-line showing the foundation supports.

"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, Oregon, in open country in the spring of 1932.

<u>Flow of water through pipes</u>. "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 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 on 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; especially, 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 per cent larger than would be indicated if there were no loss due to friction. Flow through an unobstructed pipe-line of uniform diameter can be calculated from a number of formulas. The Kutter modification of the Chezy formula appears to be preferred by hydraulic engineers. The Chezy formula may be stated as:

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

The Kutter modification of the Chezy formula is:

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

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

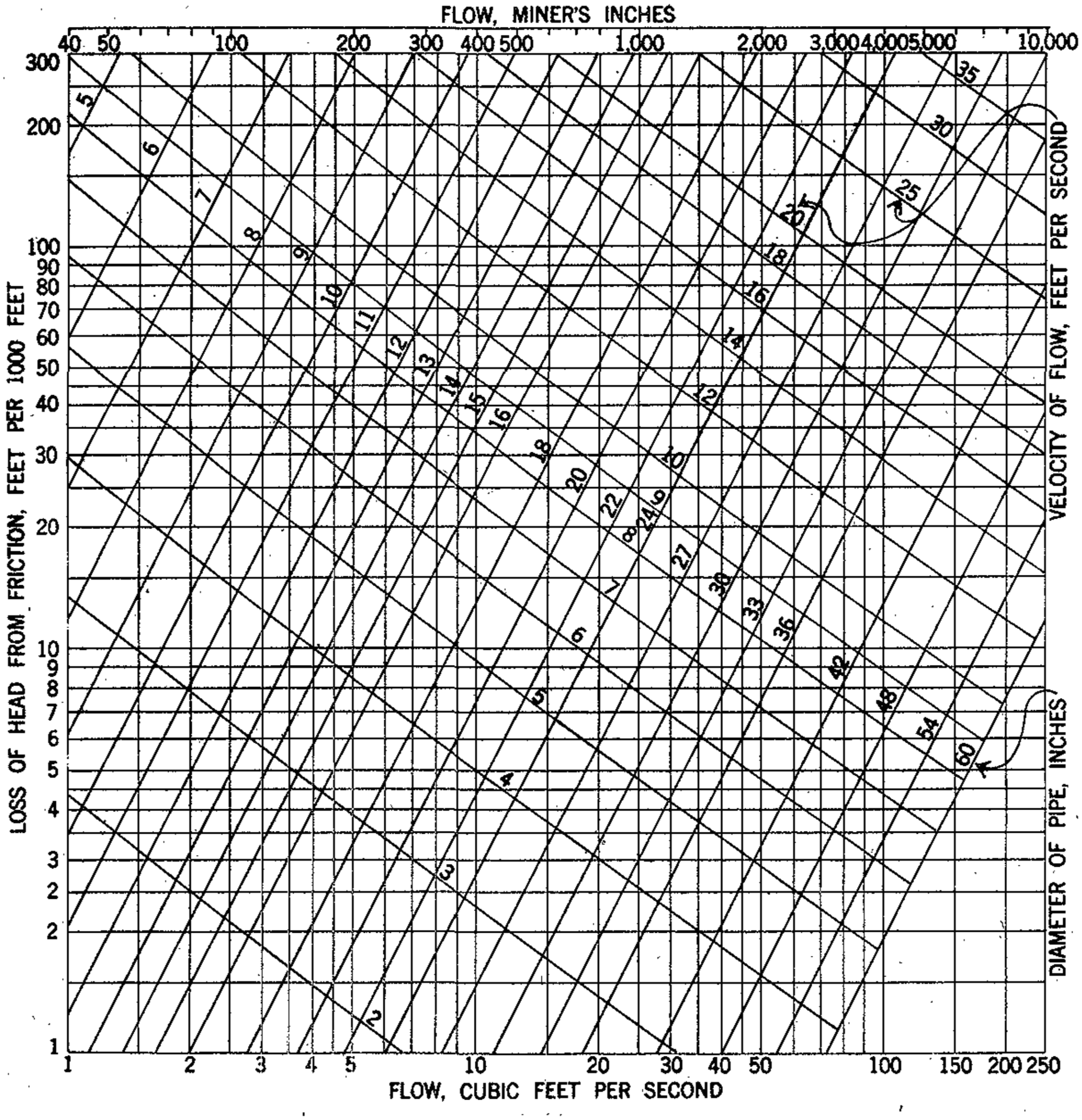

Figure 5.—Chart showing loss of head in pipes due to friction (N=0.015).

Reprinted from United States Bureau of Mines Information Circular 6787.

"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 5.

Selection of diameter of pipe-line for a given flow of water

"The chart shown in Figure 5 will assist 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 1000 feet of length, 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 1000 feet of line. If 400 feet of this diameter pipe were used in the line the loss of head would be 13 feet. With 15-inch pipe the loss per 1000 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 1000 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 per cent of the actual head. By using all 18-inch pipe the total loss would be only 10 feet. If the gravel is easy to cut and need not be swept long distances a 22-foot head loss may not be serious. In light gravel, however it probably should be economical to use just the 16- and 18-inch diameters, or possibly to construct the whole line of 18-inch. 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 because 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 machine with a nozzle for directing and controlling a stream of water under a hydraulic head. The monitor can swing horizontally through a full circle and from about 10 degrees below to 50 degrees above the horizontal. A standard monitor is shown in Plate V. The box of stones is used to counterbalance the weight of the discharge barrel. A monitor generally is set up in a hydraulic 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 allowances for variation in the quantity of water used. The monitor and nozzles are constructed with straight vanes in the discharge barrel so that a rotary motion of the jet is prevented, and the water is discharged in a solid column. Monitors are made for a wide range of service in sizes numbered 0 to 9 inclusive." Probably the largest monitor made, number S-12, is in use at the Bullion Mine, B. C., its largest nozzle is 12 1/4 inches in diameter and the total weight of the machine is 3200 pounds.

"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. 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 monitor in the opposite direction.

"Table 4 shows the sizes and weights of giants and deflectors made by one manufacturer; other companies make similar equipment.

Table 4. Sizes and weights of double-jointed, ball-bearing monitors and deflectors

Monitors				Deflectors
Size No.	Diameter of pipe inlets inches	Diameter of butts with nozzle detached inches	Shipping Weight pounds	Weight pounds
0	5	3	350	None required
1	7	4	390	30
2	9	5	520	40
3	11	6	890	45
4	11	7	1075	55
5	13	8	1475	70
6	15	9	1850	75
7	15	10	2100	80
8	18	10	2300	80
9	18	11	2450	90

Table 5. Flow of Water through Giants

Adapted from table in Catalogue of Joshua Hendy Iron Works, San Francisco, Calif.

<u>Effective Head</u>

Giant No.	Diam. of Nozzle inches	100 Cubic feet per second	100 Miners' inches	200 Cubic feet per second	200 Miners' inches	300 Cubic feet per second	300 Miners' inches	400 Cubic feet per second	400 Miners' inches
0	1 1/8	0.6	22	0.8	31				
0	1 3/8	0.8	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	1100
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	1110
6	6	13.8	550	19.6	780	24.1	960	27.9	1120
6	7	18.7	750	26.7	1070	33.2	1330	37.7	1510
7	6	14.2	570	20.0	800	24.5	980	28.3	1130
7	7	19.0	760	26.9	1080	33.3	1330	38.0	1520
8	7	19.2	770	27.2	1090	33.8	1350	38.3	1530
8	8	25.2	1010	35.3	1410	43.7	1750	48.7	1950
9	9	32.0	1280	45.0	1800	55.3	2210	63.7	2550
9	10	39.3	1570	55.3	2210	68.2	2730	78.7	3140

Sizes larger than No. 3 are substantially constructed and usually are equipped with a ball-bearing kingbolt: For heads of 400 feet or more heavy lugs may be used at the joints as a safety precaution.

Discharge through nozzles. "Table 5 gives the discharge through different sizes of nozzles under heads from 100 to 400 feet. In this table 40 miners' inches is considered as 1 cubic foot per second. The theoretical flow of water through nozzles exceeds the figures in Table 5 by about 10 per cent; 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:

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

where Q = cubic feet of water per second
A = area of nozzle in 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.

Hydraulic Elevators

"Hydraulic elevators are used to raise gravel, sand and water out of placer pits into sluice-boxes. The ground to be worked should be relatively free from big boulders and tree stumps, as these not only hinder work, but may require blasting and breaking up which adds to the cost of operation. 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 6 [1]. 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, in practice it is found that the maximum height of lift is about 17 per cent of the effective head at the nozzle of the elevator.

"The volume of gravel that can be handled by an elevator depends primarily upon the head and volume of pressure water

1 After Joshua Hendy Iron Works catalogue.

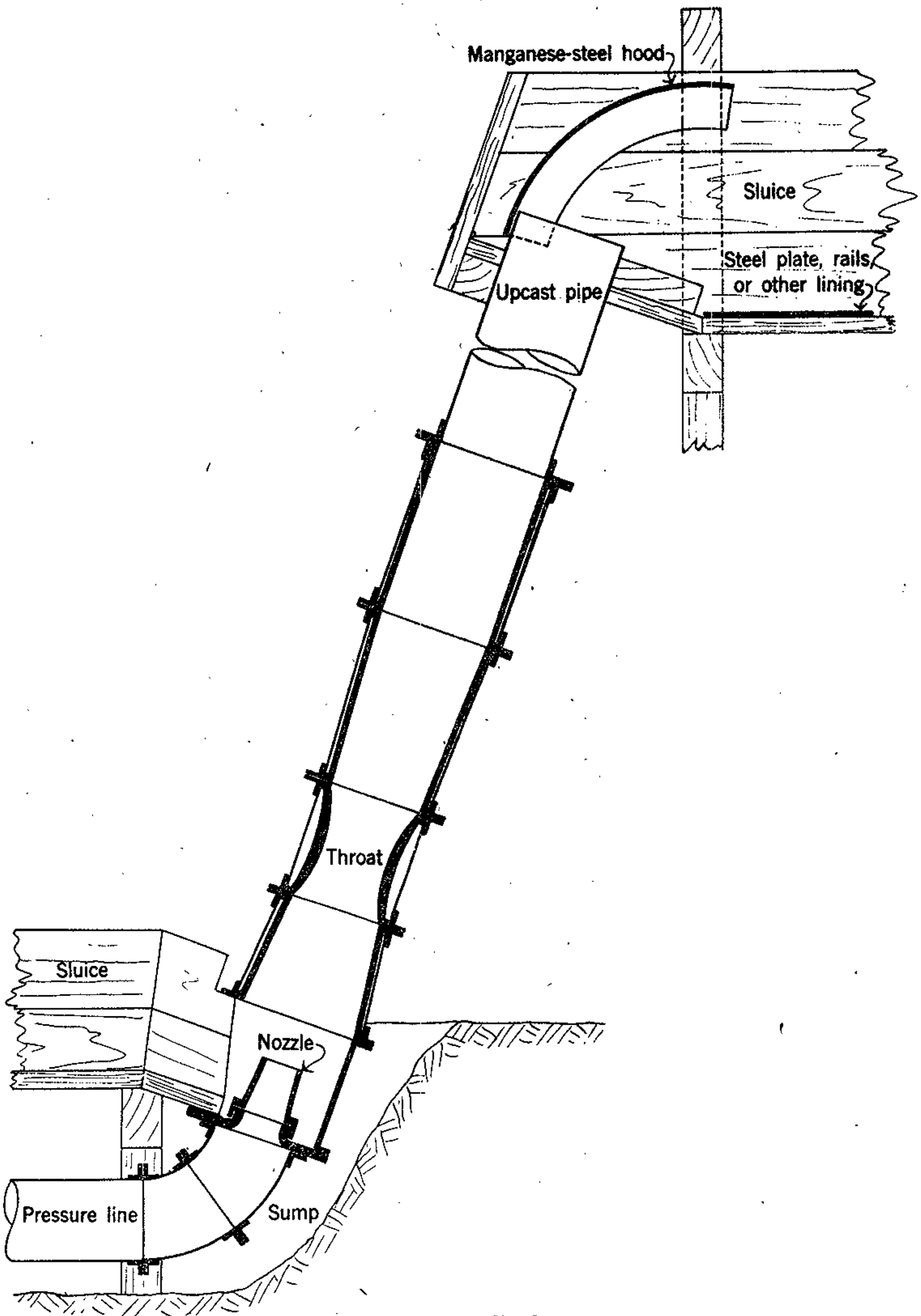

Figure 6.—Hydraulic elevator.

Reprinted from United States Bureau of Mines Information Circular 6787.

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 per cent and not more than 5 per cent of the total weight of water and gravel combined.

"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 mouth of the elevator is placed in a sump excavated in bedrock which should be 10 feet square by 5 feet deep and into this the gravel is washed by the giants. It is better practice to feed the material directly into the intake of the elevator, as this reduces the suction head, which should be kept as low as possible. To prevent the throat of the elevator becoming choked and clogged with large boulders, it is necessary to place a grizzly over the end of the sluice which delivers the material to the intake. The grizzly bars should be spaced with openings at least 1 inch less than the diameter of the throat.

"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.

"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 one placer mine in British Columbia[1]. As far as known they have not been used successfully in placer mining in the western States.

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1 Operations of B. Boe on Cedar Creek, Quesnel District: Ann. Rept. of the Minister of Mines of British Columbia, 1932, p. A-112.

Plate V A. A No. 1 giant at a small operation. This monitor has no deflector, the weight of the discharge barrel is counter-balanced by the jockey-box, at the right, weighted with rocks.

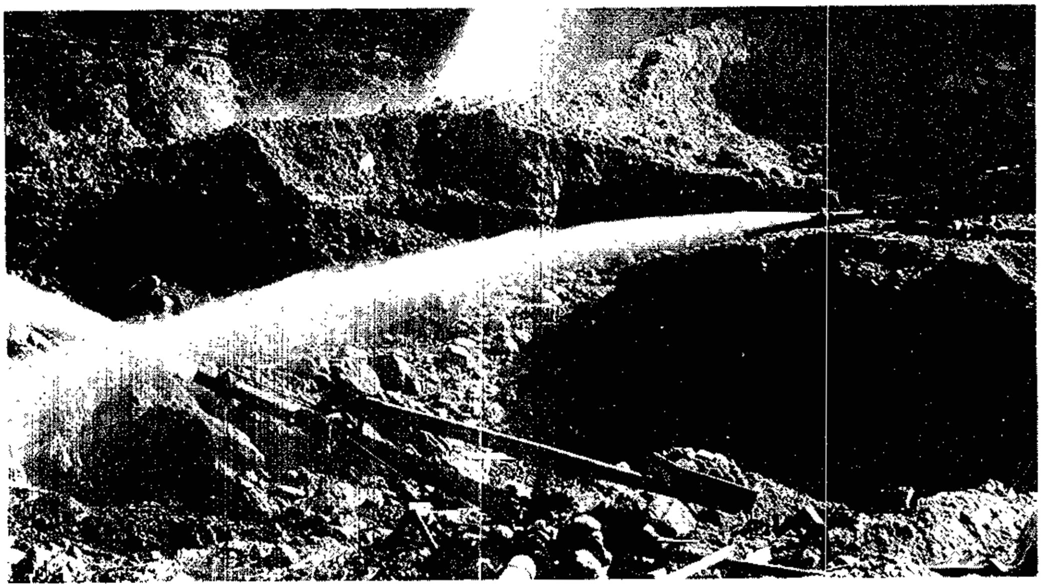

Plate V B. A large, No. 7, giant and a No. 4 machine in the No. 1 pit of Germansen Ventures Ltd. The large machine has a 9-inch nozzle. Notice the man riding the discharge barrel and the heavy load of rock in the jockey-box.

Plate VI A. Two giants at work sweeping gravel in a hydraulic pit.

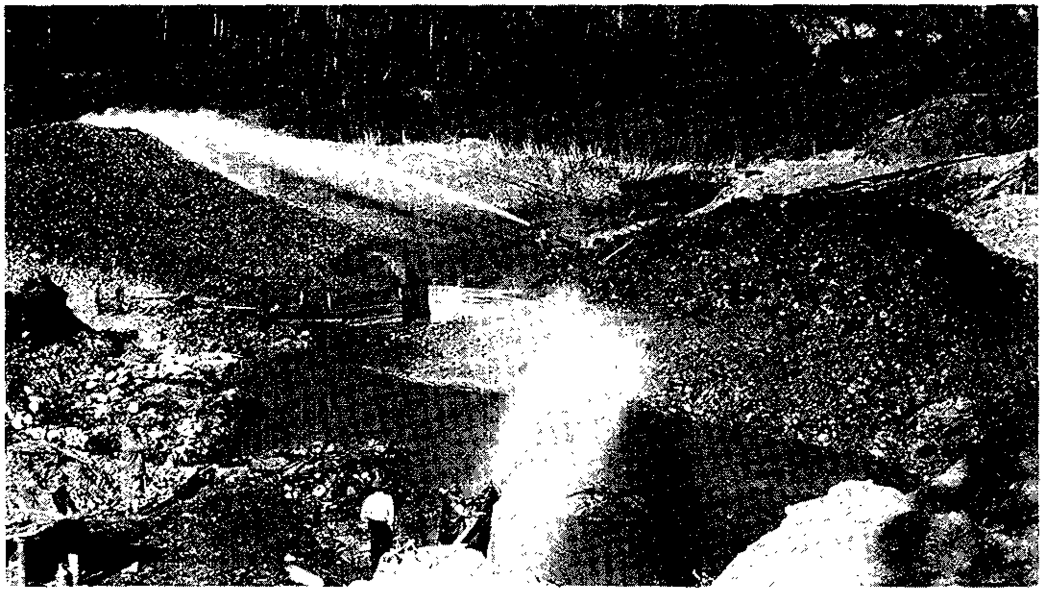

Plate VI B. A hydraulic pit with one giant feeding gravel to a short length of sluice-boxes and a second monitor stacking the coarse gravel at the end of the boxes. Sand and water run off through a sluice-flume to the right.

Hydraulic Mining Practices

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 California mines operating in 1932 ranged from 0.4 to 4.3 cubic yards per miner's inch. In these calculations by-wash water is included.

"Conditions at the mines range from the most difficult to at least average. Wimmler[1] reports a duty of as high as 10 cubic yards per miner's inch at some Alaskan placer mines."

At Germansen Ventures Ltd., Germansen Creek, B. C. an average duty for all water, including that for cleaning bedrock was 40.8 cubic yards of water[2] per yard of gravel sluiced. Whereas at Bullion Mine (using about 90 cubic feet of water per second) the duty ranged from 8.3 to 35 cubic yards of water per yard of gravel. There is a considerable difference between the duty of water used in piping overburden, in piping pay-gravel and in cleaning bedrock, consequently an average value can seldom be used except for anticipating within certain limits the yardage of gravel that will be washed.

Piping

"After a mine is opened up the gravel bank is undercut by the giant allowing the overlying material to cave into the pit (see Plate VIII B). 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

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[1] Wimmler, Norman L., Placer-Mining Methods and Costs in Alaska; Bull. 259, Bureau of Mines, 1927, p. 139.

[2] 1 cubic yard of water equals 202 gallons.

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."

If the giants are worked close to the foot of the banks, care must be exercised that a cave does not occur which might bury both workers and giants.

It is poor practice to work into a bank with a "horse-shoe" cut, for this means the giant becomes surrounded on all sides by high banks which greatly increase the danger to the operations. The best way is to work across the face of the bank maintaining a 'nose' of gravel immediately in front of the giants and working with a side cutting action both left and right of this nose, because by this means the direction of any slides would be parallel to the main face of operation, and not toward the giant.

It will generally be found that a side cutting action will excavate a far larger amount of gravel per cubic foot of water used, than by directing the water directly at the bank.

No general rule can be given for the actual location of the giant. This will depend entirely upon the conditions existing at the mine, and will vary from time to time as the work is carried forward. The actual setting of the giants, however, is a matter of considerable importance and they must be carefully and securely braced to prevent accidents which may occur causing serious trouble and loss of time.

The last length of pipe-line should not be laid so that there is a slight upward trend because there will be an upward lifting acting on the giant which will make it extremely difficult to hold firmly in place. If the last few lengths of pipe-line are laid on some slightly raised part of bedrock, or on timbers, so that at the final and terminal connection the direction of the pressure of the water and of the pipe-line has a slight downward inclination, there will be far fewer chances that the foundation of the giant will move.

It is impossible to construct permanent foundations for a giant as it has to be shifted frequently. The headblock should be a heavy square timber, proportionate to the size of the giant and from 8 to 10 feet long, laid in a trench excavated in bedrock to a depth of 12 inches or more. The giant is bolted to this timber, with the face of the timber and the face of the base of the giant on a line directly at right angles to the thrust of the water and the terminal length of pipe.

Plate VII A. The No. 1 pit at Germansen Ventures Ltd., Germansen Creek, B.C. The pit is about 1500 feet long, 150 feet deep and about 200 feet wide at the bottom. Because of the low bedrock grade, the sluice-boxes are in a bedrock cut about 25 feet deep at the wings of the sluice, in the foreground.

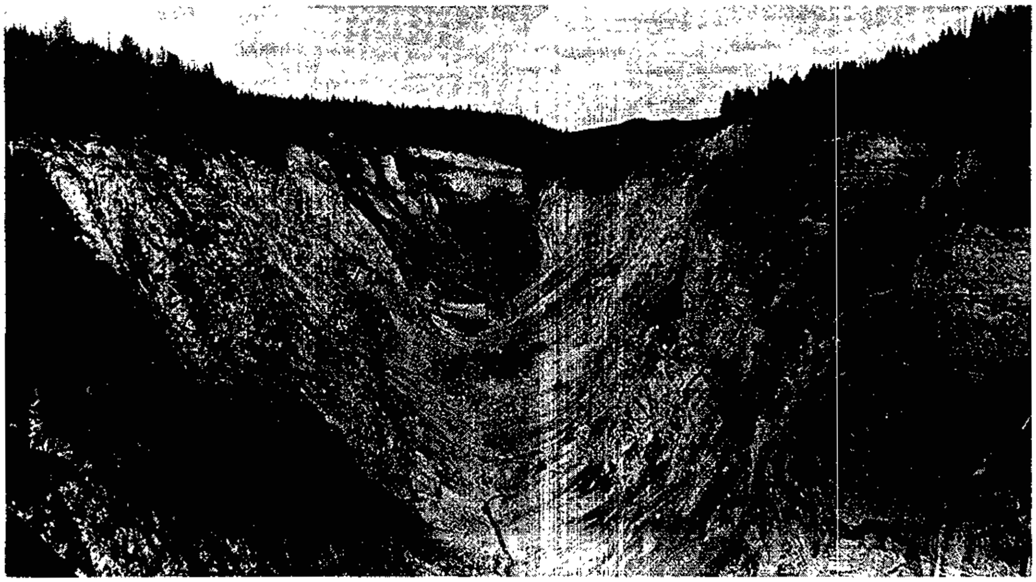

Plate VII B. The Bullion pit, on the south fork of the Quesnel River. This pit is about 3000 feet long, about 400 feet deep, from 150 to 250 feet wide at the bottom, and from 1000 to 1500 feet wide at the top.

Plate VIII A. Boulder-clay showing the irregular assemblage of boulders in a compacted clay matrix.

Plate VIII B. A cave at the head of a hydraulic pit showing the large masses into which the boulder-clay breaks.

In some cases the headblock must be secured to bedrock by cables to eyebolts driven into holes drilled in bedrock to a sufficient depth to hold the timber firmly in place. The whole timber base should then be heavily weighed to eliminate vibration.

"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[1]. Wing dams of timber, logs, or boulders generally are built to guide the water and gravel into the head of the sluice." If possible a ground-sluice grade of 8 to 15 per cent is desirable leading into the sluice-boxes.

"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.

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

"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 top soil, clay and light gravel with the heavier material from near bedrock may permit moving a larger proportion of boulders to the sluice than otherwise.

Handling Boulders

"Where the size and grade of sluices permit, all boulders that can be moved by the giant are run through the boxes. At some of the early-day large producers boulders weighing 3 or 4 tons were successfully put through the sluice.[1]

"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, the boulders may be pulled from the pit by winches or moved by a derrick mounted on a tractor. 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 drag-line. The most common method of handling boulders, however, is by means of a derrick (see Plate XIII B). The boulders that can be rolled by hand are loaded onto a sling or a stone boat

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

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."

In British Columbia where most hydraulic operations have to wash a good deal of boulder-clay the handling of large boulders takes up considerable operating time. It is particularly important in large operations where every hour the water is turned off means a considerably smaller yardage sluiced that the boulders be blasted quickly. For this purpose a compressed air operated jack hammer is essential for drilling short holes (see Plate XIV A).

Dynamite is much more efficient, and consequently rock breaking costs are lower, if it is used in this way rather than for bulldozing. Bulldozing however in the smaller operations is common practice and for this purpose 60 per cent dynamite is found more effective than 40 per cent.

Cleaning Bedrock

"Bedrock usually is cleaned by piping. As much as 2 feet or more 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 picks and shovels, brooms, small stiff brushes and small flat tools made for the purpose from strap iron or wire (see Plate XVI A).

COST OF HYDRAULICKING

The cost of hydraulicking depends on a number of factors of which the most important is the duty of the water. This is controlled by the volume, the head, the character and amount of material being sluiced, the height of the bank and the size and grade of the sluice-boxes. The lowest possible costs at any operation will be reached by attaining the greatest water duty. It is impossible to find directly comparable conditions at any two mines, consequently costs are largely governed by local conditions. For example the operating costs at 28 hydraulic mines in California, sluicing from 3,500 to 719,000 yards per season, ranged from 2.63 to 37.5 cents per cubic yard. These do not represent total costs inasmuch as it was not possible to assess depreciation or amortization charges.

In Alaska the average cost per cubic yard of 13 operations is stated by Purington[1] to be 23.8 cents.

In the Yukon, hydraulic stripping operations of the Yukon Consolidated Gold Corporation[2] are reckoned at 6 cents per yard, sluicing would be more costly because the average water duty would be less.

In British Columbia conditions differ so greatly from those in California particularly with regard to the greater number of boulders to be handled and the ever-present, tough boulder-clay that a comparison of costs cannot be made. However, in the absence of detailed cost figures which can be quoted, it is probably safe to say that in British Columbia, few, if any, operations could profitably work material averaging 8 cents or less per yard and that for most, at least 12 to 15 cents or more per yard would be necessary to sustain a profitable undertaking.

The principal item making up the per unit cost of hydraulicking is labour which in some instances may constitute 75 per cent of the cost. Other items are water, which when large ditch and flume systems are installed may be considerable; explosives, which depend largely on the size and number of boulders encountered and how they are handled; supplies and general office expenses. However, inasmuch as the reduction of costs depends on sluicing the largest possible amount of material, no effort should be spared to get all the water that is economically possible and to use it with the greatest possible efficiency.

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1 Bulletin 263. U. S. Geol. Survey, p. 38.

2 Trans., C.I.M.M. vol. XLII, 1939, p. 540.

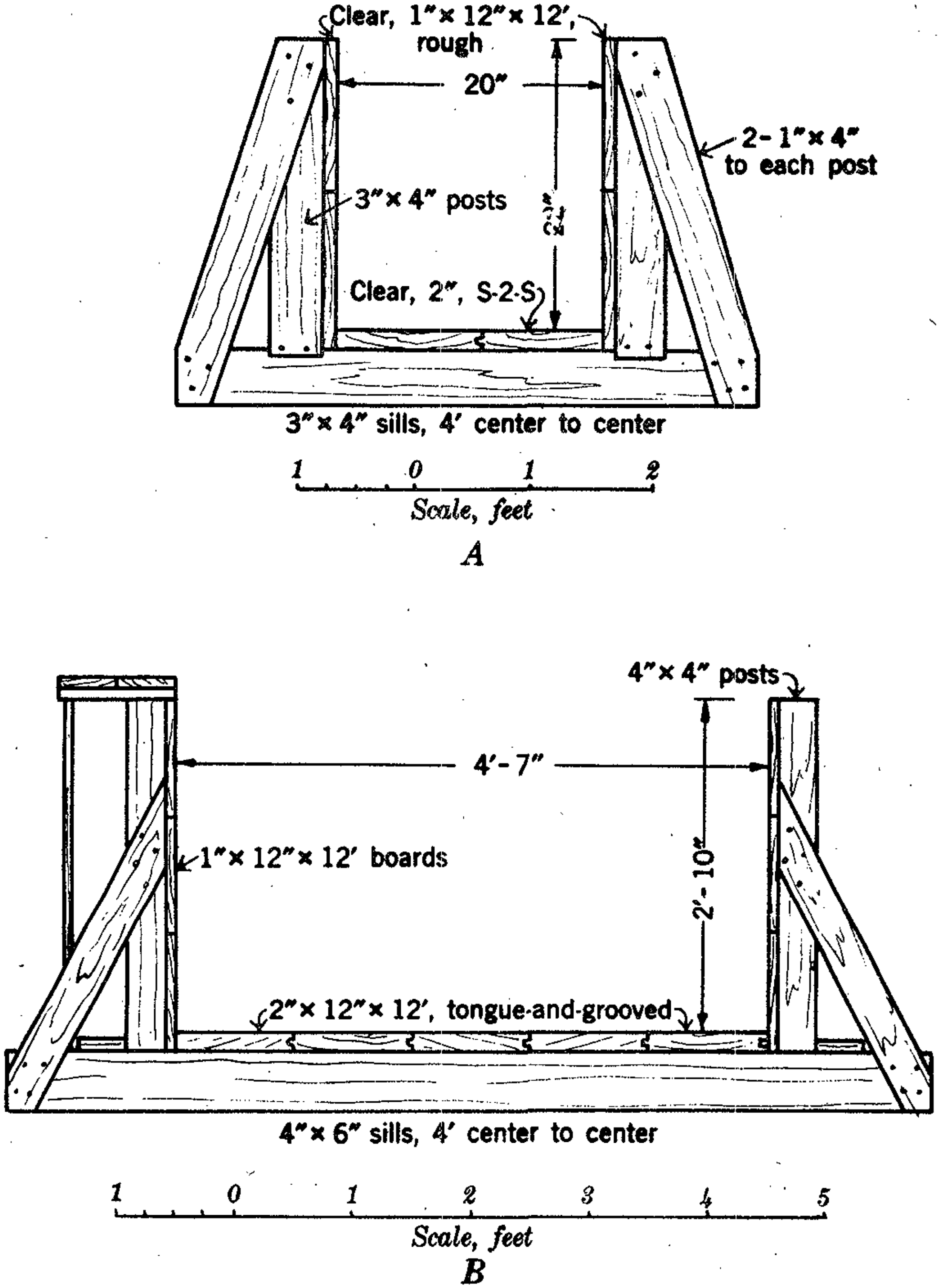

Figure 7.—Sluice-box construction: *A*, Twenty-inch box at Henderson mine, Gold Creek, Mont.; *B*, five-foot sluice box.

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SLUICE BOXES AND RIFFLES

"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.

"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.

Sluice-Boxes

Construction

"Sluice-boxes are rectangular in section and are nearly always built of lumber although steel or iron sluices are occasionally used.

"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 7.

"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, of by caulking with oakum or other material. Bowie[1] 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 customary to use surfaced lumber for boxes, inasmuch as 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 7, 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.

"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

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

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.

Maintenance

"Maintenance work on sluice-boxes consists chiefly in aligning 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 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	Depth inches	Grade %	Miner's inches of water	
			From	To
10-12	6-7	4.16	30	
12-14	10	6.2	66	
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[1]. 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[2].

	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- & 4-inch	5.3
	6- & 8-inch	6.7
	12- & 18-inch	10.0

The following table illustrates the relation of velocity and grade. The figures apply to a sluice 2 feet wide having a flow 1 foot deep. The approximate velocity in feet per second and the quantity in cubic feet per second and miner's inches are given for various grades.

Grade	1%*	2%	3%	4%	5%	6%	8%	9%	10%
Velocity, feet per second	2.7	3.8	4.6	5.3	5.9	6.5	7.6	8.1	8.6
Cu. ft. water per second	5.4	7.6	9.2	10.6	11.8	13.0	15.2	16.2	17.2
Miner's inches	216	304	386	424	472	520	608	648	688

*A grade of 1% is the equivalent of about 1 7/16 inches to a 12 foot box.

"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.

1 Gilbert, G. K., The Transportation of Debris by Running Water: U. S. Geol. Survey, Prof. Paper 86, 1914, p. 216.

2 Van Wagenen, J. F., Manual of Hydraulic Mining: Van Nostrand Co., New York, 1880, p. 88.

"Gold has a better opportunity to settle and be caught in riffles in a wide, shallow stream than in a deeper narrower stream of the same volume; the wider sluice, however, usually must be set on a steeper grade.

"Small or medium-size boxes are approximately 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 125 to 250 cubic feet per second are not unusual when the gate of the reservoir is suddenly opened wide."

A method which may be used to estimate the approximate size of a sluice-flume knowing the volume of water and the grade of the boxes, is to calculate the dimensions, assuming that it is carrying clear water, then add 30 per cent to the width and depth for the dimensions of the water stream when it is loaded with sand and gravel. Naturally several feet of freeboard will be necessary to prevent spillage through blockage of the sluice or when a surge comes through from a "double."

Furthermore the length of the sluice-flume is an important dimension. The length should be sufficient for all gravel and clay to be thoroughly broken up and disintegrated and for the gold to settle and be held by the riffles. In spite of the fact that most of the gold is found in the first half dozen head-end boxes, the sluice in any large operation should be 250 feet long to make an efficient gold saver. Whereas with small sluices (12 to 14 inches wide) and with coarse to medium gold 36 to 72 feet may be all that is necessary.

Grade

"Usually the grade of the sluice depends upon the slope and contour of the bedrock. If the gradient of bedrock, however, is too 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 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 (4.16 per cent) is the best grade for average conditions. Grades as flat as 3 inches in 16 feet can be used but only at great loss of capacity. At one 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[1] 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 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 from 1/8 to 3/8 inch per

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[1] Bowie, A. J., A Practical Treatise on Hydraulic Mining in California. Van Nostrand Co., New York, 3d ed., 1889, p. 220.

foot of sluice width 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 difficult 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 per cent of the gold recovered in the sluice was found at the drops."

At the Bullion Mine, B. C., where rail riffles are used, the three lengths of rail at the upper end are each raised 2 inches above the next down stream in order to increase the grade and give boulders an initial velocity when starting through the sluice.

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-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.

Plate IX A. Wood block riffles in a small sluice-flume. Each pair of blocks is held by a riffle stick which is toe-nailed to the side of the box.

Plate IX B. A large sluice-box paved with wood blocks wedged together by blocks in the interstices. A safety stick is nailed across at the end of each 12-foot box.

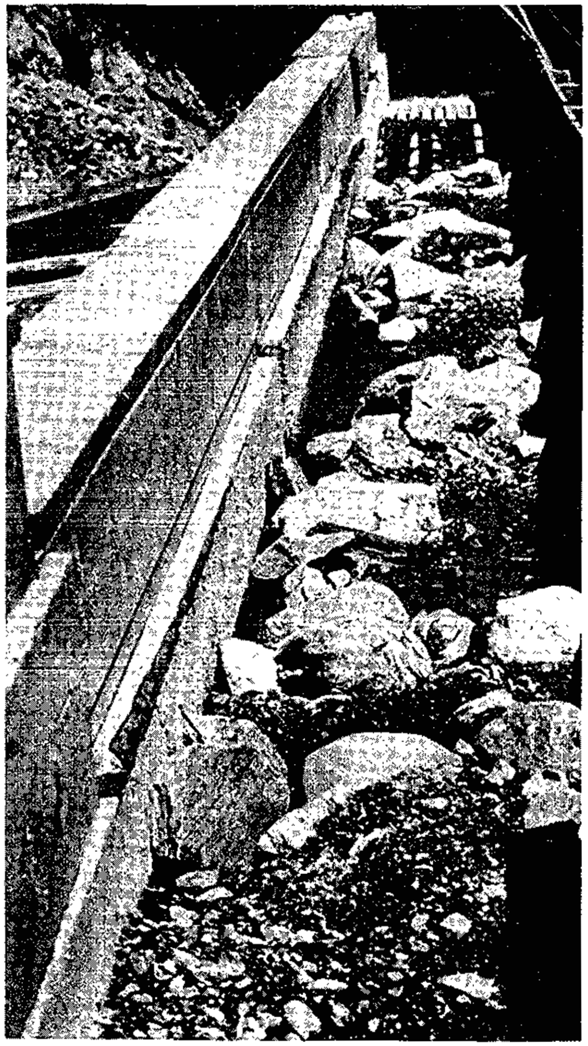

Plate X A. A 3-foot sluice-box showing the load of gravel and boulders dropped when the water was turned off.

Plate X B. A 30-inch sluice-flume showing a central groove worn in the wood block riffles after considerable service.

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 8, A, is used almost exclusively.

"Of about 80 hydraulic, ground-sluice and mechanically worked placer mines, approximately 25 per cent. 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 per cent. used the longitudinal pole type, 15 per cent. wooden blocks, and 15 per cent. 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 per cent. 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."

In British Columbia most medium and large sized hydraulic operations use wood block riffles, and a few (Bullion and Germansen Ventures) use steel rails; At the Lowhee Mine the upper 120 feet of sluice is shod with 4-foot square steel plates separated by traps 3 inches wide and 6 inches deep, the rest of the flume being paved with wood blocks.

"For a small or medium-size sluice (if lumber is costly and a plentiful supply of small timber, such as the lodge-pole pine, is available) peeled pole riffles (Fig. 8 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 one mine 6-inch pole riffles

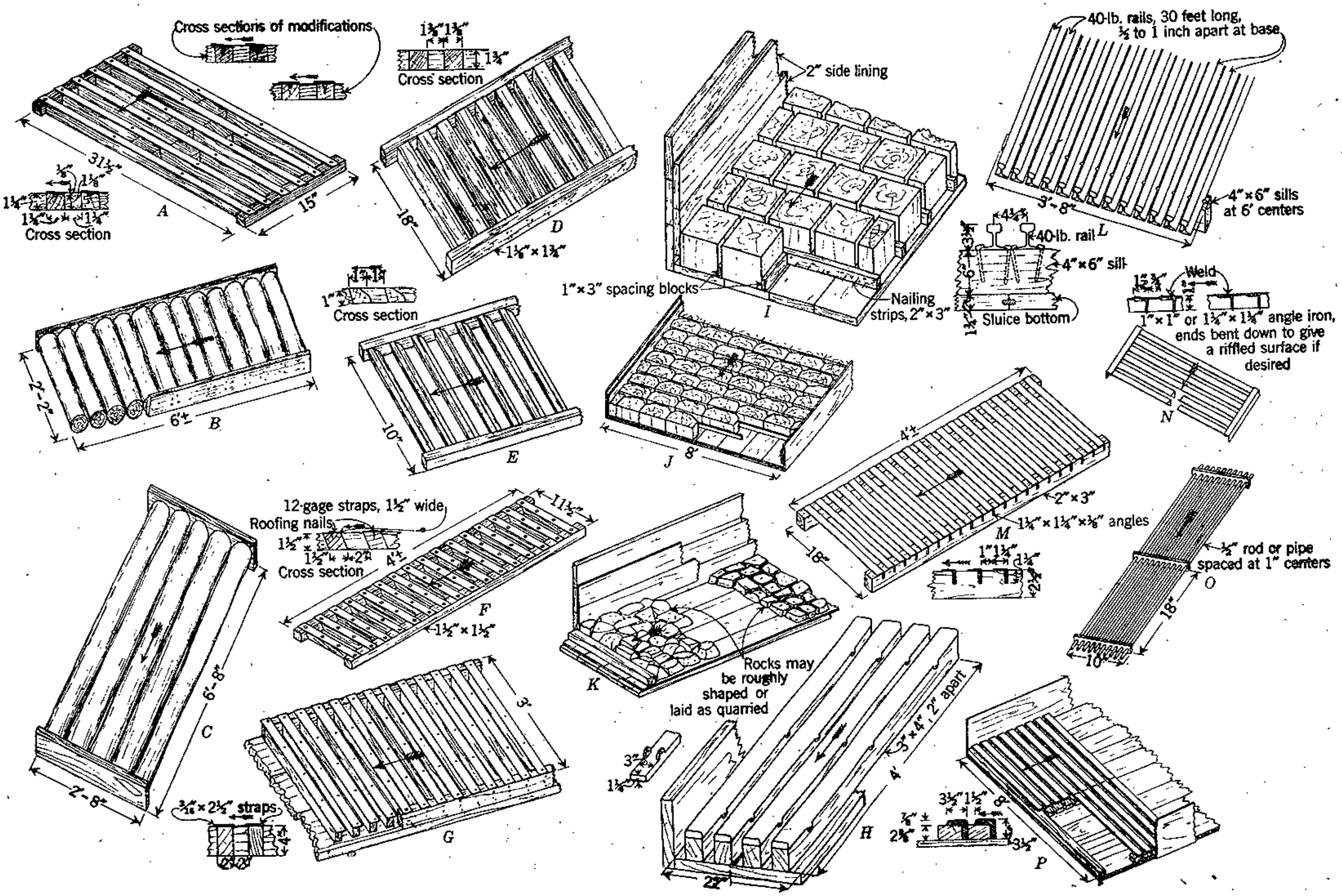

Figure 8.—Types of riffles: A, Transverse wooden, steel-capped riffles used on dredges; B, transverse pole riffles; C, longitudinal pole riffles; D, transverse wooden riffles, square section; E, transverse wooden riffles, bevelled section; F, transverse wooden riffle, steel-capped, inclined section; G, transverse wooden riffles, steel-clad, with overhang; H, longitudinal wooden riffles capped with cast-iron plates; I, wooden-block riffles for large sluices; J, wooden-block riffles for undercurrents; K, stone riffles; L, longitudinal rail riffles on wooden sills; M, transverse angle-iron riffles; N, transverse angle-iron riffles with top tilted upward; O, longitudinal riffles made of iron pipe; P, transverse cast-iron riffles used in undercurrents.

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 8, D. and E. The top surfaces of the riffles may be plated with strap iron (Fig. 8 F. and G.). Transverse riffles of this type may be slanted downstream, as shown in Figure 8, 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 8, H.

"Sluices in the Rock Creek sapphire mines were 12 inches wide and set on a grade not to exceed half an 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 (see Plate IX and Fig. 8, I and J) are held by Bowie[1] to be unexcelled in regions where the material is available cheap." In British Columbia wood blocks cost between 10 and 15 cents each delivered to the sluice-box. "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,

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1 Bowie, A. J., A Practical Treatise on Hydraulic Mining in California: Van Nostrand Co., New York, 3d ed., 1889, p. 225.

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." At the Lowhee Mine, B. C., the wood blocks must be replaced each season; these last for 200,000 to 250,000 yards of gravel moved. "The grade of the sluice apparently has much to do with the life of block riffles. At a 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.

"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 (see Plate XI). Old rails or angle iron can often be obtained cheaply in mining districts or near railroads. Various 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." Eight or 10-foot lengths are usually quite long enough. "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." Another method is to spike the rails with track spikes to 4 by 4 inch timbers set crosswise in the sluice. "At the La Grange mine in Trinity County, Calif., 40-pound rails costing about $125 per ton proved more satisfactory than wood riffles[1]. When 16- by

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

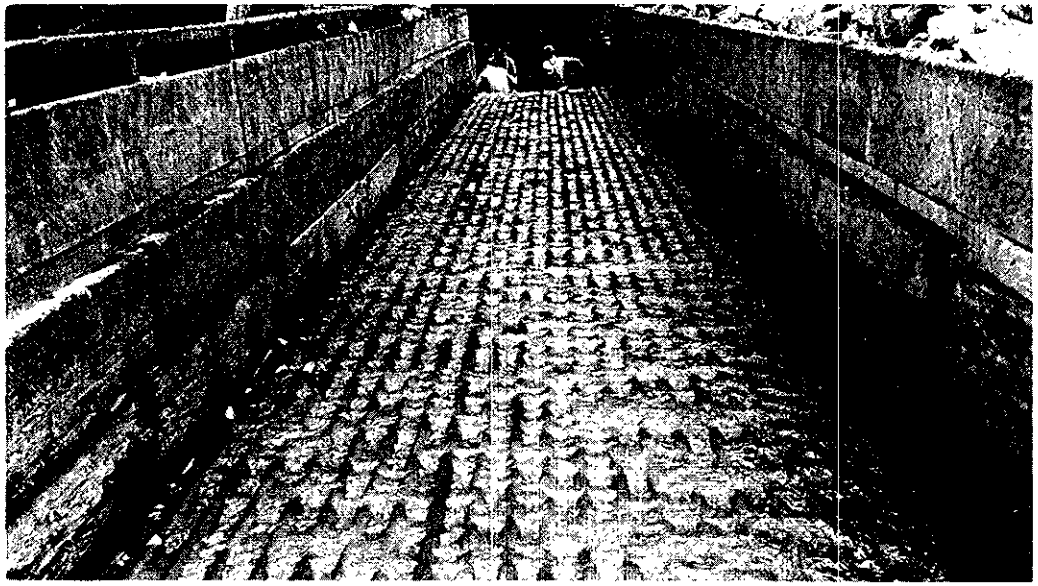

Plate XI A. The 6-foot sluice-flume at the Bullion Mine paved with rail riffles. The rails are spiked to 6 by 6 inch ties laid across the box at 3-foot intervals.

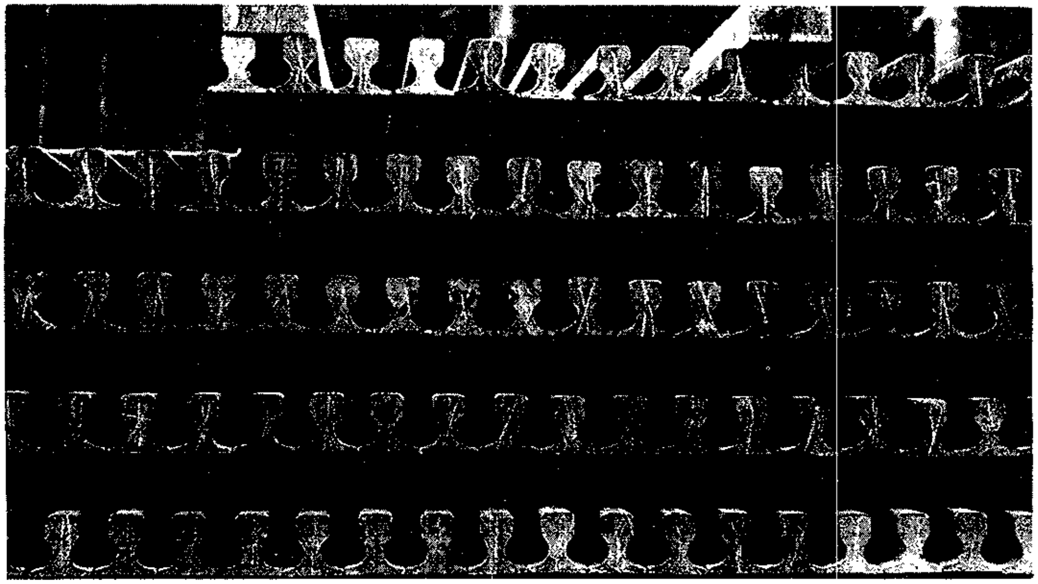

Plate XI B. Cross-section of the Bullion type rails. These weigh 37 pounds to the yard, are made of special alloy steel and of this special cross-section. Notice how thick the web is as compared to standard railway steel.

Plate XII A. Loosening up the sand and gravel packed between block riffles.

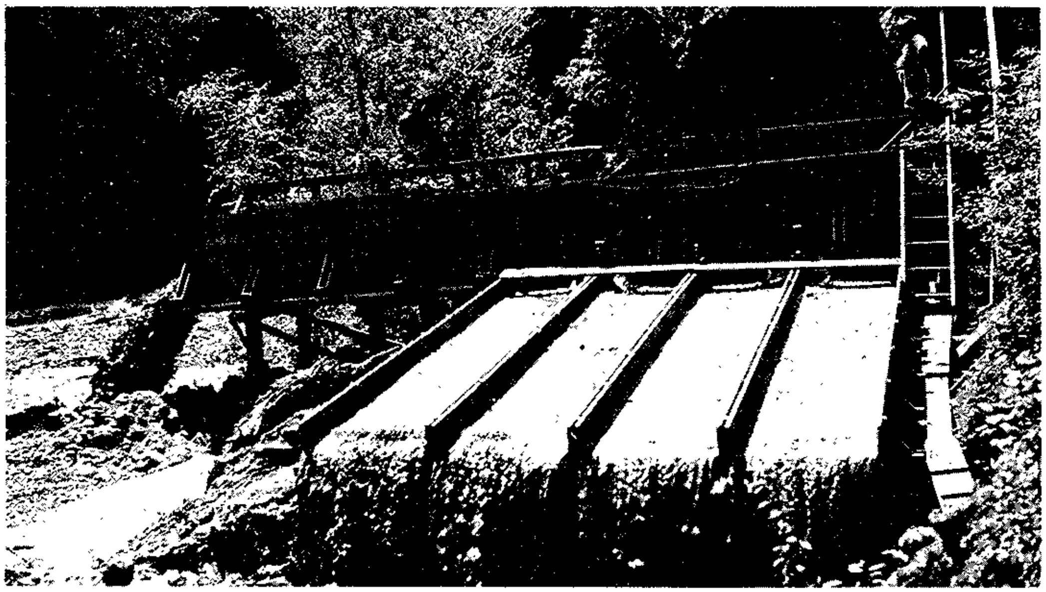

Plate XII B. Undercurrent tables at the end of the sluice at Cariboo Cottonwood Placers Ltd. Each table is 5 feet wide, 20 feet long and on a grade of 1 1/2 inches to the foot. The riffles are expanded metal lath over corduroy.

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."

One of the important advantages claimed for rail riffles as compared with wood block riffles in the same sluice (on the same grade) is that the rail riffles allow about 20 per cent more material being sluiced with the same amount of water. Conversely the same amount of gravel can be washed through a sluice shod with rail riffles on a flatter grade than the same size sluice paved with wood blocks. Rail riffles should not be laid in a sluice with a steeper grade than 5 per cent because the water velocity results in excessive wear on the rails, and a scouring action loses a greater amount of fine gold. A grade of 4.5 per cent gives good results although flatter grades as low as 3.0 per cent may be used where sufficient water is available.

Two mines in British Columbia, the Bullion and Germansen Ventures, handle large yardages each year, 1,000,000 cubic yards or more, and both have sluices lined with 37 pound manganese steel rails of a special cross-section (see Plate XI B). It is believed that the life of the rails will be from 5 to 6,000,000 cubic yards on the scale of operation, although with a smaller sluice and handling less yardage per season with somewhat slower water velocity and handling smaller boulders the life might be almost doubled. Ordinary steel railroad rails, however, will have a shorter life than the above illustration which is for a special alloy steel of special cross-section.

"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 8, 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 8, 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 8, 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, California, 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. 8, P).

"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 especially 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.

"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."

For catching fine gold, such as occurs in benches along parts of the Fraser River, B. C., and in installations where material coarser than 3/8 to 1/2 inch is screened out, tufted, short-piled carpet, not protected by expanded metal lathing, appears to give most satisfactory results when on a grade of 1 1/4 to 1 1/2 inches to the foot. This type is used on the washing plant of North American Goldfields Limited at Alexandria Ferry as well as by many of the "snipers" along the Fraser River.

"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 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 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 (see Plate XII B). 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."

Two important physical factors may however prevent an undercurrent being installed. First, it requires several feet or more of space below the sluice-box grade and often where dumping head is restricted this headroom is not available. Secondly, a certain amount of water is drawn off from the main sluice and in instances where the dump needs to be kept clear by hand it may not be profitable to divert the water through the undercurrent when it could be used more effectively in assisting to keep the dump clear of accumulated gravel and boulders.

"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 or reduces the amount so that difficulty is encountered in keeping the dump clear. 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

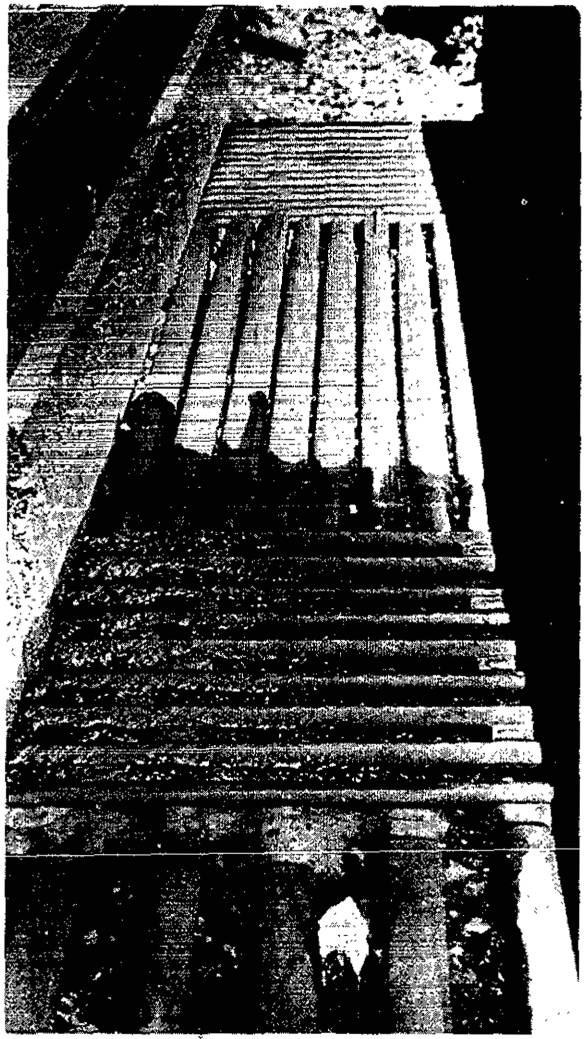

Plate XIII A. The end of a sluice-flume showing longitudinal and transverse rail riffles followed by an under-current grizzly made from 3 inch shafting, then a short section of transverse rail riffles at the end of the box.

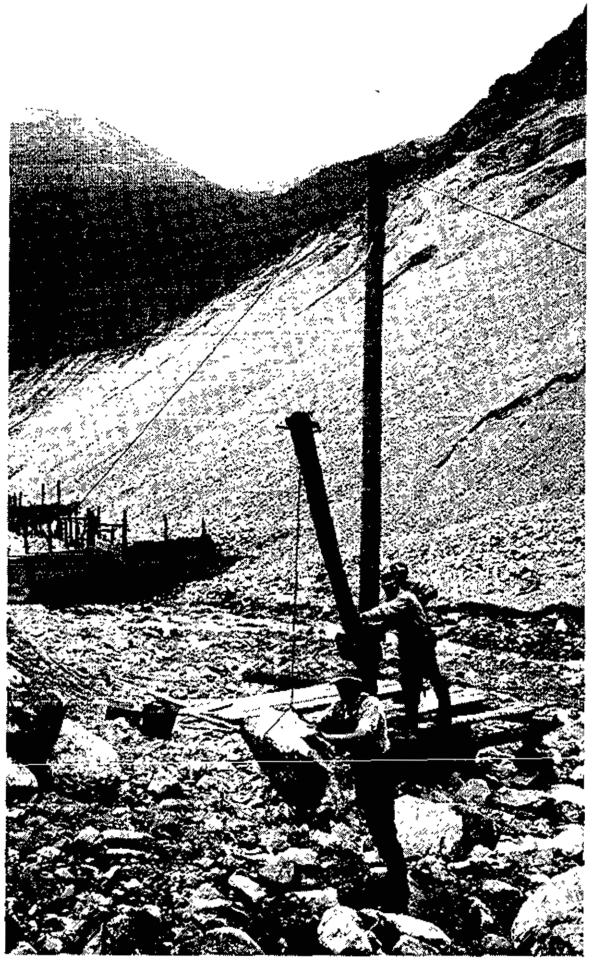

Plate XIII B. Handling boulders in a hydraulic pit with a gin pole and a hand winch at Harvey Creek Mines Ltd., on Nigger (Pine) Creek.

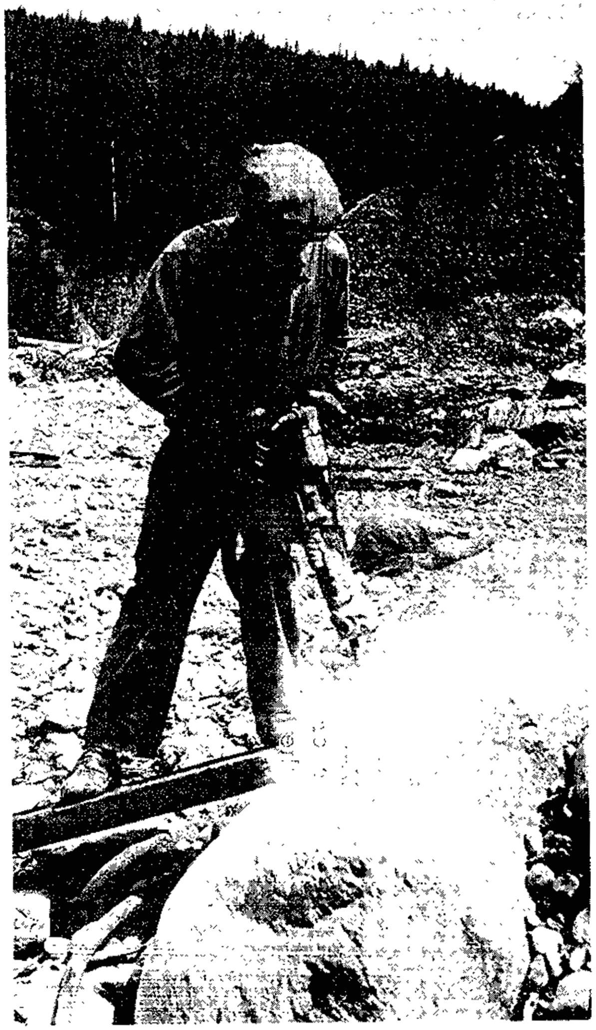

Plate XIV A. Drilling a large boulder with a jack-hammer in the hydraulic pit at Spanish Creek, B. C.

Plate XIV B. Block riffles at the head of the sluice-box lifted preparatory to making a clean-up.

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 per cent of the total clean-up but more often was less than 5 per cent. 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[1] 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 if many were justifying their installation.

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.

"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 rock hooks, 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

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1 Bowie, A. J. A Practical Treatise on Hydraulic Mining in California; Van Nostrand Co., New York, 3d ed. 1889, pp. 252-262.

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 over-size 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, any packed clay broken up and then 'streamed down' with a light flow of water (see Plate XV A). 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 (see Plate XV B), 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 one 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 another the concentrate from the lower section of the sluice was treated in a quartz mill.

Distribution of Gold in the Sluice

The distribution of the gold in a sluice is controlled by its coarseness, the length of the sluice, and by its grade, the amount of water and probably to some extent by the type of riffle used. The sluices used in hydraulic mining are as a rule much longer than those common in placer mining. This greater length is not necessary for saving the gold but rather for carrying the gravel to the dump which may be a considerable distance from the pit. In the following table the results of cleaning up separate boxes along the sluice line at the La Grange mine[1] are given. The greater part of the gold was caught in the first 250 feet of the sluice. The boxes are 12 feet long, the grade 5.5 per cent, width of sluice 6 feet.

Box No.	Mesh size and weight of gold						Total, oz.
	+ 10 oz.	-10 +50 oz.	-50 +100 oz.	-100 +150 oz.	-150 +200 oz.	-200 oz.	
5	45.8	50.7	1.38	0.36	0.31	1.45	100.0
6 - 16 incl.	18.0	83.30	2.33	1.00	0.31	0.83	105.77
22	1.73	20.22	3.08	0.70	0.25	0.62	26.60
48	0.18	2.18	1.06	0.12	0.05	0.16	3.75
88	0.018	0.12	0.47	0.008	0.026	0.005	0.647
136	None	0.053	0.027	0.043	0.011	0.01	0.144

Use of Quicksilver in Sluicing

"Quicksilver is used at nearly all placer mines in California. 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 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

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1 Bouery, P., Eng. and Min. Journ., vol. 95, No. 21, 1913, p. 1059.

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 degrees 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 riffles 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 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 per cent 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 per cent.

"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.

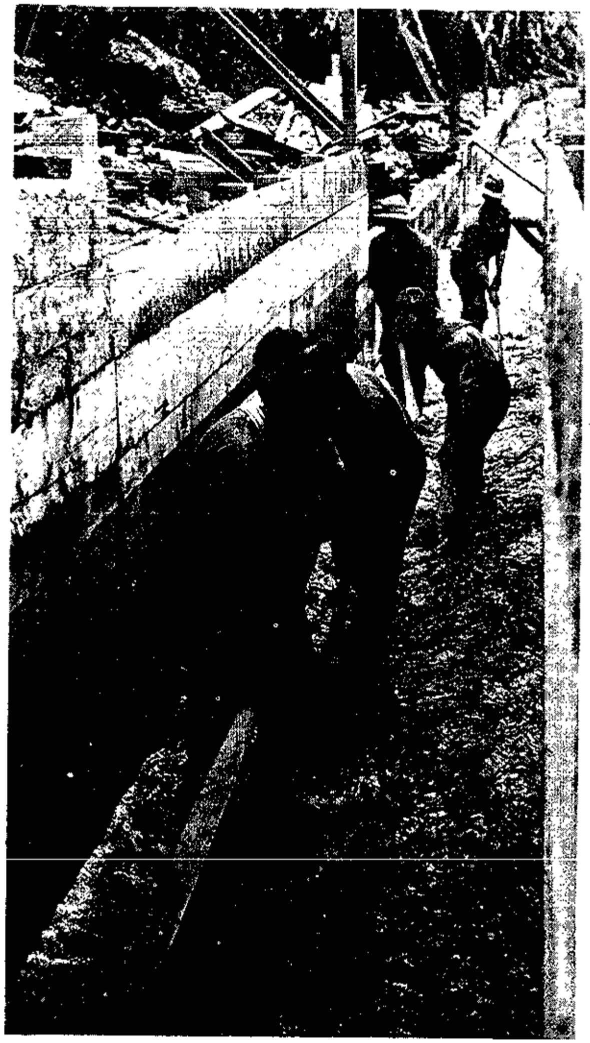

Plate XV A. Clean-up at Germansen Ventures Ltd. Streaming-down and working over the concentrates using wooden paddles.

Plate XV B. Cleaning up the gold in the bottom of a sluice-box to remove as much black sand as possible before scooping up the gold into a pan.

Plate XVI A. Cleaning bedrock by hand. At Germansen Ventures the bedrock is hard and irregular so that it is impossible to clean it adequately with the giant. One man is continuously employed cleaning out all the cracks by hand.

Plate XVI B. Retorting amalgam from the clean-up barrel over a blacksmith's forge. The mercury is condensed by the sacking immersed in the water bucket at the right.

"Wilson[1] 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.

"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.

"The quantity of quicksilver used differs according to conditions and custom. According to Bowie[2], 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[3], 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."

In British Columbia the universal practice is not to use mercury in the sluice-boxes, but only for amalgamating the concentrates from the clean-ups. The probable reason for this is that with an efficient sluice-box the gold is coarse enough to be saved readily and that the mercury lost costs more than the additional amount of fine gold that it might recover. This is particularly true at present with the considerably increased cost of mercury.

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1 Wilson, E. B., Hydraulic and Placer Mining: John Wiley & Sons, New York, 3d ed., 1918, p. 240.

2 Bowie, A. J., work cited, p. 244.

3 Janin, Charles, Gold Dredging, in the United States; Bull. 127, Bureau of Mines, 1918, p 143.

"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 a quarter of an 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.

"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 bed-rock 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 characteristics of the gold. The general methods of cleaning concentrate with pans, rockers, or small sluices are the same as those in small-scale operations 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 rather than assayed by the usual fire method.

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, offers an ideal surface for the purpose.

"The concentrates are placed at the upper end, and a small stream of water is poured over the sand while the rocker is swayed gently back and forth. The lighter material is washed down to the riffle at the lower end, and the coarser particles of gold are left behind. These are picked up with a scraper, and the operation is repeated, a portion of the concentrates presently being discarded with each washing until at length all gold of appreciable value has been recovered. This method is satisfactory with ordinary concentrates, but if the gold is very fine, flaky, 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 half an 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. The material should 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 per cent 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 far 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 large-scale operations where most of the gold is amalgamated in the sluice-boxes or on the riffle-tables, 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 per cent 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 amalgam barrel, generally is used at large stamp mills and is employed in placer operations, particularly in dredging and large hydraulic operations 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 or rocks and the barrel is turned slowly for an hour or several hours."

If, however, the gold is tarnished and does not amalgamate readily the barrel should be charged without the mercury and rotated for an hour or two to scour the gold. Then the mercury should be added and the rotation continued for a further 1 to 2 hours. In this way excessive "flouring" of the mercury can be prevented.

"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, or better still by running through a very small, riffled sluice containing a mercury trap. Potassium cyanide sometimes is added to brighten the gold, using only enough to make a very weak solution, also a small amount of lye may be added if the concentrates are greasy.

To make up a cyanide solution dissolve 1 ounce of 98 per cent potassium cyanide in half a gallon of water, then use 4 ounces (or about half a teacup) of this solution to 10 gallons of water.

"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 degrees 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 mine, particularly if part of the gold is rusty. 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 per cent 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 per cent 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 degrees F., a temperature about half-way 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 red hot and then 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-quarters of an 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. 9, 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. 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 degrees; 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 (see Plate XVI B), 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.

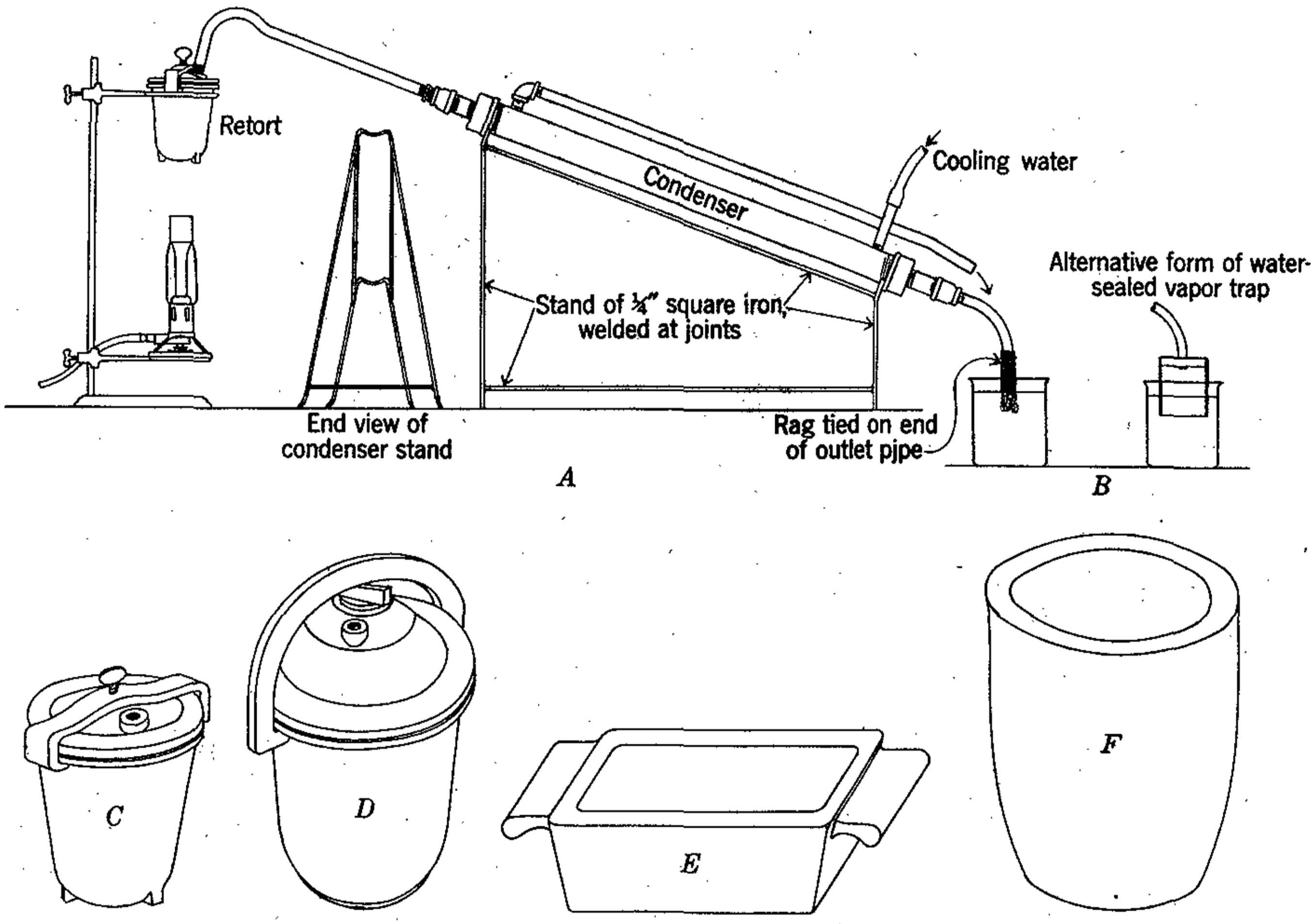

Figure 9.—Apparatus for retorting amalgam and quicksilver: *A*, Amalgam retort; *B*, Nevada-type retort; *C*, set-up of small retort; *D*, water-sealed vapor trap; *E*, graphite crucible; *F*, bullion mould.

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"A rigid, strong stand for the retort and condenser (Fig. 9, 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.

"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 9, B.

"The simplest method 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 9, 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 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:[1]

"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 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 per cent gold and silver, is retorted in a standard make of gasoline-fire 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

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1 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.

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[1] states that platinum-group metals can be recovered on zinc-amalgam plates by using a solution of 0.05 per cent copper sulphate and 0.05 per cent sulphuric acid or by agitating with zinc amalgam in such a solution. At the Onverwacht mine in South Africa a process[2] 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.

"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

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1 Zachert, V. J., Process for Recovering Platinum: Min. and Sci. Press, vol. 117, Oct. 12, 1918, pp. 489-490.

2 Wagner, P. A., Platinum Deposits and Mines of South Africa: London, 1929, p. 274.

a product assaying about 70 per cent of platinum-group metals, which is shipped.

"The recovery by amalgamation is about 98 per cent and the all-over recovery of the plant ranges from 82 to 85-56 per cent."

An alternative method of treating sluice-box concentrates containing both gold and platinum would be to extract all the recoverable gold by treating the concentrates in an amalgam barrel. The rejects from the barrel would contain platinum and the heavy sands. This concentrate could in turn be treated on a Wilfley table to concentrate the platinum further and produce a high grade concentrate which could be shipped and sold to a platinum buyer.

Canadian producers of platinum can obtain information regarding prices and acceptable grade of concentrates from Johnson, Matthey & Co. (Can.) Ltd., 198 Clinton St., Toronto.

BIBLIOGRAPHY

Bouery, P. — Study of Riffles for Hydraulicking, Eng. and Min. Journ., Vol. 95, No. 21, 1913, p. 1055.

Bowie, A. J. — Practical Treatise on Hydraulic Mining - 8th Edition, Van Nostrand, New York, 1898.

Gardner, E. D. and Johnson, C. H. — Placer Mining in the Western United States (Part I). United States Bureau of Mines, Inf. Circular No. 6786, 1934.
Placer Mining in the Western United States (Part II). United States Bureau of Mines, Inf. Circular No. 6787, 1934.

Griffith, S. V. — Alluvial Prospecting and Mining, Mining Publications Ltd., London, 1938.

Haley, C. S. — Gold Placers of California. Bull. No. 92, California State Mining Bureau, 1923.

Peele, Robert — Mining Engineers Handbook. 3rd Edition, John Wiley and Sons, New York, 1941.

Purington, C. W. — Gravel and Placer Mining in Alaska, U.S.G.S. Bull. No. 263, 1905.

Sharpe R. F. — The Bullion Hydraulic Mine, Trans. C.I.M.M. vol XLII, 1939, p. 593-598.

Thorne W. E. and Hooke, A. W. — Mining of Alluvial Deposits Mining Publications Ltd. London, 1929.

Wilson, E. B. — Hydraulic and Placer Mining, John Wiley & Sons, New York, 1918.

Wimmler, N. L. — Placer-mining Methods and Costs in Alaska: Bull. No. 259, U. S. Bureau of Mines, 1927.

Young, George J. — Elements of Mining, McGraw-Hill, New York, 1932.