Stoping Methods at the Nevada Wonder Mine

By THOMAS M. SMITHER

The Nevada Wonder vein is of the fissure type, inclined at an angle of 75°; the foot-wall is rhyolite; and the hanging wall rhyolite, or occasionally dacite. The orebody itself, in most cases, is much fractured and crushed; the gangue consists principally of quartz, feldspar, and kaolin. The average width is five or six feet. No water is encountered. The walls in the lower levels are insecure and cave away in large masses, thus rendering useless both the stull system of support and the shrinkage system of stoping; although the shrinkage system was used to great advantage from the 400-ft. level to the surface, where the walls were fairly firm and stood well.

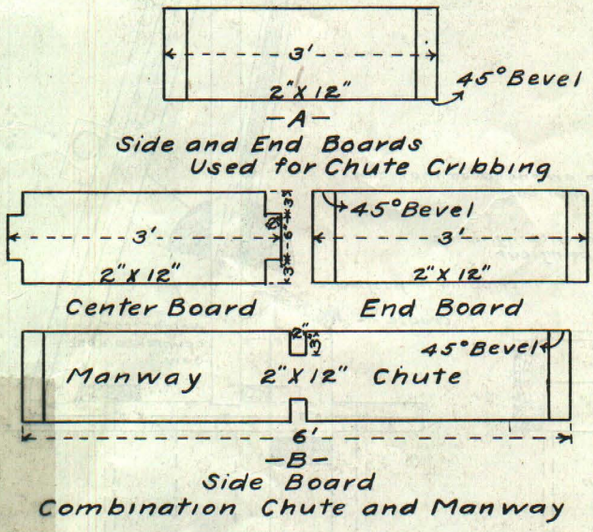
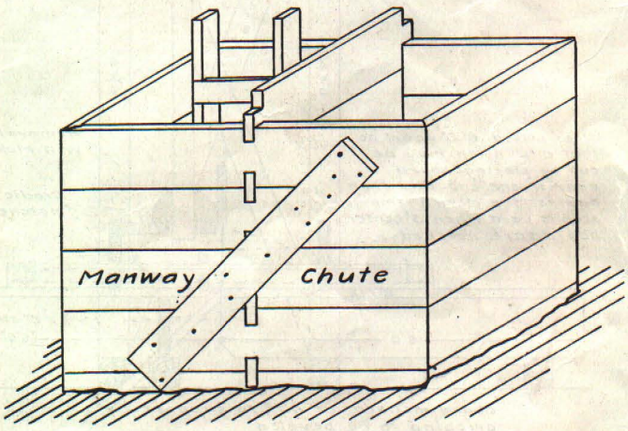


Fig. 1.



Chute and Manway in Position

Fig. 2.

The method at present employed is a waste-filling system. The salient features are:

1. The mining of waste material for filling, from the walls of the stope, by inclined rises.
2. The cribbing system of chutes and manways, using a minimum amount of timber.

As the waste broken during development cannot always be economically transported to the stopes requiring filling, the method of breaking waste by means of inclined rises was adopted. The small amount of timber required by this method is an important consideration, as all supplies are hauled from Fallon, Nevada, the nearest railroad point, which is 60 miles distant, thus making the cost of timber somewhat excessive.

The cribbing used for ore passages are cut from 2 by 12-inch Oregon pine lagging, to the dimensions shown in A, Fig. 1, and are built up by placing upon the 2-in. edge (Fig. 2). When both a chute and manway are desired, the cribbing is cut as in B, Fig. 1. A 24-in. swinging circular saw is used to cut the cribbing; it is of the American, Style D, type, and is self-contained and electrically driven. This saw has been arranged for the purpose at the mine, and is also used to cut stulls and other timber. The saw and the details of the arrangement for cutting the cribbing are shown in Fig. 3 and Fig. 4.

The first thing done is to start the stope by making one cut above the back of the drift. A row of stulls and posts is then placed in position. The stulls are covered with a layer of 3 by 12-in. planking, and chutes and foundations for cribbing are built in (Fig. 5). The first few sets of cribbing are constructed of 3 by 12-in. lagging instead of the usual 2 by 12, to secure a firm foundation. A rise for waste is next started and sufficient waste is broken to cover the flooring to a depth of 2 or 3 ft., for protection of the timbers. The waste is then leveled off and covered with a plat of 2 by 12-in. plank to receive the broken ore. The waste-rises are started as near the back of the stope as possible, by this means securing a greater length of service.

The stoping now proceeds in a series of steps. The ore in one end is drilled and shot down; sufficient is then drawn out to leave just enough head-room for a miner and a Waugh drill to operate efficiently. The back is next drilled with 5-ft. holes, and the holes left unloaded until required. This drilling upon the broken ore gives the miner a safe position close to the back and at the same time prevents falling masses of ore from mixing with the filling. As soon as the drilling is completed, as much ore as possible is drawn out through the chutes; the remainder is shoveled into them and where necessary wheel-barrows are used. The plat is next taken up and waste-filling begun.

The chutes and manways are first built up several sets and temporarily held in position by a brace of

plank, spiked across them, as shown in Fig. 2. The edges of the cribbing are toe-nailed together. The inclined rises are next driven upward until enough waste is broken down. This waste runs into the stope by gravity and is leveled off by the shovelers, and the plat of 2 by 12-in. lagging laid upon it. The filling is carried to within three or four feet of the back, and is level with the topmost edge of the chute-cribbing.

While the waste-filling is in progress, the miners drilling the ore are continually advancing toward the opposite end. A uniform procedure is maintained, one end being filled with waste, while ore is broken at the other; upon arriving at the far end the miners return to the original starting point and proceed as before. A 5-ft. strip, the full length of the stope, is removed each time, 40% 1⅛-in. E.L.F. Hercules gelatin being used for blasting. When speed is necessary more miners are used and stoping proceeds from both ends to the middle.

The waste-rises are driven alternately into the foot and hanging walls, by Waugh machines, at an angle of from 40 to 50°. The rises are placed so that one is between every two chutes, and they are used for two or three fillings. The heights of the rises vary, depending upon the amount of waste required, but they seldom exceed 30 or 40 feet.

A combination chute and manway is carried up at each end of the stope, with as many as are necessary at intermediate distances. One manway is always open, and more when possible. When ore is to be shot down above a manway, the opening is closed with a cover of 2 by 12-in. plank, nailed together. This cover is removed as soon as possible after blasting, for ventilation and entrance purposes.

When about to break through to the level above, the system of timbering shown in Fig. 6 is used to support the filled stope overhead. By this means all of the ore is removed and the ground solidly filled.

The advantages of this method, where weak and swelling walls are present, are:

1. The secure and safe support of the stopes.

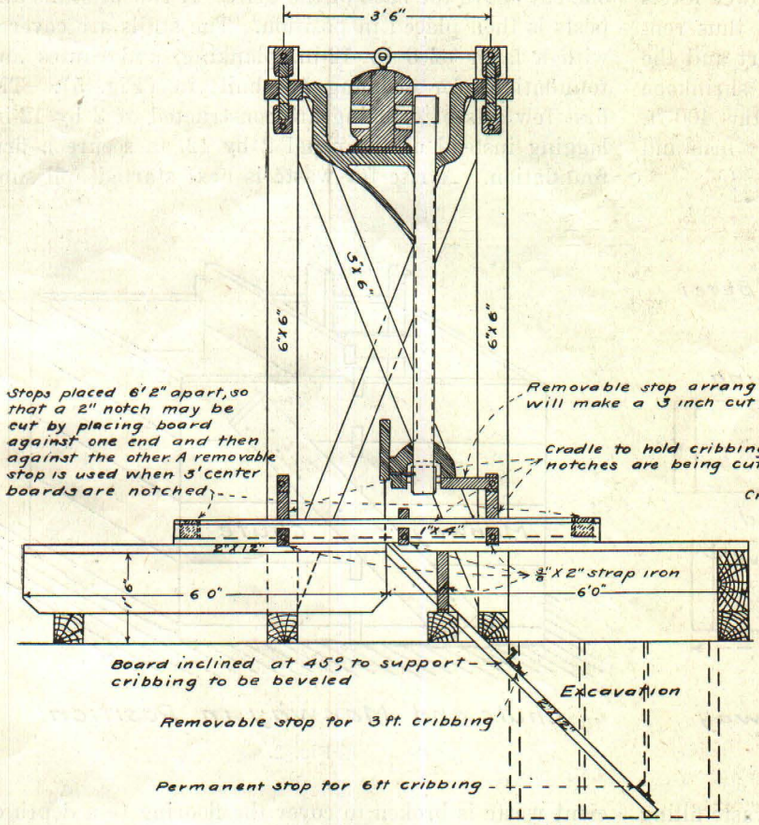


FIG. 3.

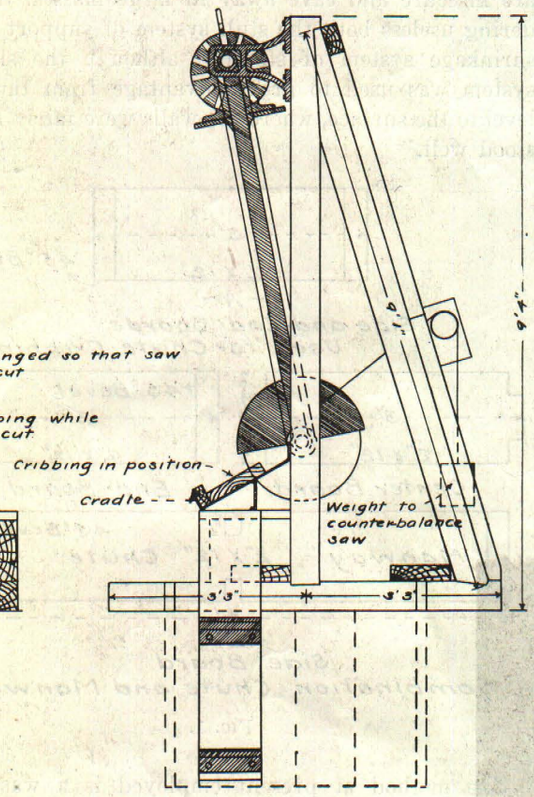


FIG. 4.

2. The minimum use of timber, where timber is an expensive item.

3. The waste-rises serve as cross-cuts, for prospecting.

4. The elasticity of the method. The required tonnage can be broken down at any time, since the back is kept drilled ahead of requirements.

5. The removal of the ore, as no pillars are left.

The disadvantages are:

1. Low-grade material, which might be worked economically at a future date, must be left. At the Nevada Wonder mine, however, little is lost in this manner.

2. The shoveling required.

These disadvantages are more or less present in all waste-filling systems and under the conditions described are of minor importance.

In conclusion, I wish to state my appreciation for information given by Mr. J. A. Burgess, superintendent, and Mr. W. Frazee, foreman.

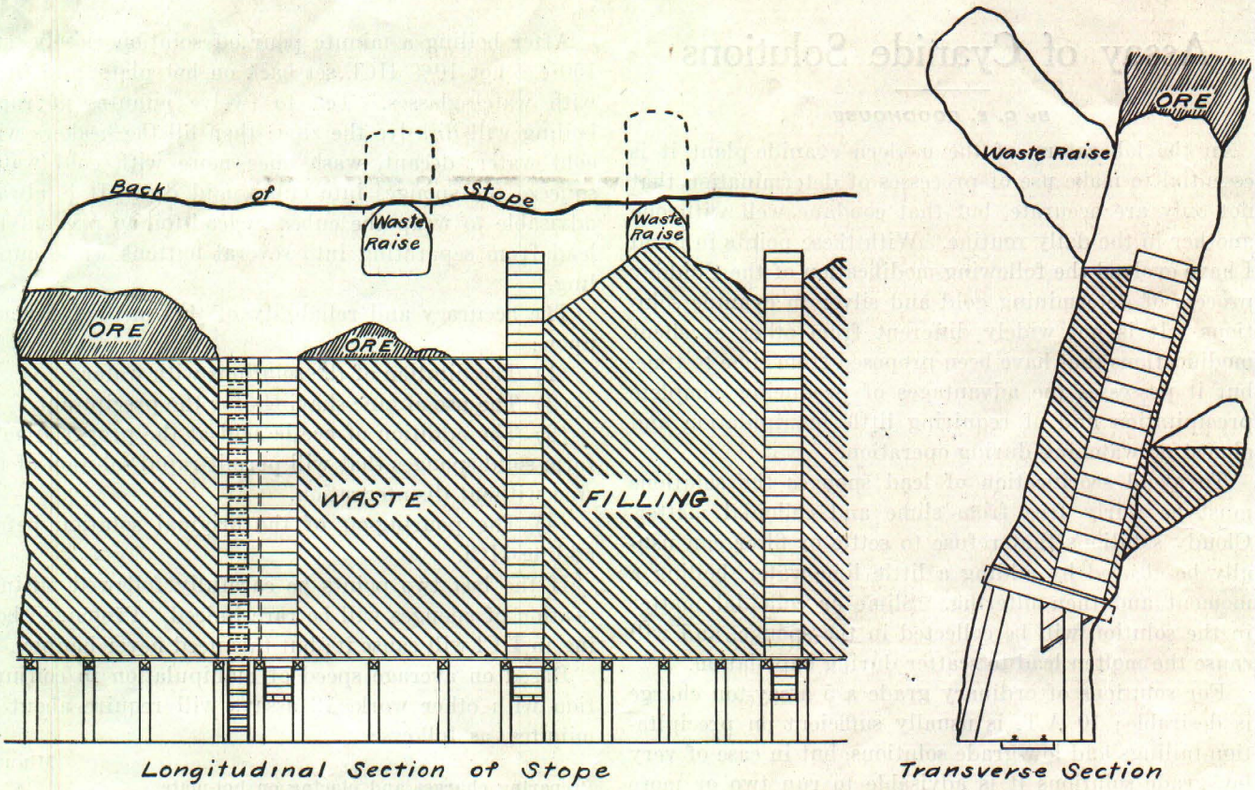

Back of Stope
Waste Raise
Waste Raise
ORE
ORE
WASTE
FILLING
Longitudinal Section of Stope
ORE
Waste Raise
Transverse Section

FIG. 5.

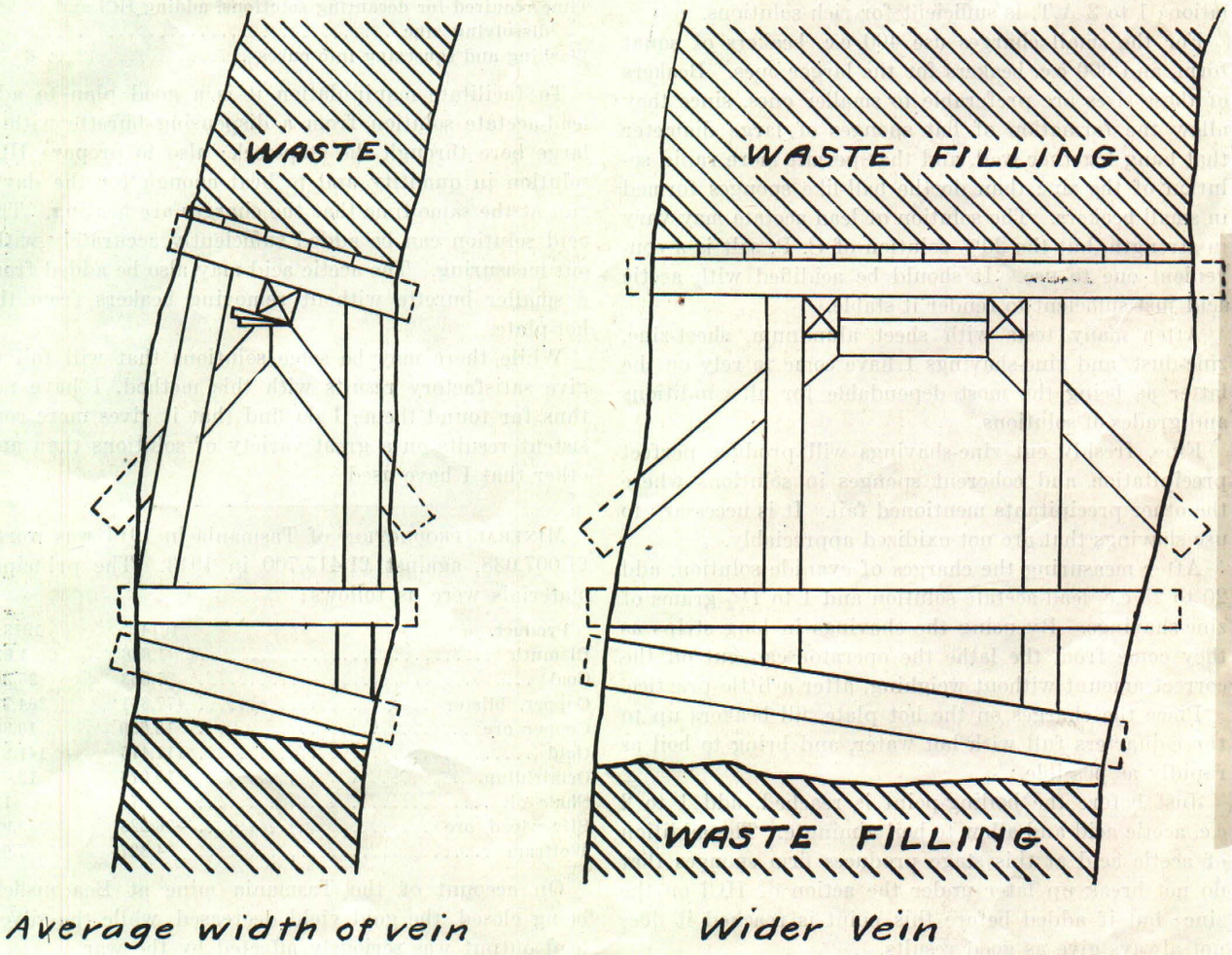

WASTE
Average width of vein
WASTE FILLING
WASTE FILLING
Wider Vein

FIG. 6.

Assay of Cyanide Solutions

By C. E. ROODHOUSE

In the laboratory of the modern cyanide-plant it is essential to make use of processes of determination that not only are accurate, but that combine well with one another in the daily routine. With these points in mind I have evolved the following modification of the Chiddey process of determining gold and silver in cyanide solutions. It is not widely different from other excellent modifications that have been proposed from time to time, but it possesses the advantages of producing complete precipitation and of requiring little manipulation and almost no watching during operation.

To avoid scorification of lead sponges the solutions must be fairly free from slime and colloidal matter. Cloudy solutions that refuse to settle or filter can usually be cleared by adding a little lime-water, boiling a moment and then filtering. Slime or colloidal matter in the solution will be collected in the sponge, and will cause the molten lead to scatter during cupellation.

For solutions of ordinary grade a 5 assay-ton charge is desirable; 10 A.T. is usually sufficient on precipitation-tailings and low-grade solutions, but in case of very low-grade solutions it is advisable to run two or more 10 A.T. charges, and to combine the buttons for cupellation; 1 to 2 A.T. is sufficient for rich solutions.

For the small charges use 400 c.c. beakers of squat form, and 600 c.c. beakers for the larger ones. Beakers of these sizes are preferable to smaller ones, since they allow the formation of flat sponges of large diameter that hang together well, and that permit more rapid solution of the zinc than do the ball-like sponges formed in small beakers. The solution of lead acetate may vary in strength, but the 20% solution of C. P. salt is a convenient one to use. It should be acidified with acetic acid just sufficient to render it stable.

After many tests with sheet aluminum, sheet-zinc, zinc-dust, and zinc-shavings I have come to rely on the latter as being the most dependable for all conditions and grades of solutions.

Fine, freshly cut zinc-shavings will produce perfect precipitation and coherent sponges in solutions where the other precipitants mentioned fail. It is necessary to use shavings that are not oxidized appreciably.

After measuring the charges of cyanide solution, add 20 to 25 c.c. lead-acetate solution and 1 to 1½ grams of zinc-shavings. By using the shavings in long strips as they come from the lathe the operator can cut off the correct amount without weighing, after a little practice.

Place the charges on the hot plate, fill beakers up to three-quarters full with hot water, and bring to boil as rapidly as possible.

Just before the boiling-point is reached, add 1 to 2 c.c. acetic acid and allow to boil a minute. The addition of acetic acid at this stage produces firm sponges that do not break up later under the action of HCl on the zinc; but if added before this point is reached it does not always give as good results.

After boiling a minute pour off solution closely, add 150 c.c. hot 10% HCl, set back on hot plate, and cover with watch-glasses. Ten to twelve minutes of rapid boiling will dissolve the zinc; then fill the beakers with cold water, decant, wash once more with cold water, squeeze the sponges into cubes, and dry. It is always advisable to wrap the cubes in lead-foil to prevent the lead from separating into several buttons while cupelling.

The accuracy and reliability of this method depends upon:

(1) The dilution of the solutions with water.

(2) The use of zinc-shavings for the precipitant.

(3) The addition of acetic acid at the proper time to form solid sponges that will permit rapid solution of the zinc without disintegration.

(4) The pouring-off of the original solution before adding HCl.

If the last two points be carefully followed, disintegration of sponges will be rare indeed. Prolonged boiling in HCl will cause it; but this need never happen.

Based on average speed of manipulation in conjunction with other work, 12 assays will require about 45 minutes, as follows:

	Minutes.
Preparing charges and placing on hot-plate	6
Time required to bring to boiling and to boil 2 min	15
Time required for decanting solutions, adding HCl and dissolving zinc	18
Washing and squeezing into cubes	6

To facilitate manipulation it is a good plan to add lead-acetate solution from a dispensing burette with a large bore through the stop-cock; also to prepare HCl solution in quantity and to heat enough for the day's run at the same time that the charges are heating. The acid solution can be added sufficiently accurately without measuring. The acetic acid may also be added from a smaller burette without removing beakers from the hot-plate.

While there may be some solutions that will fail to give satisfactory results with this method, I have not thus far found them; I do find that it gives more consistent results on a great variety of solutions than any other that I have used.

MINERAL PRODUCTION of Tasmania in 1914 was worth £1,007,038, against £1,415,700 in 1913. The principal materials were as follows:

Product.	1914.	1913.
Bismuth	£ 1,666	£ 1,627
Coal	27,853	25,367
Copper, blister	477,361	364,732
Copper ore	18,680	10,932
Gold	111,475	141,876
Osmiridium	10,076	12,016
Shale oil	75	130
Silver-lead ore	96,225	319,997
Wolfram	4,327	7,040

On account of the Tasmania mine at Beaconsfield being closed, the gold yield decreased, while the silver-lead output was seriously affected by the war.